# NI 43-101 Technical Report on the Pitarrilla Project

## Durango State, Mexico

**December 14, 2012**

**Prepared by:**

Kelly G. Boychuk P.Eng.

Dawn H. Garcia P.G., C.P.G.

Andrew W. Sharp, FAusIMM

Jeremy D. Vincent, P.Geo.

Trevor J. Yeomans, P.Eng., ACSM

Suite 1400, 999 West Hastings Street Vancouver, BC V6C 2W2 Tel: (604) 689-3846, Fax: (604) 689-3847.

## Date and Signature Page

The effective date of this Technical Report, titled "NI 43-101 Technical Report on the Pitarrilla Project, Durango State, Mexico", is December 14, 2012.

Signed,

"signed and sealed"

Kelly G. Boychuk, P.Eng.

Dated: December 14, 2012

Signed,

"signed and sealed"

Dawn H. Garcia , P.G., C.P.G.

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Andrew W. Sharp, FAusIMM

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Jeremy D. Vincent, P.Geo.

Dated: December 14, 2012

Signed,

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Trevor J. Yeomans, P.Eng., ACSM

Dated: December 14, 2012

# TABLE OF CONTENTS

# LIST OF FIGURES

# LIST OF TABLES

# DEFINITIONS

## Exchange Rate

Currency used in this report is in United States dollars. The Mexican peso (MXN) to United States dollar (USD) forecast exchange rate was determined based on publicly published forecasts by leading US and Canadian financial institutions. For time periods beyond the forecasts available, a purchase power parity assumption was utilised whereby the MXN devalued in nominal terms at the inflation rate differential between a Mexican long term inflation assumption of 3.8% and an US long term inflation assumption of 1.5%. In real terms, utilizing purchase power parity the MXN is constant. Long term inflation assumptions were determined based on publicly published forecasts by leading US and Canadian financial institutions.

## Abbreviations and Units of Measure

Metric units of measure are used throughout this report, except as noted here:

- Metal prices are converted from USD prices per pound (lb) or per ounce (oz) using the appropriate metric conversion for the production and sales quantities.
- Diameter sizes for some piping are stated in inches, according to ANSI national pipe schedules, because those are the sizes available at the project site.
- It should be noted that all tonnages are metric tonnes, where 1 tonne equals 1,000 kg, 2,205 lbs, or 1.102 imperial tons.
- Major elements and chemical compound abbreviations referred to in this study are shown below.

## Element and Chemical Compound Abbreviations

| | |
|---|---|
| Aluminum | Al |
| Antimony | Sb |
| Arsenic | As |
| Bismuth | Bi |
| Cadmium | Cd |
| Calcium | Ca |
| Carbon | C |
| Cobalt | Co |
| Copper | Cu |
| Fluorine | F |
| Gold | Au |
| Iron | Fe |
| Lead | Pb |
| Magnesium | Mg |
| Manganese | Mn |
| Mercury | Hg |
| Molybdenum | Mo |
| Nickel | Ni |
| Phosphorus | P |
| Potassium | K |
| Selenium | Se |
| Silver | Ag |
| Sodium | Na |
| Silica | $SiO_2$ |
| Sulphur | S |
| Thorium | Th |
| Uranium | U |
| Zinc | Zn |

Standard abbreviations and terms used throughout the study are shown below.

## Standard Abbreviations and Terms Used in the Report

| | |
|---|---|
| Abrasion Index | Ai |
| Acid Rock Drainage | ARD |
| Acidity | pH |
| Ampere | A |
| Annum (year) | a |
| Ball Mill Work Index | BWi |
| Billion | G |
| Brinell Hardness Number | BHN |
| Canadian Dam Association | CDA |
| Canadian Institute of Mining | CIM |
| Centimetre | cm |
| Centimetres per second | cm/s |
| Commercial Operation Declaration | COD |
| Comisíon Nacional del Agua | CONAGUA |
| Crushing Work Index | CWi |
| Counter Current Decantation | CCD |

| | |
|---|---|
| Cubic centimetre | $cm^3$ |
| Cubic metre | $m^3$ |
| Cubic metres per day | $m^3/d$ |
| Cubic metres per hour | $m^3/h$ |
| Day | d |
| Days per week | d/wk |
| Days per year (annum) | d/a |
| Decibel | dB |
| Degree | ° |
| Degrees Celsius | °C |
| Development Rock Stockpile | DRS |
| Dry Bulk Density | dbd |
| Dry metric tonne | dmt |
| Engineering, Procurement and Construction Management | EPCM |
| Equivalent (metal grades) | Eq |

| | |
|---|---|
| Formation | Fm |
| General & Administration | G&A |
| Giga (billion) | G |
| Gram | g |
| Grams per litre | g/L |
| Grams per tonne | g/t |
| Greater than | > |
| Hectare (10,000 $m^2$) | ha |
| Hertz | Hz |
| horizontal to vertical | H:V |
| Horsepower | hp |
| Hour | h |
| Hours per day | h/d |
| Inductively coupled plasma | ICP |
| Internal Rate of Return | IRR |
| Joule | J |
| Kelvin | K |
| Kilo (thousand) | k |
| Kilogram | kg |
| Kilograms per cubic metre | $kg/m^3$ |
| Kilograms per day | kg/d |
| Kilograms per tonne | kg/t |
| Kilojoule | kJ |
| Kilometre | km |
| Kilometres per hour | km/h |
| Kilonewton | kN |
| Kilopascal | kPa |
| Kilovolt | kV |
| Kilotonne per day | ktpd |
| Kilovolt ampere | kVA |
| Kilowatt | kW |
| Kilowatt hour | kWh |
| Kilowatt hours per metric tonne | kWh/mt |
| Kilowatt hours per short ton | kWh/st |
| Kilowatt hours per tonne | kWh/t |
| Kilowatt hours per year | kWh/a |
| Less than | < |
| Life of Mine | LOM |
| Litre | L |
| Litres per day | L/d |
| Litres per hour | L/h |
| Litres per minute | L/min |
| Litres per second | L/s |
| Load-Haul-Dump | LHD |
| Mega (million) | M |
| Megabyte | MB |
| Megabytes per second | MB/s |
| Megapascal | MPa |
| Megavolt ampere | MVA |
| Megawatt | MW |
| Megawatt hours | MWh |
| Metre | m |

| | |
|---|---|
| Metres above sea level | masl |
| Metres per day | m/d |
| Metres per minute | m/min |
| Metres per second | m/s |
| Micrograms per cubic metre | $\mu g/m^3$ |
| Micrometre (micron) | µm |
| Microsiemen (electrical) | S |
| Milliamperes | mA |
| Milligram | mg |
| Milligrams per litre | mg/L |
| Millilitre | mL |
| Millimetre | mm |
| Million cubic metres | $Mm^3$ |
| Million litres | ML |
| Million tonnes | Mt |
| Million Years Ago | Mya |
| Million | M |
| Minute (plane angle) | ' |
| Minute (time) | min |
| Month | mo |
| Movement Magnitude (of an earthquake) | Mw |
| National Instrument 43-101 | NI 43-101 |
| Net Present Value | NPV |
| Net Smelter Return | NSR |
| Neutralization Potential | NP |
| Newton | N |
| Newtons per square metre | $N/m^2$ |
| Operating hour | oph |
| Percent (80%) passing a mesh size | $P_{80}$ |
| Parts per billion | ppb |
| Parts per million | ppm |
| Pascal (newtons per square metre) | Pa |
| Peak Ground Acceleration (earthquake) | PGA |
| Percent | % |
| PITMET | Pitarrilla site meteorological station |
| Pound(s) | lb |
| Preliminary Economic Assessment | PEA |
| Quality Assurance / Quality Control | QA/QC |
| Qualified Persons | QPs |
| Rod Mill Work Index | RWi |
| Run of Mine | ROM |
| Second (plane angle) | " |
| Second (time) | s |
| Sierra Madre Occidental | SMO |
| Silver Standard Resources (Durango, Mexico) | SSD |
| Silver Standard Resources | SSR |
| Specific gravity | SG |
| Square centimetre | $cm^2$ |
| Square kilometre | $km^2$ |
| Square metre | $m^2$ |
| Square metres per day | $m^2/d$ |
| Tailings Storage Facility | TSF |

| | |
|---|---|
| Thousand tonnes | kt |
| Tonne (metric, 1,000 kg = 2205 lb) | t |
| Tonnes per cubic metre | $t/m^3$ |
| Tonnes per day | tpd |
| Tonnes per hour | t/h |
| Tonnes per year | t/a |
| Troy ounce (31.1035 g) | oz |
| Unspecified scale magnitude for earthquakes | M |
| Volt | V |
| Volume per volume | v/v |
| Watts per square metre | $W/m^2$ |
| Week | wk |
| Weight percent | wt% |
| Weight/weight | w/w |
| Wet metric tonne | wmt |
| Wet metric tonnes per year | wmt/a |
| Year (annum) | a |

# 1 SUMMARY

## 1.1 PROPERTY DESCRIPTION AND LOCATION

The Pitarrilla Project (or the "Property") is located within the Municipality of Inde, on the eastern flank of the Sierra Madre Occidental mountain range in the central part of Durango State, Mexico, and is centred at 25 degrees 25 minutes south latitude and 104 degrees 57 minutes west longitude.

## 1.2 OWNERSHIP

Silver Standard Resources Inc. ("Silver Standard" or "SSR") holds a 100% interest in the Property through its wholly-owned Mexican subsidiary Silver Standard Durango, S.A. de C.V. ("SSD").

## 1.3 MINERAL CONCESSION STATUS

The mine property is formed by 12 contiguous mineral concessions entitled to SSD and covers a total of approximately 136,191 hectares. SSD, as a wholly-owned subsidiary of SSR, is a legal entity that qualifies to obtain and hold such ownership in Mexico. The Property overlies approximately 1,395 ha of the overall mining concession.

## 1.4 SURFACE RIGHTS, LAND OWNERSHIP, AND RIGHTS OF WAY

SSD controls the majority of the surface rights required for mining, milling and all surface facilities at the project site. Some tracts of land are still required as of the publishing of this Technical Report. During the permitting process in Mexico, clear title or land access agreements must be presented to the regulatory authority in order to obtain mining or operating permits. In addition to the Pitarrilla Project site lands, rights-of-way are also required for the access road and power transmission line associated with the Pitarrilla Project, and access rights are currently being negotiated.

## 1.5 WATER RIGHTS

Water resources in Mexico are controlled by the National Water Commission (Comisión Nacional de Agua or CONAGUA). The Pitarrilla Project site is located in an arid to semi-arid climate that receives an average annual precipitation of 407 mm. The project will require, at its peak, approximately 150 L/s of fresh water.

In order to identify an independent reliable water source, local aquifers were located and tested by Ideas en Agua, a Mexican hydrogeological consultant, and sources of a significant portion of the required water have been identified. Additional water resource development for the full requirements of the project is in progress. The Ideas en Agua investigation (Ideas, 2010) concluded that local aquifers could provide the quantities of water required to operate the project. Ideas en Agua also provided guidance and advice related to the process of obtaining sub-surface water rights.

## 1.6 GEOLOGY AND MINERALISATION

The Property is located on the eastern flank of the Sierra Madre Occidental mountain range. This mountain range is the erosional remnant of one of the Earth's most voluminous accumulations of intermediate to felsic volcanic rocks, which formed a calc-alkaline magmatic arc that was built during Eocene to early Miocene time, roughly 52 to 25 million years ago, in response to subduction of the Farallón tectonic plate beneath North America, this mountain building event is known as the Laramide Orogeny. A large number of medium to high-level hydrothermal systems variably enriched in Ag, Au, Pb, and Zn were intermittently generated during this extended period of volcanism, including the epithermal mineral systems that formed the great Mexican silver mining districts at Guanajuato, Real de Angeles in Zacatecas, Fresnillo, and Santa Barbara-San Francisco del Oro. The silver-lead-zinc mineralisation found on the Pitarrilla property is situated in *Central Mexican Silver Belt*, a metallogenic province defined by the four previously-noted silver mining districts along with the mining districts of Parral, Santa Maria del Oro, and Sombrerete-Chalchihuites.

The Pitarrilla Project Ag-Zn-Pb deposit is hosted by deformed Cretaceous marine sediments and unconformably overlying Eocene (52 to 40 Ma) and Oligocene (32 to 28 Ma) volcanics volcaniclastics and intrusives. Eocene volcanics and volcaniclastics were derived from arc volcanism and from the erosion of subaerial arc volcanoes, and deposited into a back-arc basin. Uplift of the basin was accompanied by extension and voluminous bi-modal volcanism with the emplacement of andesitic and felsic sills and dykes during the early Oligocene. The culmination of the volcanism was the development of a rhyolitic dome which crops out on Cerro La Pitarrilla.

Ag-Zn-Pb mineralisation at the Pitarrilla Project occurs as a vertically stacked mineralised system centered on rhyolitic dykes and sills that constitute the feeder system for an early Oligocene volcanic center manifest by the rhyolitic dome. Sulphide-associated mineralisation is rooted in the basement Cretaceous sedimentary strata and is represented by an aerially restricted but vertically extensive zone of disseminated and veinlet Ag-Zn-Pb (-Cu-As-Sb) sulphide mineralisation and strata-bound massive replacement mineralisation within a polymictic conglomerate that occur at the Cretaceous-Eocene unconformity.

The sulphide mineralisation extends into the overlying Eocene and Oligocene volcaniclastic rocks and felsic sills, where it grades into mixed sulphide–oxide or transitional mineralisation and a more laterally extensive zone of disseminated iron oxide-associated mineralisation. The Ag-Zn-Pb mineralisation is interpreted to have occurred during or after emplacement of the early Oligocene rhyolitic dome.

## 1.7 EXPLORATION

Available records of mineral exploration conducted on the Property and immediately adjacent ground date back to 1996. In 2002, Silver Standard contracted F. Hillemeyer and P. Durning of La Cuesta International, Inc. ("LCI") to acquire mineral properties in Mexico which showed good exploration potential for silver. One of the areas LCI recommended for claiming was the ground covered by the Pitarrilla Project claim group. Between November 2002 and March 2003, a total of 12 concessions covering 136,191 hectares were claimed by Explominerals, S.A. de C.V. on behalf of Silver Standard.

Beginning in 2002, several programs of rock-chip sampling were completed over the core of the Property, where multiple zones of silver mineralisation eventually came to be outlined. The outlined zones represented exploration targets that were eventually drill-tested, resulting in the discovery of the five zones of oxide silver mineralisation that form the upper part of the Pitarrilla Project deposit.

A number of diamond and reverse circulation ("RC") drilling campaigns were undertaken by SSR on the Property between September 2003 and July of 2012. A total of 680 exploration related drillholes for a sum of 227,731 m were drilled. There are currently no plans for additional resource drilling on the Property, although infill drilling will be required as the project progresses.

## 1.8 MINERAL RESOURCES

Silver Standard has prepared an updated Mineral Resource estimate for the Property following completion of the recent (2012) drilling program. The Mineral Resource estimate is effective as of December 4, 2012 and is classified in accordance with CIM (2010) Definition Standards (Table 14-9 and Table 14-10). It forms the basis for the Mineral Reserve estimate as a part of the Pitarrilla Feasibility Study completed by SSR and M3 Engineering in 2012. This updated Mineral Resource estimate is based on all available drilling data since 2003 for the Pitarrilla Project, including the results from over 15,000 m of drilling completed at the Property during 2012.

**Table 1-1: Pitarrilla December 4, 2012 Global Mineral Resources**

| Classification | Cut-off Ag (g/t) | Tonnes (Mt) | Ag (g/t) | Pb (%) | Zn (%) | Ag (Moz) | Pb (Mlbs) | Zn (Mlbs) |
|---|---|---|---|---|---|---|---|---|
| Measured | 20.00 | 23.62 | 85.56 | - | - | 65 | - | - |
| | **30.00** | **20.31** | **95.42** | **-** | **-** | **62** | **-** | **-** |
| | 40.00 | 16.90 | 107.62 | - | - | 58 | - | - |
| Indicated | 20.00 | 268.73 | 75.89 | - | - | 656 | - | - |
| | **30.00** | **240.00** | **81.94** | **-** | **-** | **632** | **-** | **-** |
| | 40.00 | 199.61 | 91.41 | - | - | 587 | - | - |
| | 20.00 | 292.35 | - | 0.31 | 0.71 | - | 2,009 | 4,581 |
| | **30.00** | **260.31** | **-** | **0.32** | **0.72** | **-** | **1,815** | **4,146** |
| | 40.00 | 216.51 | - | 0.33 | 0.75 | - | 1,574 | 3,590 |
| Measured + Indicated | 20.00 | 292.35 | 76.67 | 0.31 | 0.71 | 721 | 2,009 | 4,581 |
| | **30.00** | **260.31** | **82.99** | **0.32** | **0.72** | **695** | **1,815** | **4,146** |
| | 40.00 | 216.51 | 92.68 | 0.33 | 0.75 | 645 | 1,574 | 3,590 |
| Inferred | 20.00 | 26.48 | 55.98 | 0.21 | 0.48 | 48 | 123 | 281 |
| | **30.00** | **22.08** | **62.12** | **0.21** | **0.49** | **44** | **101** | **236** |
| | 40.00 | 17.09 | 70.00 | 0.21 | 0.49 | 38 | 79 | 186 |

Notes:

1. Jeremy D. Vincent, B.Sc. (Hons), P.Geo., is the Qualified Person for the reported Mineral Resources estimate.
2. All Mineral Resource estimates have been classified in accordance with current Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definition standards.
3. Ag was estimated using Localised Uniform Conditioning (LUC). Pb and Zn were estimated using Ordinary Kriging (OK).
4. Mineral Resource estimates of Pb and Zn are not classified as Measured to account for the added uncertainty introduced by the volume-variance effect when using different estimation techniques (Ag by LUC; Pb and Zn by OK).
5. A silver cut-off grade of 30 g/t Ag is considered at this time to be the most likely economic cut-off grade for large-scale open-pit mining of the Pitarrilla deposit.
6. The reported Measured and Indicated Mineral Resources are regarded as sufficient for medium to long term production open pit planning and mine scheduling on a quarterly basis. Grade control drilling and a mine blending strategy to control grade variations are recommended for short-term mine planning.
7. Mineral Resources situated below the current open-pit shell design are considered potentially economically viable in an underground mining scenario, and are therefore included in the total reported Pitarrilla Mineral Resources. A Preliminary Economic Assessment (PEA) or higher level study validating the economics of the underground mining scenario has not been undertaken at this time.
8. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
9. The reported tonnes, grade, and metal content may not tally precisely due to rounding.

**Table 1-2: Pitarrilla December 4, 2012 Global Mineral Resources by Mineralisation Style**

| Material Type | Classification | Ag Cut-off (g/t) | Tonnes (Mt) | Ag (g/t) | Pb (%) | Zn (%) | Ag (Moz) | Pb (Mlbs) | Zn (Mlbs) |
|---|---|---|---|---|---|---|---|---|---|
| Oxide | Measured | 30.00 | - | - | - | - | - | - | - |
| | Indicated | 30.00 | 118.19 | 80.45 | 0.10 | 0.34 | 306 | 268 | 891 |
| | **Measured + Indicated** | **30.00** | **118.19** | **80.45** | **0.10** | **0.34** | **306** | **268** | **891** |
| | Inferred | 30.00 | 12.97 | 59.96 | 0.06 | 0.19 | 25 | 17 | 56 |
| Transitional | Measured | 30.00 | - | - | - | - | - | - | - |
| | Indicated | 30.00 | 57.57 | 74.13 | 0.28 | 0.60 | 137 | 351 | 763 |
| | **Measured + Indicated** | **30.00** | **57.57** | **74.13** | **0.28** | **0.60** | **137** | **351** | **763** |
| | Inferred | 30.00 | 4.92 | 67.28 | 0.15 | 0.60 | 11 | 16 | 65 |
| Sulphide | Measured | 30.00 | 20.31 | 95.42 | - | - | 62 | - | - |
| | Indicated | 30.00 | 64.24 | 91.68 | - | - | 189 | - | - |
| | Indicated | 30.00 | 84.55 | - | 0.64 | 1.34 | - | 1,196 | 2,492 |
| | **Measured + Indicated** | **30.00** | **84.55** | **92.58** | **0.64** | **1.34** | **252** | **1,196** | **2,492** |
| | Inferred | 30.00 | 4.19 | 62.73 | 0.73 | 1.25 | 8 | 67 | 116 |

**Notes:**

1. Jeremy D. Vincent, B.Sc. (Hons), P.Geo., is the Qualified Person for the reported Mineral Resources estimate.
2. All Mineral Resource estimates have been classified in accordance with current Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definition standards.
3. Ag was estimated using Localised Uniform Conditioning (LUC). Pb and Zn were estimated using Ordinary Kriging (OK).
4. Mineral Resource estimates of Pb and Zn are not classified as Measured to account for the added uncertainty introduced by the volume-variance effect when using different estimation techniques (Ag by LUC; Pb and Zn by OK).
5. A silver cut-off grade of 30 g/t Ag is considered at this time to be the most likely economic cut-off grade for large-scale open-pit mining of the Pitarrilla deposit.
6. The reported Measured and Indicated Mineral Resources are regarded as sufficient for medium to long term production open pit planning and mine scheduling on a quarterly basis. Grade control drilling and a mine blending strategy to control grade variations are recommended for short-term mine planning.
7. Mineral Resources situated below the current open-pit shell design are considered potentially economically viable in an underground mining scenario, and are therefore included in the total reported Pitarrilla Mineral Resources. A Preliminary Economic Assessment (PEA) or higher level study validating the economics of the underground mining scenario has not been undertaken at this time.
8. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
9. The reported tonnes, grade, and metal content may not tally precisely due to rounding.

## 1.9 MINERAL RESERVES

The Mineral Reserves estimate is classified in accordance with CIM (2010) Definition Standards and is presented in Table 1-3. Andrew W. Sharp, FAusIMM, Vice President, Technical Services for Silver Standard, is the Qualified Person responsible for the mining parameters and Mineral Reserves estimate. Trevor J. Yeomans, P.Eng., ACSM, is the Qualified Person who provided the metallurgical parameters incorporated in the Mineral Reserves estimate.

**Table 1-3: Pitarrilla Mineral Reserves Estimate (Effective as of December 4, 2012)**

| Category | Process Type | Tonnage | Mined Grade | | | Contained Metal | | |
|---|---|---|---|---|---|---|---|---|
| | | (Mt) | Ag (g/t) | Pb (%) | Zn (%) | Ag (Mozs) | Pb (Mlbs) | Zn (Mlbs) |
| Probable | Direct Leach | 43.4 | 91.5 | 0.17 | 0.42 | 127.5 | 161.6 | 403.9 |
| Probable | Flotation/Leach | 113.2 | 96.5 | 0.34 | 0.93 | 351.2 | 851.8 | 2,317.7 |
| Total Probable | All | 156.6 | 95.1 | 0.29 | 0.79 | 478.7 | 1,013.5 | 2,721.5 |

Notes to Mineral Reserves Table 1-3:

1. Mineral Reserves are contained within Measured and Indicated pit designs using metal prices for silver, lead and zinc of US$25/oz, US$0.90/lb, and US$0.95/lb, respectively.
2. The pit designs are generated from appropriate mining costs, processing costs, metal recoveries and inter ramp pit slope angles (varying from 36° to 48°)
3. The Mineral Reserve uses a net smelter return (NSR) calculation to determine the cut-off. The Mineral Reserve contains two ore types: direct leach ore and flotation/leach ore. The constant cut-off value for direct leach ore is $16.38 /tonne and for flotation/leach ore is $16.40/tonne.
4. Average process recovery within the total Mineral Reserves of silver, lead and zinc are 69.6%, 57.4% and 61.3% respectively.
5. No mining dilution is applied to the grade of the resource. Dilution intrinsic to the resource model is considered sufficient to represent the mining selectivity considered.
6. The life of mine strip ratio is 5.96.
7. Tonnage and grade measurements are in metric units. Contained silver ounces are reported as millions of troy ounces (Mozs). Contained lead and zinc are reported as millions of imperial pounds (Mlbs).
8. The reserve is 100% in-situ; no mining of the ore has occurred.
9. Table may not sum due to rounding.

Total Probable Mineral Reserves of silver have increased to 479 million ounces of silver at Pitarrilla, which is 5.2 times greater than the 91.7 million ounces originally reported in SSR's September 21, 2009 NI 43-101 Technical Report (Wardrop, 2009).

## 1.10 MINING OPERATIONS

The Pitarrilla Project will use standard truck and shovel open-pit mining methods. The expected mining life is 20 years, including three pre-production years. The pit will be mined in five phases, starting with Breccia Ridge and Cordon Colorado. Over the life of the Pitarrilla Project, a fleet of trucks is expected to haul approximately 1.1 billion tonnes of material and 157 million tonnes of ore, at a strip ratio of 5.96:1. After mining is completed, the plant will continue to mill ore from stockpiles for an additional 12 years.

The potential to mine Mineral Resources located below the open pit was not evaluated in the Pitarrilla Feasibility Study (M3, 2012), but may be evaluated later in the Pitarrilla Project's life.

## 1.11 MINERAL PROCESSING

The Pitarrilla Project's plant is expected to use standard crush, grind, float and agitated leach equipment to process 16,000 tonnes per day of flotation/leach ore or 12,000 tonnes per day of direct leach ore. The two ore types will utilize a common crushing and grinding circuit. Initially, highly-oxidised ore will be direct leached in an agitated leach circuit. Silver will then be extracted from the pregnant leach solution using the Merrill-Crowe process to produce silver-rich doré bars. Subsequently, the less oxidised and un-weathered sulphide ores will be processed in sequential lead and zinc flotation circuits to produce separate silver-bearing lead and zinc concentrates. Tailings from the flotation circuits will be processed in the agitated leach circuit to recover additional silver.

Over the Pitarrilla Project's 32 year life, the plant is expected to produce an estimated 604,000 tonnes of lead concentrate, with grades averaging 43% lead and 9,500 g/t silver, and an estimated 1.5 million tonnes of zinc concentrate, with grades averaging 46% zinc and 604 g/t silver. The leach circuit will produce 118.5 million ounces of silver in silver-rich doré bars. The plant is expected to produce an estimated 333 million ounces of silver. Some summary average production schedule values are presented in Table 1-4.

**Table 1-4: Average Production Schedule**

| Production Schedule | Units | Annual Average (years 1-9) | Annual Average (years 10-18) | Annual Average (life of project) |
|---|---|---|---|---|
| Total mined | Kt | 74,631 | 35,457 | 33,039 |
| Waste mined | Kt | 64,206 | 29,250 | 28,293 |
| Ore mined | Kt | 10,425 | 6,207 | 4,745 |
| Strip ratio | | 6.2 | 4.7 | 6.0 |
| Silver grade mined | g/t | 93 | 97 | 95 |
| Lead grade mined | % | 0.2 | 0.4 | 0.3 |
| Zinc grade mined | % | 0.5 | 1.4 | 0.8 |
| Lead concentrate | Kt | 17 | 42 | 18 |
| Zinc concentrate | Kt | 30 | 119 | 46 |
| Silver recovery | % | 66.7 | 83.1 | 69.6 |
| Lead recovery | % | 49.0 | 75.2 | 57.4 |
| Zinc recovery | % | 46.8 | 78.7 | 61.3 |
| **Silver produced** | **Koz** | **14,090** | **15,852** | **10,102** |
| - Doré | Koz | 5,659 | 2,587 | 3,591 |
| - Concentrate | Koz | 8,431 | 13,265 | 6,511 |
| Lead produced | lbs | 15,044 | 43,639 | 17,633 |
| Zinc produced | lbs | 30,355 | 136,947 | 50,572 |

## 1.12 ENVIRONMENTAL CONSIDERATIONS

The Pitarrilla Project has been designed to comply with Mexican mining regulations. An Environmental Impact Assessment ("EIA") is expected to be completed and ready for submission to Mexico's environmental agency in the first half of 2013. Studies conducted at the Pitarrilla Project during the EIA preparation process included characterization of the topography, geomorphology, geology, soils, water (surface water and groundwater), climate, air quality, and flora and fauna. Several environmental, land use, and operating permits and agreements are required before construction begins.

Silver Standard has implemented a community relations program that includes environmental, medical, educational, infrastructure development, and social support services. This year, SSR provided medical assistance to members of local communities, completed clean-up projects around local rivers, planted trees and completed construction projects to improve infrastructure (including the installation of livestock fences, improvements to a suspension bridge, improvements to a water well and the installation of a media room at a local high school).

## 1.13 TAILINGS STORAGE FACILITY

The Tailings Storage Facility ("TSF") is located in an area south of the process plant known as Boca de Alamo, which provides the most efficient site near the process plant with a storage-to-dam fill ratio of 14.3 (by volume). The TSF is contained by natural ridges with additional containment provided by four dams: the Main Dam and three Saddle Dams. Dams will be built predominantly with rockfill with zones of finer grained material on the upstream portions. The TSF will be constructed in five stages by downstream construction. The TSF can be expanded from 112 to 159 Mt of capacity with the addition of two extra dam raises. An ultimate dam crest elevation of 1,690 m would be reached for the 159 Mt capacity TSF.

Tailings and water containment will be constrained by a geosynthetic liner underlying the impoundment. The lining system selected consists of 1.5 mm thick, linear low-density polyethylene ("LLDPE") geomembrane liner underlain by fine-grained soil liner bedding fill. Underlying the geomembrane on the upstream dam slope will be a geosynthetic clay liner ("GCL") to provide a secondary containment redundancy through the dam(s). An underdrain system constructed underneath the liner will collect fluids in the event of seepage through the liner system.

Tailings will be delivered to the TSF, as thickened slurry, via a pipeline from the plant. Full perimeter tailings deposition will be utilised to maximize the TSF storage capacity and to maintain a supernatant water pool centred in the impoundment. Water will be reclaimed to the plant using a floating pump barge.

## 1.14 WATER BALANCE

A site-wide water balance analysis was performed to provide an estimate of freshwater make-up requirements for the mine operations and to estimate the expected fluctuations in the size of the reclaim water pond in the TSF. The water balance analysis was performed based on a 23-year mine life, which corresponds to processing 112 Mt of ore through the process plant. A revised mine plan, which increased the amount of potential mineral reserves for the Pitarrilla Project (and correspondingly increased processing to 157 Mt over a project life of 32 years), was subsequently envisioned for the project. The water balance presented in the Pitarrilla Feasibility Study (M3, 2012) accounts for the 23-year mine life only; however, based on the findings of this water balance, the latter years of processing show a reduction in the amount of water required as mining operations cease and the plant processes from stockpiles only. Consequently, the overall conclusion for peak water use will remain unchanged, between the 23- and 32-year cases.

Freshwater make-up is required seasonally for the majority of the facility life, under the base case climate conditions, and ranges from 0 L/s up to 133 L/s, averaging 67 L/s. Peak freshwater make-up requirements are 85 L/s during Stage 1, and 110 L/s during Stage 2 of the TSF. The corresponding peak total freshwater demand (process make-up plus the net demand from the rest of the site) is 115 L/s during Stage 1, 140 L/s during Stage 2, and 150 L/s for the life of the project (for a total of 7 stages of TSF construction). Later in the TSF life, the larger active beach and water pool areas, coupled with higher production rates, lead to increasing freshwater demand.

The TSF receives more upland runoff (due to increased liner area in the storage basin), as it is enlarged by successive dam raises, but suffers increased evaporation losses owing to the larger active beach and water pool area. The increase in evaporation losses exceeds the increase in precipitation runoff, so that more of the process water demand must be satisfied from freshwater supply, during the dry season. Ore moisture is a minor component of the water balance, contributing 1.9 to 7.5 L/s according to fluctuations in the rate of ore delivery from the mine; while mine dewatering flows are largely consumed for dust control. The net effect is increasing freshwater demand throughout the facility life.

## 1.15 INFRASTRUCTURE

The infrastructure facilities include the ancillary buildings, offices and support buildings, access roads into the plant site, source of electrical power and power distribution, fuel supply, storage and distribution, source of fresh water and water distribution, dewatering and drainage facilities, waste management, transportation and shipping, communications, and mobile equipment.

The primary access road to the project site will be 47 km long and will be developed and/or upgraded. Approximately 36.7 km of an existing, unpaved, narrow public road north from Highway 45 will be upgraded. A 150 m long concrete bridge will be constructed to cross the Nazas River. From the bridge, a new 9.7 km unpaved access road will be constructed to the entrance guard station of the project site.

The Pitarrilla Project will utilize an electrical interconnection to the national transmission grid to supply power to the mine site. SSR has requested that up to 40 megawatts (“MW”) of power be provided by Comisión Federal de Electricidad (“CFE”), Mexico’s national transmission utility, along a 115 kV transmission line. CFE has stated that it could provide power to the project in two stages: an initial 17 MW from its existing Nuevo Ideal substation and the final 40 MW upgrade, once the build-out of CFE’s Canatlán substation is completed. CFE is also investigating an overall upgrade to 230 kV from its Canatlán substation to the Pitarrilla Project site; however, information on this upgrade was not complete by the date of the Pitarrilla Feasibility Study (M3, 2012).

The freshwater make-up requirement for the Pitarrilla Project facilities is estimated to be approximately 115 to 150 L/s over the life of the mine, and the water will be sourced from a well field. The proposed well field comprises a minimum of three individual wells, each designed to pump 83 L/s of ground water to a booster station. From the booster station, water will be pumped to a fresh/fire water storage tank via an 11km long, 254 mm diameter water line. The above ground water supply pipeline route from the well field site will be primarily located along the main access road to the site.

### 1.16 ENVIRONMENTAL & REGULATORY AGENCY CONSIDERATIONS

Potential environmental impacts to surface soils, water, ecology, and air quality will be mitigated as part of the mining operations, which have been developed to comply with the Mexican environmental regulations. The studies conducted at the site included characterization of the topography, geomorphology, geology, soils, water (surface water and groundwater), climate, air quality, flora, and fauna.

### 1.17 MEXICAN LEGAL FRAMEWORK

Mine permitting in Mexico is administered by the federal government body Secretaría de Medio Ambiente y Recursos Naturales ("SEMARNAT"), the federal regulatory agency that establishes the minimum standards for environmental compliance. Guidance for the federal environmental requirements is derived from the Ley General del Equilibrio Ecológico y la Protección al Ambiente ("LGEEPA"). Article 28 of the LGEEPA specifies that SEMARNAT must issue prior approval to parties intending to develop a mine and mineral processing plant.

An EIA (by Mexican regulations called a Manifestación de Impacto Ambiental, or "MIA") is the document that must be filed with SEMARNAT for its evaluation and, if applicable, further approval by SEMARNAT through the issuance of an Environmental Impact Authorization, whereby approval conditions are specified where works or activities have the potential to cause ecological imbalance or have adverse effects on the environment. This is supported by Article 62 of the Reglamento de la Ley Minera. Article 5 of the LGEEPA authorizes SEMARNAT to provide the approvals for the works specified in Article 28. The LGEEPA also contains articles that speak directly of conservation of soils, water quality, flora and fauna, noise emissions, air quality, and hazardous waste management. The Ley de Aguas Nacionales provides authority to the Comisión Nacional del Agua ("CONAGUA" or "CNA"), an agency within SEMARNAT, to issue water extraction concessions, and specifies certain requirements to be met by applicants.

Another important piece of environmental legislation is the Ley General de Desarrollo Forestal Sustentable ("LGDFS"). Article 117 of the LGDFS indicates that authorizations must be granted by SEMARNAT for land use changes to industrial purposes. An application for change in land use or Cambio de Uso de Suelo, must be accompanied by a technical study that supports the environmental permit application (Estudio Técnico Justificativo). Mining projects also need to include a risk analysis of the environmental impacts (Análisis de Riesgo).

Guidance for the environmental legislation is provided in a series of Normas Oficiales Mexicanas. These regulations provide specific procedures, limits and guidelines and carry the force of law.

## 1.18 ENVIRONMENTAL PERMITS, LICENSES, AND AUTHORIZATIONS

There are three SEMARNAT permits required prior to construction: EIA, Change of Land Use, and Risk Analysis. A construction permit is required from the local municipality and an archaeological release letter is required from the National Institute of Anthropology and History ("INAH"). An explosives permit is required from the Ministry of Defense ("SEDENA") before construction begins. Water discharge and usage must be granted by CONAGUA. A project-specific environmental license (Licencia Única Ambiental), which states the operational requirements, is issued by SEMARNAT when the agency has approved the project operations. The key permits and the stages at which they are required are summarised in Table 1-5.

**Table 1-5: Permitting Requirements**

| Permit | Mining Stage | Agency |
|---|---|---|
| Environmental Impact Assessment | Construction/Operation/Post-operation | SEMARNAT |
| Land Use Change | Construction/Operation | SEMARNAT |
| Risk Analysis | Construction/Operation | SEMARNAT |
| Construction Permit | Construction | Municipality |
| Explosive & Storage Permits | Construction/Operation | SEDENA |
| Archaeological Release | Construction | INAH |
| Water Use Concession | Construction/Operation | CNA |
| Water Discharge Permit | Operation | CNA |
| Project-specific License | Operation | SEMARNAT |
| Accident Prevention Plan | Operation | SEMARNAT |

## 1.19 ENVIRONMENTAL IMPACTS AND MITIGATION MEASURES

A preliminary analysis indicates that the greatest potential for adverse environmental impacts of the Pitarrilla Project will be the impacts on the water resources. Water is a limiting factor for many environmental components in the region. It is unlikely that make-up water requirements will greatly affect regional water availability; however, contamination of ground water or surface water sources would be very detrimental. Primary mitigation against this impact is the proper engineering of the project, including spill containment mechanisms, run-off water diversions, and containment of any potentially acid-generating tailings or waste rock.

Some vegetation and animal species in the area are protected under Mexican regulations, and they will be disturbed during project construction. To mitigate this, flora and fauna "rescue" programs will be implemented to remove those protected species from the project footprint, for relocation.

Finally, impacts may occur to the socio-economy of the area, as the project becomes a source of jobs and attracts people from the region and beyond. Immigration into the area may positively influence the economy, but may also attract negative elements, such as drugs, prostitution and other criminal activity. The primary mitigation for these potential negative impacts will be the continued open communication with the communities and ejidos to provide for realistic expectations of available jobs and other community investments.

## 1.20 PROJECT EXECUTION PLAN

The project execution plan describes how the Pitarrilla Project will be carried out for construction. The overall project execution schedule is planned to take approximately 36 months, from project approval to declaration of commercial operation. The criteria for declaration of commercial production were set as at the end of the third consecutive month of mill tonnage throughput at or above 80% of nameplate.

Pre-production mining will initiate as soon as the construction permit is issued. Mining work that will be performed during the project construction time frame will include mobilization and assembly of a fast tracked fleet of five 100 tonne haul trucks, a 12 $m^3$ bucket capacity loader together with pioneering dozers and drill rigs. This will be followed by a constant stream of the arrival of 150 tonne trucks, 19 $m^3$ bucket capacity loaders, and then 21 $m^3$ bucket capacity shovels as clear mining locations are developed. Development of haul roads to access the first phase of the Breccia Ridge pit, the Cordon Colorado pit, a quarry for ARD mitigation rock, and the initial stripping and establishment of the west and south west waste dumps will constitute a large part of this expenditure.

The proposed project execution plan incorporates an integrated strategy for engineering, procurement and construction management (“EPCM”). The primary objective of the execution methodology is to deliver the project at the lowest possible capital cost, on schedule, and consistent with the project standards for quality, safety, and environmental compliance. SSR will secure the services of an EPCM contractor, through a tender offering. The EPCM contractor will be responsible for management and control of the various project activities. SSR will exercise ultimate control of the project by assigning a Project Director to work in close cooperation with the Project Manager assigned by the EPCM contractor. The Project Director will be assisted by the necessary experts in the main disciplines of engineering, construction, and project control.

## 1.21 CAPITAL AND OPERATING COST ESTIMATE

A capital cost estimate was developed to evaluate the economic feasibility of the Pitarrilla Project. All project costs prior to declaration of commercial production are considered as initial capital costs. Capital costs that occur after declaration of commercial production are considered as sustaining capital.

The capital cost estimate is based upon an open cut mine operation treating flotation/leach and direct leach ores. Crushing, grinding, flotation, leaching, and tailings storage are based on treatment and production of an average of 16,000 tonnes of ore per day for flotation/leach ore and 12,000 tonnes per day for direct leach ore. Thickened tailings will be pumped to a conventional TSF, located to the south of the process plant.

The capital cost estimate is based on second quarter 2012 US funds and it is considered to be within a ±15% level of accuracy. The estimate accuracy is a separate issue from contingency. Specifically, contingency is intended to account for costs that are expected to be incurred, but which cannot be quantified with the level of information available. The detailed, initial capital cost estimate is summarised in Table 1-6.

**Table 1-6: Summary of Initial Capital Costs (including contingency)**

| Capital costs | Millions |
|---|---|
| Mine development and mobile equipment | $155.7 |
| Plant | $308.3 |
| Infrastructure | $51.4 |
| Direct costs | $515.4 |
| Indirect costs | $80.9 |
| Owners costs | $37.3 |
| Total indirect costs | $118.2 |
| Contingency | $81.5 |
| Construction capital | $715.1 |
| Pre-operating mine, plant and G&A | $156.5 |
| Pre-operating net revenue | ($130.9) |
| Total pre-operating capital | $740.6 |

***Notes**: (1) A contingency of 5% has been applied to mine equipment and to light vehicles. (2) A contingency of 15% has been applied to all other expenditures during the first two years of the pre-production period, with the exception of capitalised operating mining cost which has the 15% contingency applied only to the first year of pre-production. (3) The capital cost estimates are based on second quarter 2012 pricing and will be subject to inflation that may occur prior to the construction decision and during the construction period.*

The base case financial indicators have been determined assuming 100% equity financing for the initial capital. Any acquisition cost or historical capital expenditures prior to project approval have been treated as “sunk” costs, and have not been included in the analysis.

The total initial capital carried in the financial model for new construction and pre-production mine development totals $740.6 million, expended over a three-year period. The initial capital includes all pre-production capital expenditures for design, procurement and construction of project facilities, including owner’s costs and contingency. Of this initial capital amount, $715.1 million is the total construction capital, which excludes the capitalised operating costs and initial revenue generated during the end of the construction period. The cash flow is shown being expended in the three years before production and a small amount is carried into the first production year.

The total sustaining capital costs over the life of the Pitarrilla Project mine are $403.9 million.

The operating and maintenance cost centres include mine operations, process plant operations, laboratory, and the general and administration area. Operating costs were determined on an annual basis for both flotation and leaching operations. Table 1-7 summarizes the operating costs for the Pitarrilla Project.

**Table 1-7: Life of Mine Operating Costs (Inclusive of Capitalised Operating)**

| | | | |
|---|---|---|---|
| Direct Leach Ore (kt) | 43,356 | | |
| Flotation/Leach Ore (kt) | 113,234 | | |
| Total Ore Mined (kt) | 156,590 | | |
| Waste Mined (kt) | 933,685 | | |
| Total Ore and Waste Mined (kt) | 1,090,275 | | |
| | **Life of Mine Cost (K$)** | **$/ore tonne mined** | **$/total material mined** |
| **Mine Operations** | | | |
| Drilling | 162,516 | 1.04 | 0.15 |
| Blasting | 269,534 | 1.72 | 0.25 |
| Loading | 188,143 | 1.20 | 0.17 |
| Haulage | 491,354 | 3.14 | 0.45 |
| Mine Support | 245,200 | 1.57 | 0.22 |
| General Mine | 214,755 | 1.37 | 0.20 |
| Total Mine Operations | **1,571,523** | **10.04** | **1.44** |
| **Mill Operations** | | | |
| Total Primary Crushing | 75,219 | 0.48 | |
| **Flotation/Leach Ore** | | **$/flotation/leach ore tonne milled** | |
| Grinding | 606,726 | 5.36 | |
| Flotation | 303,472 | 2.68 | |
| Concentrate Thickening, Filtration | 102,834 | 0.91 | |
| Tailings Leaching CCD | 479,853 | 4.24 | |
| Tailings Refinery | 67,076 | 0.59 | |
| Tailings | 32,285 | 0.29 | |
| Total Flotation/Leach Ore | 1,592,245 | 14.06 | |
| **Direct Leach Ore** | | **$/direct leach ore tonne milled** | |
| Grinding | 288,663 | 6.66 | |
| Leach/CCD | 210,678 | 4.86 | |
| Merrill Crowe/Refinery | 34,762 | 0.80 | |
| Tailings | 21,733 | 0.50 | |
| Total Direct Leach Ore | 555,835 | 12.82 | |
| Plant Administration* | 196,741 | 1.26 | |
| Total Mill* | **2,420,040** | **15.45** | |
| General Administration* | **282,108** | **1.80** | |
| **Grand Total*** | **4,273,671** | **27.29** | |

Note:* Unit cost expressed as dollar per combined ore tonne milled.
Any difference in summation is due to rounding.

## 1.22 MARKET CONSIDERATIONS

The Pitarrilla Project is a poly-metallic project containing three principal metals: silver, lead, and zinc. Production will be in the form of a silver doré and two separate concentrates: a lead and a zinc concentrate. Of the two concentrates, the lead concentrate will contain most of the recovered payable silver metal and will be the more valuable, comprising approximately 57% of the recovered payable silver. Trace amounts of minor or non-marketable elements will also be present in the two concentrates. The anticipated contribution of revenue, by metal, is approximately 82% silver, 13% zinc and 5% lead.

The Pitarrilla Project concentrates are commodities that will be sold and traded to global markets. Sales could either be made directly to smelter operations or through commodity traders. Transportation costs will vary widely depending on whether the concentrates are sold to local or overseas smelters. It was assumed that the silver doré will be sold to local smelters, and the lead and zinc concentrate will be sold on the global market to offshore smelters. Alternatives exist to sell to local smelters.

The recommended silver refining costs, for use in the Pitarrilla Feasibility Study (M3, 2012), were derived through data collected from a survey of NI 43-101 Technical Reports of precious metals projects being developed in North and South America. Most of the NI 43-101 Technical Reports surveyed were published from 2007 to 2012.

The Pitarrilla Project is projected to produce a high grade silver doré, which is readily marketable to global precious metals refiners. A comparison to applicable producers in Mexico was completed.

## 1.23 ECONOMIC ANALYSIS

The results of the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements in this section include, but are not limited to, statements with respect to the future price of silver, and base metals, the estimation of Mineral Reserves and Mineral Resources, the realization of Mineral Reserve estimates, the timing and amount of estimated future production, costs of production, capital expenditures, results of the permitting process, currency exchange rate fluctuations, requirements for additional capital, government regulation of mining operations, environmental risks, unanticipated reclamation expenses, title disputes or claims and limitations on insurance coverage.

Additional risk can come from actual results of changes in project parameters as plans continue to be refined, possible variations in ore reserves, grade or recovery rates; failure of plant, equipment or processes to operate as anticipated; accidents, labour disputes and other risks of the mining industry; and potentially delays in obtaining additional governmental approvals.

The Pitarrilla Project economic model evaluates a project which will be in development and construction for a period of three years, to the point at which commercial production will be achieved. Some production will occur during the third year of construction, as the start-up of the leaching circuit commences prior to completion of the flotation circuit. By the time the flotation circuit construction is complete, the leach circuit will be at full capacity and the levels of production are forecast to be at commercial, cash-positive levels.

The production cycle of the project can be divided into four logical phases over its 32 year life.

- A three year pre-production capital construction phase.
- The first production phase lasting approximately nine years, during which the mine will extract a mixture of direct leach and flotation leach ores; both of which will be processed. This is the phase during which the project is forecast to achieve payback.
- The second production phase, also lasting approximately nine years, in which the pit deepens to the point where higher grade ore from the Basal Conglomerate begins to represent a higher percentage of total plant feed. Lead and zinc concentrate grades increase, and the average annual metal production of the mine markedly increases, while average cash cost for metal production drops significantly.
- The third production phase of the project in which the mining of the open pit is completed, and the remaining production life is from the processing of stockpiled ores. The plant will process both ore types of a gradually decreasing grade, which maximizes the project's NPV.

### 1.24 REVENUE

Annual revenue is determined by applying estimated metal prices to the estimated annual payable metal for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The revenue is the gross value of payable metals sold before treatment factors and transportation charges. Sensitivity analyses and project economics were estimated for a wide range of paramenters including metal prices, grade, recovery, various costs and other assumptions. The Project was determined to be most sensitive to metal price.

Table 1-8 provides a tabular summary of the base case and three other metal price scenarios that were utilised in the economic analysis. All four metal price scenarios are based on the consensus forecasts. The downside and upside metal price assumptions were determined by taking the lower and upper ranges of the consensus forecasts excluding certain outliers. The spot price case is equal to the closing market price for each metal as of November 23, 2012.

**Table 1-8: Metal Price Scenarios for Sensitivity Evaluation**

| | | Downside Case | Base Case* | Upside Case | Spot Price Case |
|---|---|---|---|---|---|
| Silver Price | $/oz | $22.50 | $25.00 | $30.00 | $34.13 |
| Lead Price | $/lb | $0.80 | $0.90 | $1.10 | $0.99 |
| Zinc Price | $/lb | $0.85 | $0.95 | $1.10 | $0.87 |

* The base case silver price is assumed to $30 per oz in Yr -3 to Yr -2, $27.50 per oz in Yr-1 to Yr 2 and $25 per oz thereafter.

## 1.25 TOTAL CASH COSTS

The average Total Cash Costs over the life of the mine is estimated to be $10.01 per ounce of silver processed. Total Cash Costs includes mine operations, process plant operations, general and administration cost, smelting and refining charges, as well as transportation costs. The estimated operating cost by area per ounce of silver processed is shown in Table 1-9.

**Table 1-9: Pitarrilla Life of Mine Total Cash Operating Cost**

| Operating Cost Statistics* | | Years 1-9: Yearly Average | Years 10-18: Yearly Average | Years 19-30: Yearly Average | LOM Production |
|---|---|---|---|---|---|
| Direct Mining Cost | $/oz Payable | $3.91 | $2.77 | $1.24 | $2.93 |
| Direct Process Plant | $/oz Payable | $6.29 | $6.32 | $14.51 | $7.82 |
| G&A | $/oz Payable | $0.67 | $0.60 | $1.60 | $0.81 |
| **Cash Operating Costs** | **$/oz Payable** | **$10.87** | **$9.69** | **$17.35** | **$11.56** |
| Shipping and Selling | $/oz Payable | $0.51 | $1.60 | $0.63 | $1.00 |
| Treatment and Refining | $/oz Payable | $1.91 | $4.26 | $1.89 | $2.91 |
| By-Product Credits | $/oz Payable | ($2.51) | ($9.32) | ($2.75) | ($5.46) |
| **Cash Offsite Costs** | **$/oz Payable** | **($0.09)** | **($3.45)** | **($0.23)** | **($1.55)** |
| **Total Cash Costs** | **$/oz Payable** | **$10.78** | **$6.24** | **$17.12** | **$10.01** |
| Non-Cash Costs** | $/oz Payable | $4.81 | $8.37 | $2.50 | $5.90 |
| **Total Production Costs** | **$/oz Payable** | **$15.59** | **$14.61** | **$19.62** | **$15.91** |

Notes:
*Includes ARO and employee profit sharing.
Exchange rate assumed to equal 12.86 $MXN per $USD (Year -3), 12.58 $MXN per $USD (Year -2) and 12.5 $MXN per $US thereafter.
**Non-Cash Costs includes period depreciation and amortization of physical plant and equipment, asset retirement obligation assets, and capitalised mine development and pre-operating costs (i.e. this is based upon amounts added to inventory in each period and not the from the income statement). It should be noted that the depreciation is compliant with IFRIC 20 relating to deferred stripping.

## 1.26 CORPORATE INCOME TAX

The Mexican income tax liability for the Pitarrilla Project has been calculated based upon the current income tax laws enacted in Mexico. The Pitarrilla Project is subject to a corporate income tax rate of 29% in 2012 and 28% in all subsequent years. Total current tax liability over the life of the mine, as shown in the cash flow model in Table 22-9 equals $871.3 million.

## 1.27 NET PRESENT VALUE, INTERNAL RATE OF RETURN, AND PAYBACK

The economic analyses for the project are summarised in Table 1-10 for the low, base, high and spot metal price cases.

**Table 1-10: Financial Analysis Results**

| Performance Metrics | | Downside Case | Base Case | Upside Case | Spot Price Case |
|---|---|---|---|---|---|
| Silver Price* | $/oz | $22.50 | **$25.00** | $30.00 | $34.13 |
| Average Silver Price | $/oz | $22.50 | **$25.53** | $30.00 | $34.13 |
| Lead Price | $/lb | $0.80 | **$0.90** | $1.10 | $0.99 |
| Zinc Price | $/lb | $0.85 | **$0.95** | $1.10 | $0.87 |
| Diesel Price | $/litre | $0.70 | **$0.80** | $0.95 | $0.85 |
| $MXN per $USD** | | 12.50 | **12.50** | 12.50 | 12.96 |
| Pre-tax NPV 5% | $M | $680 | **$1,176** | $1,972 | $2,552 |
| **After-tax NPV 5%** | **$M** | **$368** | **$737** | **$1,316** | **$1,741** |
| Pre-tax IRR | % | 11.5% | **15.8%** | 20.8% | 25.3% |
| **After-tax IRR** | **%** | **9.1%** | **12.8%** | **17.2%** | **21.2%** |
| Undiscounted Cash Flow | $M | $1,328 | **$2,015** | $3,187 | $3,948 |
| Payback After COD | Years | 10.4 | **7.4** | 4.8 | 3.8 |
| Cash Cost/Payable oz of Silver | $/oz Payable | $10.23 | **$10.01** | $9.65 | $10.47 |
| Production Cost/Payable oz of Silver | $/oz Payable | $16.10 | **$15.91** | $15.54 | $16.28 |

Notes:
* Base case silver price is $30 per oz (Year -3 to Year -2), $27.50 (Year -1 to Year 2) and $25 thereafter. The cases assume the stated price in all years (these prices impact the capital cost since revenues during the construction period are netted off against the capital cost).
** All cases but the Spot Price are assumed to equal 12.86 $MXN per $USD (Year -3), 12.58 $MXN per $USD (Year -2) and 12.5 $MXN per $US thereafter. The Spot Price Case is assumed to equal 12.96 $MXN in all periods.

## 1.28 INTERPRETATIONS AND CONCLUSIONS AND RECOMMENDATIONS

The conversion of Mineral Resources to Mineral Reserve was made using industry-recognised methods of determining operational costs, capital costs, and plant performance. Thus, it is considered to be representative of actual and future operational conditions. This report has been prepared with the latest information regarding environmental and closure cost requirements and has indicated that future work is in progress. All work has been completed to support the Pitarrilla Feasibility Study (M3, 2012).

Silver Standard is not aware of any significant risk or uncertainty that may materially affect the reliability or confidence in the Mineral Resources/Mineral Reserve estimates or projected economic outcomes.

Silver Standard is in the process of obtaining the remaining surface rights and rights-of-way required for the Project, and expects to complete this in 2013. All required surface rights are necessary prior to submitting the construction permit application.

The recommended development plan for Pitarrilla includes a large open cut mine with a flexible flotation and leach plant that would be capable of efficiently processing a significant portion of the resource. The mine would operate for 20 years, moving an average of 175,000 tonnes per day and mining a total of 157 million tonnes of ore over its life. The process plant would operate for 30 years including commissioning and start-up and would have primary crushing, SAG and Ball milling circuits with the capacity to process 12,000 tonnes per day of direct leach ore and 16,000 tonnes per day of flotation/leach ores. The direct leach ore would be treated in an agitated leach circuit, CCD thickening and Merrill Crowe refinery to produce silver doré. The flotation/leach ores would be processed in a two stage lead and zinc flotation circuit, with the tailings treated by agitated leaching for incremental recovery of silver. This circuit would produce lead and zinc concentrates along with incremental doré from the tailings leach. In the final 12 years of the operation, the plant would continue to process ore from stockpiles.

This mine would be one of the largest silver mines in Mexico, with production of 333 million ounces of silver, 582 million pounds of lead and 1,669 million pounds of zinc over a 32 year project life. Production of silver in the first 18 years will range from 5 to 26 million ounces per year, averaging 15 million ounces of silver per year.

### 1.29 RECOMMENDATIONS

It is recommended that the EIA document be completed and submitted to SEMARNAT for environmental review. Formal application for a construction permit from SEMARNAT cannot be made until all land impacted by the Project is under SSR control. The environmental review of the EIA document will expedite future consideration of the construction permit, when submitted, and is expected to reduce permitting risk. No significant costs remain on the application process.

Whilst the EIA process is underway, SSR will continue its community relations work, continue to investigate project financing alternatives, and advance a number of programs to reduce risk and advance critical infrastructure. The cost of these optimization programs is estimated to be approximately $1 million in total.

Following a positive construction decision, the Project would advance to detailed engineering and full project implementation as defined in the Pitarrilla Feasibility Study (M3, 2012), inclusive of refinements due to the latter risk reduction and opportunity programs. The recommendations in this Technical Report do not go beyond the construction decision.

# 2 INTRODUCTION

This Technical Report was prepared in order to fulfill Silver Standard's obligation to file a Technical Report in accordance with Section 4.2(1)(j)(ii) of Canadian National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101"). The obligation to file the Technical Report was triggered by Silver Standard's press release dated December 4, 2012 announcing the completion of a Feasibility Study on the Pitarrilla Project (the "Property"), Silver Standard's 100% owned property in the central part of Durango State, Mexico. This report has been prepared by Qualified Persons employed by Silver Standard. Silver Standard fulfills the requirements of a producing issuer as defined in NI 43-101.

Silver Standard is a Canadian-based mining, development, and exploration company, with a pipeline of projects ranging from grassroots exploration in Peru, Mexico, Canada, Chile, and the United States to production in Argentina. Silver Standard's shares are listed on the Toronto Stock Exchange under the symbol SSO and on the NASDAQ Global Market under the symbol SSRI.

The principal metals of interest at the Pitarrilla Project are silver, lead, and zinc. On December 4, 2012, Silver Standard prepared a Mineral Resources and Mineral Reserves estimate for the Pitarrilla Project following the guidelines set forth by the CIM (2010). This Technical Report is written in accordance with NI 43-101 and is suitable for filing with Canadian Securities Commissions.

## 2.1 SOURCES OF INFORMATION

This Technical Report references principal reports and the work of the following consultants and laboratories:

- ALS Metallurgy Kamloops (2012a and b)
- Clifton Associates Ltd (2012)
- Frontier Geosciences Inc. (2012)
- G&T Metallurgical Services Ltd (2007a, 2007b, 2008, 2009, 2011, 2012a and b)
- Hatch (2005, 2012a and b)
- Ideas en Agua (2010)
- Kappas Cassiday and Associates (2011)
- Knight Piésold Consulting (2008, 2010a, 2010b, 2012a, b, c, d, e and f)
- M3 Engineering & Technology Corporation (2012)
- McClelland Laboratories Inc. (2012a and b)
- McCrea (2004, 2006a and b, and 2007)
- P&E (2008)
- Boutilier (2010a and b)

- Process Research Associates Ltd. (2005, 2006, 2007, and 2008)
- SGS Canada Inc. (2009, 2011, and 2012a and b)
- SGS Mineral Services Durango (2011, 2011a, b, and c, 2012a, b, c, d, e, f, g, and h)
- Somers et al (2010)
- SRK (2012, 2012a and b)
- Tierra Group International (2012a and b)
- Wardrop (2009)
- Xstract (2012a and b)

In addition, other reports, opinions and statements of lawyers and other experts are discussed in Section 3.

The sample information used to develop the Mineral Resources and Mineral Reserves estimates and metallurgical testwork was collected over a number of years dating back to 2003 and includes only sample information acquired by Silver Standard personnel.

## 2.2 SITE VISITS AND SCOPE OF PERSONAL INSPECTION

The Mineral Resources estimate was prepared by Jeremy D. Vincent, P.Geo., and the Mineral Reserves estimate was prepared by Andrew W. Sharp FAusIMM. Mr. Vincent visited the Property on October 5 and 6, 2012 for a total of two days on site. Mr. Sharp visited the Property on September 9 and 10, 2011 for a total of two days on site. Sections pertaining to metallurgical processing and testwork, and recovery methods were prepared by Mr. Trevor J. Yeomans, P.Eng. Mr. Yeomans visited the Property most recently from October 28 to 31, 2012 for a total of four days on site. Sections pertaining to project infrastructure were prepared by Mr. Kelly G. Boychuk, P.Eng. Mr. Boychuk visited the Property on June 17 and 18, 2012 for a total of two days on site. Sections pertaining to environmental studies, permitting and social or community impacts were prepared by Ms. Dawn H. Garcia, P.G., C.P.G. of SRK. Ms. Garcia visited the Property on January 13 to 15, 2012 for a total of three days on site.

The authors and the sections for which they are responsible are summarised in Table 2-1.

**Table 2-2-1: Summary of Qualified Person Responsibilities**

| Technical Report Section | Qualified Person Responsible |
|---|---|
| 1: Summary<br>2: Introduction<br>3: Reliance on Other Experts<br>4: Property Description and Location | All QPs |
| 5: Physiography, Climate, Access, Local Resources, and Infrastructure<br>6: History<br>7: Geological Setting and Mineralisation<br>8: Deposit Types<br>9: Exploration<br>10: Drilling<br>11: Sample Preparation, Analysis and Security<br>12: Data Verification | Jeremy D. Vincent |
| 13: Mineral Processing and Metallurgical Testwork | Trevor J. Yeomans |
| 14: Mineral Resources Estimate | Jeremy D. Vincent |
| 15: Mineral Reserves Estimate<br>16: Mining Methods | Andrew W. Sharp |
| 17: Recovery Methods | Trevor J. Yeomans |
| 18: Project Infrastructure | Kelly G. Boychuk |
| 19: Markets and Contracts | Andrew W. Sharp |
| 20: Environmental Studies, Permitting, and Social or Community Impact | Dawn H. Garcia |
| 21: Capital and Operating Costs<br>22: Economic Analysis | Andrew W. Sharp |
| 23: Adjacent Properties | Jeremy D. Vincent |
| 24: Other Relevant Data and Information<br>25: Interpretations and Conclusions<br>26: Recommendations<br>27: References | All QPs |

# 3 RELIANCE ON OTHER EXPERTS

In preparing this Technical Report, Silver Standard has partly relied upon the opinions and reports of consultants as well as certain reports, opinions and statements of lawyers and other experts. These reports, opinions and statements, the makers of each such report, opinion or statement, and the extent of reliance are described below. Silver Standard considers the reliance on other experts, as described in this section, as being reasonable based on their knowledge, experience and qualifications.

## 3.1 LEGAL

For matters related to title to the property, Silver Standard has relied wholly on the opinion of Creel, Garcia-Cuellar, Aiza y Enriquez S.C., a Mexican law firm retained by Silver Standard. The firm's legal opinion, dated November 9, 2012, is discussed in Section 4.3.

## 3.2 ENVIRONMENT

Silver Standard has relied partly on the opinion of Clifton Associates Ltd (Clifton, 2012) for collection, collation, and interpretation of baseline environmental data. Clifton's advancement of EIA studies and permitting activities included the collection of baseline data on:

- Climatic conditions;
- Surveys of noise, existing mining works, soil, flora and fauna;
- Groundwater; and
- Surfacewater.

Clifton's work is reported in sub-sections 20.6.4, 20.6.5 20.6.6, 20.6.7, and 20.6.8, and is taken from the report, "Clifton Associates, 2012, Linea Base Ambiental, Proyecto Minero Pitarrilla", prepared for SSD, dated May, 2012 148p.

## 3.3 POLITICAL AND TAXATION

Silver Standard has not relied on external opinion for political or taxation matters and/or reports for this Technical Report.

# 4 PROPERTY DESCRIPTION AND LOCATION

## 4.1 PROPERTY LOCATION

The Property, is located within the Municipality of Inde, on the eastern flank of the Sierra Madre Occidental mountain range in the central part of Durango State, Mexico, and is centered at 25 degrees 25 minutes south latitude and 104 degrees 57 minutes west longitude. The city of Victoria de Durango, the capital of Durango state, is located 160 km southwest of the property and the major city of Torreón (capital of Coahuila state) 160 km to the east please refer to Figure 4-1.

The nearest population centers are San Francisco de Asís (located 12 km to the northeast of the property) and Casas Blancas (situated in the northeast portion of the project concessions). Both villages are located in Durango State. San Francisco de Asís has a population of about 800 and Casas Blancas has a population of approximately 120. The larger population centers near the project of Torreón and Victoria de Durango have approximately 1 million and 1.5 million inhabitants, respectively.

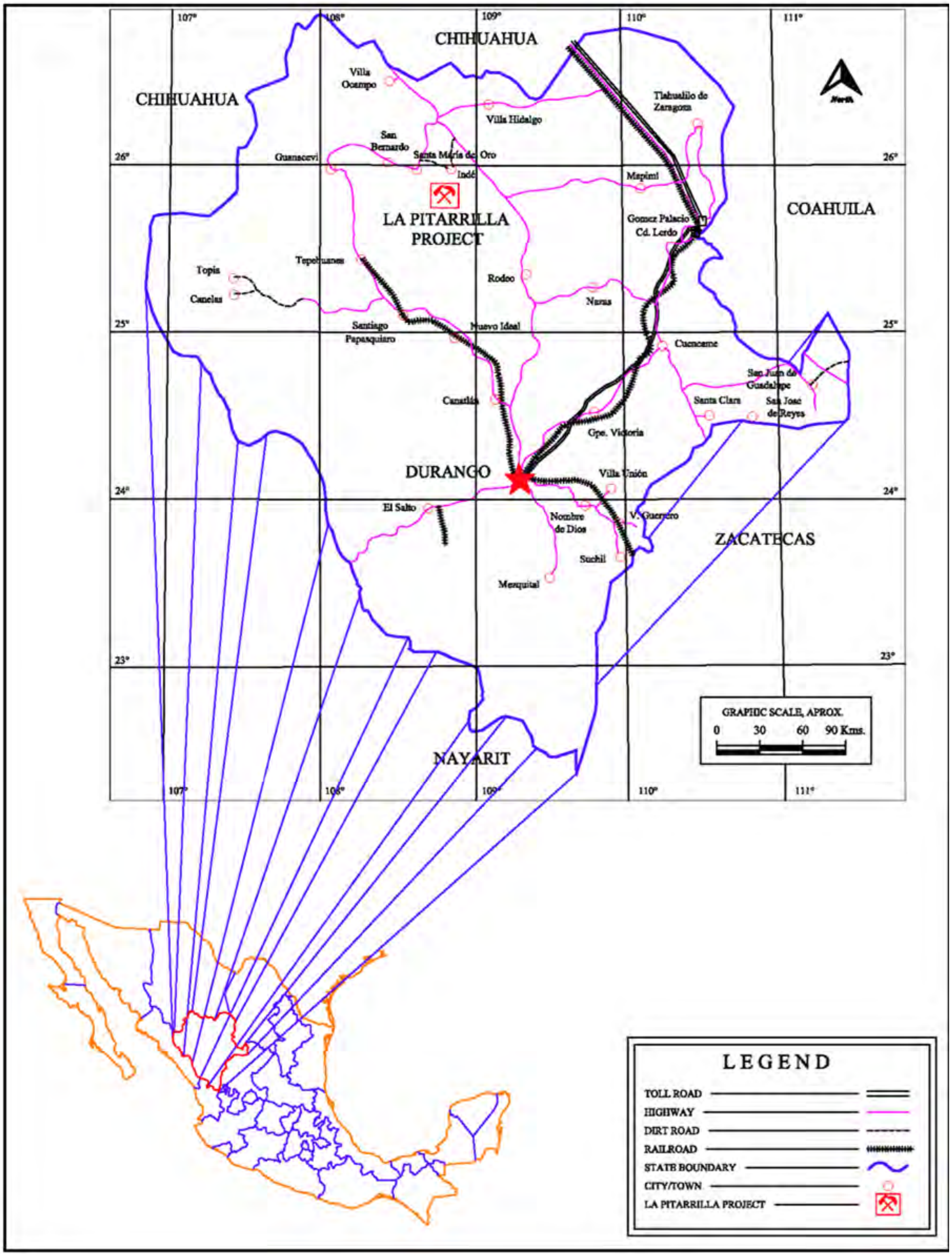



**Figure 4-1: Pitarrilla Property Location Map**

Source: M3, 2012

## 4.2 LAND TENURE

The Property is defined as the group of mining concessions and the surface rights that partially overlie the mining concessions. The mining concessions are displayed in Figure 4-2 and presented in Table 4-1, and the surface rights that partially overlie the concessions are described in Sections 4.2.2, 4.2.3 and 4.2.4 and are displayed in Figure 4-3 and Figure 4-4, and the ownership or rights thereto is presented in Table 4-2 and Table 4-3.

### 4.2.1 Mining Concessions

The property is formed by 12 contiguous mineral concessions entitled to SSD and covering a total area of approximately 136,191 hectares. SSD is a Mexican corporate entity, and a wholly-owned subsidiary of Silver Standard.

The complete set of mining concessions is shown in Figure 4-2 and the legal status of each, including expiration dates, is summarised in Table 4-1.

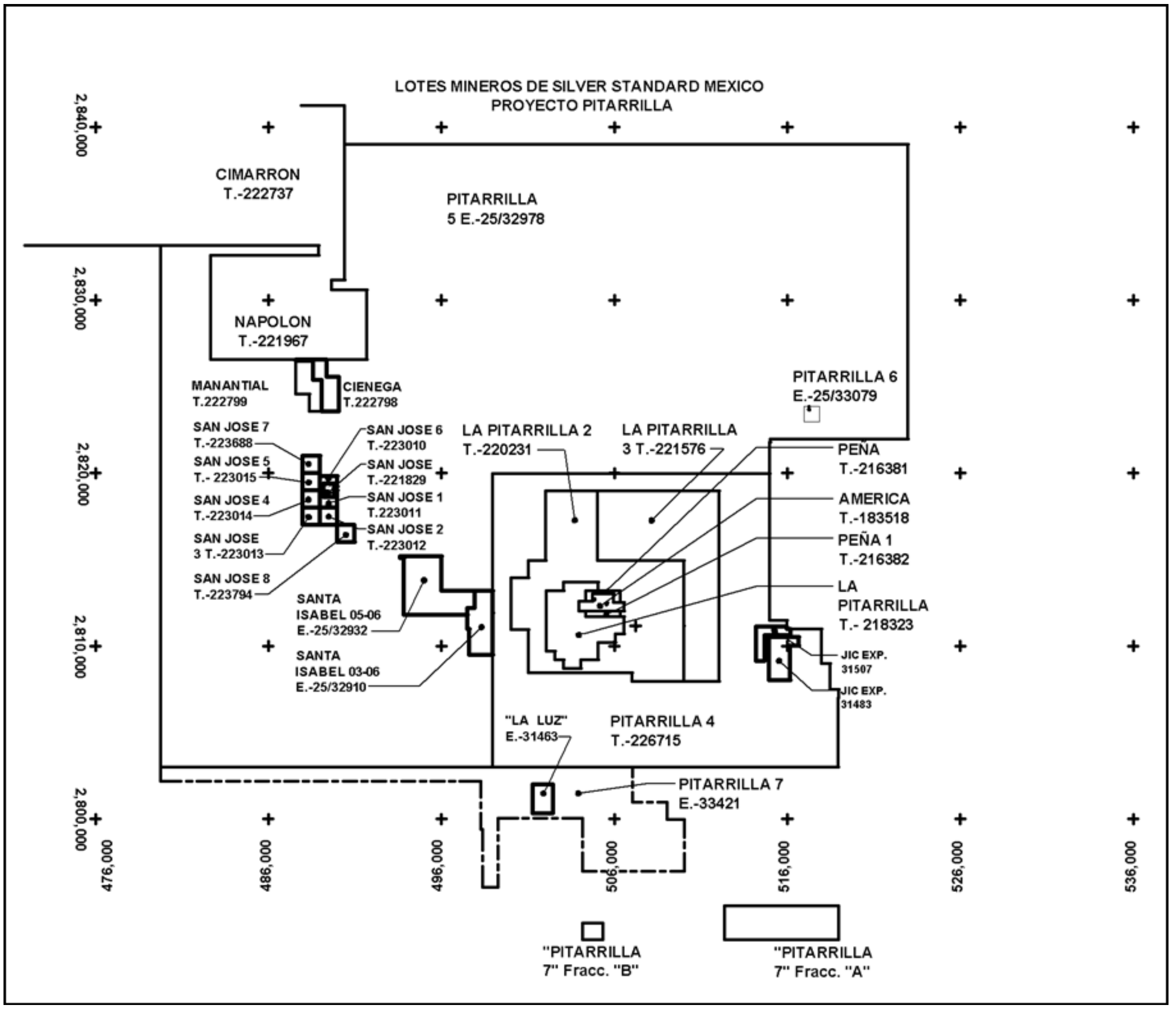



**Figure 4-2: Pitarrilla Mining Concessions**
NAD27 UTM Zone 13N Source: M3 2012

## Table 4-1: Legal Status of Pitarrilla Mineral Concessions

| No. | Claim name | File No. | Title | Area (ha) | Agency | Municipality & State | Validity | |
|---|---|---|---|---|---|---|---|---|
| | | | | | | | From | To |
| 1 | AMERICA | 321.1/1-111 | 183518 | 198 | DURANGO | EL ORO, DURANGO | 26/10/1988 | 25/10/2038 |
| Pitarrilla Group | | | | | | | | |
| 2 | LA PITARRILLA | 25/30749 | 218323 | 1,395 | DURANGO | EL ORO, DURANGO | 05/11/2002 | 04/11/2052 |
| 3 | LA PITARRILLA 2 | 31124 | 220231 | 5,771 | DURANGO | EL ORO, DURANGO | 24/06/2003 | 23/06/2053 |
| 4 | LA PITARRILLA 3 | 31254 | 221576 | 4,200 | DURANGO | INDE, DURANGO | 02/03/2004 | 01/03/2054 |
| 5 | LA PITARRILLA 4 | 31845 | 226715 | 17,960 | DURANGO | INDE, DURANGO | 21/02/2006 | 20/02/2056 |
| 6 | PEÑA | 27442 | 216381 | 73 | DURANGO | EL ORO, DURANGO | 14/05/2002 | 13/05/2052 |
| 7 | PEÑA 1 | 27443 | 216382 | 62 | DURANGO | EL ORO, DURANGO | 14/05/2002 | 13/05/2052 |
| 8 | PITARRILLA 5 | 25/32978 | 231034 | 98,796 | DURANGO | EL ORO & INDE, DURANGO | 30/11/2007 | 29/11/2057 |
| 9 | PITARRILLA 6 | 25/33079 | 230335 | 81 | DURANGO | EL ORO & INDE, DURANGO | 16/08/2007 | 15/08/2057 |
| 10 | PITARRILLA 7 | 25/33421 | 234722 | 6,242 | DURANGO | EL ORO & INDE, DURANGO | 06/08/2009 | 05/08/2059 |
| 11 | PITARRILLA 7 FRACCION A | 25/33421 | 234723 | 1,298 | DURANGO | EL ORO & INDE, DURANGO | 06/08/2009 | 05/08/2059 |
| 12 | PITARRILLA 7 FRACCION B | 25/33421 | 234724 | 115 | DURANGO | EL ORO & INDE, DURANGO | 06/08/2009 | 05/08/2059 |
| | Total | | | 136,191 | | | | |

### 4.2.2 Surface Rights Property

SSD has acquired surface rights to most of the lands required for successful project permitting, construction and operation. Figure 4-3 and Figure 4-4 provide maps that show the boundaries of the surface rights required for the Project site, and Table 4-2 and Table 4-3 provide the corresponding status of SSD's ownership of, or access to, this land.

During the permitting process in Mexico, clear title or land access agreements must be presented to the regulatory authority in order to obtain mining or operating permits.

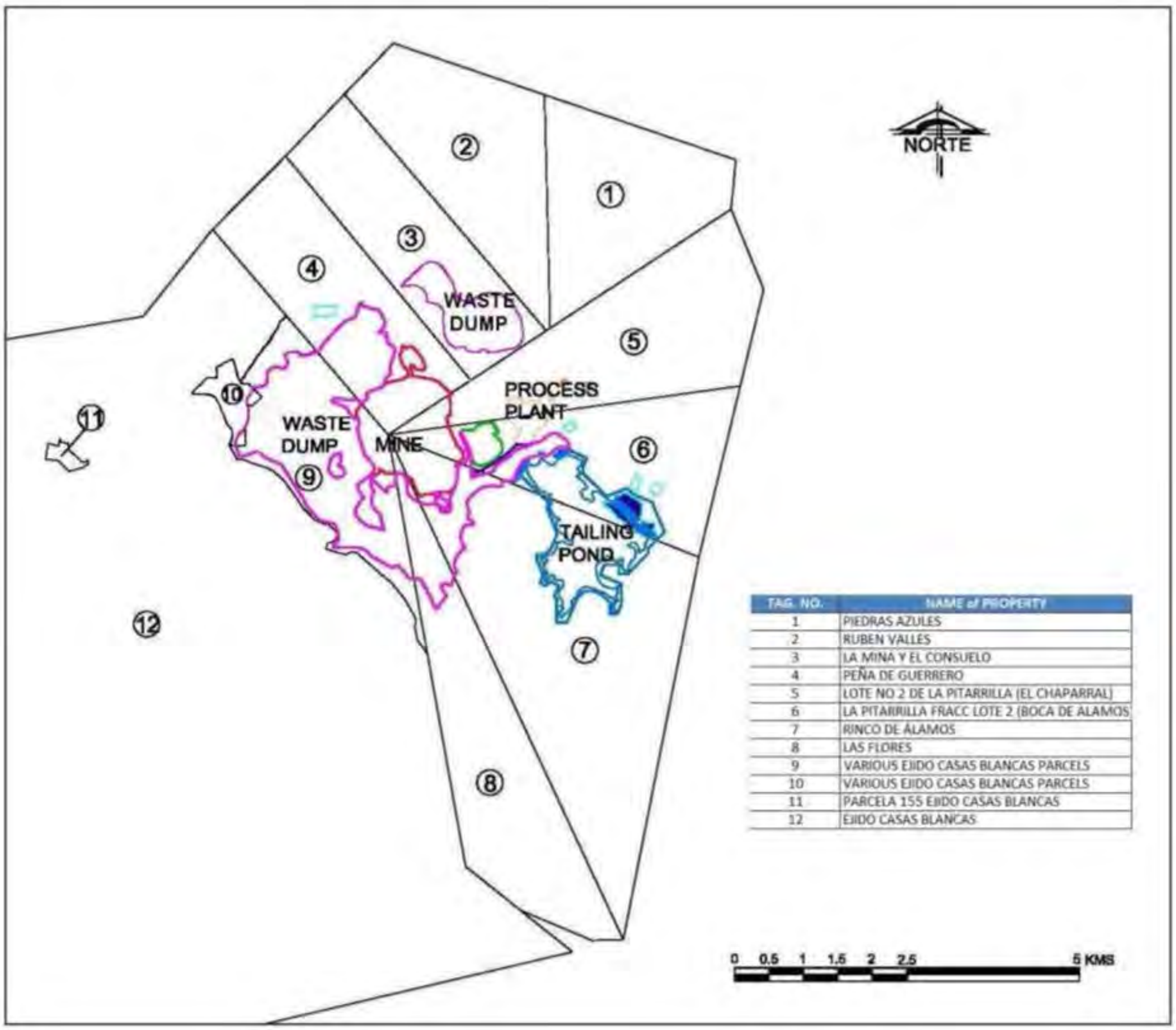



**Figure 4-3: Pitarrilla Property Ownership**

Source: SSR 2012

### 4.2.3 Private Land Required for Project Site

SSD owns clear title to the majority of the private land required for the Project site. It is currently in negotiations to obtain a long-term access agreement to one of the two remaining, privately-held plots of land required for the Project (Tag No. 3 on Figure 4-3: La Mina y el Consuelo) and has commenced an administrative process to obtain access to the other remaining, privately-held plot of land (Tag No. 8 on Figure 4-3: Las Flores).

With regards to the administrative process to obtain long-term access to the Las Flores property, the Mexican Constitution and federal laws state that mineral activities are of public interest to Mexico, and accordingly, provide the owner of a mineral concession with the legally-preferred right to the overlying surface rights. If the holder of the mineral concession is unable to acquire the surface rights required for operations through negotiation, the concession holder may request the federal government to commence an administrative process to acquire for the mineral concession holder long-term access to the required surface rights. This administrative process is known as a “temporary occupation”.

The temporary occupation is commenced with the concession-holder applying to the government agency, Administration and Appraisals Institute of National Goods (Instituto de Administración y Avalúos de Bienes Nacionales or “INDAABIN”, for its initials in Spanish), to conduct an appraisal of the value of the surface lands. SSD completed this application in November 2012, INDAABIN has conducted the appraisal of the Las Flores property, and SSD is awaiting the results of this evaluation.

### 4.2.4 Ejido Land Required for Project Site

As shown in Figure 4-3 and the inset table, a significant portion of the Project site is located on ejido land. An ejido is a communal ownership of land declared as such by Presidential Decree, regulated by the Mexican Agrarian Law, and administered by a representative board formed by members of the ejido. Although ejido land is generally held by the ejido community at large, the Agrarian Law permits the ejido community to compartmentalize the land and allocate specific parcels to individual members of the ejido for their exclusive use and usufruct. When plots of ejido land are compartmentalised, the beneficiary ejido members are permitted to enter into lease agreements over the parcels with third parties, but they may only transfer their ownership rights to the parcels to other members of the ejido community.

The Agrarian Law also permits the ejido community to authorize the privatization of the parcels and adoption by the corresponding ejido member of full title over such pieces of land. When the parcel is privatised, the ejido owner may sell the land to third parties, subject to certain rights of first refusal provided by the Agrarian Law in favour of other members of the ejido community.

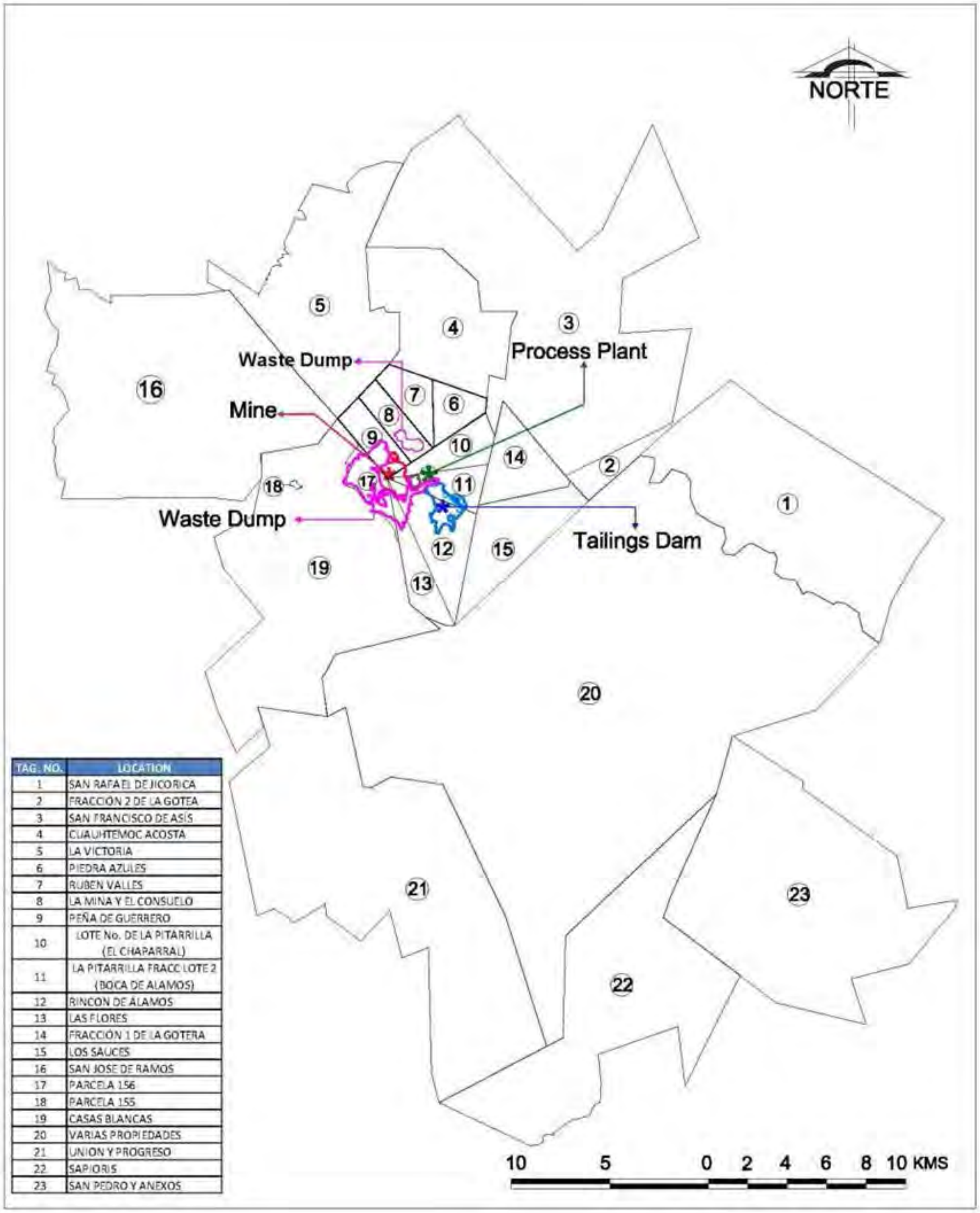
Tags No. 9 and 11 on Figure 4-3 represent five ejido parcels that have been compartmentalised, allocated to specific members of the ejido community, and approved by the members of the ejido for privatization. SSD currently has a long-term lease over all five of the ejido parcels, and an option to purchase all such lands when privatization is completed, subject to the rights of first refusal described above. The lease to such parcels, as well as the holding of the corresponding mining concessions, is sufficient for SSD’s mining purposes on such land. Silver Standard anticipates the privatization of this land to be completed in mid-2013, and, subject to the rights of first refusal, being able to acquire full title over such parcels of land at a date shortly thereafter.

Tag No. 10 in Figure 4-3 represents a number of smaller ejido parcels, of which SSD requires surface rights to two of these parcels for its planned waste dump. SSD believes it will be able to secure long-term access to these two parcels in the near future.

Overall, relations with the Ejido Casas Blancas remain strong. SSD employs a number of the members of the Ejido Casas Blancas and is actively engaged in projects improving the well-being of the community members.

**Table 4-2: Ownership of Pitarrilla Surface Properties**

| Tag No. | Location | Owner | Status | Tenure |
|---|---|---|---|---|
| 1 | Piedras Azules | Silver Standard Durango | Not Required | Private |
| 2 | Ruben Valles | Ruben Valles | Not Required | Private |
| 3 | La Mina y La Consuelo | Norberto Arreola y Hnos | Outstanding | Private |
| 4 | Pena de Guerrero | Silver Standard Durango | Owned by SSD | Private |
| 5 | Lote No. 2 de la Pitarrilla | Silver Standard Durango | Owned by SSD | Private |
| 6 | La Pitarrilla Fracc Lote 2 | Silver Standard Durango | Owned by SSD | Private |
| 7 | Rinco de Alamos | Silver Standard Durango | Owned by SSD | Private |
| 8 | Las Flores | Herederos de Enrique Padilla | Outstanding | Private |
| 9 | Various Ejido Casas Blancas Parcels | Ejido | Leasing | Ejido |
| 10 | Various Ejido Casas Blancas Parcels | Ejido | Outstanding | Ejido |
| 11 | Parcela 155 | Ejido | Leasing | Ejido |
| 12 | Ejido Casas Blancas | Ejido | Not Required | Ejido |




| TAG. NO. | LOCATION |
|---|---|
| 1 | SAN RAFAEL DE JICORICA |
| 2 | FRACCIÓN 2 DE LA GOTEA |
| 3 | SAN FRANCISCO DE ASÍS |
| 4 | CUAUHTEMOC ACOSTA |
| 5 | LA VICTORIA |
| 6 | PIEDRA AZULES |
| 7 | RUBEN VALLES |
| 8 | LA MINA Y EL CONSUELO |
| 9 | PEÑA DE GUERRERO |
| 10 | LOTE No. DE LA PITARRILLA (EL CHAPARRAL) |
| 11 | LA PITARRILLA FRACC LOTE 2 (BOCA DE ALAMOS) |
| 12 | RINCON DE ÁLAMOS |
| 13 | LAS FLORES |
| 14 | FRACCIÓN 1 DE LA GOTERA |
| 15 | LOS SAUCES |
| 16 | SAN JOSE DE RAMOS |
| 17 | PARCELA 156 |
| 18 | PARCELA 155 |
| 19 | CASAS BLANCAS |
| 20 | VARIAS PROPIEDADES |
| 21 | UNION Y PROGRESO |
| 22 | SAPIORIS |
| 23 | SAN PEDRO Y ANEXOS |

**Figure 4-4: Pitarrilla Property Ownership – Expanded Area**
Source: SSR, 2012

**Table 4-3: Ownership of Pitarrilla Surface Properties – Expanded Area**

| Tag No | Property Name | Owner | Status | Tenure |
|---|---|---|---|---|
| 1 | San Rafael de Jicorica | Ejido | Agreement | Ejido |
| 2 | Fraccion 2 de la Gotea | Victor Manuel Medina Castanos | Agreement | Private |
| 3 | San Francis de Asis | Ejido | Agreement | Ejido |
| 4 | Cuauhtemoc Acosta | Cuauhtemoc Acosta | Agreement | Private |
| 5 | La Victoria | Ejido | Not Required | Ejido |
| 6 | Piedra Azules | Silver Standard Durango | Owned by SSD | Private |
| 7 | Ruben Vales | Ruben Vales | Not Required | Private |
| 8 | La Mina y el Consuelo | Norberto Arreola y Hermanos | Outstanding | Private |
| 9 | Pena de Guerrero | Silver Standard Durango | Owned by SSD | Private |
| 10 | Lote No 2 de la Pitarrilla (El Chaparral) | Silver Standard Durango | Owned by SSD | Private |
| 11 | La Pitarrilla Fracc Lote 2 (Boca de Alamos) | Silver Standard Durango | Owned by SSD | Private |
| 12 | Rincon de Alamos | Silver Standard Durango | Owned by SSD | Private |
| 13 | Las Flores | Heirs of Enrique Padilla Herrera | Outstanding | Private |
| 14 | Fraccion 1 de la Gotera | Humberto Delgado Monte y EstellaMontes F | Agreement | Private |
| 15 | Los Sauces | Sr Nicolas Delgado | Not Required | Private |
| 16 | San Jose de Ramos | Ejido | Not Required | Ejido |
| 17 | Various Parcels | Ejido | Leasing | Ejido |
| 18 | Parcella 155 | Ejido | Leasing | Ejido |
| 19 | Casas Blancas | Ejido | Outstanding | Ejido |
| 20 | Various Properties | Various Owners | Not Required | Private |
| 21 | Union y Progresso | Ejido | Agreement | Ejido |
| 22 | Sapioris | Ejido | Agreement | Ejido |
| 23 | San Pedro y Anexos | Ejido | Agreement | Ejido |

In addition to the Project site lands, rights-of-way are also required for the access road and power transmission line associated with the Project.

For the access road and bridge across the Nazas River, the status of the associated rights-of-ways is shown in Table 4-4 and the positions of the properties are presented in Figure 4-5.

**Table 4-4: Pitarrilla Access Road and Bridge Rights of Way**

| Tag No. | Property Name | Owner(s) | Status | Tenure |
|---|---|---|---|---|
| 1 | Abasolo and Anexus | Ejido | Agreement in Place | Ejido |
| 2 | La Esperanza | Ejido | Agreement in Place | Ejido |
| 3 | Fraccion 20 del Refugio | Pedro Victor Lopez Lopez | Agreement in Place | Private |
| 4 | San Rafael de Jicorica | Ejido | Agreement in Place | Ejido |
| 5 | Fraccion 2 de la Gotera | Victor Manuel Medina Castanos | Agreement in Place | Private |
| 6 | San Fransisco de Asis | Ejido | Agreement in Place | Ejido |
| 7 | Fraccion 1 de la Gotera | Humberto Delgado Monte y Estela Montes F. | Agreement in Place | Private |
| 8 | La Pitarrilla Lote No 2 (El Chaparral) | Silver Standard Durango | Owned by SSD | Private |
| 9 | Lote No 2 La Pitarrilla (Boca de Alamos) | Silver Standard Durango | Owned by SSD | Private |

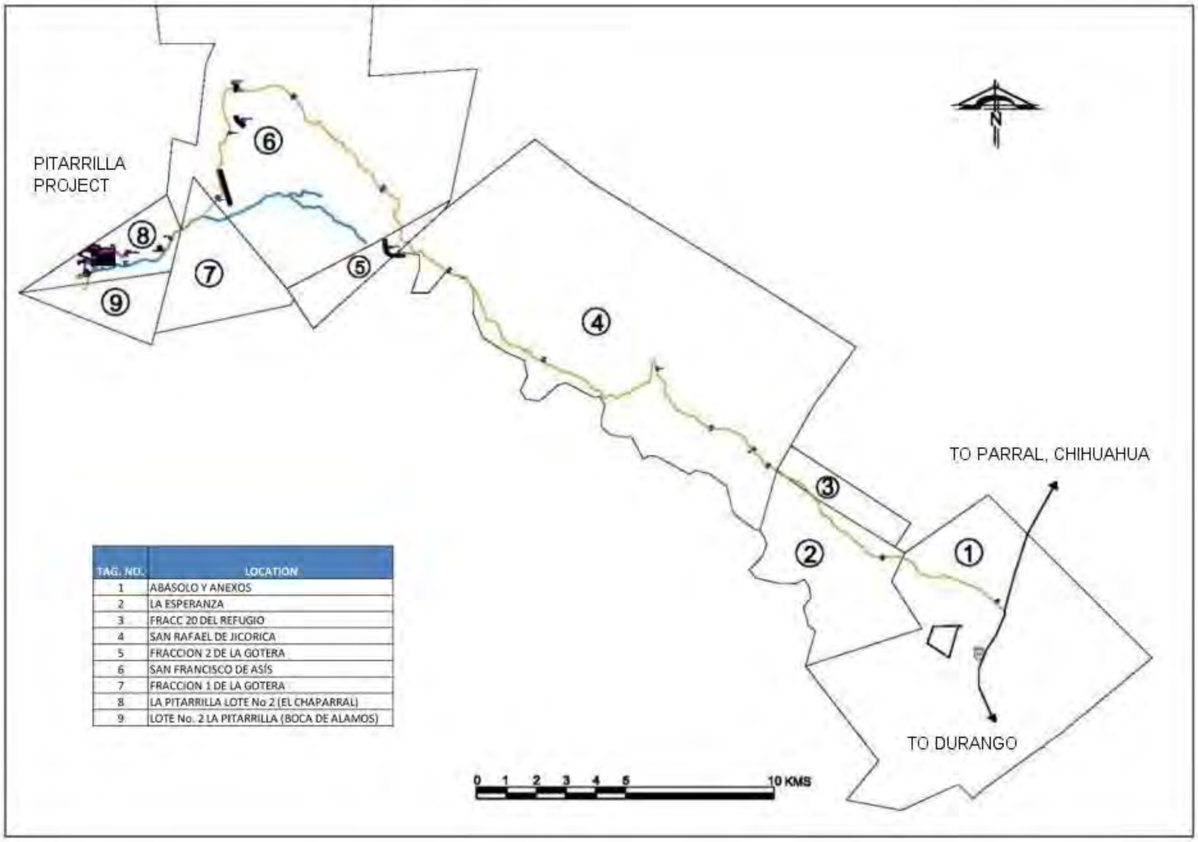



**Figure 4-5: Pitarrilla Property Ownership – Access Road and Bridge Rights of Way**

Source: M3, 2012

A power transmission line route has been identified by CFE, it extends through the city of Nuevo Ideal to a grid connection at the Subestacion Electrica Canatlán II (substation). The route for and land ownership for the power transmission line corridor is presented in Figure 4-6 and Figure 4-7 and Table 4-5.

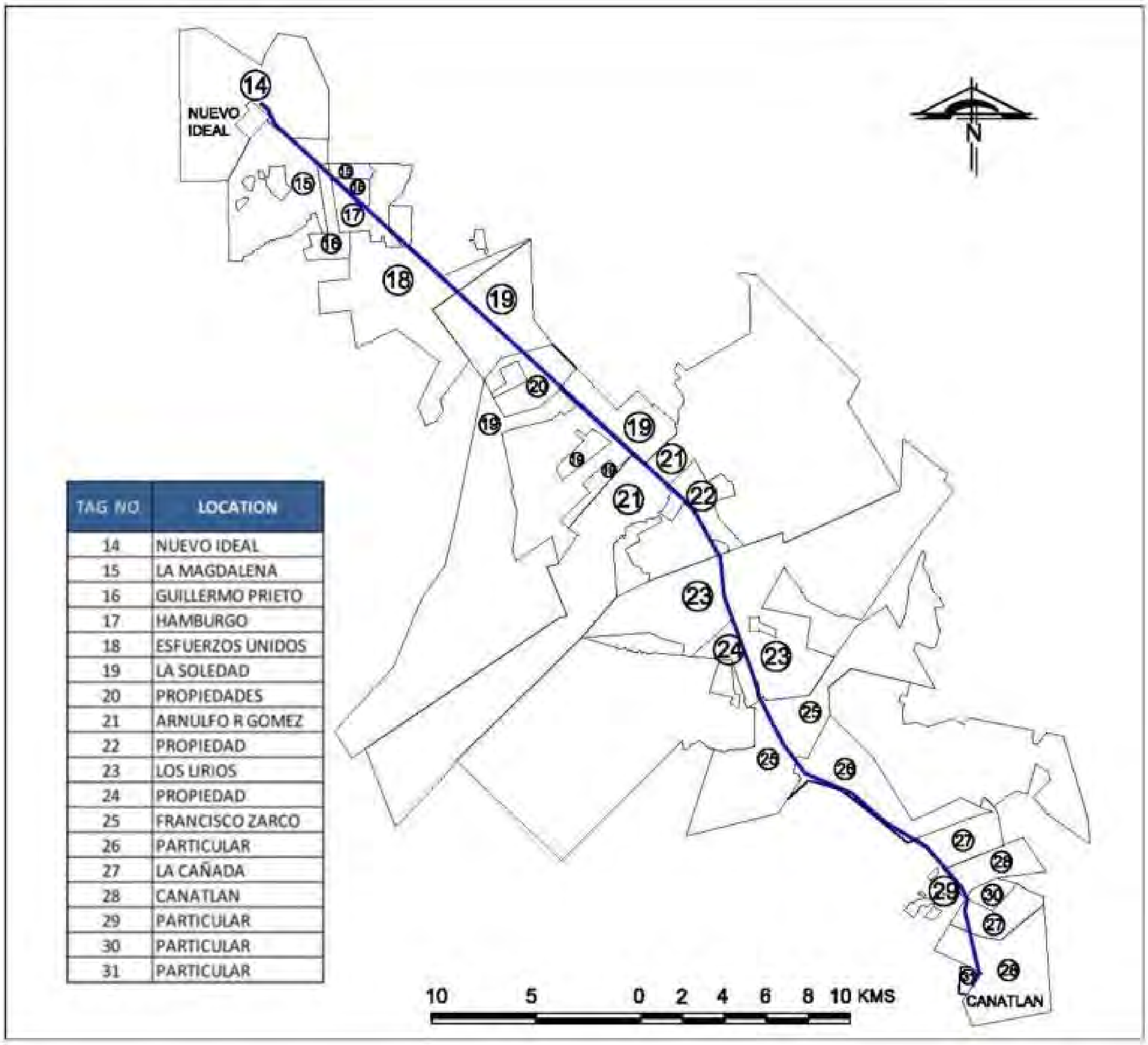



**Figure 4-6: Pitarrilla Power Transmission Route SE Canatlán II to Nuevo Ideal**

Source: M3, 2012

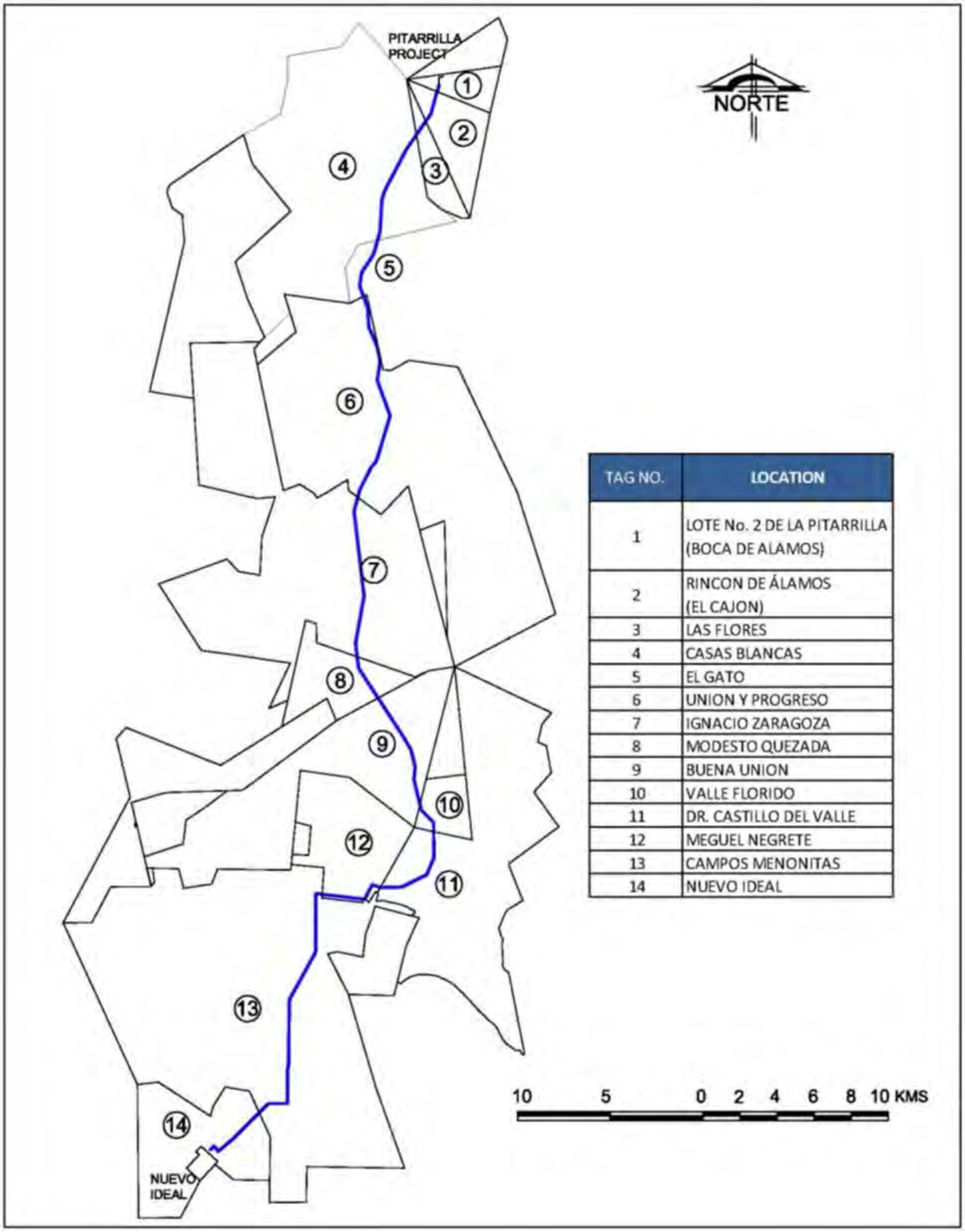



**Figure 4-7: Power Transmission Route Nuevo Ideal to Pitarrilla**

Source: M3, 2012

**Table 4-5: Land Status for Power Transmission Line Rights-of-Way**

| Tag No. | Property Name | Owner | Status | Tenure |
|---|---|---|---|---|
| 1 | Lote No 2. De La Pitarrilla | Silver Standard Durango | Owned by SSD | Private |
| 2 | Rincon de Alamos (El Cajon) | Silver Standard Durango | Owned by SSD | Private |
| 3 | Las Flores | Heirs of Enrique Padilla | Outstanding | Private |
| 4 | Casas Blancas | Ejido Owners | Outstanding | Ejido |
| 5 | El Gato | Sr Molino | Outstanding | Private |
| 6 | Union y Progresso | Ejido Owners | Agreement | Ejido |
| 7 | Zaragoza | Ejido Owners | Agreement | Ejido |
| 8 | Modesto Quezada | Ejido Owners | Agreement | Ejido |
| 9 | Buena Union | Ejido Owners | Agreement | Ejido |
| 10 | Valle Florida | Ejido Owners | Agreement | Ejido |
| 11 | Dr. Castillo del Valle | Ejido Owners | Agreement | Ejido |
| 12 | Meguel Negrete | Ejido Owners | Agreement | Ejido |
| 13 | Mennonite Colony | Mennonite Colony | Agreement | Menonite |
| 14 | Nuevo Ideal | Ejido Owners | Outstanding | Ejido |
| 15 | La Magdalena | Municipality of Nuevo Ideal | Outstanding | Ejido |
| 16 | Guillermo Prieto | Municipality of Nuevo Ideal | Outstanding | Ejido |
| 17 | Hamburgo | Municipality of Nuevo Ideal | Outstanding | Menonite |
| 18 | Esfuerzos Unidos | Municipality of Nuevo Ideal | Outstanding | Ejido |
| 19 | La Soledad | Municipality of Canatlan | Outstanding | Ejido |
| 20 | Propiedades | Municipality of Canatlan | Outstanding | Private |
| 21 | Arnulfo R Gomez | Municipality of Canatlan | Outstanding | Ejido |
| 22 | Propiedad | Municipality of Canatlan | Outstanding | Private |
| 23 | Los Lirios | Municipality of Canatlan | Outstanding | Ejido |
| 24 | Propiedad | Municipality of Canatlan | Outstanding | Private |
| 25 | Francisco Zarco | Municipality of Canatlan | Outstanding | Ejido |
| 26 | Particular | Municipality of Canatlan | Outstanding | Private |
| 27 | La Canada | Ejido Owners | Outstanding | Ejido |
| 28 | Canatlan | Ejido Owners | Outstanding | Ejido |
| 29 | Particular | Under Investigation | Outstanding | Ejido |
| 30 | Particular | Under Investigation | Outstanding | Private |
| 31 | Particular | Under Investigation | Outstanding | Private |

### 4.2.5 Obligations to Retain the Property

The main obligations which arise from a mining concession, and which must be kept current to avoid its cancellation, are:

- carry out the exploitation of those minerals expressly subject to applicability of the mining law;

- performance and filing of evidence of assessment work; and
- payment of mining taxes (technically called "mining duties").

The compliance with environmental laws is also relevant, as any un-fulfillment of such obligations may result in the shutting-down of the mining operations.

The regulations establish minimum amounts that must be invested in the concessions; and minimum expenditures may be satisfied through sales of minerals from the mine for an equivalent amount. A report must be filed in May of each year that details the work undertaken during the previous calendar year.

Mining duties must be paid in advance in January and July of each year, and are determined on an annual basis under the Mexican Federal Rights Law. Duties are based on the surface area of the concession, and the number of years that have elapsed since the mining concession was issued.

### 4.3 LEGAL TITLE

Silver Standard obtained a legal opinion on the titles of the exploration concessions, i.e. the Property, (defined in the opinion as "Pitarrilla"), from the Mexico City based law firm of Creel, Garcia-Cuellar, Aiza y Enriquez a Mexican law firm, with offices at Paseo de los Tamarindos 60 Bosques de las Lomas 05120 Mexico, Districto Federal, Mexico. The firm's legal opinion, dated 9 November, 2012, stated the following:

"Based upon the foregoing, and subject to the examinations and assumptions set forth herein, and without any further investigation, we are of the opinion that:

1. The Company is the lawful and registered holder of the Concessions.
2. Each Concession was duly granted by the General Bureau of Mining and the corresponding mining concession title was duly recorded before the Public Registry of Mining. The recordation information of each of the Concessions before the Public Registry of Mining is attached hereto as Exhibit "C".
3. The Concessions: (i) are not subject to any registered lien, encumbrance or ownership limitation; (ii) are not subject to any registered claim, suit or legal proceeding; and (iii) are in force.
4. The mining duties with respect to each of the Concessions have been paid up to the second semester of 2012.
5. The annual reports of proof of mining works for each of the Concessions have been submitted with the General Bureau of Mining up to the year 2011."

### 4.4 ROYALTIES AND ENCUMBRANCES

There are presently no royalties paid on lead, zinc, silver or gold in Mexico. Pitarrilla is not encumbered with a royalty payment to a third party.

### 4.5 ENVIRONMENTAL LIABILITIES

The Pitarrilla Project has no outstanding environmental liabilities from prior mining activities. Environmental commitments for future mining activities and mine closure requirements are discussed in detail below.

During mining activity and upon mine closure minimal environmental standards specified by the Mexican government in mine permitting must be met. Permits and Permitting are discussed in full in section 20.3

### 4.6 MEXICAN LEGAL FRAMEWORK

Environmental permitting of the mining industry in Mexico is mainly administered by the federal government body SEMARNAT, the federal regulatory agency that establishes the minimum standards for environmental compliance. Guidance for the federal environmental requirements is mainly derived from the LGEEPA. Article 28 of the LGEEPA specifies that SEMARNAT must issue prior approval to parties intending to develop a mine and mineral processing plant. An EIA (by Mexican regulations called a Manifestación de Impacto Ambiental, or "MIA") is the document that must be filed with SEMARNAT for its evaluation and, if applicable, further approval by SEMARNAT through the issuance of an Environmental Impact Authorization, whereby approval conditions are specified where works or activities have the potential to cause ecological imbalance or have adverse effects on the environment. The need for the mining industry to comply with Mexican environmental laws and regulations is supported by Article 27 section IV of the Ley Minera and Articles 23 and 57 of the Reglamento de la Ley Minera. Article 5 Section X of the LGEEPA authorizes SEMARNAT to provide the approvals for the works specified in Article 28. The LGEEPA also contains articles that speak directly to soil protection, water quality, flora and fauna, noise emissions, air quality, and hazardous waste management. The Ley de Aguas Nacionales provides authority to the CONAGUA, an agency within SEMARNAT, to issue water extraction concessions, and specifies certain requirements to be met by applicants.

Another important piece of environmental legislation is the LGDFS. Article 117 of the LGDFS indicates that authorizations must be granted by SEMARNAT for land use changes to industrial purposes. An application for change in forestry land use or Cambio de Uso de Suelo Forestal ("CUSF"), must be accompanied by a technical study that supports the ETJ. In cases requiring a CUSF, a MIA for the change of forestry land use is also required.

Mining projects also must include an AR and PPA.

The Ley General para la Prevención y Gestión Integral de los Residuos or LGPGIR also regulates the generation and handling of hazardous waste coming from the mining industry.

Guidance for the environmental legislation is provided in a series of NOM. These regulations provide specific procedures, limits and guidelines and carry the force of law.

## 4.7 MINE CLOSURE AND RECLAMATION

In accordance with the mine plan, the mine will cease operation after the 17th production year but stockpile re-handle operations continue to the 29th production year. Plans to facilitate closure have been incorporated into the design of the facilities, such that the need for extensive re-sloping and re-handling of materials is minimised at closure. Progressive reclamation will be included in the mine planning and operations, which will minimize the effect of the mine on the environment during operations. As part of the permitting requirements, a detailed closure plan will be prepared and submitted to SEMARNAT prior to operations. Detailed mine closure plans and regulations are presented in Section 20.9.

## 4.8 OPERATING PERMITS

In accordance with Mexican law certain permits must be acquired in order to commence mining and construction activities on the Pitarrilla property.

### 4.8.1 Environmental Permits, Licenses and Authorizations

There are three SEMARNAT permits required prior to construction: EIA, Change of Land Use, and Risk Analysis. A construction permit is required from the local municipality and an archaeological release letter is required from the National Institute of Anthropology and History ("INAH"). An explosives permit is required from the Ministry of Defense ("SEDENA") before construction begins. Water discharge and usage must be granted by CONAGUA. A project-specific environmental license (Licencia Única Ambiental), which states the operational requirements, is issued by SEMARNAT when the agency has approved the project operations. The key permits and the stages at which they are required are summarised in Table 4-6.

**Table 4-6: Permitting Requirements**

| Permit | Mining Stage | Agency |
|---|---|---|
| Environmental Impact Statement – MIA | Construction/Operation/Post-operation | SEMARNAT |
| Land Use Change – ETJ | Construction/Operation | SEMARNAT |
| Risk Analysis – RA | Construction/Operation | SEMARNAT |
| Construction Permit | Construction | Municipality |
| Explosive & Storage Permits | Construction/Operation | SEDENA |
| Archaeological Release | Construction | INAH |
| Water Use Concession | Construction/Operation | CNA |
| Water Discharge Permit | Operation | CNA |
| Project-specific License (LUA) | Operation | SEMARNAT |
| Accident Prevention Plan | Operation | SEMARNAT |

The Project has acquired permits for mineral exploration and construction of initial project works, including water concessions, ramp, hazardous waste generator and the archaeological release. The permitted activities and the corresponding permit numbers are listed in detail in Section 20.3.

An environmental permit application was submitted on July 4, 2012 for the construction of various new facilities, including the principal access road, a permanent camp for operations personnel, a power line, a metallurgical laboratory, a maintenance workshop, a landfill, and other minor works. An additional environmental permit for construction and operation for a bridge over the Nazas River, airport runway for private airplanes, and Telmex - Telcel communications tower was submitted on October 9, 2012. Review of the permits by SEMARNAT typically requires 60 to 120 days.

Environmental permitting documents for the open pit, crusher, beneficiation plant, waste rock dumps and TSF will be ready for submittal to SEMARNAT once all of the surface land right acquisitions are completed.

### 4.9 OTHER SIGNIFICANT FACTORS AND RISKS

There are no other known significant risks that may affect access, title or the right or ability to perform mining-related work on the Property.

# 5 PHYSIOGRAPHY, CLIMATE, ACCESS, LOCAL RESOURCES AND INFRASTRUCTURE

## 5.1 PHYSIOGRAPHY

The physiography in the immediate area of Pitarrilla (Figure 5-1) is rugged, with rocky cliffs and steep walled gullies, with surficial soil conditions represented by shallow soils and weathered bedrock. Elevations on the property range from about 1600 masl in the valley floors to 2140 masl at the top of Cerro La Pitarrilla.

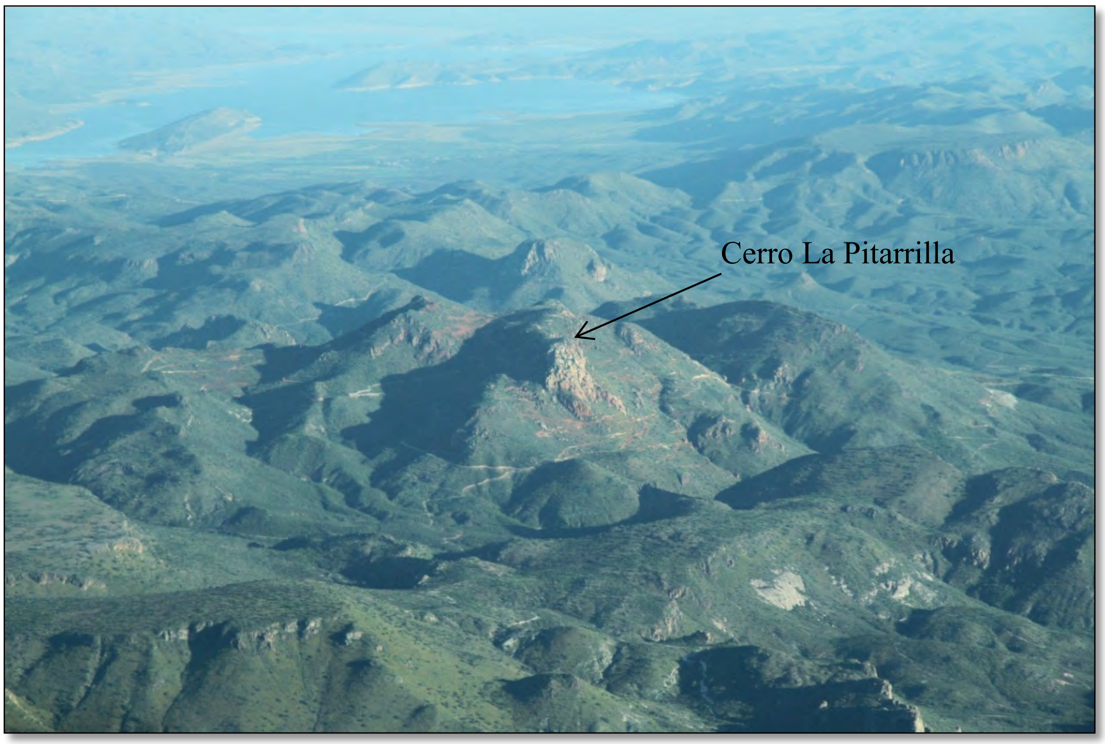



**Figure 5-1: Aerial View of the Pitarrilla Project, Showing Landforms (view to the north)**
Source: SSR, 2012

## 5.2 CLIMATE

The area of Pitarrilla falls in an area of characterised as a steppe (that is, grassland plains without trees, except at water sources). Typically the climate at Pitarrilla is dry, winter days are cool with minimum temperatures dropping slightly below 0° C before warming to daily maximums of about 23°C. From May to the beginning of July the daytime maximum temperatures rise to a maximum of about 35°C before the onset of a wet season that lasts until about mid-September. After mid-September there is another short hot dry period before cool autumn days begin in mid-October.

The total annual precipitation varies between 375 to 405 mm, based on public weather stations. The local station has an annual average precipitation of 407 mm. Hail, snow and electrical storms occur in the region which can be impacted by tropical storms or depressions, but would be on the edge of the hurricane trajectories and is too far inland to be adversely affected by intense winds. It is however, from these storm systems that the region typically receives the bulk of its annual rainfall. The local weather station has registered wind velocities averaging 3.9 m/s with gusts up to about 30 m/s. Winds are generally from the east and are strongest in the months of April and May.

A mine at Pitarrilla will have a 12 month operating season.

## 5.3 VEGETATION

The Project is located in the high plains (Altiplano) region of Mexico, a large area noted for its high altitude and low winter temperatures. The region is comparable to high desert and can include vegetation such as chaparral, mesquite-grassland, or arid tropical scrub. There are a considerable number of endemic species within the Altiplano.

A vegetation baseline study was completed by Centro de Ecología Regional A.C. (2010).Three vegetation types were identified in the Project area: 1) pine and oak forest, 2) "matorral xerofilo", which includes high desert-chaparral, and 3) riparian forest, which is a forested area adjacent to a water source.

An inventory of the vegetation types included 29 plant families and 66 species. The most abundant families present are *Asteraceae* (commonly referred to as the aster, daisy, or sunflower family) and *Cactaceae* (cactus family). Three species (*Mamillaria marksiana*, *Pinus pinceana*, and *Thelocactus heterochromus*) are classified as at risk per the Mexican regulation NOM-059-SEMARNAT-2010, which lists native species and their risk status. An additional 14 species were considered of special interest due to potential for commercial or decorative use, or due to the difficulty to propagate the species.

The relative importance of the species was calculated based on the species value. The species with the highest value were ocotillo, cat claw mimosa, acacia, and mesquite. The calculated species diversity was 1.791, which is considered to be low. It is attributed to the degradation experienced by the area due to decades of use for agriculture and grazing.

## 5.4 ACCESS

Access is currently accessible through a network of public roadways in the area. Road access is available from two unpaved public roads, with a paved national Highway-45 extending to within 47 km of the plant site. The main access to the Project site is planned to be along approximately 47 km of public and private dirt roadways, from the junction with paved Highway 45, to the Project's southeast gate. The primary site access road will utilize the existing roadway serving the nearby local community of San Francisco de Asís, with secondary access via the existing road to Casas Blancas. Improvements are required for the main road, prior to plant construction, the most significant of which is the addition of a permanent bridge over the Nazas River, approximately 11 km from the plant site.

## 5.5 SURFACE RIGHTS FOR MINING OPERATIONS

SSR has acquired surface rights to most of the lands required for successful project permitting, construction, and mining operations, including those lands required for the process plant, TSF, and waste dumps. Surface rights, land ownership, and rights of way are discussed in detail in Section 4.2.2. The position and design of the Pitarrilla TSF, waste disposal sites and processing plant and are presented in other sections of this Technical Report. The locations of these relative to surface land holdings are presented in Section 4.2.2.

## 5.6 LOCAL RESOURCES

Mexico has a large mining economic sector and well trained human resources are available in the country. It is planned that human resources required to operate a mine at the Project will be sourced mainly from Durango, Coahuila, Chihuahua and Zacatecas states, and that these people will work rotating shifts.

Located 160 km southwest of the Property is the city of Victoria de Durango, the state capital of Durango, with a population of approximately 1 million. Additionally the city of Torreón, the capital of Coahuila state and with a population of approximately 1.5 million, is located approximately 160km to the east. There are large active mines and developed mining infrastructure in the states of Durango and Coahuila. Both the state capitals have sufficient populations and support services to adequately provide the Pitarrilla Project with general goods, services and labour.

The closest population centers to the Property are San Francisco de Asís (located 12km to the northeast) and Casas Blancas (situated in the northeast portion of the Project concessions). San Francisco de Asís has a population of about 800 and Casas Blancas has a population of approximately 120.

## 5.7 INFRASTRUCTURE

Power for the Project is available from the national power grid at the Subestacion Electrica Canatlán II (substation) located approximately 139 km south of the plant site. The power will be provided by the national power utility, CFE.

Fresh make-up water to the plant will be provided from several wells located on the property near the Nazas River, approximately 10 km from the plant site. Water from the wells will be pumped to a booster tank and, from there, be pumped to the plant and other project water consumers.

Detailed information on the TSF is presented in Section 18.11. The waste disposal areas for the Project are discussed in detail in Section 16.4 and the processing plant facility is discussed in Section 17.2.

# 6 HISTORY

## 6.1 PAST EXPLORATION WORK

Available records of mineral exploration conducted on the Property and immediately adjacent ground date back to 1996. Any earlier exploration on these lands by mining or exploration companies appears to have gone undocumented.

Table 6-1 summarizes the most significant exploration work conducted at Pitarrilla between 1996 and 2002, before Silver Standard acquired the property.

Recognition of the mineral potential of the Pitarrilla area was first established by F. Hillemeyer and P. Durning of LCI, while conducting regional reconnaissance gold exploration for Monarch Resources de Mexico (Monarch) in 1996. Based on encouraging prospecting results obtained by LCI, Monarch stake-claimed the Pitarrilla concessions and commenced exploration in the area. Rock-chip and grid-controlled soil sampling programs were completed along with the collection of stream-sediment samples. Monarch's soil geochemistry survey identified a gold anomaly to the southeast of Cerro La Pitarrilla, which the company tested with an RC drilling program. Monarch's drillholes are not located within the area of the current Ag-Pb-Zn resource.

Due to the relatively weak assay results for gold that were obtained by its drilling program, Monarch returned the mineral rights for the claims back to LCI. In the following years, until 1999, Hillemeyer and Durning returned to the Property on a number of occasions to prospect the area on behalf of companies potentially interested in acquiring the claims from LCI.

In 1999, LCI conducted a comprehensive evaluation of the Property for the Mexican subsidiary of Hecla Mining Company ("Hecla"), which involved the excavation, mapping and sampling of shallow trenches mostly located on the northeastern slopes of Cerro La Pitarrilla. Notwithstanding the sporadic gold anomalies in trench samples, Hecla was not sufficiently encouraged by the results of the property evaluation to acquire the mineral rights to the property from LCI. A few months after Hecla's evaluation, LCI decided to allow the exploration licenses for the Pitarrilla claims to expire.

In 2002, Silver Standard contracted LCI to acquire mineral properties in Mexico which showed good exploration potential for silver. One of the first areas LCI recommended for claiming was the ground covered by the former Pitarrilla claim group. Between November 2002 and March 2003, a total of 12 concessions covering 136,191 ha were claimed by Explominerals, S.A. de C.V. on behalf of Silver Standard.

**Table 6-1: Summary of Work Conducted on Pitarrilla Property 1996 to 2002**

| | |
|---|---|
| 1996 | Monarch Resources de Mexico, S.A. de C.V. completed rock-chip sampling along with soil and stream-sediment surveys (Durning and Hillemeyer, 2002). Monarch completed a 22 Reverse Circulation (RC) holes program, totaling 2,842 m (Durning and Hillemeyer, 1997b). Monarch's exploration was concentrated outside of the current Ag-Pb-Zn resource. |
| 1997 | La Cuesta International, Inc. (LCI) re-acquired the Pitarrilla concessions from Monarch and collected a total of 30 rock-chip samples in a follow-up program (Durning and Hillemeyer, 1997a). |
| 1998 | LCI collected 14 channel and grab samples. The samples were sent to Chemex Labs, Inc. for chemical analysis (Thurow, 1998). |
| 1999 | LCI conducted additional reconnaissance rock sampling and basic geological mapping. Seven trenches, between 60 m and 200 m in length, were excavated and mapped in detail. A total of 637 samples were sent to Bondar-Clegg in Hermosillo, Sonora for multi-element analysis (Durning and Hillemeyer, 1999). |
| 2002 | Explominerals, S.A. de C.V. acquired Pitarrilla concessions on behalf of Silver Standard, and together with LCI, collected 34 rock-chip samples in a work program (Durning and Hillemeyer, 2002). |

## 6.2 HISTORICAL MINERAL RESOURCE ESTIMATES

There are no historical Mineral Resource estimates for the Property.

## 6.3 PRIOR MINERAL PRODUCTION

There is no recorded prior mineral production from the Pitarrilla property.

# 7 GEOLOGICAL SETTING AND MINERALISATION

## 7.1 REGIONAL AND LOCAL GEOLOGY

The Property is located on the eastern flank of the Sierra Madre Occidental ("SMO") mountain range that extends for more than 1,500 km in a north-westerly direction through the northern half of Mexico. This mountain range is the erosional remnant of one of the Earth's most voluminous accumulations of intermediate to felsic volcanic rocks, which formed a calc-alkaline magmatic arc that was built during Eocene to early Miocene time, roughly 52 to 25 million years ago, in response to subduction of the Farallón tectonic plate beneath North America, (Ferrari *et al*., 2007). Medium to high-level hydrothermal systems variably enriched in Ag, Au, Pb, Zn and to a lesser extent Cu, Sb, As, Hg, and F were intermittently generated during this extended period of volcanism, including the epithermal mineral systems that formed the Mexican silver mining districts at Guanajuato, Zacatecas, Fresnillo, and Santa Barbara-San Francisco del Oro (Figure 7-1). The Ag, Pb, Zn mineralisation found on the Property is situated in the central section of the globally important *Central Mexican Silver Belt*, a north-westerly aligned, 900 kilometre-long metallogenic province defined by the four previously-noted silver mining districts along with the mining districts of Parral, Santa Maria del Oro, and Sombrerete-Chalchihuites.

In the area of the Pitarrilla property, the Tertiary volcanic rocks of the SMO overlie marine sedimentary rocks that were deposited in a back-arc basin from early to middle Cretaceous time. These marine sediments suffered compressional deformation during the Laramide Orogeny which peaked during the late Cretaceous. A major unconformity separates the deformed Cretaceous sedimentary rocks from the overlying Eocene to Oligocene volcanic and volcaniclastic rocks. Along the eastern margin of the SMO, this unconformity is commonly marked by the presence of continental clastic sedimentary rocks, mainly conglomerates and sandstones. Eocene (52 to 40 Ma) andesitic flows and domes, as well as minor silicic lavas and ignimbrites generally form the lower volcanic stratigraphy of the SMO (Ferrari *et al*., 2007). Following a hiatus in arc magmatism at the end of the Eocene, extensive volcanism in the SMO resumed, with voluminous Oligocene (32 to 28 Ma; Ferrari *et al*., 2007) silicic ignimbrites and rhyolitic domes deposited. Basaltic-andesitic volcanism followed most of the major ignimbritic episodes, as evidenced by mafic flows locally overlying the Oligocene sequences (Aguirre-Díaz and McDowell, 1993).

The core of the SMO can be viewed as a relatively un-extended crustal block that separates two NNW-SSE trending belts, marked by extensional deformation, which occur along its western and eastern flanks (Henry and Aranda-Gómez, 1992). The eastern flank of the central sector of the SMO, where the Pitarrilla deposit is located, is interpreted as having undergone two major extension events. The first event, directed ENE-WSW, occurred during the early Oligocene (32.3 to 30.6 Ma) at about the same time as the main episode of ignimbritic volcanism (Luhr *et al*., 2001). A subsequent, late Oligocene ENE-WSW directed extension began around 24 million years ago, post-dating the Oligocene silicic volcanism but coinciding with a mafic, alkaline volcanic event (Aguirre-Díaz and McDowell, 1993; Luhr *et al*., 2001). NW-trending faults generated during early Miocene extension are interpreted to be reactivated early Oligocene structures (Aranda-Gómez and McDowell, 1998).

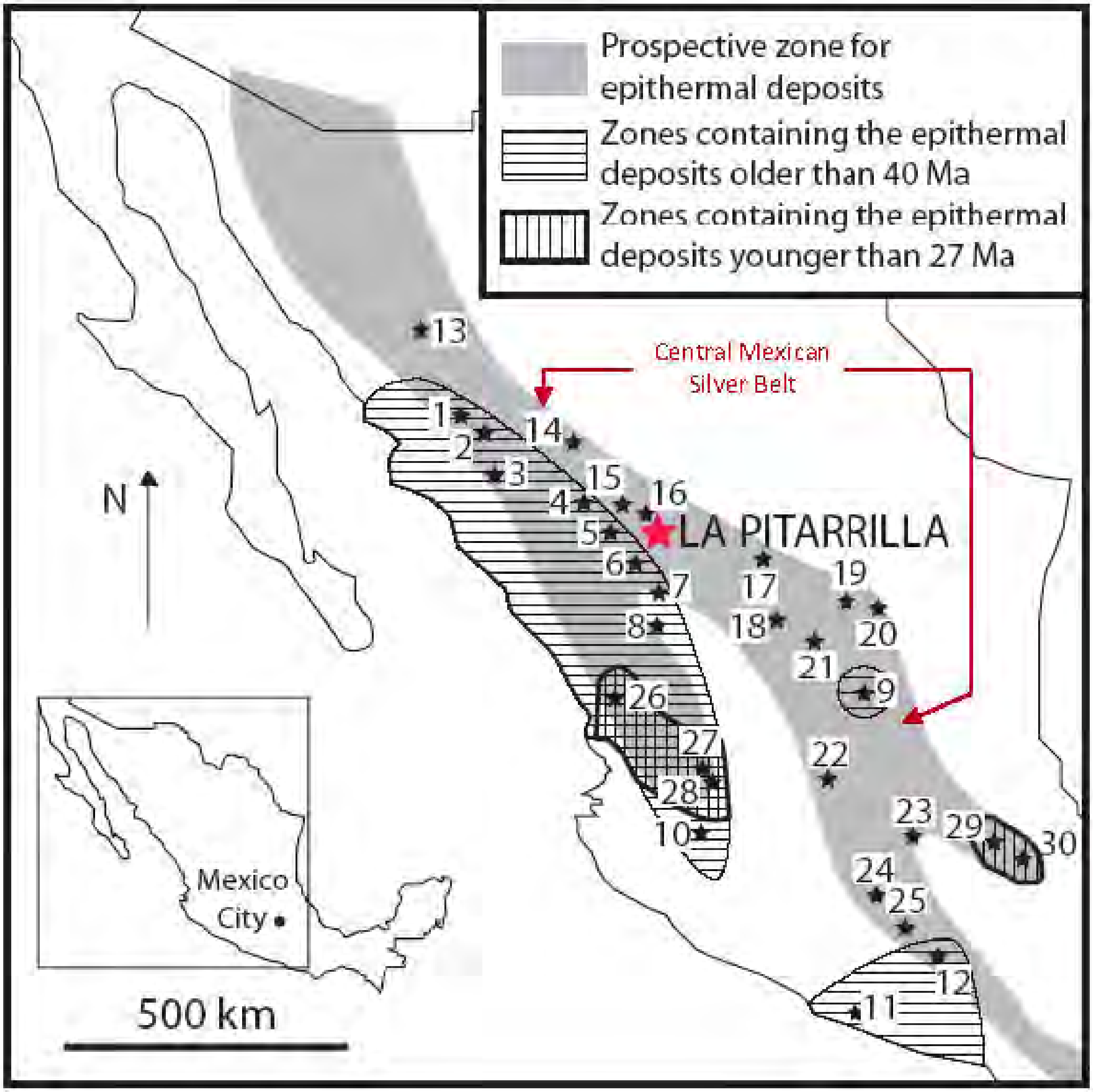



**Figure 7-1: Location of the Pitarrilla Ag-Pb-Zn deposit in Relation to Other Silver Mining Districts and Deposits in Central Mexico**

(Modified from Camprubi and Albinson, 2007). 1: Batopilas, 2: Los Angeles, 3: Guanacevi, 4: Topia, 5: Orion, 6: San Dimas (Tayoltita), 7: Mala Noche, 8: Lluvia de Oro, 9: Real de Angeles, 10: El Barqueño, 11: Real de Guadalupe, 12: Taxco, 13: Ocampo, 14: San Fransisco del Oro, 15: La Ciénega, 16: Bacis, 17: Velardeña, 18: Sombrerete, 19: Real de Cartorce, 20: La Paz, 21: Fresnillo, 22: Guanajuato, 23: San Martin, 24: El Oro-Tlapujahua, 25: Temascaltepec (La Guitarra), 26: El Indio-Huajicori, 27: Bolaños, 28: San Martin de Bolaños, 29: Pachuca, 30: Ixtacamacaxtitlan.

## 7.2 PROJECT AREA GEOLOGY

### 7.2.1 Property Geology

The interpreted geology of the Property is presented in the geologic map Figure 7-2. The Ag-Pb-Zn mineralisation is spatially associated with a Tertiary rhyolite dome complex that was emplaced over a sequence of intermediate to felsic volcaniclastics and pyroclastics, which overlie a Cretaceous marine sedimentary basement. Four informal formations are defined at Pitarrilla, which from oldest to youngest are the Peña Ranch, Pitarrilla, Cardenas, and Casas Blancas Formations. These are described in detail below.

#### 7.2.1.1 The Peña Ranch Formation

The Peña Ranch Formation is dominated by thinly inter-bedded Cretaceous mudstone and siltstone with lesser limestone and pebble conglomerate lithofacies. Economically significant Ag-Pb-Zn mineralisation occurs in the form of disseminated and fracture-fill sulphides, which in order of abundance include pyrite, sphalerite, pyrrhotite, galena, chalcopyrite, arsenopyrite, stibnite, and tetrahedrite.

#### 7.2.1.2 The Pitarrilla Formation

The Pitarrilla Formation unconformably overlies the Peña Ranch Formation and consists of well-stratified, volcaniclastic rocks and a single massive lava flow of presumed Eocene age.

At the base of the Pitarrilla Formation and immediately overlying the Cretaceous-Tertiary unconformity is a polymictic conglomerate unit, referred to as the Manto Rico member (Somers *et al*., 2010). The Manto Rico conglomerate is a key lithology because it hosts important semi-massive replacement mineralisation composed of varying combinations of pyrrhotite, sphalerite, pyrite/marcasite, chalcopyrite and lesser galena.

Overlying the Manto Rico member is a mainly volcaniclastic succession, which displays vertical composition variation from andesitic at the base to dacitic in the upper horizons. Intercalated with the volcaniclastic units are thinner deposits of primary pyroclastic ejecta sourced from the Eocene arc eruptions. In the central part of the Pitarrilla Formation is an interpreted massive, fine-grained intermediate volcanic flow.

The various members and rocks of the Pitarrilla Formation are interpreted to have been deposited in a back-arc sedimentary basin that had formed along the eastern flank of the Eocene volcanic arc, now represented by the Sierra Madre Occidental. Silver mineralisation hosted by volcaniclastics and pyroclastics of the Pitarrilla Formation is extensive, typically being associated with disseminated sulphide and sulphosalt phases at lower elevations, and with disseminated iron and manganese oxides in weathered rocks closer to surface.

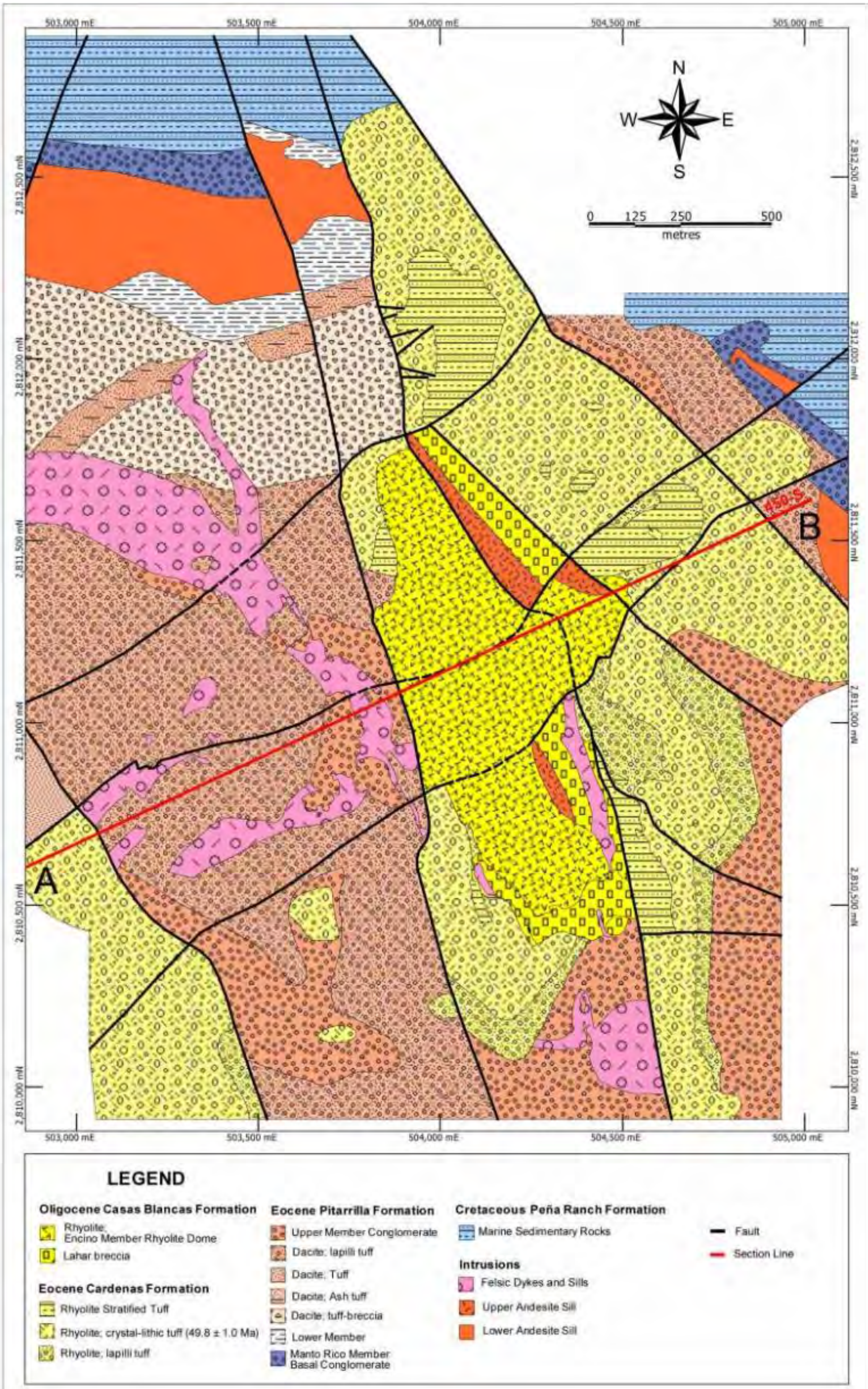



**Figure 7-2: Interpreted Surface Geologic Map of the Pitarrilla Ag-Pb-Zn deposit**

Based on the 1:2000 scale map of Somers et al, 2010. NAD27 UTM Zone 13N

#### 7.2.1.3 The Cardenas Formation

The Cardenas Formation unconformably overlies the Pitarrilla Formation and consists predominantly of sub-aerial, crystal-rich, non-welded to welded ignimbrites and surge deposits of presumed Oligocene age. Following the eruption and emplacement of the felsic pyroclastics, the ignimbrite and surge deposits were eroded and the depositional environment returned to a shallow marine or lacustrine one, in which the stratified lithofacies of the Cardenas Formation were deposited. The Cardenas Formation rocks have undergone extensive weathering. Silver mineralisation is associated with disseminated hematite ± manganese oxides occurring in pyroclastic and volcaniclastic facies of the formation in two areas, at the Javelina Creek Zone and at the South Ridge Zone.

#### 7.2.1.4 The Casas Blancas Formation

The Casas Blancas Formation unconformably overlies the Cardenas Formation and is composed of volcaniclastic rocks and an overlying rhyolitic flow-dome, referred to as the Encino member, also of presumed Oligocene age. The flow-dome crops out on the eastern ridge of Cerro La Pitarrilla. Field observations suggest the rhyolitic flow-dome was emplaced a relatively short time before the main hydrothermal event that deposited the Ag-Pb-Zn mineralisation, although the rocks forming the dome only rarely contain geochemically significant concentrations of silver or associated base metals.

#### 7.2.1.5 Intrusives

Two andesitic sills intrude the volcaniclastic-pyroclastic succession hosting the Pitarrilla deposit; the larger of the two (Lower Andesite Sill), with 100 to 130 m true thickness, was intruded into the basal section of the Pitarrilla Formation, whereas the smaller one (Upper Andesite Sill) occurs at the base of the Casas Blancas Formation Figure 7-2. The Upper Andesite Sill is an important host to disseminated and veinlet sulphide mineralisation containing silver. In addition, sub-horizontal lenses of semi-massive to massive, silver-rich base metal mineralisation occur at or close to the upper and lower contacts of the Upper Andesite Sill. These mineralised lenses, or *mantos*, have lateral extents of tens to hundreds of metres and are a few metres thick. Sulphide phases found in the lenses include, in order of decreasing abundance, pyrite, sphalerite, chalcopyrite, pyrrhotite, galena, arsenopyrite and tetrahedrite.

Quartz-feldspar porphyry (felsic) dykes cross-cut all strata at Pitarrilla, except the Encino dome, and are interpreted to be the igneous ‘feeders’ that supplied magma to the flow-dome. The felsic dykes also fed a large sill that was emplaced into the Pitarrilla Formation beneath the western flank of Cerro La Pitarrilla **(**Figure 7-3.). The dykes have two preferred strike orientations: NE-SW and NNW-SSE, parallel to the orientation of two main fault sets. The felsic dykes and sills are concentrated and converge at the highest elevation of Cerro La Pitarrilla, where the Encino rhyolitic flow-dome occurs. Disseminated sulphide mineralisation occurs within the felsic dykes and sill.

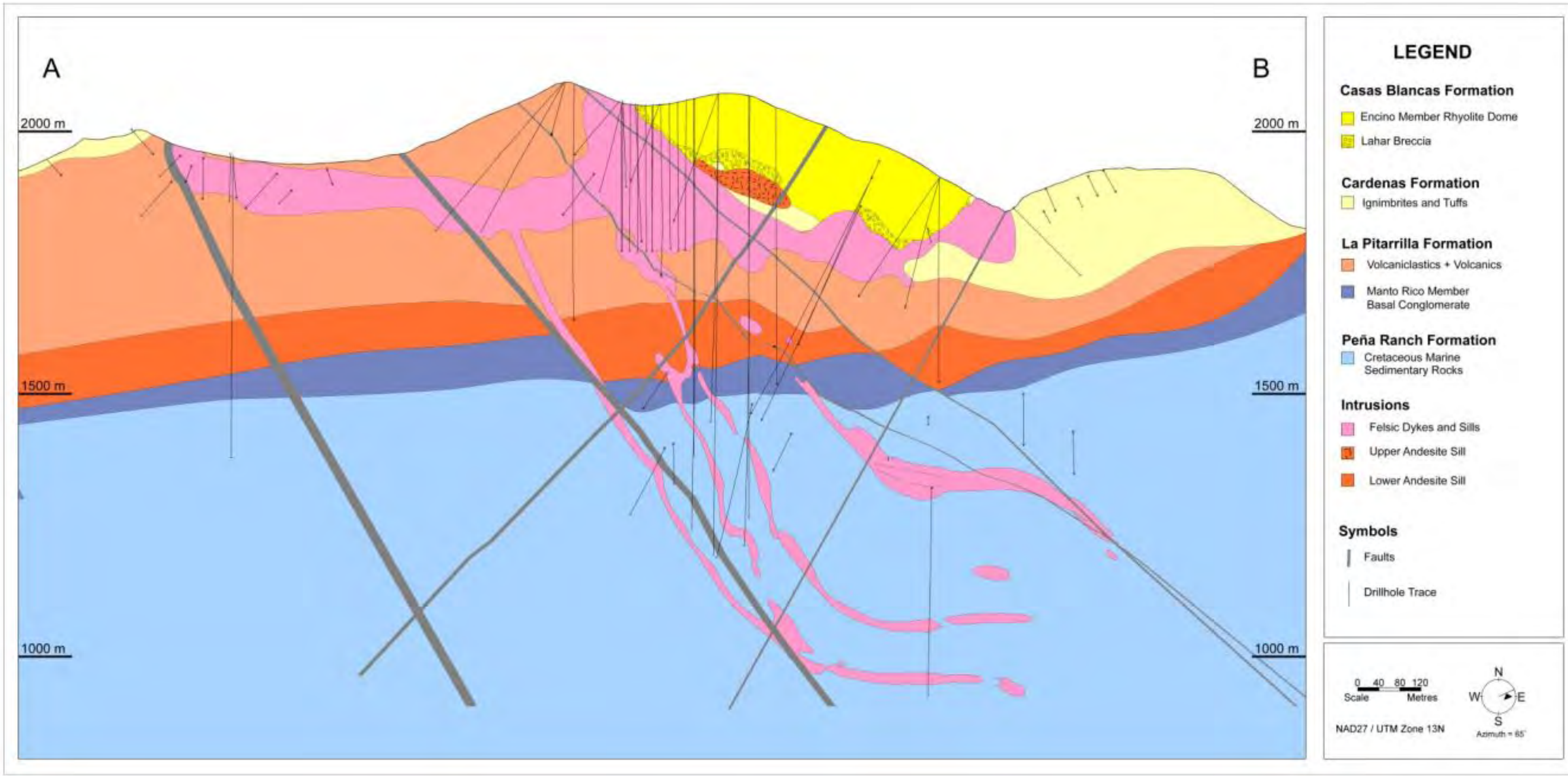



**Figure 7-3: ENE-WSW Cross-Section Showing the Interpreted Pitarrilla Geology, Selected Faults, and Drillhole Traces**

Source: M3, 2012

## 7.3 STRUCTURAL GEOLOGY

At Pitarrilla, two erosional events and one protracted extensional structural event are recognised as having taken place during the Tertiary age (Somers, 2010). Compressional structural features are developed only in the Cretaceous sedimentary rocks of the Peña Ranch Formation. These include upright folds, listric reverse faults, probable thrust faults, and intense fracturing. These structural features are believed to be related to the Laramide Orogeny (80 to 55 mya) (Somers *et al.*, 2010). They are not discussed in this report as they do not appear to play a significant role in the localization of the Pitarrilla Ag-Pb-Zn deposit.

The first major erosional event is represented by a well-defined angular unconformity between folded Cretaceous sedimentary basement rocks belonging to the Peña Ranch Formation and the shallowly dipping Pitarrilla Formation of Eocene age. The unconformity is generally defined by the presence of Manto Rico conglomerate. Moderate to shallow dipping faults define the main structural setting at Pitarrilla, with three principal fault sets recognised:

i. NE-striking, NW- dipping faults
ii. NNW-striking, NE- dipping faults (Figure 7-2 and Figure 7-3)
iii. NNW-striking, SW-dipping faults

Moderately dipping NNW-striking faults tilt the originally sub-horizontal strata of the Pitarrilla and Cardenas Formations up to 35° to the southwest, whereas the NE-striking faults tilt these same strata up to 28° to the south-southeast. Offsets of 20 to 150 m of the shallow dipping strata of the Pitarrilla and Cardenas Formations along the interpreted faults indicate mainly normal displacement. Horizontal slip-components along the principal faults appear to be minor.

The relatively flat-lying Casas Blancas Formation overlies the faulted and tilted Cardenas Formation and its emplacement appears to have post-dated the faulting that inclined and displaced the older formations. The contact between these two formations is marked by an erosional unconformity that is much less pronounced than the Cretaceous-Tertiary unconformity.

The felsic dykes at Pitarrilla, interpreted as feeders for the rhyolitic Encino dome, have essentially the same trends as the two main fault sets, suggesting that these faults, or ancillary parallel structures, were reactivated during the volcanism that deposited the Casas Blancas Formation and served as structural conduits for the felsic magma.

## 7.4 PITARRILLA GEOLOGICAL MODEL

In order to generate a practical geological model of the Pitarrilla deposit, the lithologies and litho-facies identified at Pitarrilla were grouped into the rock packages that together make up the Pitarrilla Mine Sequence shown in Table 7-1.

Using the lithological groupings of the Pitarrilla Mine Sequence and Leapfrog software (Version 2.4.5.17), SSR generated a three-dimensional model of the main rock formations found at Pitarrilla. A cross-section of the Pitarrilla geological model is shown in Figure 7-3. Co-incident with the modeling of the Pitarrilla geological mine sequence, three dimensional (wire-frame) models of interpreted faults in the area of the deposit were created using Minesight software (version 7.0-3) and identified drillhole intercepts of faulted rock.

Given the importance of accounting for faults in pit-wall design, a seismic reflection geophysical survey was carried out in the area of the Pitarrilla deposit in order to validate the wire-frame fault models and to test for unrecognised structures. SSR contracted Frontier Geosciences Inc. to survey along five lines with a combined length of 7,024 m. The seismic work validated the position of interpreted faults and helped to locate two additional west dipping structures.

**Table 7-1: Simplified Pitarrilla Mine Sequence**

| Mine Sequence Units | Equivalent Mapped Rock Units ( Somers, 2010) | Attributes |
|---|---|---|
| Encino Rhyolite Dome | Encino Member, Casas Blancas Fm. | Rhyolite flow-dome; locally flow-banded, locally flow-brecciated |
| Lahar Breccia and Volcaniclastics | Sarape member and Javelina and Andesite members of the Casas Blancas Fm. | Volcaniclastic breccias and tuffs |
| Ignimbrites and Tuffs | Ignimbrite and tuffaceous units of Cardenas Fm | Mainly rhyolite pyroclastics and reworked tuffs with minor volcaniclastics |
| Volcaniclastics and Volcanics | Pitarrilla Fm tuffaceous, volcaniclastic and flow units | Andesitic to dacitic volcaniclastics, minor tuffs, along with single coherent flow unit minor carbonate component in upper horizon |
| Basal Conglomerate | Manto Rico member, Pitarrilla Fm | Polymictic conglomerate, limestone clasts |
| Cretaceous Sedimentary Rocks | Peña Ranch Fm, | Folded and fractured, marine clastic sedimentary rocks; siltstone, sandstone, pebble conglomerate and shale beds |
| Felsic Dykes and Sills | Felsic Intrusions | Quartz +/- feldspar porphyritic felsic sills and dykes intruding all other lithologies |
| Upper Andesite Sill | Upper Andesite Sill | Fine-grained andesite intruded subconformably at base of Casas Blancas Fm |
| Lower Andesite Sill | Lower Andesite Sill | Massive, fine-grained andesite intruded conformably within Lower member of Pitarrilla Fm |

## 7.5 PITARRILLA MINERALISATION MODEL

Ag-Zn-Pb mineralisation at Pitarrilla occurs as a vertically stacked mineralised system centered on rhyolitic dykes and sills that constitute the feeder system for an early Oligocene volcanic centre manifest by a rhyolitic dome. Mineralisation is interpreted to have occurred during or shortly after emplacement of this dome. Ag-Zn-Pb mineralisation is rooted in the basement Cretaceous sediments where it is represented by an aerially restricted but vertically extensive zone of disseminated and veinlet Ag-Zn-Pb (-Cu-As-Sb) sulphide-associated mineralisation. Overlying the Cretaceous basement, strata-bound massive replacement mineralisation occurs within a polymictic conglomerate at the Cretaceous-Eocene unconformity. The hypogene (fresh) or sulphide-associated mineralisation extends into the overlying Eocene to Oligocene, volcanic and volcaniclastic rocks as well as felsic and intermediate sills, where it grades into partly weathered, or transitional mineralisation, and a more laterally extensive zone of disseminated highly weathered, or oxide-associated, mineralisation.

Sulphide-associated mineralisation at Pitarilla was weathered under near surface oxidising conditions, resulting in the destruction of primary sulphide and sulphosalt minerals and liberation of ions into the weathering environment where they re-precipitated as secondary mineral phases. The destruction of pyrite resulted in the release of iron and sulphuric acid. The released iron was re-precipitated as iron oxide species including limonite and goethite. Argentiferous galena was broken down as a result of weathering and silver was liberated to re-form in minerals such as acanthite and silver halides (chlorargyrite, iodargyrite, bromargyrite; LeCouteur, 2006) which deposited along with the iron (and manganese) oxides thus producing oxide-associated mineralisation.

Typically, the oxide-associated mineralisation is accompanied by pervasive argillization of the originally feldspathic intrusive and volcanic host rocks. The felsic intrusive rocks and near surface dacitic volcaniclastics that make up the bulk of the mineralised rocks in the zones of oxide-associated silver mineralisation show evidence of moderate to strong acid-leaching and consequent mass reduction. Acid-leaching is inferred to have affected these rocks on the basis of their highly depleted levels of calcium, sodium, and magnesium as well as their highly porous and commonly ‘vuggy’ textures. The leaching is believed to have resulted from the acidification of weakly acidic oxidised meteoric waters as the weathering of pyrite resulted in the production of sulphuric acid.

Weathering of the host rocks and mineralisation was gradational, in places remnant sulphide species remained surrounded by minerals precipitated as a result of the oxidation process. Mixed oxide and sulphide mineral species form a material type that is called transitional mineralisation.

In summary, for metallurgical treatment purposes three main silver bearing material types are recognised at the Pitarrilla Ag-Pb-Zn deposit, these are called Oxide, Transitional, and Sulphide mineralisation. It is important to note that the total Mineral Resources and Mineral Reserves defined as part of the Pitarrilla Feasibility Study (M3, 2012) are formed by a combination of these three mineralisation types. The following sections provide an overview of the distribution and characteristics of the defined zones or domains of mineralisation.

The total extent of the oxide-associated mineralisation is considerable, about 1.9 km in the NNW-SSE direction and 2.9 km in the NE-SW direction. The six zones of oxide-associated mineralisation are, in chronological order of discovery, Cordon Colorado, Peña Dyke, Javelina Creek, Breccia Ridge, South Ridge and South Ridge East (Figure 7-4).

Based on host-rock lithologies and style of mineralisation, one transitional mineralised domain and three sulphide mineralised domain were outlined at depth beneath Breccia Ridge Zone, with a number of subdomains designated in each of these four domains. Each subdomain within the Transitional and Sulphide domains represents a separate Mineral Resource domain as defined in Section 14.3.3.


**Figure 7-4: Plan of Main Mineralised Zones that are Centered on Cerro La Pitarrilla with Drillhole Collars and Traces**

NAD27 UTM Zone13N. Source: M3, 2012

### 7.5.1 Cordon Colorado Zone

The Cordon Colorado Zone is relatively flat-lying and the mineralisation lies close to surface. The NE-SW axis of maximum length is approximately 575 m, with the NW-SE axis being about 400 m. The deposit ranges in thickness from 30 to 85 m, with the average being about 50 m. Disseminated oxide silver mineralisation is hosted entirely within a massive, fine-grained and weakly quartz feldspar-porphyritic felsic sill. Silver is the only metal of economic interest in this zone. Lead and zinc concentrations are typically low.

### 7.5.2 Peña Dyke Zone

The Peña Dyke Zone lies 500 m north of Cordon Colorado beneath a northwesterly trending ridge that extends from the western peak of Cerro La Pitarrilla. The length of the zone is approximately 500 m, with the width averaging 125 m. Mineralisation crops out; however, the silver-rich core of the deposit lies roughly 60 m below surface. The disseminated oxide silver mineralisation occurs within a weakly quartz feldspar-porphyritic felsic intrusive. Silver is the only metal of economic interest.

### 7.5.3 Javelina Creek Zone

In the Javelina Creek Zone, disseminated oxide silver mineralisation occurs in two separate sub-zones that are sub-conformable to the southwest dipping volcanic strata; the upper sub-zone is hosted by thinly bedded, well-stratified tuffs of the Javelina member of the Casas Blancas Formation, while the lower sub-zone is found in quartz crystal and pumice rich tuff of the Cardenas Formation. The oxide mineralisation crops-out, the total length of the combined zone is approximately 600 m NS, with the combined width averaging 300 m and an overall average thickness of 80 m.

### 7.5.4 Breccia Ridge Oxide and South Ridge Oxide Zones

Together, the Breccia Ridge Oxide and South Ridge Oxide zones extend in the NNW-SSE direction for approximately 1,300 m, with an average width in the WSW-ENE direction of approximately 600 m.

The cross-section of Figure 7-5 shows the complex of felsic dykes and sills that converge beneath the Encino flow dome at Breccia Ridge. It is predominantly in these intrusive rocks that the disseminated, oxide silver mineralisation occurs. Volcaniclastic rocks of the Pitarrilla Formation also contain oxide and transitional mineralisation, where these rocks are proximal to the contacts with the felsic dykes and sills. The oxide silver mineralisation of the Breccia Ridge Oxide Zone grades downwards into transitional mineralisation, and with increasing depth into true sulphide mineralisation.

The core of mineralisation within the South Ridge Oxide Zone is situated about 600 m south-southeast of the peak of Cerro La Pitarrilla Vertical thickness of the South Ridge Oxide Zone mineralisation varies considerably, from less than 20 m up to more than 100 m. Felsic dykes and sills are the most important host rock for mineralisation in the South Ridge Oxide Zone, with the lahar breccia and stratified tuff litho-facies of the Casas Blancas Formation also locally mineralised. Along the eastern part of South Ridge, disseminated, oxide silver mineralisation is hosted by crystal and pumice rich welded tuff of the Cardenas Formation and by volcaniclastics of the upper Pitarrilla Formation. Silver is the only metal of economic importance in the Breccia Ridge Oxide and South Ridge Oxide zones.

### 7.5.5 South Ridge East Oxide Zone

The South Ridge East Zone is situated immediately east of the South Ridge Zone. The two zones are connected by, low-grade oxide mineralisation and for resource estimation purposes the two zones are combined. This zone is best described as a strongly elongated and horizontally flattened ellipsoid that has its primary axis oriented NNW-SSE. The deposit is at least 700 m long and 75 m to 100 m thick. Disseminated, oxide-associated silver mineralisation, locally in high-grade concentrations, is found in crystal- and pumice-rich welded tuff of the Cardenas Formation and in volcaniclastics belonging to the upper section of the Pitarrilla Formation.

### 7.5.6 Breccia Ridge Transitional and Sulphide Domains

There are four main domains of transitional and sulphide silver mineralisation in the Pitarrilla Ag-Pb-Zn deposit, which together, define a vertically-stacked mineralised system that is centered on the cluster of felsic dykes and sills representing the feeder complex and vent area for the Encino rhyolitic flow dome (Figure 7-5 ). From highest to lowest in the system, the four domains consist of the AB domain (transitional), the Andesite (C) domain (sulphide), the Basal Conglomerate (D) domain (sulphide), and the Basement domain (sulphide), as discussed below. Figure 7-5 is a NE-SW cross-section through the Pitarrilla deposit showing the distribution of oxide disseminated, silver mineralisation, transitional mineralisation, and three types of sulphide Ag-Pb-Zn mineralisation.

#### 7.5.6.1 AB Domain (Transitional)

The AB Domain encompasses mineralisation with both oxide and sulphide characteristics, i.e., incomplete weathering of sulphide mineralisation. When the limits of the AB Domain are projected to surface it has a maximum known lateral extent of approximately 1,000 m in the NW-SE direction, extending from the northern boundary of Breccia Ridge to approximately 300 m north of the southern margin of South Ridge. It extends approximately 625 m laterally in the NE-SW direction. The top of the AB Domain ranges from approximately 20 m to 200 m below the surface. The contact between the base of the Breccia Ridge Oxide and South Ridge Oxide Zones and the top of the AB Domain is highly irregular and isolated pods of AB Zone transitional material are present within the oxide mineralisation near the surface. The base of the AB Domain is generally considered to be the upper contact of the Lower Andesite Sill which intrudes the lower strata of the Pitarrilla Formation (Figure 7-6). The interpretation of the AB Domain relies upon detailed work undertaken by SSR to understand the weathering of the deposit, which is discussed in detail in Section 7.6.1.

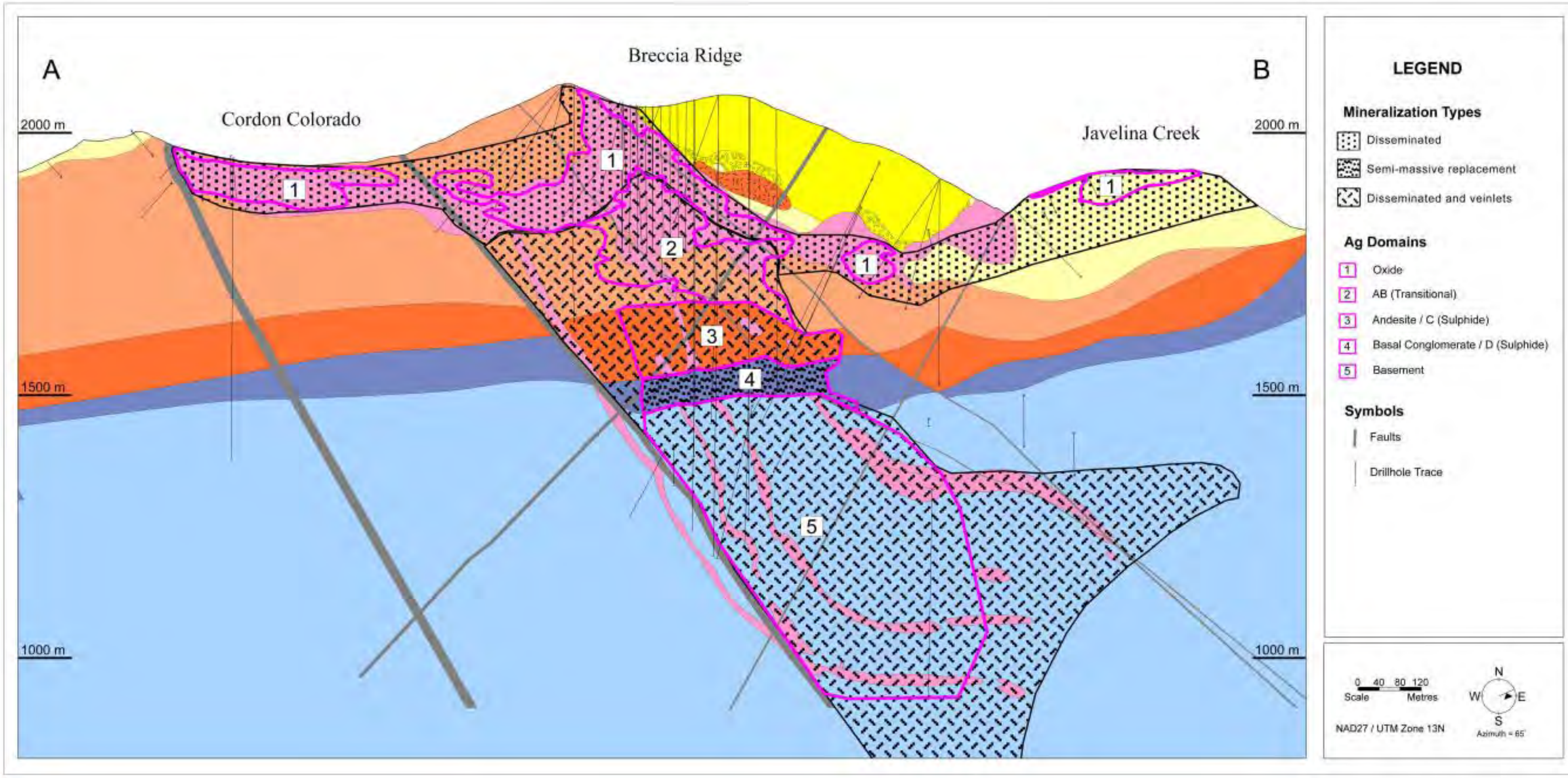



**Figure 7-5 Cross-Section of the Pitarrilla Ag-Pb-Zn Deposit – Zones of Mineralisation and Ag Domains**

(Refer to Figure 7-3for geology legend. Source: M3 2012)

#### 7.5.6.2 Andesite (C) Domain

Andesite-hosted mineralisation has been subdivided into four subdomains: the C, C-1, C-2 (NW) and C-2 (SE) subzones (Note: mineralised subzones correspond to resource domains described in Section 4.3). The mineralisation forming these subdomains generally consists of disseminated aggregates and stockwork veinlets of the same sulphide phases found elsewhere in the deposit. The exception is the C subzone, which in addition to having the disseminated and fracture-filling sulphides contains a sub-horizontal body of massive, relatively coarse-grained base metal sulphides, mainly pyrite, chalcopyrite, sphalerite, with lesser amounts of pyrrhotite, galena, arsenopyrite and tetrahedrite. Only minor amounts of hydrothermal gangue minerals, mostly quartz and calcite, are mixed with the sulphides. The massive sulphide mineralisation of the C subdomain, which can attain thicknesses of up to 5 m, has the appearance of a vein; however, it is also possible that it represents a body of pervasive sulphide replacement which was localised along the upper contact of the andesite sill. In terms of size, the C subdomain has lateral dimensions of 350 m in the NNW-SSE direction, about 400 m in the NE-SW direction, and reaches a maximum thickness of about 20 m. Beneath the C subzone and lying just above the lower contact of the andesite sill is the C-1 subdomain. This subdomain of disseminated and veinlet sulphide mineralisation is 15 to 40 m thick and has maximum lateral dimensions of 520 m and 420 m in the NNW-SSE and NE-SW directions, respectively. Lying between the C and C-1 subdomains within the central part of the andesite sill are the C-2 (NW) and C-2 (SE) subdomains. Although they are contiguous, the two C-2 subdomains have been distinguished on the basis of metal grades and dip orientations; the C-2 (NW) subdomain contains lower grade mineralisation and lies sub-horizontally, while the C-2 (SE) subdomain is richer in Ag, Zn and Pb, and dips moderately to the southeast. Viewed together, the C-2 subdomains have a NNW-SSE dimension of 800 m and a 600 m extent in the NE-SW direction.

#### 7.5.6.3 Basal Conglomerate (D) Domain

The Basal Conglomerate or D domain is probably the most important mineralised domain at Pitarrilla, as it contains some of the highest grade silver bearing base metal sulphide mineralisation presently known on the property. It is stratabound within the Manto Rico and is characterised by replacement style mineralisation. The D domain generally has a thickness of between 15 to 25 m but attains a maximum thickness of 65 m in its central part. Drillhole intersections of this domain have determined that the maximum lateral dimension is about 800 m. Sulphide phases found in the Basal Conglomerate (D) Domain, in order of abundance; include pyrite, sphalerite, galena, arsenopyrite, pyrrhotite, stibnite and tetrahedrite.

#### 7.5.6.4 Basement Domain

Multiple, lenticular bodies of disseminated and stockwork veinlet sulphide mineralisation have been delineated below the Cretaceous-Tertiary unconformity, which represent the deepest part of the Pitarrilla Ag-Pb-Zn ore system and together constitute the Basement domain. They are sub-parallel and with an average strike of 330° and dip 55° to 70° to the east. Weakly porphyritic felsic dykes typically separate the mineralised lenses of the Basement Zone, with the dykes having strike and dip orientations that mimic those of the mineralised bodies. Silver-bearing, disseminated and fracture-controlled sulphides are found within the felsic dykes, typically close to their contacts, but the bulk of the Basement Zone mineralisation (>95%) is hosted by the thinly inter-bedded siltstone, sandstone, and minor pebble conglomerate strata of the Peña Ranch Formation. Disseminated mineralisation tends to favour the more porous litho-facies such as pebble conglomerate and sandstone, while the beds of siltstone and shale commonly contain sulphide veinlets millimetres to centimetres thick, along with volumetrically restricted zones of mineralised hydrothermal breccia. Sulphide phases found in the Basement domain, in order of abundance, include pyrite, sphalerite, galena, arsenopyrite, pyrrhotite, stibnite and tetrahedrite. The veinlets contain relatively minor amounts of gangue minerals, mostly quartz, calcite and chlorite.

Five separate lenses of mineralisation were outlined as sub-domains within the Basement domain, and these represent different resource domains in the Cretaceous basement rocks. From west to east, they have been designated as the G, F, E, H and I subzones. The G, F, and E subdomains have comparable strike lengths, 450 to 520 m, whereas the H and I subdomains are 100 to 150 m in length. Down-dip extents range from 690 m for the F subdomain to about 250 m for the I subdomains. Maximum thicknesses of the subdomains range from 60 to about 100 m, with only a few tens of metres separating adjacent subzones. The down-dip limits to the mineralisation of the Basement domain have not been determined due to the considerable depths below surface that would need to be drilled.

### 7.6 ALTERATION

In order to understand the spatial distribution of hydrothermal alteration at Pitarrilla, an alteration study was conducted in 2012. A hand portable TerraSpec mineral spectrometer was used to identify the alteration minerals present in drillcore samples for a selected 26 drillholes distributed on NS and EW sections throughout the deposit, at a spacing of approximately one TerraSpec measurement every 20 m down the lengths of the drillholes, with 801 measurements in total. The results of this study enabled the definition of five main types of hydrothermal alteration and determined their distribution in relation to zones of silver and base metal mineralisation. Existing PIMA spectrographic analyses from the PhD study of C. Somers (Somers, 2010) were also incorporated into this study.

The TerraSpec and PIMA analyses of the drillcores resulted in the definition of five alteration types, which from oldest to youngest have been designated as the quartz-tourmaline (Qz-To), smectite-chlorite (Smec-Chl), illite-chlorite (Ill-Chl), siderite-chlorite (Sid-Chl), and buddingtonite ± kaolinite (Budd±Kaol) alteration types. These alteration types are characterised by particular suites of secondary minerals, with each assemblage dominated by two phases which define the respective alteration type. Some of the key aspects of the distribution of hydrothermal alteration at the Pitarrilla Project are:

- Quartz-Tourmaline: Designated as the Qz-To type, occurred prior to the main Ag-Pb-Zn mineralisation event and the emplacement of the felsic dykes. The quartz-tourmaline alteration is apparently associated with the formation of a phreatomagmatic breccia at core of the Pitarrilla magmatic hydrothermal system.

- Smectite-Chlorite: This type of alteration is found at the outer edges of the deposit in weakly or non-mineralised rocks. The rocks showing smectite-chlorite alteration are composed largely of low-temperature clay minerals, predominantly layered smectite-illite and montmorillonite, along with chlorite, minor quartz and accessory iron oxides.

- Illite-Chlorite: This type of alteration is the dominant form of alteration throughout the deposit, affecting most of the lithologies hosting both the oxide and sulphide types of silver mineralisation.

- Siderite-Chlorite: In the deeper levels of the deposit, especially within the Basal Conglomerate, iron carbonate and chlorite alteration, defined as the Sid-Chl alteration type, accompanied the deposition of iron and base metal sulphides.

- Buddingtonite-Kaolinite: Felsic dykes and sills and the felsic tuffs in the upper parts of the volcanic pile contain the secondary minerals kaolinite, illite, ammonium illite, dickite and ammonium feldspar (buddingtonite).

### 7.6.1 State of Oxidation

Weathering of host rocks at the Pitarrilla Ag-Pb-Zn deposit has resulted in the partial to complete destruction of primary sulphide and sulphosalt minerals as well as the hydrolysis, hydration, and oxidation of the main rock-forming minerals. Through these processes, metals of economic interest were variably mobilised and re-distributed into secondary minerals that formed under supergene conditions.

The weathering of the primary rock-forming minerals such as plagioclase, K-feldspar and amphibole to assemblages of clay minerals and chlorite generally results in the physical weakening of the rock mass. The intensity of weathering that a mineralised rock displays may be reflected in the degree or efficiency of metal extraction during ore processing and in the mechanical strength of the rock. Consequently, determining the degree to which a rock is weathered, is important in the design of an open-pit mine (for establishing cost-effective and safe pit-wall angles) and in the selection of the metallurgical processes that are to be employed by the planned operation.

Drillhole intervals were assigned state of oxidation values of 5 through 0 on a six point scale (Table 7-2) to denote the intensity of weathering. The final product of the semi-quantitative rock weathering study was used as the input for the estimation of oxidation values within the resource block model. Refer to Section 14.6.2 for a discussion of the estimation procedure.

Figure 7-6 shows a typical cross section with drillholes colour coded by state of oxidation state and interpreted boundaries for transitional (A-B domain) and oxide Ag mineralisation at a 20 g/t cutoff grade.

**Table 7-2: Pitarrilla State of Oxidation Logging Scale Used to Record Intensities of Weathering Shown by Variably Oxidised Rocks**

| Code | Label | Rock Description |
|---|---|---|
| 5 | Extremely Weathered | Decomposed; discolored; resembles a soil; original rock textures may be preserved. |
| 4 | Highly Weathered | Discoloration throughout; rock is friable but hard; cores remain intact; rock textures may be preserved. |
| 3 | Moderately Weathered | Discoloration extends from fractures generally throughout the rock; the rock is not friable; rock textures are preserved. |
| 2 | Slightly Weathered | Discoloration extends out from fractures into the rock; discoloration affects less than 40% of rock, or very weak pervasive iron staining. |
| 1 | Fresh Jointed | Discoloration or oxidation is limited to surfaces of, or for short distances from fractures; less than 10% of rock is discolored; rock rings when struck with hammer. |
| 0 | Fresh | No discoloration; rock rings when struck with hammer. |

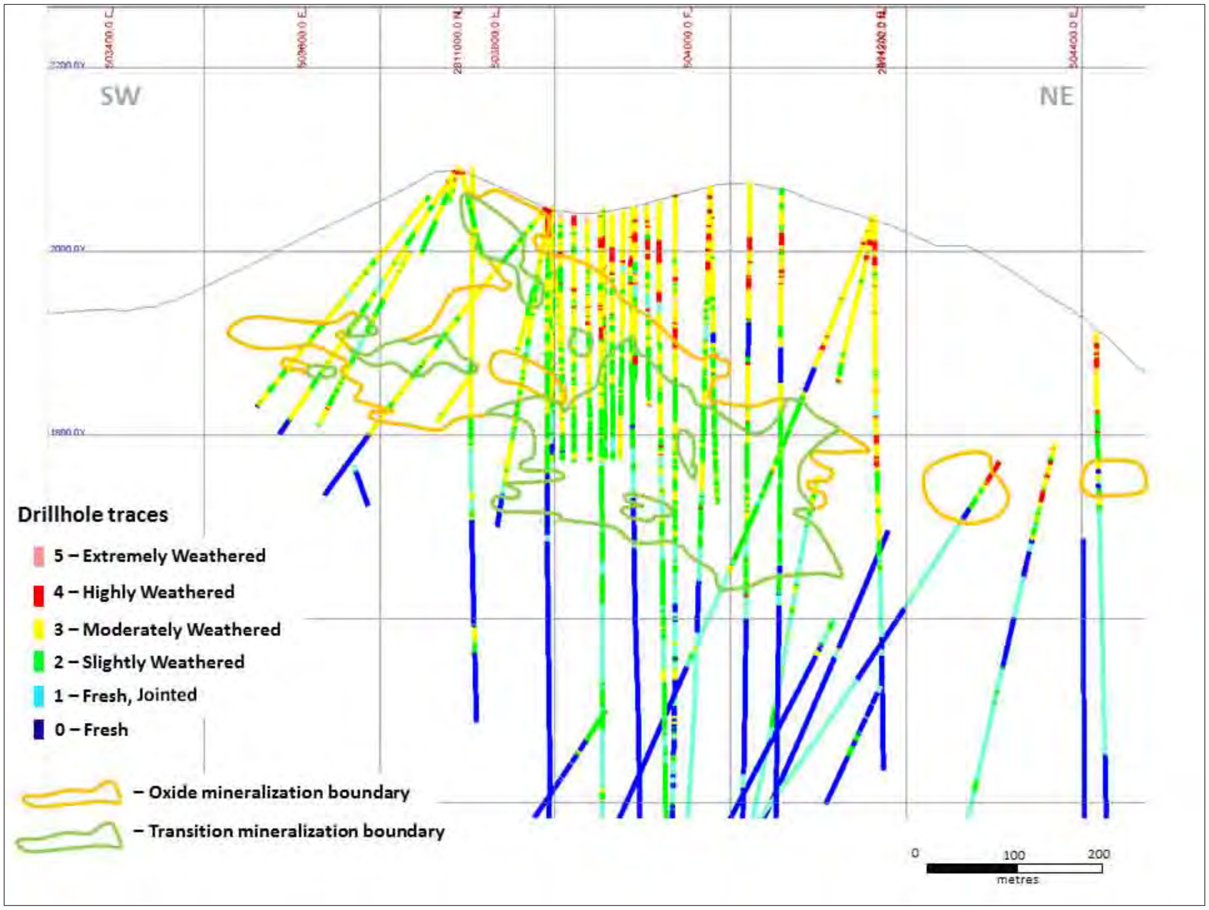



**Figure 7-6: ENE-WSW Cross-section through the Breccia Ridge Oxide Zone, with the Traces of Drillholes Colour-coded by Oxide State**

Source: M3, 2012 Note: the small zones of transitional material located within the oxide boundary, with transitional material near the surface.

# 8 DEPOSIT TYPE

The Property is centrally located within the *Central Mexican Silver Belt*, which is defined by numerous Ag-Pb-Zn (±Au-±Cu) deposits that are classified as intermediate sulphidation epithermal deposits (Hedenquist *et al*., 2000). This includes the world class silver ore systems at Fresnillo, Zacatecas, and Guanajuato. These Mexican polymetallic silver deposits consist mainly of vein systems that occupy faults and major fractures affecting Mesozoic sedimentary and marine volcanic rocks and to a lesser degree unconformably overlying Tertiary volcaniclastics and pyroclastics. The sequences of Tertiary volcanic and volcaniclastic formations found in the majority of the historic Mexican silver districts are significantly less voluminous, i.e., less well preserved than the Eocene-Oligocene succession that is mapped at Pitarrilla.

The Mexican intermediate sulphidation vein deposits are characterised by economically significant concentrations of Ag, Zn, Pb, Au, and occasionally Cu, with these metals occurring in base metal sulphides, accessory amounts of acanthite-argentite, freiburgite, pyrargyrite, tetrahedrite-tennantite, trace amounts of electrum and a variety of Ag-Pb-As-Sb-Cu sulphosalts. Where the hypogene mineralisation has been weathered, the sulphides and sulphosalts are replaced by iron oxides, which are accompanied by minor amounts of various Zn, Cu and Pb carbonates, hydroxides, and sulphates along with acanthite, silver halides and trace amounts of native silver and gold. Gangue minerals in the veins include, in order of decreasing abundance, quartz, chalcedony, calcite, pyrite, adularia, barite, fluorite, Ca-Mg-Mn-Fe carbonates (e.g. rhodochrosite, siderite), amethyst, sericite, and chlorite. Characteristic vein textures include multiple stages of brecciation, colloform banding and crustiform crystallization. Hydrothermal alteration of wall-rocks is generally restricted to vein halos a few metres in width, where silicification occurs immediately next to the veins and grades outwards into an assemblage of sericite-illite-kaolinite, then into illite-smectite-montmorillonite and finally into a low-temperature alteration assemblage dominated by smectite-chlorite. Larger veins have kilometres of strike-length, are several metres wide and have vertical extents in the hundreds of metres, with a few cases of veins extending more than one kilometre below surface. Vertical metal zonation is a common feature of larger veins, with three principal mineralisation zones, from shallowest to deepest, being defined by the following metal suites: Ag-(Au)-As-Sb-Hg, Ag-Pb-Zn-(Cu-Au), and Pb-Zn-(Ag). Age dating and lead isotope studies indicate that the Ag-Pb-Zn-(Au-Cu) vein deposits of the *Central Mexican Silver Belt* are mainly Tertiary in age (36 to 28 Ma), and are genetically related to rhyolitic magmatism, which in the mineral districts, is manifested as relatively small porphyry stocks, dyke systems and/or flow-dome complexes.

Superficially, the Pitarrilla Ag-Pb-Zn mineralisation does not display features generally considered to be characteristic of intermediate sulphidation epithermal deposits, especially considering the occurrences of near-surface oxide silver mineralisation. However, when the different forms of sulphide mineralisation found in the Lower Andesite Sill, Basal Conglomerate, and Basement Zones of the Pitarrilla deposit are examined, it is evident that these bodies of sulphide silver mineralisation, do in fact, share mineralogical features with the major polymetallic vein deposits in Mexico and elsewhere in the world. Specifically, the sulphide mineral suite of pyrite/marcasite-sphalerite-galena-chalcopyrite-pyrrhotite-arsenopyrite-tetrahedrite (-freiburgite), that is found in all of the hypogene mineral zones at Pitarrilla, is fairly diagnostic of the intermediate sulphidation subclass of epithermal deposits (Hedenquist *et al*., 2000). The fact that the mineral resources and reserves at Pitarrilla are not defined on major veins, except perhaps for the C Zone within the Lower Andesite Sill, does not necessarily preclude the Pitarrilla deposit from being classified as an intermediate sulphidation type of epithermal deposit, since there are analogies of the Pitarrilla mineralised zones in a number of deposits within the *Central Mexican Silver Belt*. For example, the Ag-Pb-Zn ore that was mined at the Real de Angeles open-pit mine in Zacatecas State (Pearson *et al*., 1988) is quite similar, in terms of host rock and styles of mineralisation, to the mineralisation forming the resource domains (subzones) defined in the Basement Zone at Pitarrilla. As well, the replacement style sulphide mineralisation of the Basal Conglomerate (D) Zone is presumed to be comparable to the manto mineralisation that was historically mined at Fresnillo (Ruvalcaba-Ruiz and Thompson, 1988). Furthermore, while recognizing that hydrothermal phases such as quartz, calcite, barite, and fluorite, which form gangue minerals in most Mexican epithermal veins, are only minor to accessory components in the sulphide ores at Pitarrilla, it should be noted that unmineralised calcite, barite, and fluorite veins do exist on the Property, even proximal to zones of silver mineralisation. Thus, while not representing a "classic" example of an intermediate sulphidation epithermal mineral system, the zones of sulphide mineralisation at Pitarrilla do have a mineralogical signature that is consistent with these zones belonging to this subclass of epithermal deposit. Moreover, the overall geological setting at Pitarrilla and the perceived genetic association of the Ag-Pb-Zn mineralisation with middle Tertiary felsic magmatism are again consistent with the deposit being classified as an intermediate sulphidation epithermal deposit. A schematic geological cross-section is included as Figure 8-1, illustrating various settings of intermediate sulphidation epithermal deposits found in the *Central Mexican Silver Belt*.

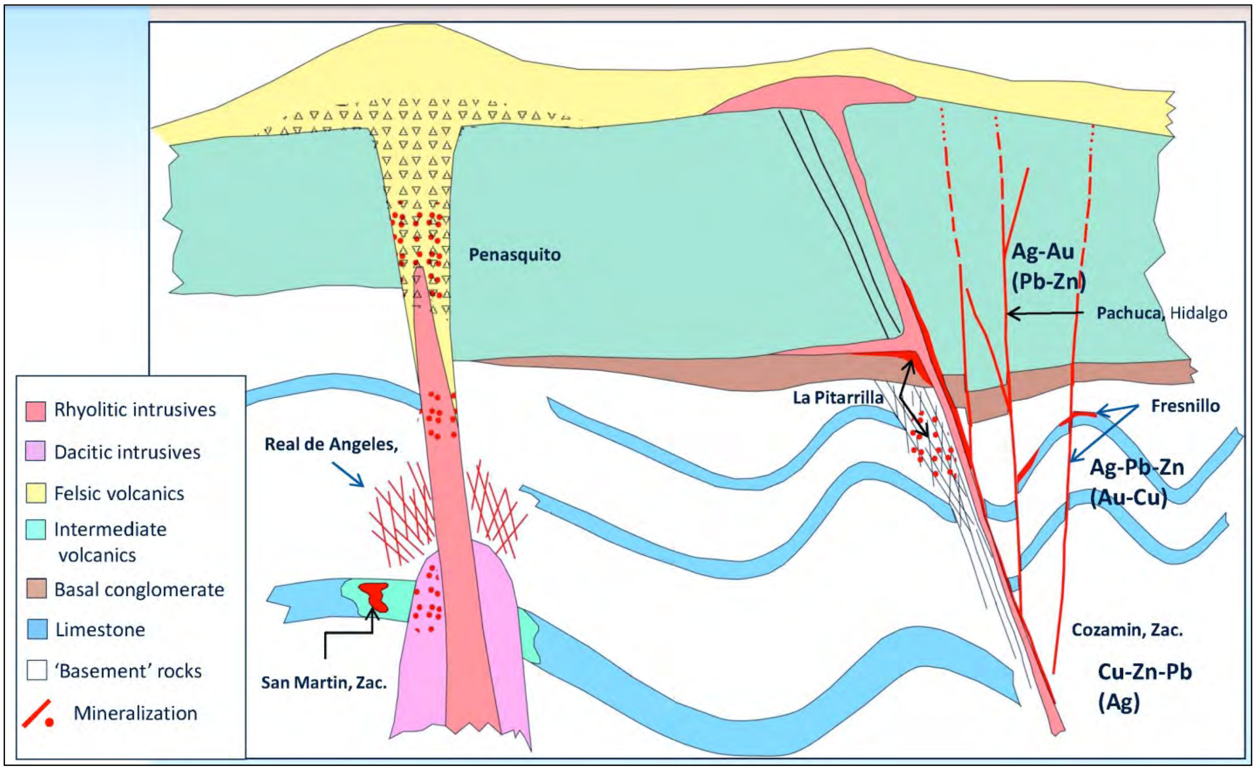



**Figure 8-1: Schematic Geological Cross-section Showing Geological Settings of Some Mexican Silver Deposits, Including the Pitarrilla Ag-Pb-Zn Mineralisation.**

Source: SSR 2011.

# 9 EXPLORATION

## 9.1 PAST EXPLORATION (2002 TO 2012)

For a discussion of exploration prior to Silver Standard's acquisition of the property in 2002, refer to Section 6.1.

### 9.1.1 Rock Chip Sampling Programs

In 2002, Silver Standard contracted F. Hillemeyer and P. Durning of LCI to acquire mineral properties in Mexico that exhibited good exploration potential for silver. One of the first areas LCI recommended for staking was the ground covered by the Pitarrilla claim group previously held and explored by LCI as described in Section 6.1. Between November 2002 and March 2003, a total of 12 concessions covering 136,191 hectares were claimed by Exploминerals, S.A. de C.V. on behalf of Silver Standard. Beginning in late 2002 and continuing until May 2003, Silver Standard, using the services of Exploминerals, carried out extensive rock-chip sampling over the slopes of Cerro La Pitarrilla. Silver anomalies were identified on the western slope of the hill (Cordon Colorado prospect) and this became Silver Standard's first drilling target defined on the Property.

Beginning in 2002, several programs of rock-chip sampling were completed over the core of the property, where multiple zones of silver mineralisation eventually came to be outlined. In the Cordon Colorado and Javelina Creek Zones, chip samples were collected along a grid pattern with an approximate sample spacing of 20 m. At each sampling location, samples were collected over an area approximately 4 $m^2$ in size and comprised chips totaling between 1 to 2 kg. In the Peña Dyke Zone, chip samples were taken in the same manner, but were not taken along a systematic grid pattern. In the Breccia Ridge Zone, chip sampling was focused along topographic contours around the exposed rhyolite dome (Burk, *pers. comm.*, 2012). The quality of the sampling was considered to be sufficient for target delineation, but chip samples are not considered truly representative due to issues with sample delimitation and extraction. The chip samples collected during these programs were not used as part of the December 4, 2012 Mineral Resource estimation. In July 2003, Silver Standard tested the Casas Blancas ASTER anomaly identified by Durning and Hillemeyer (2003) approximately three km to the southwest of Cerro La Pitarrilla by taking five rock-chip and eight stream-sediment samples.

In 2004, road-cut chip sampling, along with additional surface chip sampling, was completed along the La Colorado area of the property, to the northwest of the Cordon Colorado Zone. The surface samples were vertical channels cut at intervals along the northwest face of the ridge (McCrea, 2007). More than 5,500 rock-chip samples were collected and geochemically analyzed. The bulk of these samples were analyzed by ALS Chemex Laboratories in North Vancouver, Canada, with sample preparation and some assaying done at this laboratory's sample preparation facilities in Chihuahua, Mexico. The majority of the surface rock samples were analyzed for concentrations of 31 trace and minor elements using the inductively coupled plasma mass spectrometry ("ICP-MS") analytical method. Samples that yielded silver values greater than 100 ppm were re-assayed using the fire assay plus atomic absorption spectrometry (“AAS”) method. Gold analyses were not carried out on a large percentage of the samples as it was determined early on that the mineral system at Pitarrilla lacked economically significant gold mineralisation. When the analytical results of geochemical sampling programs were compiled and plotted on property maps, areas of rock exposures with anomalous concentrations of silver, arsenic, antimony, lead and zinc were outlined, with these trace metal ‘anomalies’ representing exploration targets that were eventually drill-tested, resulting in the discovery of the five zones of oxide silver mineralisation that form the upper part of the Pitarrilla deposit.

### 9.1.2 Geophysical Surveys

A number of geophysical surveys were completed on and over the Property, although none of these surveys were instrumental in the discovery of the deposit. In 2007, a helicopter-facilitated radiometrics (K, Th, U and total gamma count) plus magnetics survey was undertaken by Servicio Geológico Mexicano (“SGM”). The area covered by the survey extended eastwards beyond the boundary of the Pitarrilla claim group.

In 2010, Zonge Engineering Inc. of Tucson, USA., performed ground-based geophysical surveys in NW to NE trending lines over an area of approximately two km NW and three km NE extending NW from a central point on Cerro La Pitarrilla. Zonge collected induced polarization chargeability and resistivity responses as well as magnetotelluric (“NSAMT”), magnetic and gravimetric data. These geophysical surveys failed to discover any previously undefined zones of mineralisation.

A seismic reflection study, conducted by Frontier Geosciences Inc. of North Vancouver, Canada, was completed in 2012 in order to enhance the understanding of the structural geology of the Pitarrilla deposit. This involved seismic reflection data collection and interpretation, with the objective of delineating major fault planes that cut through and locally displace the Pitarrilla stratigraphy. This geophysical survey was employed as an aid to identify faults with potentially major influences on mine design and future pit stability. The seismic survey consisted of 7.024 linear km of surveying and was conducted along five survey lines. The area covered by this work is represented in Figure 9-1. The methodology employed, and quality of the geophysical data obtained from the survey are excerpted from the report completed by Frontier (2012).

> “*The seismic reflection investigation was carried out with three Geometrics Geodes, 24 channel signal enhancement seismographs, and Mark Products Ltd. 4.5 Hz geophones. Energy input was provided by small dynamite charges. In this survey, an ‘at end’ configuration was used with the energy source located at the end of an array of 48 geophone receivers. This receiver array spanned a survey line length of 235 metres and captured a broad spatial range of energy reflected from the horizons at depth. The survey procedure entailed collection of a 48 geophone record, then advancing the energy source 5 metres down the survey line and repeating*

*the discharge and record process. This method, known as the common mid-point gather (CMP) technique, provides a very high degree of redundancy of sampling of the energy received from a given reflector at depth. The redundancy is used during the data processing procedure to develop an image of the subsurface reflectors of high fidelity. The seismic data acquired in this survey was generally of "good to excellent quality*".

Cross-section seismic profiles were generated and interpreted (Frontier, 2012). Cliff lines prevented the continuity of two of the seismic traverses. As such, the resulting seismic profiles have gaps, however, the application of the seismic survey data to the geological model of the Pitarrilla Project confirmed the position of faults previously recognised and helped define faults in another key orientation, which were difficult to recognize due to the spatial arrangement of drillholes that were optimised for intersecting mineralisation.

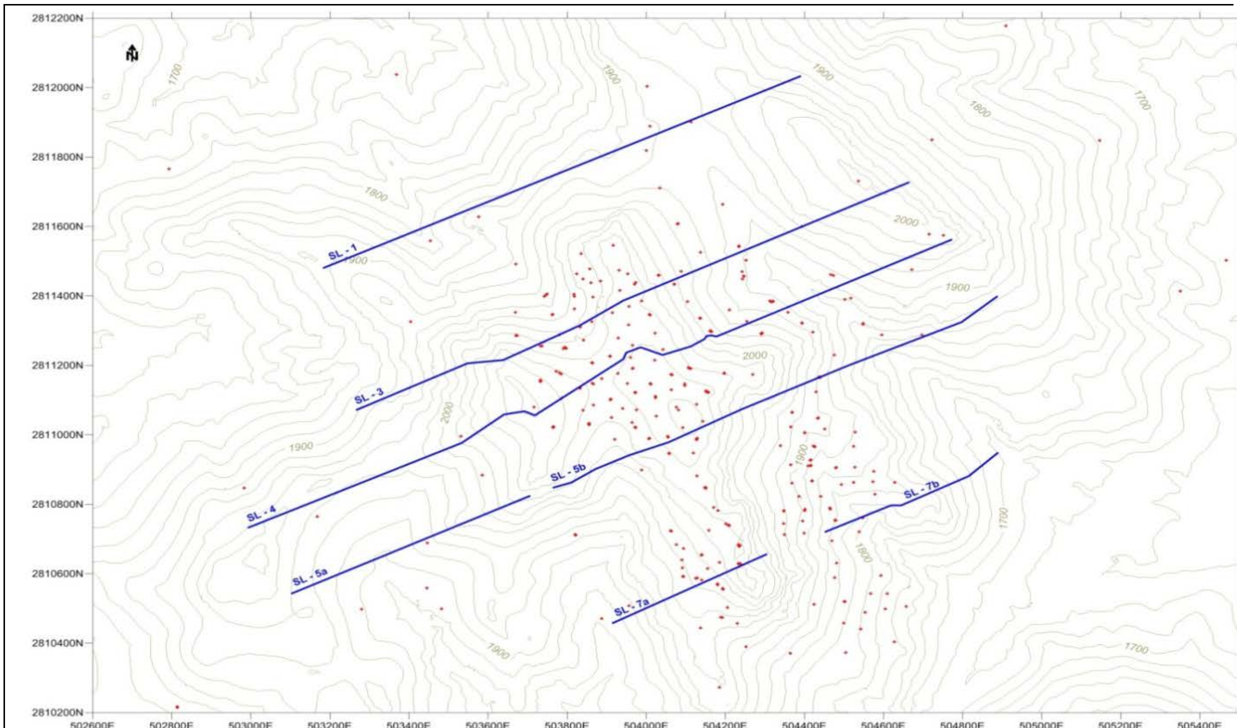



**Figure 9-1: Plan Map Showing Locations of Seismic Reflection Lines** **Note:**

Seismic reflection lines (blue), as surveyed by Frontier Geosciences; topographic contours (grey lines), and drillhole collar locations (red dots). Source: Frontier Geosciences, 2012. NAD 27 UTM Zone13N

In order to understand the spatial distribution of hydrothermal alteration at Pitarrilla, an alteration study was conducted in 2012. A hand portable TerraSpec mineral spectrometer was used to identify the alteration minerals present in drillcore from 26 drillholes chosen to be spatially representative along NS and EW sections throughout the deposit, at a spacing of approximately one TerraSpec measurement every 20 m down the lengths of the drillholes, with 801 measurements in total. Existing PIMA spectrographic analyses from the PhD study of C. Somers (Somers, 2010) were also incorporated into this study. The results of this study enabled the definition of five main types of hydrothermal alteration and determined their distribution in relation to zones of silver and base metal mineralisation.

By far the greatest amount of exploration-related data has come from the campaigns of reverse circulation and diamond drilling that was completed by Silver Standard on the property between September 2003 and July 2012. This drilling information is reported in detail in Section 10.

# 10 DRILLING

Monarch Resources de Mexico, S.A. de C.V. completed a Phase I drilling program on the Fluorite Mine Target in 1996, including 22 RC drillholes totalling 2,842 m. The drilling was on the Property, but not in the area of the current Mineral Resource.

The greatest amount of exploration-related data has come from the several campaigns of reverse circulation and diamond drilling completed by Silver Standard on the Property between September 2003 and July 2012.

From September 2003 until October 2005, 186 reverse circulation holes with a combined length of 20,619 m were drilled on the Property. The RC drillholes targeted oxide mineralisation in the Cordon Colorado, Peña Dyke, and Javelina Creek Zones. Between 2005 and July 2012, 428 diamond drillholes were drilled for exploration and resource infill purposes, with a total of 183,358 m being completed. The majority of the drillcore was of HQ diameter, though core samples from depths below surface greater than about 450 m were generally of NQ diameter. To provide a sufficient amount of core from different types of mineralisation for metallurgical testing, nine drillholes of HQ diameter were cored into the deposit in 2008 for a total of 6,126 m. An additional four holes of PQ diameter were drilled into four of the five zones of oxide silver mineralisation to obtain core samples for communition tests. In the area of the deposit, 31 drillholes (including re-drills), totalling 12,834 m, were drilled for mining-related geotechnical information between 2010 and 2012. Condemnation, water well, piezometer, and short geotechnical holes drilled for the investigation of foundations for site facilities were also completed during the history of the project (Table 10-1).

Most recently, during May and June of 2012, 33 closely-spaced diamond drillholes totaling 8,914 m were completed as part of a study to investigate the short distance variability of oxide and transitional silver mineralisation in the upper 200-250 m of the Pitarrilla deposit. These holes were drilled along three control lines, two oriented ENE-WSW with the third line crossing the other two lines perpendicular to them.

The orientation of drillholes varied in order to drill perpendicular to the interpreted orientation of the mineralised bodies. The dips of all drillholes were between 45° and 90°. In the Breccia Ridge Zone, drillholes were generally oriented vertically or at azimuths of 240° dipping at an average of 55°. In the South Ridge Zone, the drillholes were oriented at 100° and 274° with dips averaging 60°. In the Peña Dyke Zone, drillholes were drilled at azimuths of 200° and 025° degrees with dips at 60°. In the Cordon Colorado and Javelina Creek Zones, there were no preferred drillhole orientations.

## 10.1 DRILLHOLE COLLAR SURVEY BY LICENSED SURVEYOR

The positions of all drillhole collars were surveyed in the co-ordinate system NAD 27 UTM Zone 13N, by licensed surveyors employed by SSR, using differential GPS. The survey co-ordinates generated were added to the electronic data files for the Pitarrilla Project.

## 10.2 DOWNHOLE SURVEYS

All diamond drillholes were surveyed using downhole survey tools at 50 m intervals downhole, where it was possible to do so. The downhole survey information was passed from the drilling contractor, Major Drilling, to the geologists daily for incorporation into electronic data files.

## 10.3 DRILLING CONTRACTOR

Major Drilling was the drilling contractor used for all drilling on the Property. Major Drilling are independent of SSR.

## 10.4 DRILL COLLAR MONUMENTS

Concrete monuments with the drillhole identification and co-ordinates in NAD 27 UTM Zone 13N, were placed around the drillhole collar of each hole drilled.

## 10.5 DRILLING PLANS AND DRILL SECTIONS THROUGH PITARRILLA

A plan view of the Pitarrilla drilling is shown overlaying satellite topography imagery in Figure 10-1. Figure 10-2 shows the position of typical drill section lines through the deposit, and Figure 10-3, Figure 10-4, and Figure 10-5 illustrate typical sections through the deposit.

### Table 10-1: Pitarrilla Drilling (Entire Project)

| Drilling Method | Reason | Planned By | Prefix | No. Holes | Metres Drilled | Years |
|---|---|---|---|---|---|---|
| Reverse Circulation (RC) | Monarch | Monarch | BDA | 22 | 2,842 | 1996 |
| Reverse Circulation | Exploration | SSR | RC | 186 | 20,619 | 2003 - 2005 |
| Diamond | Exploration/Resource | SSR | DDH | 428 | 183,358 | 2005 - 2012 |
| Diamond | Statistical | SSR | DDH | 33 | 8,914 | 2012 |
| Diamond | Condemnation | SSR | DDH | 33 | 14,840 | 2008 |
| Diamond | Metallurgy | SSR | DDH-M | 9 | 6,126 | 2008 |
| Diamond Triple Tube | Geotechnical | KPL | BPG | 31 (incl. 4 redrills) | 12,834 | 2010 - 2012 |
| Diamond and RC | Piezometers | KPL | PZ | 11 | 213 | 2007 - 2008 |
| Diamond | Piezometers | KPL | MW08 | 3 | 73 | 2008 |
| Diamond Triple Tube | Geotechnical Site Construction | KPL | DH | 18 | 582 | 2008 |
| RC/Tricone | Water Wells | KPL | WW | 16 | 2,644 | 2008 |
| Diamond | Tailing Site Condemnation | MWH | BPT | 17 | 1,120 | 2010 |
| Diamond | Water Wells | MWH | WW10 | 6 | 2,486 | 2010 |
| Diamond | Monitoring Well | MWH | MW10 | 4 | 164 | 2010 |
| Diamond | Geotechnical Site Construction | MWH | BPB | 5 | 61 | 2010 |
| Diamond | Tailing Site Geotechnical | MWH | BPP | 3 | 38 | 2010 |
| Diamond | Comminution | SSR | BPC | 4 | 541 | 2011 |
| Diamond  Triple Tube | Plant and Waste Dump Geotechnical | KPL | BPG | 12 | 231 | 2012 |
| Diamond | Tailing Site Geotechnical | Tierra Group | TGI-12 | 10 | 445 | 2012 |
| **Total (excluding Monarch)** | | | | **829** | **255,287** | |

Note: values may not sum correctlydue to rounding

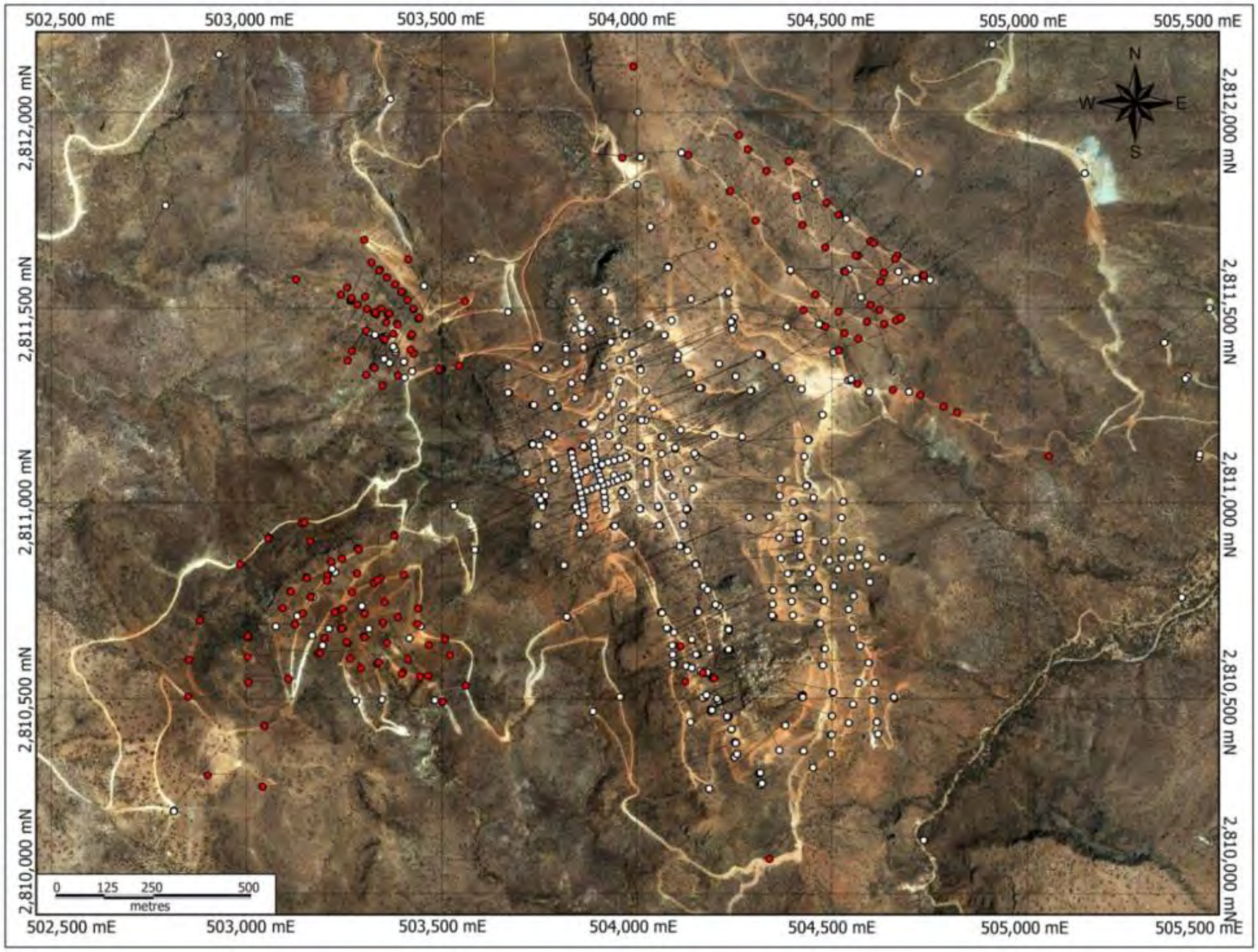



**Figure 10-1: Drillhole Location Plan Pitarrilla**

Notes.Red Dots Represent RC collars; White Dots Represent Diamond Drillhole Collars.

Source: SSR, 2012

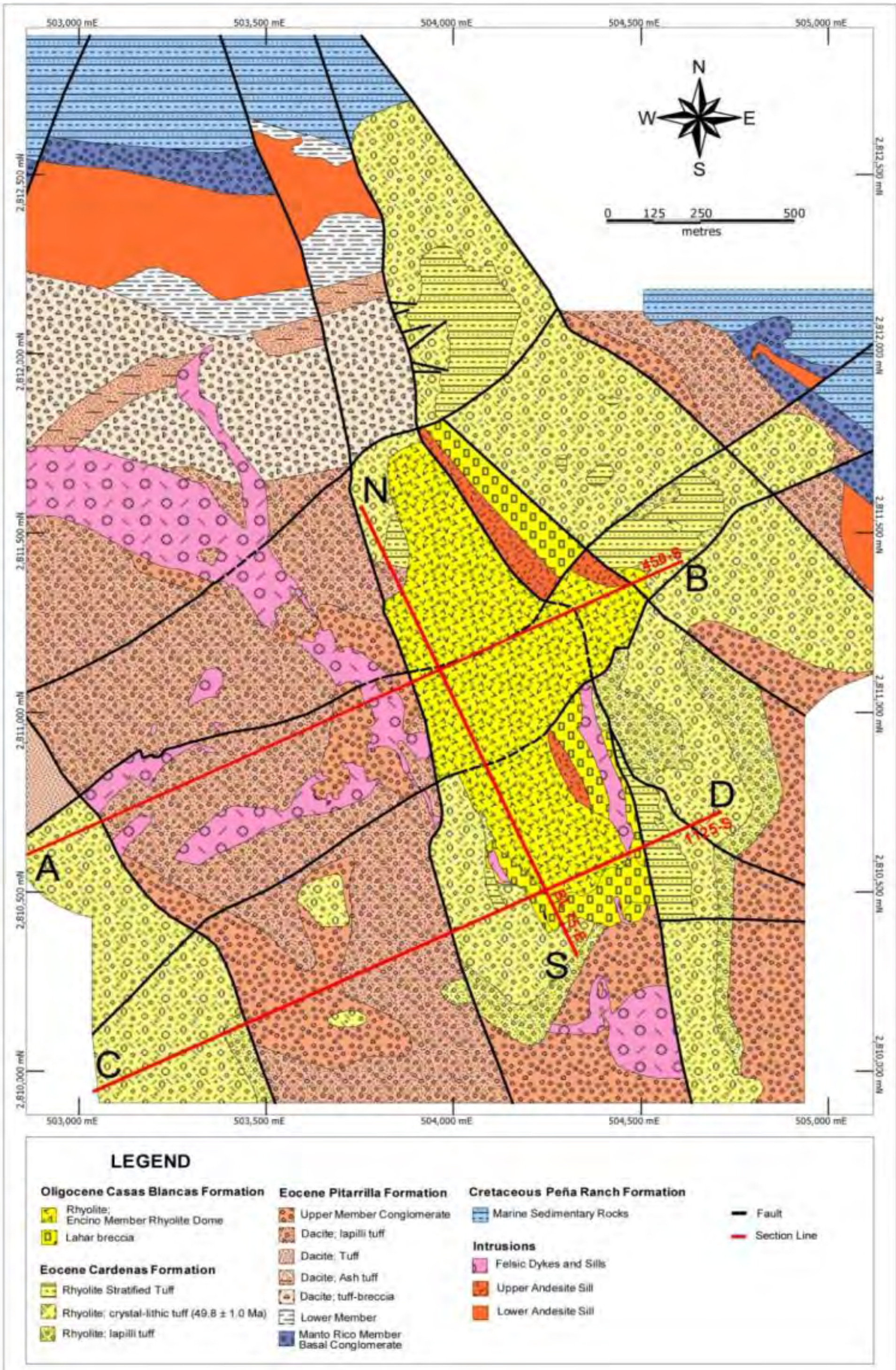



**Figure 10-2: Pitarrilla Geologic Map: Locations of Section Lines in Figs. 10-3, 10-4, 10-5**

Source: SSR, 2012

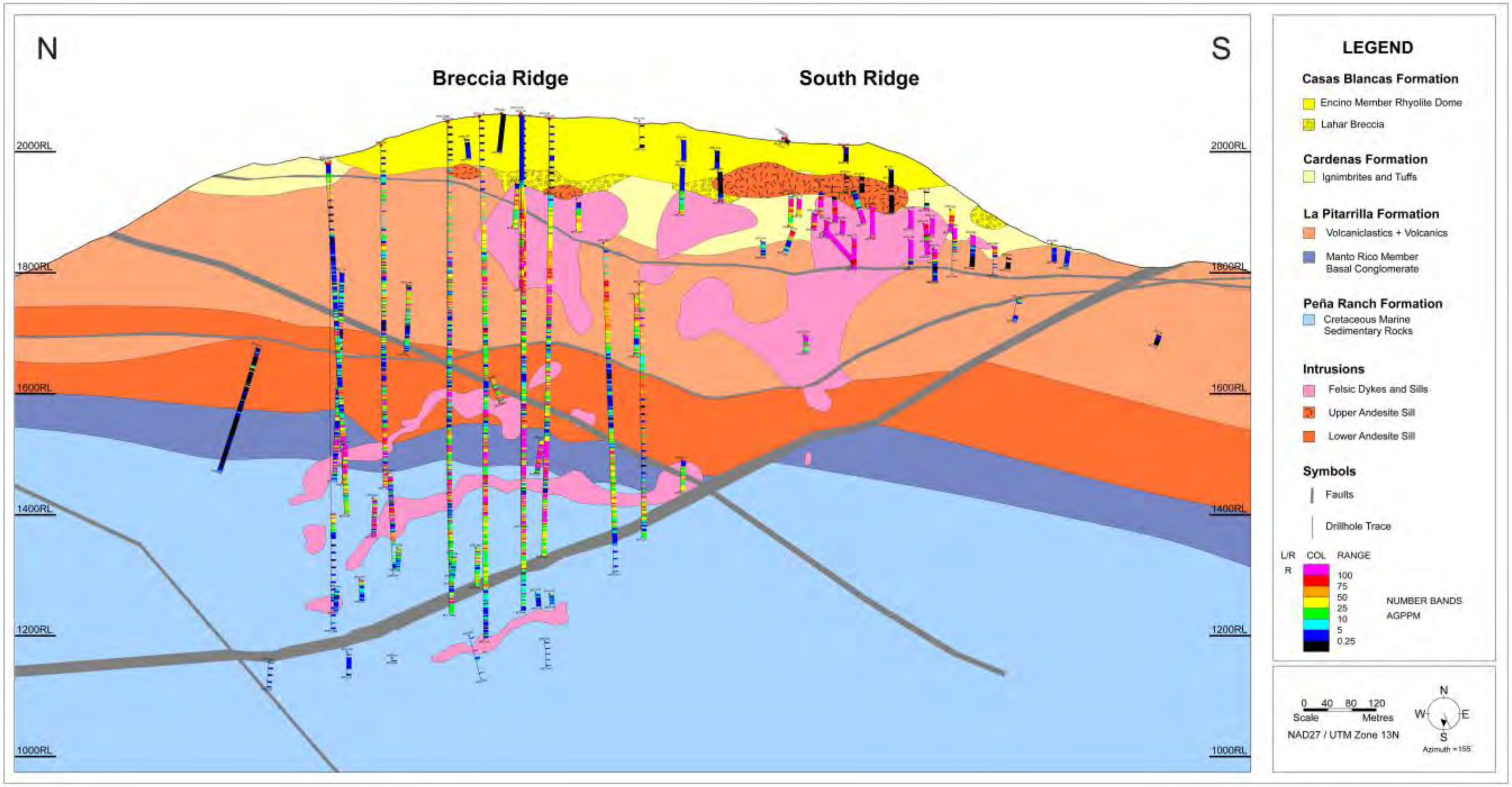



**Figure 10-3: Drill Section N-S Showing Interpreted Geology, and Drillhole Traces with Ag Grades (ppm)**
Source: SSR, 2012

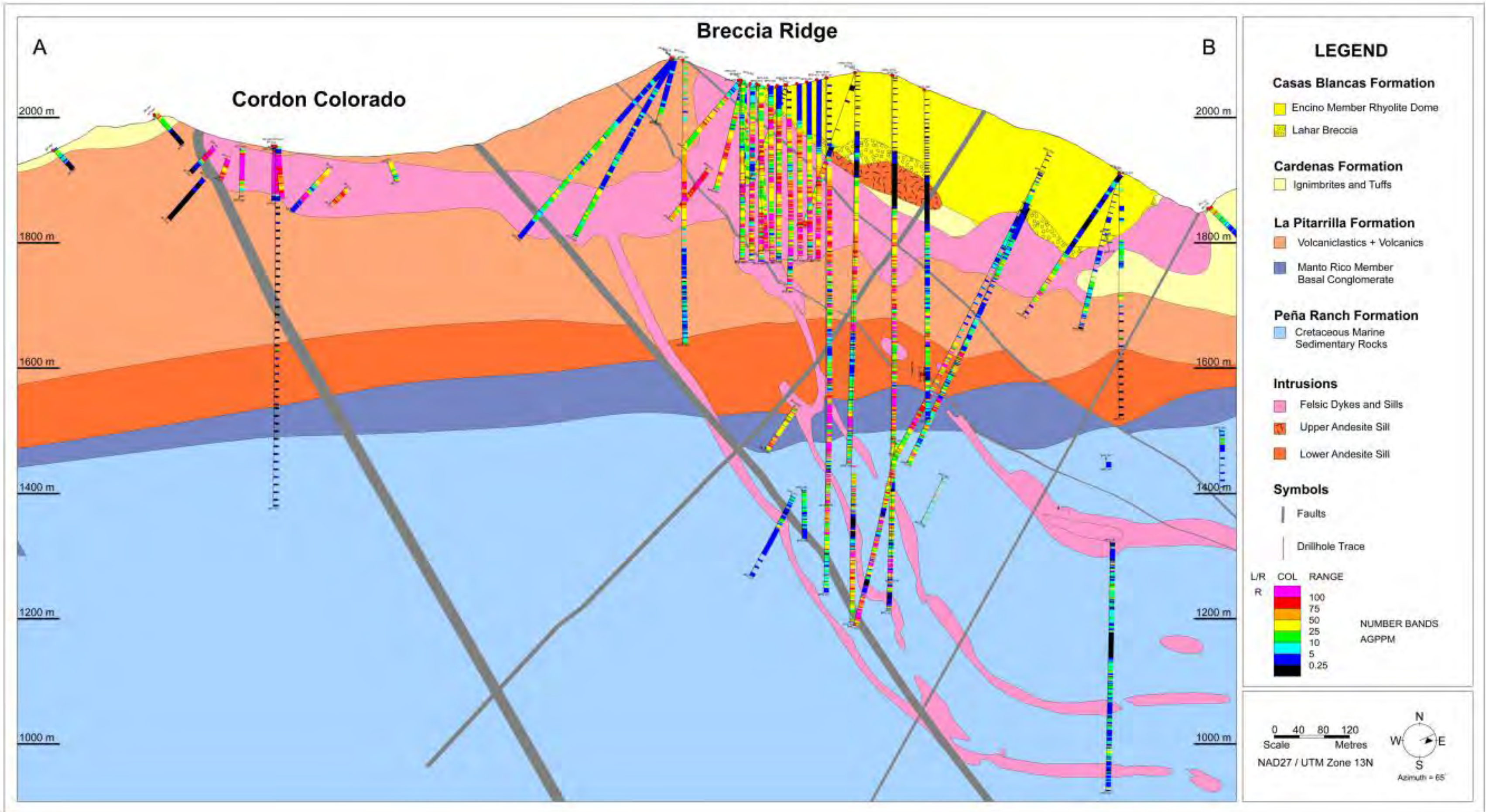



**Figure 10-4: Drill Section A-B Showing Interpreted Geology and Drillhole Traces with Ag Grades (ppm)**
Source: SSR 2012

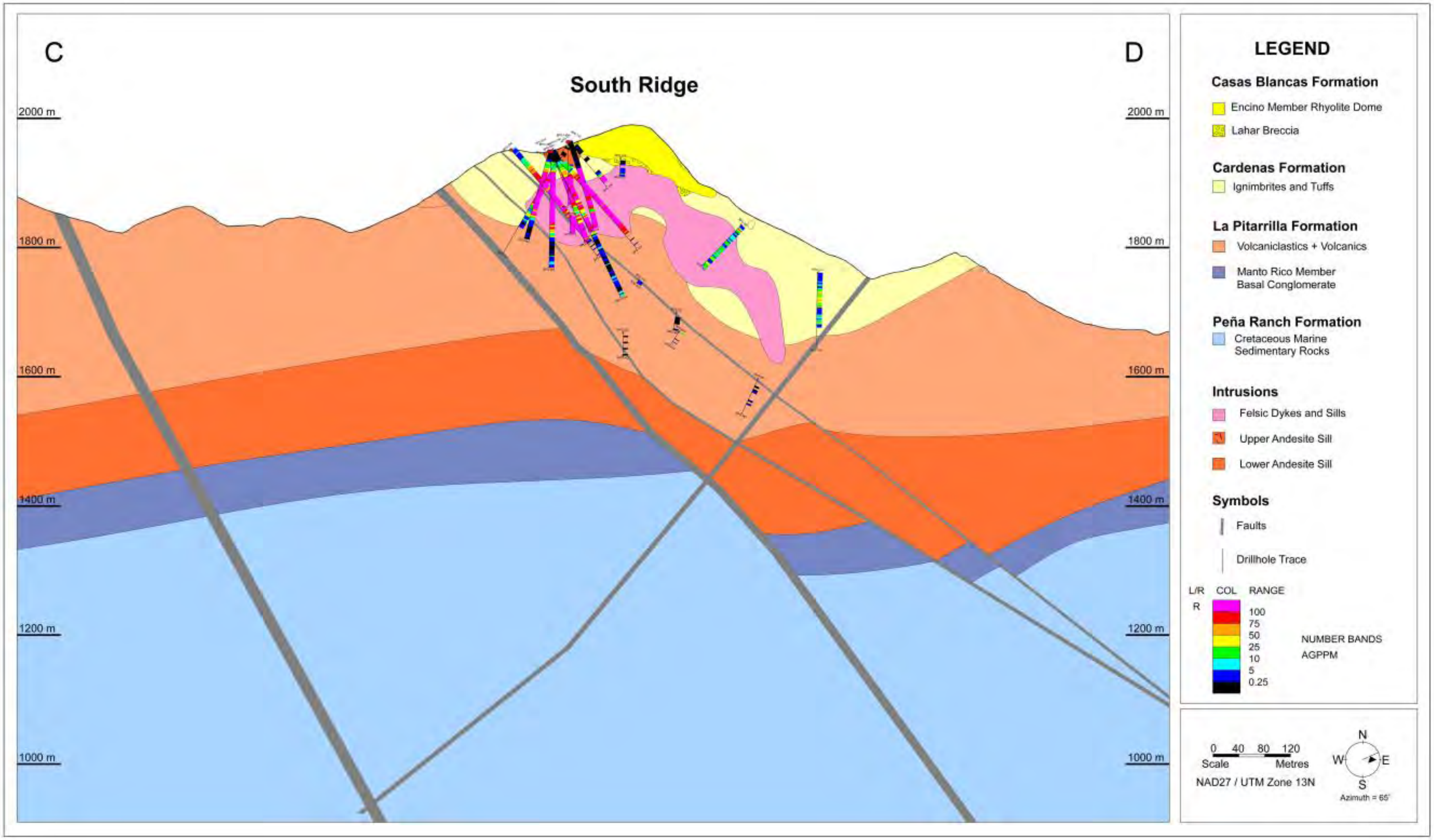



**Figure 10-5: Drill Section C-D Showing Geology and Drillhole Traces with Ag Grades (ppm)**
Source: SSR 2012

### 10.6 POTENTIAL DRILLING, SAMPLING OR RECOVERY FACTORS

The recovery from diamond drilling is generally very good with the average drilling recovery of 98.5%.

There are no obvious drilling, sampling, or recovery factors that would materially affect the reliability of the samples.

# 11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

## 11.1 2003 TO 2008 DRILLING PROGRAMS

### 11.1.1 Sampling

#### 11.1.1.1 RC Drillhole Samples

RC drillhole samples were collected at the drill site at 1 m lengths, from the collar, down to the final drillhole depth. Sampling intervals were dependent on the drilling equipment selected and not based on geological controls or other features of the zone of interest. The RC samples were split three times using a Jones splitter down to 1/8 of the original weight. The sample weight ranged from approximately two to 10 kg, with every 20$^{th}$ sample taken as a field duplicate. The samples were collected in numbered, heavy duty plastic bags along with sample tickets, which carried numbers referring back to a digital data file of drillhole identification, sample number and sample intervals. All samples were stored in the company warehouse in Casas Blancas prior to shipping. Periodically, staff from ALS laboratories collected and transported the samples to Guadalajara for sample preparation. After mid-August of 2005 (after BP-179), the samples were shipped by SSR personnel to Chihuahua for sample preparation, prior to analysis at the ALS laboratory in North Vancouver, BC, Canada. Once analysed, remaining pulps and coarse rejects were returned from ALS and catalogued and stored in a secured warehouse in the city of Parral, Chihuahua, located 180 km north-northwest of Pitarrilla.

#### 11.1.1.2 Diamond Core Drillhole Samples

Digital core photographs were taken after the core was cleaned and measured in the core boxes. After geological logging, diamond core samples were marked by the geologist and then split using a diamond saw. Sample lengths were approximately 1.52 m. Geological contacts were generally respected during sampling. The maximum sample length was just over 3 m in zones considered to be weakly mineralised or unmineralised. The half core samples were put in numbered heavy duty plastic bags, along with sample tickets. The sample tickets carried a number which referred back to a digital data file of drillhole identification and sample intervals. The bags were labelled with sample numbers and collected into rice sacks, which were sealed with a tamper proof seal and labelled prior to shipment. The geologists on site completed sample shipment and tracking forms, such that the reported assays could be tracked through the transport and analytical system and matched back to the appropriate drillhole identification and sample interval. Field duplicates comprised quarter core samples. Staff from ALS laboratories collected the samples and transported them to Guadalajara. After mid-August of 2005 (after BPD-037), the samples were shipped to Chihuahua by either ALS or SSR personnel for sample preparation prior to analysis at the ALS laboratory in North Vancouver.

#### 11.1.1.3 Geotechnical Drilling and Oriented Core

Geotechnical diamond drillholes were drilled by Major Drilling using triple tube wireline techniques. The drillcore was transported from the drilling rig in core boxes and taken to the core logging shed in a secure compound near the Casa Blancas village. Digital core photographs were taken after the core was cleaned and measured in the core boxes. It was then logged for geotechnical features by a trained geotechnical engineer or geologist from Knight Piésold Consulting. The geotechnical diamond drillholes were additionally logged for geological information by a SSR geologist who defined sample intervals for geochemical analysis. Sampling followed the procedure for non-oriented diamond drillcore described in Section 11.1.1.2.

### 11.1.2 Sample Preparation

Samples were received at the laboratory, given bar codes, and then entered into the ALS laboratory information management system for tracking purposes. Samples were then weighed, dried, and crushed to >70% passing -2 mm (ALS Code: CRU-31) screen. The crushed material was then riffle-split (ALS Code: SPL-21) to produce a representative 250 g split sample for pulverization to >85% passing 75 µm (ALS Code: PUL-31) screen. The pulps were then shipped to the ALS laboratory in North Vancouver for digestion and analysis. The ALS sample preparation facilities in Guadalajara and Chihuahua have maintained ISO 9001 certification for sample preparation since 1998.

### 11.1.3 Sample Analysis

Digestion and analyses were completed on a standard 30 g split of the 250 g pulverised sample. At the ALS laboratory in North Vancouver, samples were digested using the four-acid "near total" digestion, followed by inductively coupled plasma atomic emission spectroscopy ("ICP-AES") analysis for 27 elements (ALS Code: ME-ICP61). Mercury was added to the standard package and analyzed by cold vapour atomic absorption ("AA") following digestion in aqua regia (ALS Code: Hg-CV41). Sample values above the analytical detection limit (over limit) were re-run by atomic absorption for zinc, lead, and copper, and fire assay with a gravimetric finish for silver. These provided upper detection limits of 30% for zinc, 20% for lead, 40% for copper (all overlimit samples used ALS Code: –OG62), and 1,500 ppm for silver (ALS Code: AgGRA21). Gold analyses were requested during the early stages of the program, but were dropped for lack of results. Gold analyses were occasionally requested in deep drillholes in base metal zones. The ALS Chemex facility in North Vancouver was ISO 9001 certified between 2003 and 2005, and then obtained ISO 17025 certification in 2005 for the analytical procedures described in this section.

### 11.1.4 Silver Standard Quality Assurance/Quality Control (QAQC) Samples

Silver Standard initiated and implemented a QAQC program in November 2005. It utilised standard reference material, blanks, field duplicates, and third party check laboratories, where every tenth sample submitted was a QAQC sample (either a standard or a blank). To monitor precision field duplicates were inserted at a rate of one in 20. Samples for analysis by a check laboratory totalled 5% of the original number of assays.

11.1.4.1 Property Reference Materials

Silver Standard used eight reference materials to monitor laboratory accuracy, which were composed of coarse reject material left over from the initial RC drilling program. Each batch of reference material was given a number and called Standard 1, Standard 2, and Standard 6 through Standard 11. (Standard 3 through Standard 5 were reference materials certified by WMC Minerals (Lloyd Twaites Registered Assayers; Section 11.1.4.2). Early results showed a number of samples falling outside of the three standard deviation tolerance limit of the certified mean value. After review, the scattered results were attributed to mislabelling of standards and blanks.

In 2008, P&E Mining Consultants Inc. (P&E, 2008) determined that the number of samples in each round robin characterization of the certified property reference standards was found to be too few to provide a representative mean. As the data set provided by Silver Standard contained between 200 and 700 samples, P&E (2008) recalibrated the mean for each of the property standards using this larger dataset. They determined a revised mean and standard deviation for each standard by using all available analyses for each one (P&E, 2008). All values greater or less than two standard deviations from the calculated mean were removed from the data set. With these data points removed, a new mean and standard deviation were calculated. Results from this were graphed in order to view the warning limits (±2 standard deviations from the mean) and the tolerance limits (±3 standard deviations from the mean). All values falling within the warning limits were considered acceptable and those falling outside the tolerance limits were declared failures (P&E, 2008). All results from the 2005 through 2008 drilling programs were using the recalibrated property standard reference values (and standard deviations), which are summarised in Table 11-1.

**Table 11-1: Results of 2005 through 2008 Property Standard Samples after Recalibration**

| | Ag | | Pb | | Zn | |
|---|---|---|---|---|---|---|
| | No. Samples | No. Failed | No. Samples | No. Failed | No. Samples | No. Failed |
| STD-1 | 276 | 6 | 276 | 4 | 276 | 4 |
| STD-2 | 211 | 5 | 211 | 2 | 211 | 2 |
| STD-6 | 206 | 6 | 206 | 4 | 204 | 1 |
| STD-7 | 869 | 22 | 870 | 9 | 870 | 5 |
| STD-8 | 234 | 6 | 234 | 6 | 234 | 2 |
| STD-9 | 859 | 2 | 859 | 18 | 859 | 18 |
| STD-10 | 819 | 14 | 819 | 7 | 819 | 14 |
| STD-11 | 857 | 20 | 857 | 18 | 857 | 21 |
| **Total** | **4,331** | **81** | **4,332** | **67** | **4,332** | **68** |

Standard 7, Standard 10, and Standard 11 continued to exhibit an unacceptable failure rate after recalibration, suggesting that insufficient homogenization of the material before round robin testing was responsible for the high failure rate.

11.1.4.2 Certified Reference Materials

Silver Standard purchased three reference material standards certified by WMC Minerals (Lloyd Twaites Registered Assayers), which were interspersed with the property standards. These were designated as Standard 3, Standard 4, and Standard 5. The initial round robin characterization of these standards, such that they could be used to monitor laboratory accuracy, was also found to be inadequate by P&E (2008), as it involved only two laboratories using between five and 16 samples. In June 2008, P&E (2008) recalibrated these standards following the procedure outlined in Section 11.1.4.1. All results from the 2005 through 2008 drilling programs were since analysed using the recalibrated property standard reference values (and standard deviations), which are summarised in Table 11-2.

**Table 11-2: Results of 2005 through 2008 Certified Reference Samples after Recalibration**

| | Ag | | Pb | | Zn | |
|---|---|---|---|---|---|---|
| | No. Sample | No. Failed | No. Sample | No. Failed | No. Sample | No. Failed |
| STD-3 | 175 | 1 | 175 | 4 | 175 | 2 |
| STD-4 | 121 | 0 | 121 | 1 | 121 | 0 |
| STD-5 | 170 | 7 | 170 | 3 | 169 | 2 |
| **Total** | **466** | **8** | **466** | **8** | **465** | **4** |

Standard 3 and Standard 4 results plotted within acceptable tolerance limits, but Standard 5 results continued to exhibit a high failure rate. Silver Standard was recommended by P&E to reduce the number of reference standards to three, as 11 in total were considered to be more than necessary (P&E, 2008).

Silver Standard's Senior Geologist, Jeremy D. Vincent, P.Geo., has reviewed the procedure followed by P&E (2008) to recalibrate the certified reference standards and considers the work to have been conducted in accordance with acceptable industry standards.

11.1.4.3 Blank Samples

Silver Standard used three different blanks throughout the drill programs from 2003 to 2008. The material used as a source for Blank 1 came from an area immediately west of the South Ridge Zone and immediately south of the Breccia Ridge Zone, before these zones were discovered. After this material was found to return values greater than three times the detection level for Ag, Pb, and Zn, Blank 1 was discarded. Silver Standard procured a second blank material (Blank 2) in 2007, from a dacite tuff located approximately 3.8 km west of the mineralised areas. The third blank material (Blank 3) was sourced in 2008, from an intermediate volcanic located approximately 6 km west of the mineralised areas. Results from Blank 2 and Blank 3 also returned a wide range of silver values (upwards of 100 g/t Ag). Although some mislabelling of samples had been identified, these poorly performing blanks returned values that did not match any of those of the certified reference material. It was therefore assumed that they were either sourced from mineralised material, or resulted from cross-contamination.

11.1.4.4 Field Duplicate Samples

Silver Standard's early quality control program included shipping field duplicates of RC drillhole samples to BSI Inspectorate in Victoria de Durango (Section 11.1.4.5). Diamond drillhole field duplicates (quartered drillcore) were inserted at a rate of approximately one in 20.

Silver Standard evaluated duplicate (paired) sample data for bias through analysis of quantile-quantile (QQ) plots, X-Y scatter plots, and cumulative distribution function ("CDF") plots. Precision was monitored through analysis of ranked half absolute relative difference ("HARD") plots, paired precision plots, and half absolute difference ("HAD") plots. Field duplicate results are summarised in Table 11-3. Taking into account the style of mineralisation, the elements of interest, and the volume difference between the half-core original sample and quarter-core duplicate sample, the expected ranked HARD statistic for a field duplicate, which is selected at the 90$^{th}$ percentile of the duplicate data distribution, should be lower than approximately 20-25% (i.e., 90% of the paired data should vary by less than 20-25%). Values greater than this threshold may indicate that incorrect sampling error (as opposed to correct sampling error, which cannot be controlled by the sampler) is becoming more significant in the sampling process.

**Table 11-3: Results of 2005 through 2008 Field Duplicate Samples**

| Year | No. Samples | Ranked HARD statistic (90$^{th}$ Percentile) | Bias |
|---|---|---|---|
| 2005-2006 | 1,112 | Ag:33%<br>Pb:32%<br>Zn:27% | None |
| 2007 | 2,487 | Ag:42%<br>Pb:42%<br>Zn:37% | None |
| 2008 | 1,135 | Ag:45%<br>Pb:39%<br>Zn:29% | None |

11.1.4.5 Umpire Laboratories

Umpire laboratories are employed as an additional check on the accuracy of the primary laboratory. Silver Standard has employed several laboratories during the course of the QAQC program

*BSI Inspectorate/Rocky Mountain Geochemical*

In the early part of Silver Standard's quality control program, every 20$^{th}$ RC drill sample was split twice. The second was sent to BSI Inspectorate de Mexico, S.A. de C.V ("BSI") in Victoria de Durango, Mexico for preparation. Samples were crushed to -10 mesh and split with a riffle splitter (McCrea, 2006). A 300 g pulp was prepared and then shipped to Rocky Mountain Geochemical ("Rocky Mountain") in Sparks, Nevada for digestion in aqua regia and analysis by multi-element ICP. The samples were analyzed for a total of eight elements. Overlimit silver samples were re-run by fire assay with a gravimetric finish. Mercury was added to the analysis package and was analyzed by cold vapour atomic absorption. Rocky Mountain is a part of the BSI group of companies and obtained an ISO 9001:2000 certification, between 2004 and 2005 (McCrea, 2006). At the time of writing, Silver Standard has been unable to locate and verify these results.

*American Assay Laboratories*

In 2006, American Assay Laboratories ("AAL"), located in Sparks, Nevada, was used to analyze 537 pulp duplicate samples from 186 RC drillholes, completed between 2003 and 2006 (McCrea, 2007). These samples were sent in order to establish quality control data on the results from the early drilling campaigns that were not subject to a QAQC program. From McCrea (2007), the scatter plots of the paired data indicated no sample bias for silver. The Thompson-Howarth chart showed 37% absolute relative percent difference at the 90$^{th}$ percentile, which indicated poor precision, as the industry standard for pulp duplicates was 10%. McCrea (2007) discovered a mix up with the sample submission and attributed some of the poor precision to this. At the time of writing, Silver Standard has been unable to verify these results.

AAL does not have ISO certification, but their website states they participate in several certification and testing processes twice a year, including CANMET PTP-MAL, GEOSTATS, SMA (US and Canada), and IAG. AAL is a "reputable" laboratory under the Mineral Exploration Best Practices Guidelines and has participated in CANMET-PTP MAL studies since their inception in 1998 (www.aallabs.com/cms/index.php?page=quality-control).

*Assayers Canada*

Assayers Canada, located in Vancouver, BC was used for check sample submission for diamond drillcore in 2007 and 2008. The laboratory had an ISO 9001:2008 certification and also held certificates for Laboratory Proficiency from the Standards Council of Canada for precious and base metals analysis.

Check samples revealed generally poor levels of precision in comparison to the original samples (Table 11-4), as the ranked HARD statistics for pulp duplicates are expected demonstrate precision better than 10% at the 90$^{th}$ percentile (i.e., 90% of the data should vary by less than 10%). The value for lead was caused by poor precision results from approximately 230 samples. This precision issue did not appear to affect the silver or zinc results.

**Table 11-4: Results of 2006-2008 Diamond Drillcore Pulp Duplicate Samples**

| Years | No. Samples | Ranked HARD statistic (90$^{th}$ Percentile) | Bias |
|---|---|---|---|
| 2006-2008 | 1,853 | Ag:14%<br>Pb:35%<br>Zn:14% | None |

### 11.1.5 ALS Quality Control Samples

In light of quality control results generated between 2005 and 2008, Silver Standard requested the results of ALS's internal quality control samples to assess sample bias, accuracy, precision, and evidence of cross-contamination. ALS employed 42 different standards for silver, lead, and zinc during these years. The analytical results of the standard samples did not indicate any significant source of bias or deviation outside of accepted thresholds. A review of the internal blank sample results indicated no evidence of cross-contamination, suggesting the blank material used in Silver Standard's quality control program was sourced from mineralised rock, and not adequately confirmed to be barren of mineralisation before use. ALS sample duplicates for silver did not exhibit evidence of sampling bias, while precision levels were within acceptable industry limits (Table 11-5).

**Table 11-5: Results of ALS internal QC Duplicate Samples**

| Years | No. Samples | Ranked HARD statistic ($90^{th}$ Percentile) | Bias |
|---|---|---|---|
| 2005-2008 | 3,767 | Ag:6% | None |

Though ALS's quality control samples were not "blind", they demonstrate strong analytical control over the assay results they were supporting. This suggests an ineffective implementation of the QAQC program by Silver Standard during these years, not an inherent problem with the quality of the assay data generated by the laboratory.

### 11.1.6 Sample Security

Drillcore samples were in Silver Standard's custody from collection and bagging until pickup by an ALS transport vehicle from the warehouse in Casas Blancas. Samples were transported to the ALS sample preparation facility in Guadalajara until mid-August 2005. After this date they were shipped to the ALS sample preparation facility in Chihuahua by SSR personnel. Upon arrival at the laboratory, each sample was given a bar code label and logged into the laboratory information management system. This permitted sample tracking and provided a complete chain of custody record after receipt at the laboratory. Sample bags were sealed on site and none of the seals were reported tampered by the receiving analytical laboratory. Silver Standard is not aware of any deliberate attempts to compromise samples.

## 11.2 2010 TO 2012 DRILLING PROGRAMS

### 11.2.1 Sampling of Diamond Core Drillholes

Sampling of diamond core drillholes followed the procedure outlined in Section 11.1.1.2. The plastic sample bags were then placed in larger labelled rice bags, then sealed, and then sent by SSR trucks to the ALS Chemex facility in Zacatecas, Mexico.

### 11.2.2 Sample Preparation

The sample preparation facility in Zacatecas has maintained ISO 9001 certification since 2012. Sample preparation followed the procedure outlined in Section 11.1.2.

### 11.2.3 Sample Analysis

Sample analysis followed the procedures outlined in Section 11.1.3, with the exception that mercury and gold were not analyzed.

### 11.2.4 Silver Standard QAQC Samples

QC sample data were monitored on a monthly basis to ensure that sample batches with control sample data outside of acceptable limits were re-submitted for analysis in a timely manner.

#### 11.2.4.1 Certified Reference Materials

Silver Standard utilised three reference standards during the drilling campaigns from 2010-2012. These covered a range of medium and high grade silver and zinc values. Two reference standards (STD-13 and STD-14) were created for the Pitarrilla deposit by CDN Resources Laboratories Ltd., and certified by Smee & Associates Consulting Ltd. following a round robin analysis at five independent analytical laboratories. One reference standard (STD-12) was created for the Pitarrilla deposit by Minerals Exploration & Environmental Geochemistry following a round robin analysis at six independent laboratories; however, this reference material has not been certified.

Silver Standard has reviewed the results of the three certified standards from the 2010 through 2012 drilling programs. A total of 1,293 standard samples were submitted during this time representing a rate of one standard for each 20 assay samples. In total there were seven failed Ag results, four failures for Zn, and two failures for Pb. No significant evidence of bias was observed.

#### 11.2.4.2 Blank Samples

Sample blank material comprised unmineralised sand sourced from the Casas Blancas area. A total of 1,291 sample blanks were inserted throughout the drilling campaigns between 2010 and 2012, representing a rate of one blank sample for each 20 assay samples. A threshold of 10 times the analytical detection limit of Ag (i.e., a threshold of 5 ppm) was used discriminate samples showing evidence of cross-contamination. Results of field blank control samples indicated that there was no significant source of cross-contamination during analytical work during Silver Standard's 2010 to 2012 drilling campaigns.

#### 11.2.4.3 Field Duplicate Samples

Between 2010 and 2012 Silver Standard inserted approximately 1,285 field duplicate samples, which equates to a rate of one field duplicate in 20 assay samples (Table 11-6). Silver Standard analyzed the field duplicate data for bias and precision using the methodology outlined in Section 11.1.4.4. The field duplicate ranked HARD statistic precision value at the 90$^{th}$ percentile was within acceptable industry standards of approximately 20%. No significant evidence of bias was observed.

**Table 11-6: Results of 2010 through 2012 Field Duplicate Samples**

| Years | No. Samples | Ranked HARD statistic (90th Percentile) | Bias |
|---|---|---|---|
| 2010-2012 | 1,285 | Ag:19%<br>Pb:21%<br>Zn:17% | None |

11.2.4.4 Umpire Laboratory – Assayers Canada

Silver Standard sends 5% of assay pulps to the Assayers Canada laboratory (now SGS Canada) in Vancouver for an independent third party check. For the 2010 and 2011 drilling campaigns (Silver Standard is currently awaiting the results of the 721 check samples from the 2012 drilling), the ranked HARD statistic at the 90th percentile indicated levels of precision within industry limits (better than 10%) for pulp duplicates.

**Table 11-7: Results of 2010-2011 Diamond Drillcore Pulp Duplicate Samples**

| Years | No. Samples | Ranked HARD statistic (90th Percentile) | Bias |
|---|---|---|---|
| 2010-2011 | 363 | Ag:8%<br>Pb:8%<br>Zn:9% | Weak (5% high, original vs duplicate) |

### 11.2.5 Sample Security

Drillhole core samples were in Silver Standard's custody from collection and bagging until delivered to the ALS Chemex sample preparation facility in Zacatecas. Upon arrival, each sample was given a bar code label and logged into the laboratory information management system. This permitted sample tracking and provided a complete chain of custody record after receipt at the laboratory. Sample bags were sealed on site with tamper proof seals. None of the seals were reported tampered by the receiving analytical laboratory. Silver Standard is not aware of any deliberate attempts to compromise samples.

## 11.3 RELATIONSHIP OF THE LABORATORIES TO THE ISSUER

All laboratories contracted for sample analysis from 2002 to 2012 were and are independent of Silver Standard.

## 11.4 OPINION ON ADEQUACY OF SAMPLE PREPARATION, SECURITY, AND ANALYTICAL PROCEDURES

Silver Standard's Senior Geologist, Jeremy D. Vincent, P.Geo., has reviewed the sample preparation, analytical, and security procedures for the various drilling programs conducted on the Pitarrilla deposit and considers them as having been conducted in accordance with industry standards.

Results generated from Silver Standard's QAQC program from 2005 through 2008 indicate it was not effectively implemented. The lack of adequate external quality control data is mitigated, however, by the strong analytical control demonstrated by ALS's internal QC samples during these years.

In 2010, monthly monitoring of QC results was initiated to ensure that sample batches with control sample data outside of acceptable limits were re-submitted for analysis in a timely manner. This led to an improvement in accuracy and precision, with few failed samples, and there was no evidence of cross-contamination.

The procedures discussed in Section 11 pertaining to the treatment of the quality control data employed throughout Silver Standard's drilling campaigns are considered adequate for the generation of data suitable for use in Mineral Resources and Mineral Reserves estimation, and for mine planning purposes.

# 12 DATA VERIFICATION

The following data verification steps were conducted as part of the generation of the December 4, 2012 Mineral Resource estimate presented in this report:

- A visit to inspect geology and mineralisation at the Pitarrilla Property;
- Detailed review of selected drillholes to assess the nature of the mineralisation, and the effectiveness of the selected drilling orientation in the delineation thereof;
- Select drillhole collar locations were confirmed by GPS during the visit to site;
- Downhole survey data were reviewed for all drillholes to assess drillhole traces;
- Quality Control (QC) information for all exploration drilling programs (see Sections 10 and 11) conducted on the Pitarrilla Property was analysed;
- Approximately 10% of the pre-2011 drilling assay dataset was checked and compared to the original assay certificates, to generate additional confidence in this data;
- Detailed checks of assay data from the 2011 and 2012 drilling programs in conjunction with Silver Standard's database manager, with iterative correction for any anomalies (generally typographical errors, including mislabelled samples and mislabelled sample intervals);
- Review of monthly QC data monitoring by Silver Standard's database manager, especially timing and effectiveness of remedial action taken with respect to failed batches;
- Comparison analyses were conducted between data derived from different drilling generations and types (e.g., RC, diamond core drilling) to validate their use in a single database; and
- Data was validated at each manipulation stage throughout the database compilation until the completion of Mineral Resources grade tonnage estimates (see Section 14.11).

All assay data are provided directly to Silver Standard by the relevant analytical laboratory, and directly imported into the DataShed database management system. Silver Standard checked a randomly selected proportion (10%) of the 2011 and 2012 assay data in the DataShed Pitarrilla assay database against the assay certificates provided by ALS and no errors were noted.

Based on the data verification steps outlined above, Silver Standard's Senior Geologist, Jeremy D. Vincent, P.Geo., considers the Silver Standard exploration drilling data (including, collar, survey, lithology, and assay data) to be adequate for classification of the Mineral Resources and Mineral Reserves summarised in this Techncial Report.

# 13 MINERAL PROCESSING AND METALLURGY TESTWORK

The Pitarrilla Project will produce lead and zinc flotation concentrates with contained silver values from sulphide ore, and doré from both flotation tailings and oxide (direct leach) ore. Both of the two flotation concentrates and the doré will be shipped offsite for sale or further processing.

This section summarizes the testwork performed to evaluate the metallurgical aspects of the project. The interpretation of the testwork is also discussed and an estimate is presented for the consumption of reagents and other consumables.

## 13.1 GENERAL

In 2004, Silver Standard initiated testwork to provide a better understanding of the Pitarrilla deposit metallurgy and to establish design criteria for the mineral extraction process. The test programs have included initial scoping studies, flotation process development for sulphide ore, cyanide leaching development for oxide ore, and a combination of processes for the transitional (located between sulphide and oxide ore zones) and sulphide ores. Within the testwork, four pilot flotation tests of sulphide ore were completed. The test results are reported in the following documents and relevant tests are summarised below.

### 13.1.1 Initial Testwork Performed On All Ore Types - 2004 to 2007

- "Pitarrilla Metallurgical Testwork, Project No 0401902, February 24, 2005, Process Research Associates Ltd, Richmond, British Columbia, Canada.
- "Metallurgical Studies Pitarrilla Project, Project No 0503804", February 27, 2006, Process Research Associates Ltd, Richmond, British Columbia, Canada.
- "Pressure Leach Study of Silver Bearing Samples Pitarrilla Project, Mexico, Project 0507810", January 22, 2007, Process Research Associates Ltd, Richmond, British Columbia, Canada.

### 13.1.2 Process Testwork Performed On Sulphide Ore Types - 2008 to 2012

- "Preliminary Cyanidation and Flotation studies Breccia Ridge, Project No 0604906", March 11, 2008. Process Research Associates Ltd., Richmond, British Columbia, Canada.
- "A Preliminary Assessment of Metallurgical Response, Pitarrilla Project – Breccia Ridge Zone, Durango State, Mexico, KM1889", January 5, 2007, G&T Metallurgical Services Ltd., Kamloops, British Columbia, Canada.

- “Flotation Process Design and Metallurgical Response, Pitarrilla Project, Durango State, Mexico, KM1971”, April 30, 2007, G&T Metallurgical Services Ltd., Kamloops, British Columbia, Canada.
- “Metallurgical Response – Pitarrilla Project, Silver Standard Resources Inc., Pitarrilla Project, Durango State, Mexico, KM2056”, March 13, 2008, G&T Metallurgical Services Ltd., Kamloops, British Columbia, Canada.
- “Advanced Process Design Studies, Silver Standard Resources Inc. Pitarrilla Project, Durango State, Mexico, KM2232”, January 30, 2009, G&T Metallurgical Services Ltd., Kamloops, British Columbia, Canada.
- “An Investigation Into La Pitarrilla, Prepared for Silver Standard Resources, Project 50014-001 – Final Report”, May 14, 2009, SGS Canada Inc., Vancouver, British Columbia, Canada.
- “A Laboratory Investigation Into The Recovery of Lead, Zinc and Silver From Pitarrilla Samples, Project 12526-001 Final Report”, April 11, 2011, SGS Canada Inc., Lakefield, Ontario, Canada.
- “Dacite Test KM 3433”, October 11, 2012, G&T Metallurgical Services Ltd., Kamloops, British Columbia, Canada.

### 13.1.3 Mineralogy Reports On Oxide Ore Types - 2006 to 2012

- “Deportment Of Silver In BP-29 Ore Composite From Pitarrilla”, May 23, 2006, AMTEL, London, Ontario, Canada.
- “Mineralogical Assessment Of A Silver Ore Sample, KM 3017”, August 23, 2011, G&T Metallurgical Services Ltd., Kamloops, British Columbia, Canada.
- “The Mineralogical Characteristics Of Two Composites And A Cyanide Leached Tails Sample, Project 50141-101”, March 7, 2012, SGS Canada, Vancouver, British Columbia, Canada.
- “Diagnostic Leach Report” August 20, 2012, Kemetco Research, Richmond, British Columbia, Canada.

### 13.1.4 Process Testwork Performed On Oxide Ore Types – 2011 to 2012

- “La Pitarrilla Project Report Of Metallurgical Testwork AVR And Detoxification Studies”, December 11, 2011, Kappas, Cassidy and Associates, Reno, Nevada, USA.
- “Flocculant Screening, Gravity Sedimentation, Pulp Rheology, Vacuum Filtration And Pressure Filtration Studies Conducted For McClelland/Pitarrilla Project”, July 2011, Pocock Industrial Inc., Salt Lake City, Utah, USA.

- “Report On Ore Variability And Optimisation Testing, Pitarrilla Drill Core Composites”, Report No. 3553, March 12, 2012, McClelland Laboratories, Sparks, Nevada, USA.

- “Determinar La Cinetica De Extraction De Ag A Neuve Meustrars De McClelland Labs”, August 2012, SGS de Mexico, Victoria de Durango, Durango, Mexico.

- “Report On Follow-up Bottle Roll Testing, 8 Drill Core Composites From The Pitarrilla Project”, Report No. 3553-01, September 2012, McClelland Laboratories, Sparks, Nevada, USA.

#### 13.1.5 Process Testwork Performed On Transition Ore Types - 2012

- “Determinar La Suceptibilidad De 118 Meustras Al Proceso Lixivication, EDTA Y Analisis De S”, June 2012, SGS de Mexico, Victoria de Durango, Durango, Mexico.

- “Cyanidation Of Flotation Tailings From The Pitarrilla Deposit”, September 7, 2012, SGS Canada Inc., Lakefield, Ontario, Canada.

- “Transition Tests KM3513”, October 15, 2012, G&T Metallurgical Services Ltd., Kamloops, British Columbia, Canada.

### 13.2 SUMMARY OF RESULTS FOR ALL PROJECTS

#### 13.2.1 Process Research Associates – Project No 0401902 - 2005

These metallurgical studies were conducted upon 14 representative composite samples of the mineralisation. The objective of these test programs was to evaluate processes to recover silver. The program included evaluation of leaching, gravity separation, magnetic separation, and froth flotation processes. The first lot of samples consisted of ten samples, differentiated as either silver or zinc mineralised material, in the form of coarse drillcore assay reject samples.

The results of this test program indicated that the samples represented material refractory to direct cyanide leaching methods and did not upgrade using typical mineral processing procedures.

The results of diagnostic leaching procedures indicated that silver mineralisation is either within manganese minerals that will require dissolution or within clay silicate minerals that will require fine grinding.

#### 13.2.2 Process Research Associates – Project No 0503804 - 2006

These metallurgical studies were conducted upon 15 representative composite samples of the mineralisation in the form of coarse drillcore assay reject samples. The objective of the test program was to evaluate processes to recover silver. The program included evaluation of leaching under both direct and pre-treatment leaching procedures, gravity separation, magnetic separation, and froth flotation processes.

This results of this test program indicated that concentration procedures using gravity separation, magnetic separation, or flotation techniques were only partially successful. Poor silver extraction was indicated to be the result of silver being encapsulated in alumino-silcate clay type minerals. Procedures attempted to deal with these clay minerals was ineffective.

#### 13.2.3 Process Research Associates – Project No 0507810 - 2007

These metallurgical studies were conducted upon 11 representative composite samples of the mineralisation. Existing samples from the previous test program (0503804) were supplemented with new samples to create a new combined set of samples to be tested. The previously completed test programs had shown that the application of caustic pressure treatment, before cyanide leaching, gave the best extraction rates. The objective of the test program was to evaluate pressure leaching conditions such as pressure and temperature, caustic addition, slurry solids density, and chemical additives to optimize silver extraction.

The results of this test program indicated that alkaline pressure leaching effectively liberated refractory silver components from the samples. The key factor in the process is the attack of silicate minerals by high concentrations of caustic solution at process temperatures above 180°C. The lime boil technique was indicated to reliably precipitate silica from the pressure leach solution and regenerate the caustic.

Cyanide leaching, after pressure leaching with caustic, resulted in silver extraction ranging from 46% to 95%.

#### 13.2.4 Process Research Associates – Project No 0604906 - 2008

These metallurgical studies were conducted upon 14 samples and four master composites samples. Cyanide leaching and flotation were the techniques investigated for extracting silver from the samples. Samples are described as split, diamond drillcore intervals originating primarily from an oxidised cap and sulphide transition interval, both located within the Breccia Ridge Zone.

The results of this test program were as follows:

- The mixture of oxides and sulphides in the material tested, as well as other mineralogical characteristics, complicated the selection of a single optimum process.
- Whole rock cyanidation on the oxide composite samples gave silver recoveries between 33% and 84%.
- Cyanidation of sulphide concentrates and flotation tailings gave a similar variation in response.
- Bulk flotation provided silver recoveries of over 90% for the two samples that exhibited the highest sulphide content and the lowest degree of sulphide oxidation. However, as the sulphide content decreased and extent of oxidation increased, the flotation response worsened for both silver and base metals recovery.

- Further laboratory studies of mineralised samples from the oxide and transitional zones are warranted in conjunction with the process development and advancement of the project as a whole.

### 13.2.5 G&T Test Program – KM 1889 - 2007

These metallurgical studies were conducted on nine representative ore composites. The composite samples were prepared from diamond drillcore from the Breccia Ridge Zone. The shipment was subdivided into 26 sub-samples. All samples were clearly identified in terms of source by drillhole and geological designation. The samples were segregated into nine groups according to instructions provided by SSR, with each group corresponding to a drillhole and a specific geological horizon.

The objectives of the metallurgical test program were to:

- Conduct chemical, mineralogical and modal analyses of nine composites; and

- Determine mineral compositions and mineral fragmentation profiles, and thereby assess the flotation treatment options of these composites, in order to maximize silver recoveries; and

- Perform kinetic tests and open circuit batch cleaner tests to outline some of the details of the treatment scheme. The general approach to treating these materials, which was based on the observed mineral compositions of the samples, was to sequentially produce bulk copper-lead and zinc flotation concentrates; and

- Attempt to produce saleable grade lead and zinc concentrates. Determine the minor element contents of typical examples of these flotation concentrates, with particular attention being given to deleterious components, which might influence smelter acceptance.

The following observations can be made regarding the sulphide mineral contents of these samples:

- No specific silver sulphide mineral carriers were evident in the preliminary scans performed during the mineral search routines. It was concluded that most of the silver was probably present either in solid solution within one or more of the sulphide minerals, or occurred as disseminated submicroscopic inclusions.

- The dominant sulphide mineral present in all samples, and accounting for about 10% by weight of the average sample, was pyrite. Other iron sulphide minerals, which were detected, were pyrrhotite and arsenopyrite, which tended to be present in similar amounts. Traces of the iron oxides goethite/limonite were noted in most samples, as was a minor but pervasive carbonaceous component.

- A relatively low, interstitial iron content sphalerite was present as an ancillary sulphide mineral, as was galena. Preliminary estimates suggested that the sphalerite had an average interstitial iron content of about 3% by weight in the nine samples tested. This implies a stoichiometric limit for the zinc concentrate grade of 64% by weight zinc. As a generalization, the sphalerite content of the samples was appreciably higher than that of galena.

- Copper was present in all samples and occurred dominantly as chalcopyrite. Tetrahedrite group minerals were an ancillary mineral carrier for copper and possibly arsenic and antimony. Traces of enargite and chalcocite/covellite were recorded in some of the samples.

Statistical analysis of flotation test products indicated that there is an excellent correlation between lead content and silver content. It was recommended that lead concentrate grades of the order of 45% to 50% by weight lead content will ensure silver grades in the range of 6,000 g/t to 8,000 g/t for most of the samples examined. There was no evidence to suggest that a significant fraction of the silver in any of the samples was associated with any other mineral.

13.2.5.1 Mineral Fragmentation

Samples were ground in the laboratory to approximately an 80% by weight passing 100µm size distribution ($P_{80}$), and the modal assessments were conducted.

About 95% liberation of the gangue was achieved at a nominal sizing of about 100 $P_{80}$ µm for this suite of samples. Further, extrapolations of the mineral fragmentation data revealed that still coarser flotation feed sizings might be more economically appropriate for processing these materials. Based on limited data, the practical envelope of feed size for economic flotation of these materials could be in the range of 150 to 200 $P_{80}$ µm.

At the nominal 100 $P_{80}$ µm sizing, average galena and sphalerite liberation levels approached 50% to 60% when assessed in two dimensions. In G&T's experience, these liberation values are considered well within the usual range for a successful, sequential flotation separation of galena and sphalerite.

Of the un-liberated minerals present in these ground flotation feed samples, a significant fraction of the chalcopyrite was locked with sphalerite. Galena and sphalerite did not display a great affinity for each other. Typically less than 10% of the galena in these samples was locked with sphalerite. The majority of the un-liberated galena and sphalerite, in all samples, was locked with non-sulphide gangue in binary and multiphase composites.

13.2.5.2 Flotation Test Results

Based on the sample mineralogy, a simple lead-zinc sequential separation flow-sheet was devised for treating these materials. The flow-sheet employed a flotation feed sizing of approximately 100 $P_{80}$ µm and included regrinding stages, sited ahead of dilution cleaning, in both lead and zinc cleaner circuits.

A conventional lead-zinc reagent regime, based on a lime modulated process pH control strategy, was used to maximize and maintain the flotation differential between the sulphide minerals. For some samples, a pre-flotation stage, sited ahead of the lead flotation circuit, was included in the flow-sheet to remove a carbonaceous, but often silver-rich, component.

The averaged metallurgical balances, calculated for the best cleaner flotation tests, indicated a lead concentrate grade of 56% by weight lead and 8,000 g/t of silver. Lead recovery was highly variable for the samples evaluated in this program. Between 20% and 95% of the lead in the samples was recovered into the lead concentrate and similar recoveries for silver were achieved. On average, approximately 87% of the lead and 85% of the silver in the samples were recovered into the lead concentrate.

A zinc concentrate extensively diluted by pyrite and assaying 45% by weight zinc and containing 600 g/t silver was also produced. The zinc concentrate contained an estimated 75% of the zinc and approximately 5% of the silver contained in the samples. Definitive estimates of process metallurgy will depend upon the results of replicate locked cycle tests being conducted at some point in the future.

#### 13.2.5.3 Concentrate Quality

Minor element scans were conducted on composite samples of the lead and zinc flotation concentrates produced in the open circuit cleaner tests conducted in this program. The results of these scans, which were performed using assay specific techniques, are shown in Table 13-1. The following notes may be of interest when considering marketing these concentrates to conventional lead and zinc smelters worldwide:

- The lead concentrates, which contained 56% by weight lead, 4% by weight copper and 5% by weight zinc, are comparable to concentrates produced and marketed elsewhere in Mexico. The high silver content of 8,000 g/t of this product could offset many concerns about lead grade.

- The selenium, arsenic, bismuth and antimony contents of the concentrates are elevated beyond the thresholds at which some smelters may impose penalties. However, the deleterious element concentrations for Pitarrilla concentrates are comparable to those routinely observed in flotation concentrates from Mexican lead-silver producers.

**Table 13-1: Concentrate Quality**

| Element | Units | Lead Concentrate | Zinc Concentrate |
|---|---|---|---|
| Copper-Cu | % | 4.2 | 0.8 |
| Lead-Pb | % | 56.3 | 1.3 |
| Zinc-Zn | % | 5.3 | 44.5 |
| Iron-Fe | % | 6.0 | 13.5 |
| Gold-Au | g/t | 0.5 | 0.2 |
| Silver-Ag | g/t | 8,100 | 600 |
| Arsenic-As | g/t | 709 | 3,739 |
| Antimony-Sb | % | 1.5 | 0.2 |
| Cobalt-Co | g/t | 64 | 110 |
| Cadmium-Cd | g/t | 526 | 4,496 |
| Bismuth-Bi | g/t | 376 | 18 |
| Mercury-Hg | g/t | 20 | 21 |
| Nickel-Ni | g/t | 150 | 220 |
| Fluorine-F | g/t | 32 | 36 |
| Selenium-Se | g/t | 280 | 27 |
| Phosphorus-P | g/t | 4,220 | 687 |
| Silica- $SiO_2$ | % | 1.8 | 2.0 |
| Calcium Oxide-CaO | % | 0.4 | 0.2 |
| Aluminum Oxide-$Al_2O_3$ | % | 0.03 | 0.03 |
| Magnesium Oxide-MgO | % | 0.25 | 0.19 |
| Manganese Oxide-MnO | % | 0.06 | 0.24 |

Notes:

a) These composites were constructed from equal weights of lead and zinc concentrate from tests 20 to 37. There was no remaining lead concentrate from these tests.

b) Cadmium in the zinc concentrate was re-assayed at 4,200 g/t.

### 13.2.6 G&T Test Program – KM1971 - 2007

These metallurgical studies were conducted on three composites representing the lithologies of the deposit. Drillcore from the G&T Test Program-KM 1889 was re-processed to make composites representing the Breccia Contact, the Basal Conglomerate, and the Sediments lithologies that were used in this test program. The objectives of the metallurgical test program were to:

- Conduct chemical, mineralogical and modal analyses on the three composites. Determine mineral compositions and mineral fragmentation profiles and thereby assess the flotation treatment options of these composites in order to maximize silver recovery; and
- Perform kinetic rougher flotation tests, open circuit batch cleaner flotation tests, and locked cycle flotation tests to outline some of the details of the flotation treatment scheme; and
- Perform rougher flotation concentrate regrind tests and cleaner flotation tests to investigate the effect on concentrate grades and metal recovery; and
- Determine the minor element contents of typical examples of flotation concentrates; and

- Investigate silver recovery from flotation tailings using cyanide leaching technology.

All three composites represented low sulphide content mineralisation in which pyrite and a moderate interstitial iron sphalerite were the dominant sulphides. Analysis of high grade zinc concentrate samples generated by flotation testing indicates 8% to 9% by weight interstitial iron in the sphalerite lattice. Galena and a range of copper sulphides including chalcopyrite and tetrahedrite were present as ancillary minerals. Trace amounts of pyrrhotite, arsenopyrite and iron oxide minerals were also recorded in all composites.

#### 13.2.6.1 Mineral Fragmentation

The results of liberation assessment determined at simulated, flotation feed sizings of approximately 130 $P_{80}$ µm indicate that flotation feed sizings in the range of 150 $P_{80}$ µm to 200 $P_{80}$ µm will provide adequate mineral liberation for the design of a successful two-product flotation flow sheet for the separation of galena and sphalerite. Also, mutual interlocking of galena and sphalerite in these samples was a relatively rare occurrence. Binaries of this particular class accounted for about 5% of the galena but less than 1% of the sphalerite in these samples. Furthermore, instances of interlocking between galena and copper sulphides were very rare.

#### 13.2.6.2 Silver Occurrences

Statistical analyses of assay data from key flotation test products were used to relate silver deportment to that of the carrier minerals. The results of these statistical exercises for the three lithologies indicated the following trends in silver deportment:

- Almost all of the silver in the three composites tracked galena through all stages of the flotation separation process; and
- There was no statistical evidence to indicate that silver is associated with, or behaved like any other mineral, in the mineralisation matrix; and
- The selection of treatment conditions designed to maximize galena recovery will automatically maximize silver recovery into the lead concentrate.

#### 13.2.6.3 Flotation Test Results

Rougher flotation kinetic tests were initially conducted to assess reagent demand for the samples. Additional rougher tests were executed in which the principal variable probed was the effect of flotation feed size on response.

The test data revealed that, within the nominal flotation feed size range, 100 to 190 $P_{80}$ µm , flotation performance was essentially constant. Further, these same tests showed that solids mass-pulls to the lead rougher concentrate of about 5% were sufficient to ensure 90% lead recovery into that stream. In the zinc rougher circuit, solids mass-pulls of 6% to 10%, dependent upon the sample, were required to approach 90% zinc recovery.

The cleaner circuit testwork was initially focused upon determining the reagent balance required in the cleaner stages to maximize the flotation selectivity between galena and sphalerite. More critical to process design was the outcome of testwork performed at a later date which was focused upon optimization of the lead and zinc regrinding stages. The results of this work indicated that the lead regrind product size should be about 20 $P_{80}$ µm to optimize lead circuit performance, and the zinc regrind product sizing should be about 30 $P_{80}$ µm.

Using the suite of treatment conditions identified in both the kinetic and batch cleaner tests, replicate locked cycle testwork was executed on all three lithologies. The Sediments composite required the use of a pre-flotation stage to remove a naturally floatable component from the rock ahead of lead rougher flotation, but it was not necessary for the successful flotation treatment of the Basal Conglomerate or the Breccia Contact composites. All three types responded favorably to the locked cycle procedures and concentrate quality was acceptable.

Residual silver contained in the pyrite-rich tailings streams from the two-product separation process accounted for about 15% of the silver in the rock. Samples of the process tailings were subjected to limited cyanidation leaching studies to determine if any additional silver could be recovered using this technique.

#### 13.2.6.4 Concentrate Quality

The lead and zinc concentrates produced in selected locked cycle tests were subjected to minor element scans. The resultant assay data for the lead and zinc concentrates is shown in Table 13-2.

**Table 13-2: Minor Element Compositions of the Concentrates by Rock Type**

| Element | Lead Concentrate Assay (g/t) | | | Zinc Concentrate Assay (g/t) | | |
|---|---|---|---|---|---|---|
| | Sediments | Basal | Breccia | Sediment | Basal | Breccia |
| Antimony- Sb | 13,400 | 1,170 | 29,660 | 754 | 470 | 2,756 |
| Arsenic-As | 832 | 458 | 1,229 | 160 | 65 | 204 |
| Cobalt-Co | 32 | 28 | 60 | 82 | 66 | 58 |
| Cadmium-Cd | 380 | 602 | 1,480 | 4,600 | 3,720 | 3,720 |
| Bismuth-Bi | 120 | 882 | 284 | 38 | 30 | 54 |
| Mercury-Hg | 4 | 1 | 75 | 12 | 7 | 88 |
| Nickel-Ni | 168 | 72 | 104 | 50 | 52 | 60 |
| Fluorine-F | 36 | 25 | 52 | 47 | 28 | 22 |
| Selenium-Se | 122 | 259 | 163 | 19 | 72 | 22 |
| Phosphorus-P | 28 | 65 | 28 | 39 | 45 | 44 |
| Silica-$SiO_2$ | <0.01 | <0.01 | 0.08 | <0.01 | 0.19 | 0.89 |
| Calcium Oxide-CaO | 459 | 310 | 220 | 1,853 | 1,300 | 2,120 |
| Aluminum Oxide-$Al_2O_3$ | 2,404 | 1,367 | 2,434 | 3,400 | 2,189 | 5,239 |
| Magnesium Oxide-MgO | 1,016 | 615 | 836 | 1,263 | 730 | 1,503 |
| Manganese-MnO | 342 | 425 | 1,227 | 2,360 | 2,687 | 2,849 |

Notes: The assays for silica are shown as weight percent.

The mineral compositions of the lead and zinc flotation concentrates were determined using standard modal techniques. The results of these mineral composition assessments indicated that:

- The lead concentrates typically contained about 60% by weight galena. The remainder of the lead concentrate mass was occupied principally by chalcopyrite and sphalerite with lesser amounts of gangue and pyrite.

- The zinc concentrates assayed 50% by weight zinc, but they typically contained about 85% by weight sphalerite, principally because of the higher interstitial iron content of the sphalerite. The dominant diluents were pyrite and non-sulphide gangue together with smaller quantities of chalcopyrite and galena.

### 13.2.7 G&T Test Program – KM2056 - 2008

These metallurgical studies were conducted on drillcore and assay reject samples to provide information for treatment of five composites representing Basal Conglomerate, C-Horizon (Lower Andesite), and Sediments (Peña Ranch Fm) rock types. Test procedures included rougher flotation, open circuit batch cleaner flotation, and locked cycle flotation.

Samples were analyzed to determine mineral composition, mineral liberation, and silver occurrence.

The mineral composition for five of the composites was investigated by modal analysis techniques. The analysis indicated that galena, sphalerite, and copper sulphide minerals were present with pyrite.

#### 13.2.7.1 Silver Occurrence

Statistical analysis of flotation data indicates that copper sulphide and galena are the dominant silver carriers. Also, that the proportion of silver associated with the copper sulphide minerals and galena varies by composite.

#### 13.2.7.2 Mineral Fragmentation

The mineral fragmentation profile for the composites was determined using standard methodologies. Mineral deportment by class of association was determined at a nominal flotation feed sizing of 120 $P_{80}$ µm.

The analysis indicated that:

- The majority of the non-sulphide gangue host minerals are very effectively liberated from the sulphide minerals at a nominal size of 80% by weight passing $P_{120}$ µm and in all of the samples, except one composite (with high copper) , the galena and sphalerite liberation levels exceeded the sulphide liberation levels typically exhibited at successfully operating lead-zinc flotation plants; and

- Regrinding of rougher flotation concentrates will liberate galena and sphalerite from non-sulphide gangue and improve cleaner concentrate grades; and

- Interlocking of galena and sphalerite accounts for a small fraction of these mineral occurrences.

13.2.7.3 Flotation Test Results

The flotation response of the five composite samples was assessed using three laboratory flotation test types: batch rougher flotation tests, open circuit batch cleaner tests, and locked cycle tests.

Rougher flotation test results indicated that between 94% and 97% of lead was recovered into a lead rougher concentrate containing about 8% of the feed mass. Tests conducted on Composite B indicated that 92% and 95% of the copper and lead sulphides, respectively, were recovered into a bulk concentrate containing about 16% of the feed mass.

Rougher flotation tests were also performed on two composites that were determined to be oxide material. These tests yielded poor flotation results and have not been included in the sulphide rock evaluation.

Cleaner flotation testing indicated that the process will successfully produce saleable grade lead and zinc concentrates from most of the composites. On average, 91% of the lead and 85% of the zinc were recovered into lead and zinc concentrates assaying 64% by weight lead and 51% by weight zinc, respectively. The lead concentrate contained variable amounts of copper, ranging between 1.3% and 4.2% by weight. Silver contents in the lead concentrates were high, averaging 10,000 g/t.

The exception to the above findings was in the results for the concentrate produced from composite B, a high copper composite. In this case, a true, bulk copper-lead concentrate was produced assaying about 40% by weight, combined copper and lead.

Locked cycle testing indicated that:

- Except for the test results of testing composite sample B, more than 90% of the lead was recovered into a lead concentrate, assaying about 68% by weight lead and 10,500 g/t silver.

- Except for the test results of testing composite sample B, more than 90% of the zinc was recovered into a zinc concentrate, assaying about 50% by weight zinc and 796 g/t silver.

- For one composite (with high copper), 92% of the lead and 86% of the copper was recovered into a lead (bulk) concentrate, assaying 19% by weight lead, 19% by weight copper, 6.9% by weight zinc, and 4,200 g/t silver. In addition, 64% of the zinc was recovered into a zinc concentrate, assaying 25% by weight zinc and 285 g/t silver.

13.2.7.4 Concentrate Quality

Chemical analysis conducted on lead and zinc flotation concentrates produced by the locked cycle testing are presented in Table 13-3.

**Table 13-3: Chemical Compositions of Concentrate Products from Locked Cycle Tests**

| Component | | Composite D | | Composite B | | Composite A | | Composite C | | Composite E | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Lead | Zinc | Lead | Zinc | Lead | Zinc | Lead | Zinc | Lead | Zinc |
| Copper-Cu | % | 1.83 | 0.63 | 19.1 | 0.88 | 3.86 | 0.67 | 1.43 | 0.43 | 4.13 | 4.02 |
| Lead-Pb | % | 68.8 | 0.56 | 16.9 | 0.36 | 6.32 | 0.83 | 73.3 | 1.05 | 66.3 | 0.43 |
| Zinc-Zn | % | 2.27 | 50.9 | 8.18 | 48.8 | 4.29 | 51.9 | 4.38 | 57.4 | 1.42 | 47.1 |
| Iron-Fe | % | 3.70 | 12.7 | 2.18 | 13.0 | 4.10 | 10.0 | 2.80 | 10.9 | 5.10 | 11.6 |
| Silver-Ag | g/t | 10,651 | 291 | 3,910 | 177 | 18,801 | 1,071 | 3,995 | 150 | 9,046 | 1,666 |
| Antimony-Sb | g/t | 26,900 | 872 | 2,060 | 134 | 20,280 | 1,214 | 2,928 | 116 | 10,020 | 1,716 |
| Arsenic-As | g/t | 2,397 | 303 | 184 | 74 | 453 | 123 | 1,217 | 195 | 322 | 52 |
| Cobalt-Co | g/t | 30 | 42 | 34 | 80 | 8 | 48 | 52 | 82 | 8 | 68 |
| Cadmium-Cd | g/t | 176 | 4,660 | 320 | 3,960 | 346 | 4,520 | 366 | 4,980 | 132 | 4,420 |
| Bismuth-Bi | g/t | 7,440 | 120 | 154 | 36 | 686 | 26 | 274 | 22 | 702 | 24 |
| Mercury-Hg | g/t | 0.8 | 4.4 | 0.5 | 5.6 | 3.0 | 5.6 | 1.0 | 7.3 | 1.7 | 11.0 |
| Nickel-Ni | % | 0.004 | 0.002 | 0.013 | 0.007 | 0.007 | 0.003 | 0.016 | 0.007 | 0.008 | 0.005 |
| Fluorine-F | g/t | 16 | 40 | 40 | 2 | 74 | 29 | 59 | 2 | 20 | 33 |
| Selenium-Se | g/t | 299 | 53 | 955 | 99 | 122 | 70 | 157 | 22 | 111 | 64 |
| Gold-Au | g/t | 1.73 | 0.05 | 0.09 | 0.02 | 0.65 | 0.07 | 0.66 | 0.06 | 0.26 | 0.04 |
| Phosphorus-P | g/t | 75 | 31 | 30 | 29 | 24 | 39 | 26 | 30 | 57 | 36 |
| Silica- $SiO_2$ | % | 0.61 | 0.50 | 1.05 | <0.1 | 0.53 | 1.37 | 1.63 | 0.61 | 0.70 | 2.14 |
| Calcium Oxide-CaO | % | 0.89 | 0.14 | 0.18 | 0.58 | 0.04 | 0.15 | 0.03 | 0.12 | 0.76 | 0.19 |
| Aluminum Oxide-$Al_2O_3$ | % | 0.33 | 0.16 | 0.13 | 0.41 | 0.13 | 0.21 | 0.12 | 0.34 | 0.47 | 0.19 |
| Magnesium Oxide-MgO | % | 0.19 | 0.04 | 0.07 | 0.16 | 0.09 | 0.13 | 0.17 | 0.31 | 0.32 | 1.10 |
| Manganese Oxide-MnO | % | 0.05 | 0.31 | 0.03 | 0.27 | 0.03 | 0.20 | 0.02 | 0.19 | 0.03 | 0.18 |

Note: %-percent by weight

### 13.2.8 G&T Test Program – KM2232 - 2009

These metallurgical studies were conducted on multiple samples to further advance the development of the metallurgical treatment scheme for Pitarrilla rock. The objectives of the metallurgical test program were to:

- Optimize the treatment parameters and flow sheet to produce saleable grade lead and zinc concentrates;
- Conduct pilot scale simulation for the flotation process to verify laboratory bench test results and to produce concentrate samples for further evaluation; and
- Investigate variability in results across mineralised mine zones with respect to rockhardness and flotation response.

The mineral compositions of the Andesite, Basal Conglomerate, C-Horizon, and Sediments composite samples were investigated by modal analysis techniques. The analysis indicated that galena, sphalerite, and copper sulphide minerals were present with pyrite.

#### 13.2.8.1 Silver Occurrence

Analysis of Basal Conglomerate flotation test products indicated that slightly more than half of the silver in a bulk copper-zinc concentrate behaved like copper sulphide minerals through the flotation process. The remainder of the silver followed the principal lead mineral, galena.

#### 13.2.8.2 Mineral Fragmentation

The mineral fragmentation profile for the composites was determined using standard methodologies. Mineral deportment by class of association was determined at a normal flotation feed sizing of 150 $P_{80}$ µm.

The usual target mineral liberation of about 50% was achieved, or was exceeded. Further, gangue liberation easily surpassed the rougher flotation design threshold of 90%. The data indicates that mineral liberation was scarcely influenced by flotation feed sizing within the range of 100 $P_{80}$ µm to 300 $P_{80}$ µm.

Of equal importance was that interlocking between galena and sphalerite was limited, with galena-sphalerite binaries typically containing 60% galena.

#### 13.2.8.3 Rock Hardness

Tests were performed to determine the Bond ball mill work index for the four rock type composites. The results of the Bond Work Index are presented in Table 13-4.

**Table 13-4: Estimate of Rock Hardness**

| Composite | Bond Work Index | |
|---|---|---|
| | Metric kWhr/mt | Imperial kWhr/st |
| Andesite | 20.4 | 18.5 |
| Basal Conglomerate | 18.4 | 16.7 |
| C Horizon | 18.8 | 17.2 |
| Sediments | 17.9 | 16.3 |

Additional rock hardness test-work was performed on the variability testing samples. This work was done using a comparative grindability technique. The result of this work is presented in Table 13-5.

**Table 13-5: Variability in Rock Hardness**

| Composite | Particle Sizing $K_{80}$ µm | | | |
|---|---|---|---|---|
| | Min | Max | Average | Standard Deviation |
| Andesite | 73 | 666 | 182 | 138 |
| Basal Conglomerate | 67 | 352 | 138 | 70 |
| C Horizon | 75 | 287 | 143 | 60 |
| Sediments | 82 | 376 | 166 | 73 |
| UP Samples | 107 | 110 | 109 | 2 |
| RD Samples | 148 | 525 | 331 | 98 |

The relatively small scatter of the comparative index data indicates consistency in grindability across the major mineralised zones.

#### 13.2.8.4 Flotation Test Results

Flotation response was examined for each of the four major rock composite samples. The laboratory flotation test types included examination of the rougher flotation rate (or kinetic test), the open circuit batch cleaner test and the locked cycle test.

The kinetic flotation tests were used primarily to link solids mass-pulls, for lead and zinc circuits, to determine metal recoveries in the associated rougher concentrates. These tests are designed to probe a range of variables which might include flotation feed sizing, flotation time and reagent dosages. Specifically, the kinetic flotation test results here indicated that:

- For all composites, variations in the flotation feed size exerted very little influence on the flotation performance in either the lead or the zinc rougher circuits;
- In the rougher circuit, an average lead recovery of 85% to 90% was achieved at solids mass-pull of about 3% of the flotation feed mass into the lead rougher flotation concentrate. The Sediments and the Basal Conglomerate samples produced the best results in that they averaged about 10% better lead recovery than the other composites, at an equivalent solids mass-pull; and

- In the zinc rougher circuit, zinc recovery of 90% was achieved, at a solids mass-pull of 6% to 8% of the flotation feed mass into the zinc rougher flotation concentrate. The solids mass-pull recovery data has been normalised to the metal contained in the zinc rougher feed stream. The Sediments and Basal Conglomerate samples produced the best results of the samples tested. Considering all samples tested, the difference between the response extremes was equivalent to about 5% zinc recovery.

The data collected from the open circuit batch cleaner test indicated that:

- Flotation results from the Sediments and Basal Conglomerates samples were better than results from the Andesite and C Horizon samples;

- The lead and the zinc grade recovery relationships indicated that no mineralogical constraints, especially mineral inter-locking, were adversely impacted by the upgrading process;

- The lead rougher concentrates were upgraded from an average of about 15% by weight lead to about 60% by weight lead. Lead losses in the lead cleaners were 7% to 10%; and

- The zinc rougher concentrates averaged 15% by weight zinc content and were readily upgraded to 50% by weight zinc. Zinc losses in the zinc cleaners were about 5%.

Locked cycle tests were performed to test flotation parameters and response in a continuous flotation separation process. The cycle tests were conducted using flotation nominal process feed sizes of 200 $P_{80}$ µm and regrind product sizes between 15 $P_{80}$ µm and 30 $P_{80}$ µm.

Locked cycle test data indicated that:

- Despite the variations in metal contents and the flotation responses in preliminary tests, process metallurgy is consistent for all four composite types;

- Lead concentrates, marginally contaminated by copper sulphides, were an average of 50% to 65% by weight lead and 5,500 to 7,000 g/t silver. Lead and silver recoveries, ranged from 70% to 87%, and 60% to 75%, respectively; and

- The grade of the zinc concentrates were approximately an average of 48% by weight zinc. In all cases, the combined copper and lead contents were below the usual combined smelter penalty threshold of 3% by weight. Silver content ranged from 275 to 700 g/t. Zinc recoveries to the zinc concentrates ranged from approximately 80% to greater than 90%. Approximately 10% to 20% of the silver content of the composites was captured into the zinc concentrates.

#### 13.2.8.5 Variability Testwork

Determinations of flotation response in a variability program were conducted by running all of the variability composite samples through a standard open circuit batch cleaner test. In this case, the batch cleaner test involved constant grinding power input, sequential lead and zinc flotation stages, regrinding of the rougher concentrates using fixed power input, and three stages of open circuit dilution cleaning.

The results of the lead circuit variability testing indicated that:

- The average lead flotation results were similar across all six composites;
- The lead concentrates assayed between 40% and 55% by weight lead and 4,000 g/t to 10,000 g/t silver. Average lead and silver recoveries approximated 65% and 55%, respectively (not accounting for recovery or loss in the open circuit batch cleaner tailing streams);
- The lead concentrates and particularly those from the C-Horizon samples, contained elevated copper levels; and
- Generally, antimony levels were high and arsenic was recorded in perceptible amounts in all lead concentrates.

The results of the zinc circuit variability testing indicated that:

- The zinc concentrate grade consistently averaged approximately 45% by weight zinc and contained approximately 70% of the zinc content from the raw composite samples;
- The zinc concentrates contained between 200 and 1,000 g/t silver, equivalent to approximately 12% silver recovery; and
- The combined copper and lead content of the zinc concentrates was, in most cases, well below the smelter penalty threshold of 3% by weight combined metals. The combined arsenic and antimony contents of the zinc concentrates were in excess of 0.3% by weight.

#### 13.2.8.6 Phase II – Pilot Plant Operation

Several individual pilot plant test runs, each of eight to ten hours duration, were conducted on the four composite samples from KM2232. The results of these pilot scale runs confirmed that the basic treatment protocols developed in the laboratory scale testing could be used in plant operation.

The test results indicated that:

- The lead concentrates assayed about 55% by weight lead, 3% by weight copper, and 6,000 g/t silver on average. Average lead and silver recoveries were approximately 85% and 75%, respectively;
- The zinc concentrates assayed about 46% by weight zinc and 350 to 500 g/t silver. Copper and lead in the zinc concentrate assayed less than 3% by weight combined.

Average zinc and silver recoveries were approximately 90% and less than 15%, respectively; and

- Analyses of the grade-recovery relationships for both lead and zinc flotation circuits indicated that concentrate grade can be balanced against metal recovery.

Lead flotation concentrates produced in the pilot plant, from the Basal Conglomerate composite, were used in a lead-copper flotation separation test. In this test, galena flotation was suppressed and the copper bearing sulphide minerals were floated.

The results indicated that:

- The copper concentrate contained 24% by weight copper and about 15% by weight combined lead and zinc. The concentrate also contained about 85% of the copper originally recovered into the lead-copper concentrate; and
- The lead concentrate was largely devoid of copper sulphide minerals and much of the silver was removed. This indicates that a significant amount of the silver is associated with the copper sulphide minerals.

These tests indicate that the final lead concentrate would assay 57.5% by weight lead and about 0.4% by weight copper, and it would contain better than 95% of the galena and about 60% of the silver that was in the lead-copper concentrate.

#### 13.2.9 McClelland Laboratories Test Program – 3553 and 3553-01 2012

A total of 168 individual diamond drillcore interval samples were selected to prepare 45 variability composite samples based on grade and location. The locations tested were Cordon Colorado, Peña Dyke, South Ridge, East South Ridge, Breccia Ridge and Javelina Creek.

Two phases of grind size optimization testing were completed. The initial phase was performed upon both the location composite samples and the Master composite sample. Grind sizes that were tested were $P_{80}$ 75µm, 53µm, 38µm and 25 µm. A summary of the results of this testwork is presented graphically in Figure 13-1.

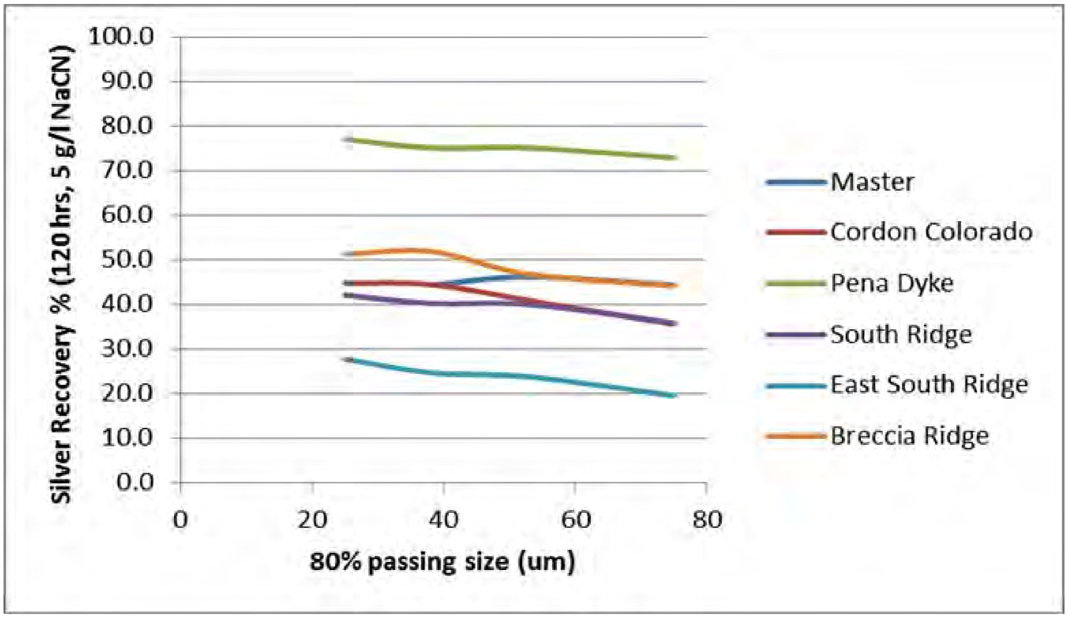



**Figure 13-1: Silver Recovery versus Passing Size (Fine Grind)**
Source: M3, 2012

The second phase of grind size optimization testing was performed at coarser grinding sizes. Grind sizes tested were $P_{80}$ 150 µm, 106 µm, and 75 µm. A summary of the results of this testwork is presented graphically in Figure 13-2.

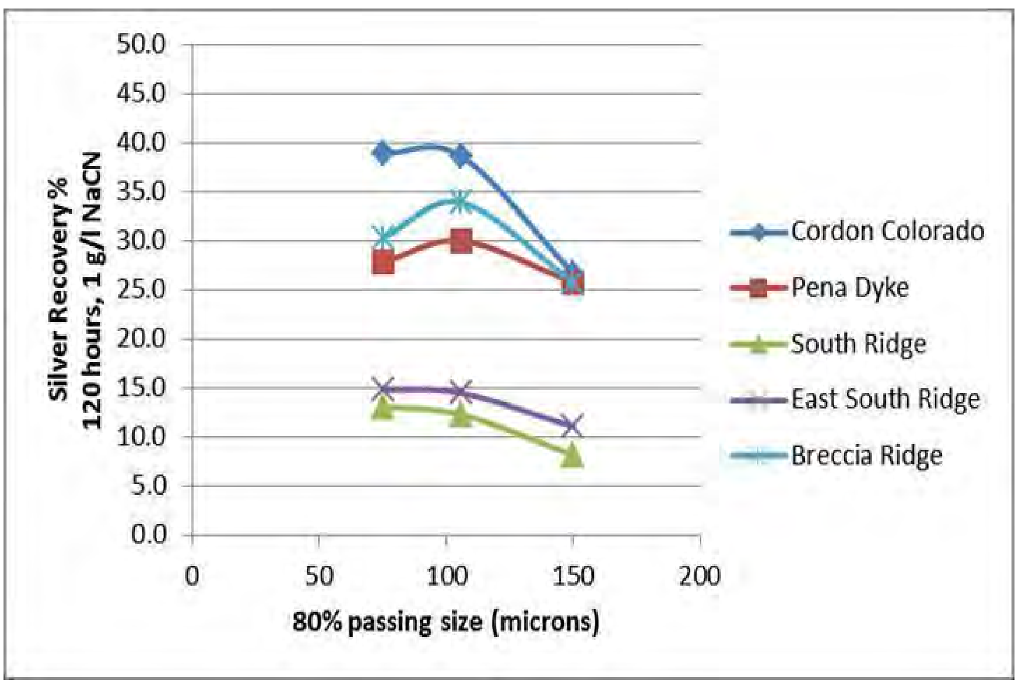



**Figure 13-2: Silver Recovery versus Passing Size (Coarser Grind)**
Source: M3, 2012.

Additional tests were performed to explore the effect of leach solution cyanide concentration on silver extraction. Tests were performed at a grind size distribution of $P_{80}$ 38 µm and cyanide solution concentrations of 0.5, 1, and 5 grams per litre for the master composite sample and 1, 2, and 5 grams per litre for the location composite samples. A summary of the results of this testwork is presented graphically in Figure 13-3.

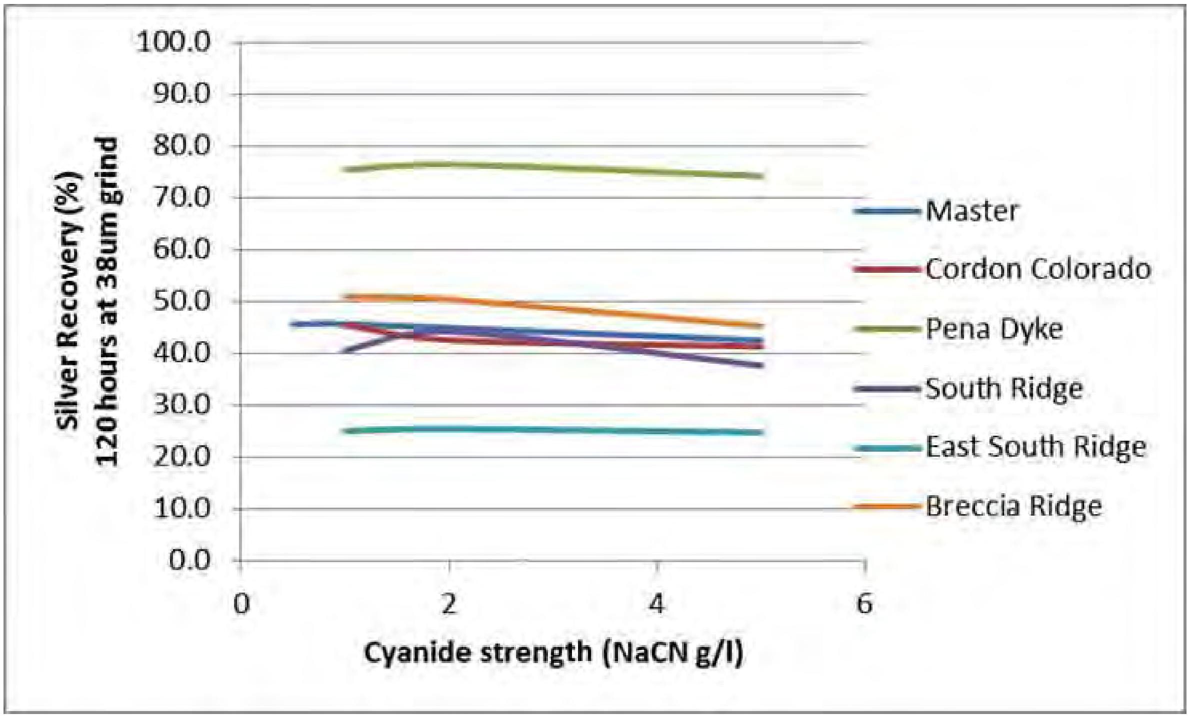



**Figure 13-3: Silver Recovery versus Cyanide Solution Strength**
Source: M3, 2012

The results from the bottle roll tests were used to assess the kinetics of silver leaching. In nearly every test, the leach extraction was determined at the 2, 4, 8, 24, 48, 72 and 96 hour leach time period and the leach time was often extended to 120 hours. In nearly every case, the shape of the silver extraction versus time plot resulted in a consistently shaped graph. The graph indicates a rapid initial extraction rate for the first 12 hours, reaching a maximum extraction value during the first 24 hours of leaching, and finishing with very little additional extraction to the completion of the test. A typical leach extraction profile is shown in Figure 13-4.

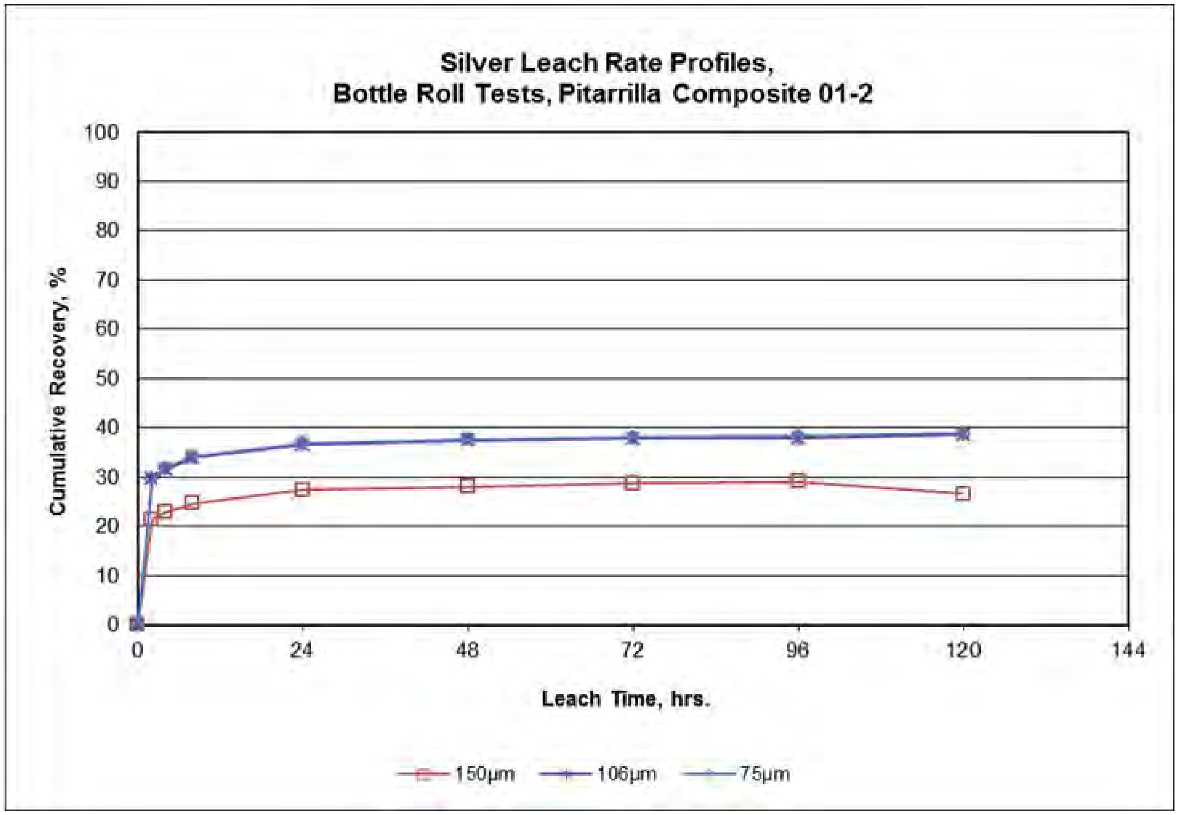



**Figure 13-4: Cumulative Silver Recovery versus Leach Time**
Source: M3, 2012

With the mineralogical understanding of the oxide mineralisation (containing coarse and fine silver halides and very fine sulphides), the kinetic leaching profile is logical. The initial, fast leaching silver is recovered from the coarse silver halide minerals and then, the much slower leaching silver is recovered from the finer, partially liberated, silver halides. The locked, very fine silver sulphides are not leached. The proportion of locked, silver in fine sulphides, limits the maximum recovery.

In addition to silver assaying and 32-element ICP analysis by drillhole interval, an additional analytical method known as a “cyanide shake test” or “Hot CN” was used. The test is described by an ALS-Chemex laboratory procedure identified by the analysis code “Ag-AA13HY”.

The cyanide shake test method uses a pulverised sample that is leached at 60°C in a cyanide/caustic solution for a duration of six hours. The leach solution is then analysed for silver by atomic absorption spectroscopy. The direct comparison of any sample’s silver content (Ag-ppm) and the corresponding Ag-AA13HY solution result (Ag-ppm) gives an indication of the possible maximum silver recovery.

An important result of all the leaching testwork was the determination of the “recovery factor”, which would convert the Ag-AA13HY drill interval result to the corresponding bottle roll recovery result at the plant design operating point (grind size/leach solution cyanide strength/leach time). The “recovery factor” was variable with grind size, cyanide strength, and leach duration. The "recovery factor" is illustrated graphically in Figure 13-5. In Figure 13-5, the hot cyanide extraction determined by Ag-AA13HY for the samples was plotted versus the extraction determined from bottle roll tests (run at 75 µm grind size, one gram per liter cyanide solution strength, and 48-hours leach time). The equation for the best fit line through the data points indicates a correlation of 0.9508.

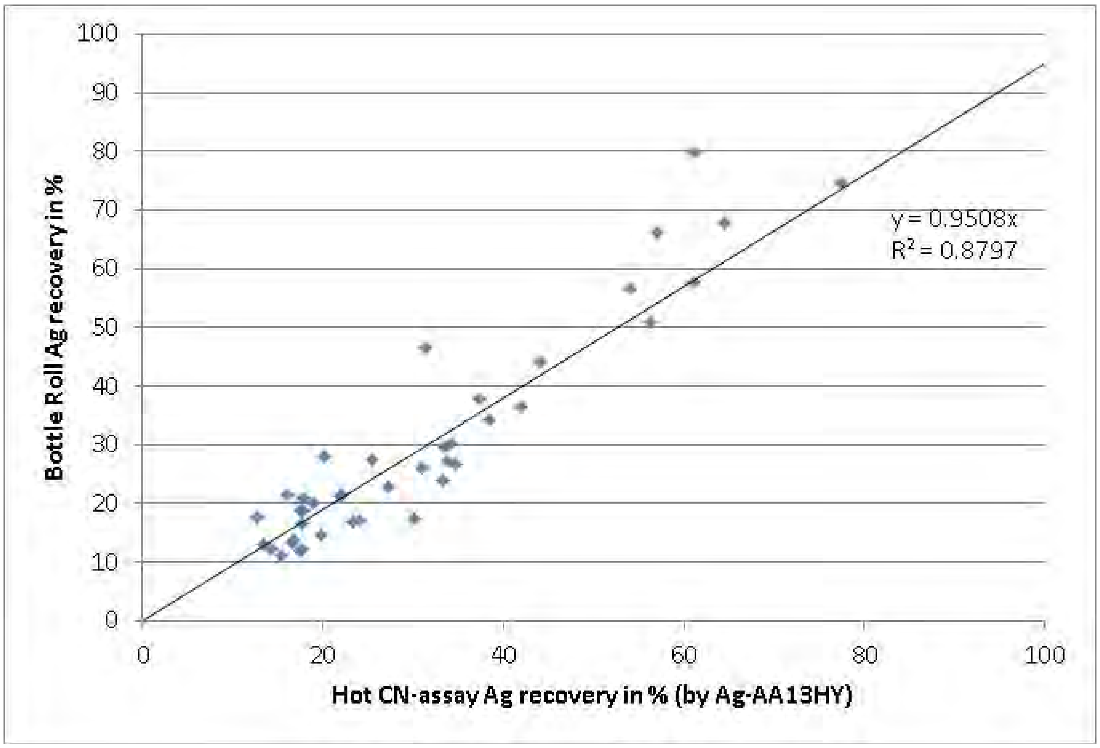



**Figure 13-5: Comparison of Hot Cyanide Ag Recovery % Versus Bottle Roll Recovery**
Source: M3, 2012

### 13.2.10 SGS Test Programs – 40-12 and 22-12 2012

These metallurgical studies were conducted on multiple samples to further advance the understanding of the metallurgical performance for Pitarrilla rock. The objectives of the metallurgical test program were to test, by using the Pitarrilla standard flotation and cyanide leaching conditions, variability samples from six drillholes.

The variability samples were generated from ten metre intervals from six drillholes. These drillholes were selected such that they represented spatially the entire deposit. The first intervals' depth location was selected based upon the geologic logging records and the assigned oxidation code.

13.2.10.1 Cyanide leaching testwork.

For every composite sample that contained silver greater than 20 g/t, a cyanide leaching test was performed. The test conditions were consistent with the Pitarrilla cyanide leach test standard conditions.

All the results demonstrated that silver recovery by cyanide leaching varied both by head grade, and by depth. As examples Figure 13-6 and Figure 13-7 display the results for drillhole BPD-313.

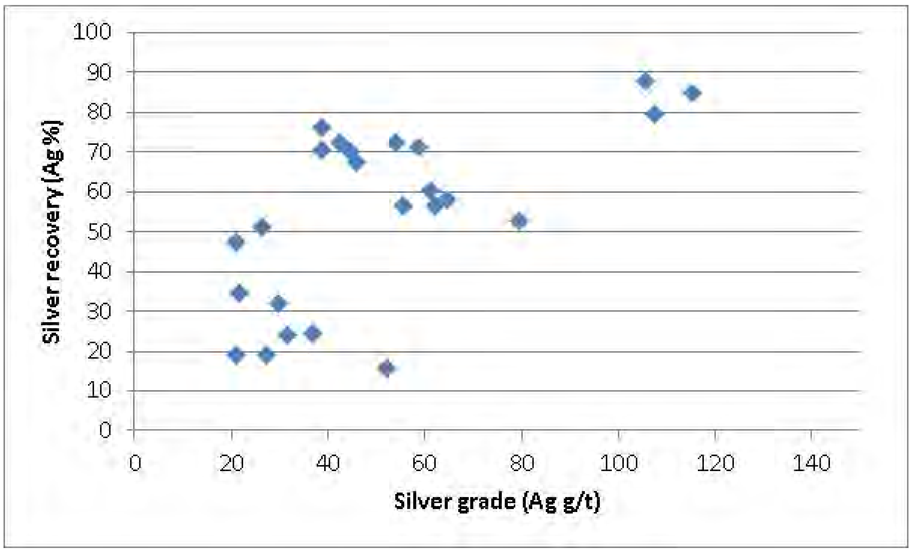



**Figure 13-6: Leaching, Silver Recovery versus SilverHead Grade, Drillhole BPD-313**
Source: M3, 2012

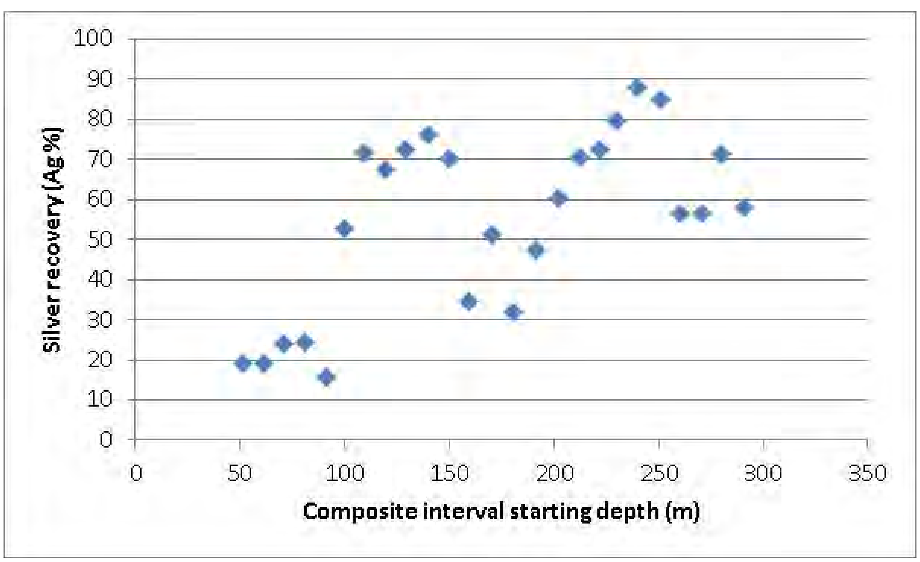



**Figure 13-7: Leaching, Silver Recovery versus Composite Depth, Drillhole BPD-313**
Source: M3, 2012

The effect of a third variable, oxidation code, upon silver recovery, is shown in on Figure 13-8. The oxidation code is a 0 to 5 scale, where 0 is fresh sulphide, and 5 is highly weathered 'oxide' material.

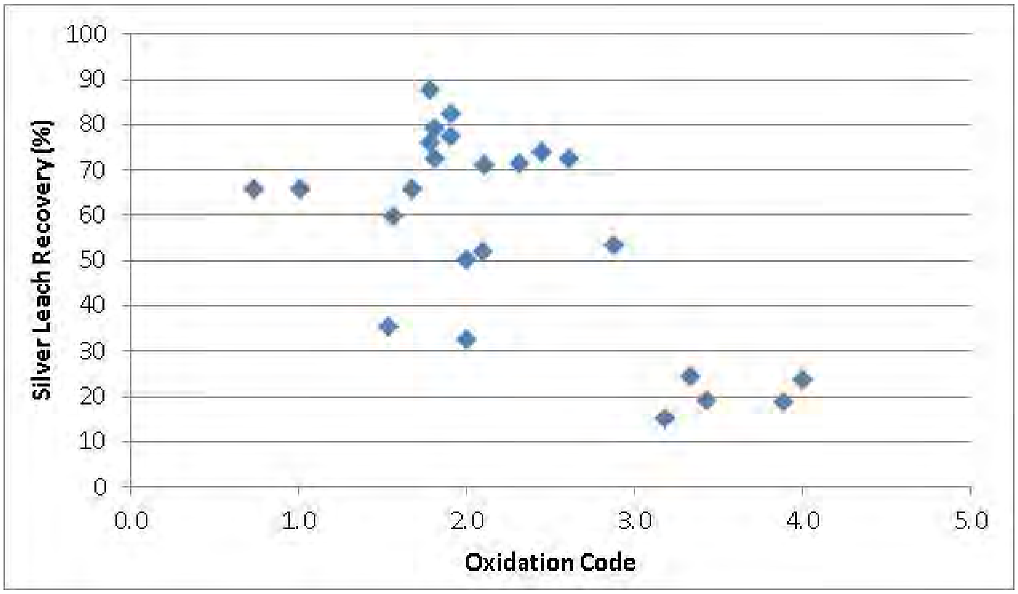



**Figure 13-8, Silver Leach Recovery versus Oxidation Code, Drillhole BPD-313**
Source: M3, 2012

13.2.10.1.1Flotation Testwork

For every composite sample that contained silver greater than 20 g/t, a rougher flotation test was performed. The test conditions were consistent with the Pitarrilla flotation standard conditions. A total of 118 tests were completed.

All the results demonstrated that total silver recovery by flotation varied both by head grade, and by depth. As examples, Figure 13-9 and Figure 13-10 display the results for drillhole BPD-313.

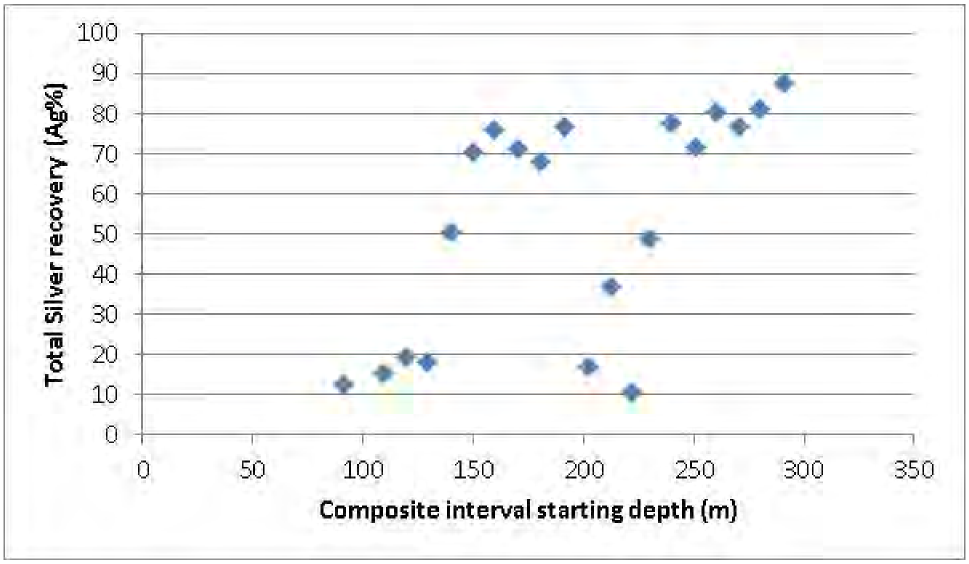



**Figure 13-9: Total Silver Recovery Versus Composite Depth, Drillhole BPD-313**
Source: M3, 2012

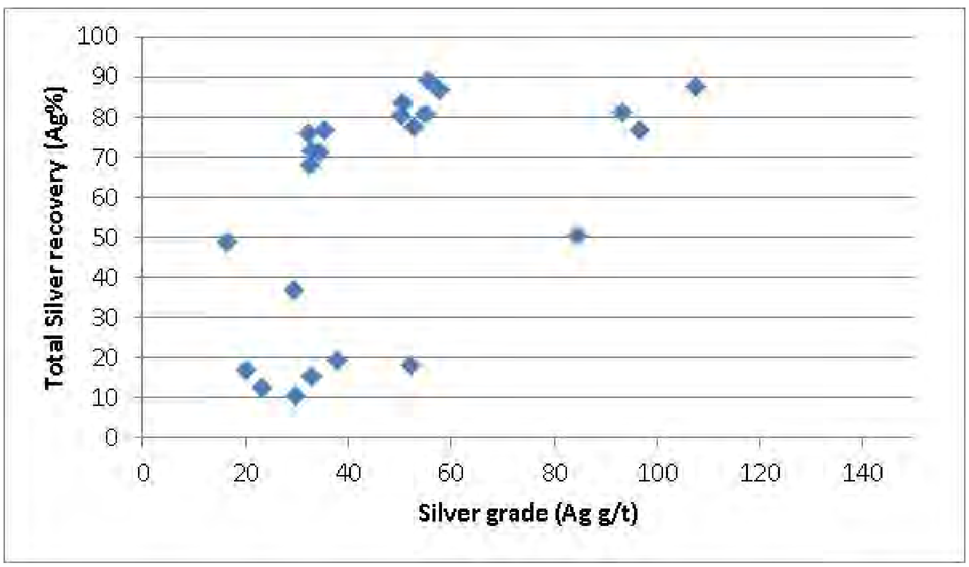



**Figure 13-10: Flotation, Silver Recovery versus Ag Head Grade, Drillhole BPD-313**
Source: M3 ,2012

The effect of a third variable, oxidation code, upon silver recovery, is shown on Figure 13-11. The oxidation code is a 0 to 5 scale, where 0 is fresh sulphide and 5 is highly weathered 'oxide' material.

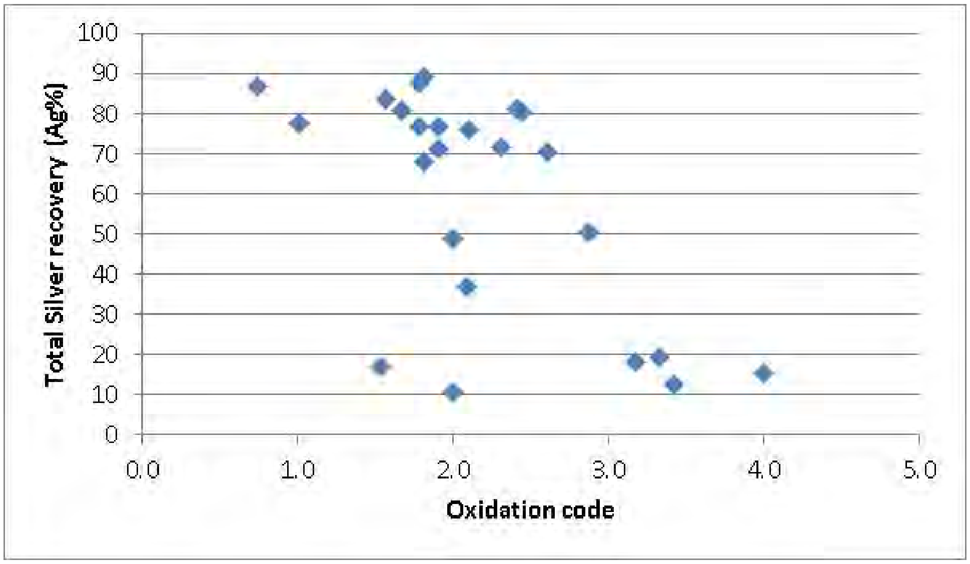



**Figure 13-11: Flotation, Silver Recovery versus Oxidation Code, Drillhole BPD-313**
Source: M3, 2012

### 13.2.11 G&T Test Program – KM3433 2012

These metallurgical studies were conducted on multiple samples to further advance the understanding of the metallurgical performance for Pitarrilla rock. The objectives were to test, by using the Pitarrilla standard flotation and flotation tailings cyanide leaching conditions, master composites and variability samples from the Dacite, Intrusive and Transitional rock types.

The samples were identified by rock type: Dacite Oxide, Dacite Fresh, Intrusive Shallow, and Intrusive Deep.

These variability samples were also used to create the Master composites for the rock types: Oxide, Fresh, Shallow, and Deep.

The chemical and mineral compositions of the Oxide, Fresh, Shallow, and Deep composite samples were investigated by modal analysis techniques. The analysis indicated that galena, sphalerite, zinc oxides and copper sulphide minerals were present with pyrite.

#### 13.2.11.1 Flotation Variability Testwork

Determinations of flotation response in a variability program were conducted by testing the variability composite samples by a standard rougher test, a batch cleaner test, and for three master composites, a complete Locked cycle batch flotation tests.

#### 13.2.11.2 Variability Batch Standard Rougher Test Summary

The results of rougher kinetic tests on the master composites are summarised below:

- For the Deep, Fresh, and Shallow Master Composites, lead recovery to the bulk circuit rougher concentrate ranged from about 65% to 90%.

- Under all conditions tested, for the Oxide Master Composite, lead recovery was poor, ranging from about 10% to 20%, and zinc recovery ranged from about 20% to 35%.

- Zinc recovery, to the combined bulk and zinc rougher concentrates, ranged between 45% and 90% for the Deep, Fresh, and Shallow Master Composites. Between 20% to 30% of this reported to the bulk rougher concentrate.

- Silver recovery, to the bulk rougher concentrate, ranged from about 45% to 90% for the four master composites tested.

13.2.11.3 Variability Batch Cleaner Test Summary

Open circuit batch cleaner flotation test results, for the master composites are summarised as follows:

- For the Deep, Fresh, and Shallow Master Composites, lead recovery to the final bulk concentrate ranged from about 40% to 60%. At these recovery levels, lead content, in the final bulk concentrate, ranged from about 40% to 60%. Lead recovery for the Oxide Master Composite was very poor at about 10 % and at a grade of 10% lead.

- Silver recovery, to the final bulk concentrate, ranged from about 35% to 65%, with silver grades ranging between about 5,000 to 10,000 g/t. The Oxide Master Composite had the lowest silver recovery and grade, at 35% and about 5,000 g/t, respectively.

- The Deep Master Composite had the best zinc metallurgical performance, with about 60% zinc recovery at 50% zinc grade. The Fresh and Shallow Master Composites had the lower zinc recoveries to the final zinc concentrate, ranging between about 30% to 60% at zinc grades ranging from about 30% to 40%. The Oxide Master Composite had less than 10% zinc recovery at less than 10% zinc grade.

13.2.11.4 Master Composite Complete Locked Cycle with Cyanide Leaching of Tailings Summary

A single locked cycle flotation test was completed on each of the Deep, Shallow and Fresh Master Composites. The zinc rougher tail from each test was further treated using a cyanidation bottle roll test. The results of these tests are discussed below:

- Between about 50% and 70% of the lead in the feed was recovered to the final bulk concentrate at lead grades between about 40% and 60%. Silver recovery, to the concentrate, ranged between 59% to 68% at silver grades ranging from 7,000 g/t and 8,500 g/t.

- Zinc recovery to the zinc final concentrate ranged between about 50% to 80%, at zinc grades ranging between 44% to 48%. For the Shallow Master Composite, most of the zinc losses were in the zinc rougher tail, at about 42%.

- After 24 hours of cyanidation, from 65% to 75% of the silver in the zinc rougher tailings' was extracted to solution. The best extraction, at about 75%, was achieved on the zinc rougher tail from the Fresh Master Composite.

### 13.2.12 G&T Test Program – KM3513 2012

These metallurgical studies were conducted on multiple samples to further advance the understanding of the metallurgical performance for Pitarrilla rock. The objectives of the metallurgical test program were to:

- Test, by using the Pitarrilla standard rougher flotation conditions, all variability samples; and

- Test, by using the Pitarrilla standard cyanidation conditions, all variability rougher flotation tailings.

The samples were identified by rock type: Dacite Oxide 2 (5 samples), Dacite Oxide 3 (5 samples), Cordon Colorado Oxide 2 (5 samples), Cordon Colorado Oxide 3 (5 samples) and Intrusive Oxide 3 (5 samples).

These varability samples were also used to create the Master composites for the rock types: Dacite Oxide 2, Dacite Oxide 3, Cordon Colorado Oxide 2, Cordon Colorado Oxide 3 and Intrusive Oxide 3.

The chemical and mineral compositions of the Dacite Oxide 2 and 3, Cordon Colorado Oxide 2 and 3, and Intrusive Oxide 3 composite samples were investigated by modal analysis techniques. The analysis indicated that galena, sphalerite, zinc oxides and copper sulphide minerals were present with minor pyrite.

#### 13.2.12.1 Variability Batch Rougher Test Summary

The flowsheets used in this test program are discussed with the following comments:

- The target primary grind sizing for the rougher flotation tests was 150 $P_{80}$ µm.. Actual grind sizings ranged between 118 $P_{80}$ µm to 151 $P_{80}$ µm.

- Cytec 3418A was used as the main sulphide mineral collector in the bulk flotation circuit. Sodium cyanide was added to the primary grind to depress zinc flotation in the bulk circuit.

- Flotation in the bulk circuit was conducted at a target pH of between 9.1 and 9.5.

- Copper sulphate was used to activate zinc flotation in the zinc rougher circuit. Sodium Isopropyl Xanthate ("SIPX") was used to collect zinc to the zinc rougher concentrate. Zinc rougher flotation was completed at a pH of 11.5.

- The zinc rougher tailings, from each flotation test, was cyanide bottle roll leached for 24 hours with interval sampling at 2, 6 and 24 hours. The target sodium cyanide concentration was 1,000 ppm.

The pH was maintained at 11.5 during the cyanidation test.

#### 13.2.12.2 Variability, Cyanide Leaching of Batch Rougher Flotation Tailings

The cyanidation bottle roll test data showed:

- The 24 hour silver extractions ranged about 21% to 76%. The average 24 hour silver extraction for all 25 variability composites was about 59%. Three of the Cordon Colorado 3 Oxide composites, V2, V3 and V5, had much lower 24 hour silver extractions than the bulk of the samples tested.

- There is a trend between the calculated silver content in the feed and the silver grade in the cyanidation tailings. The samples with lower silver feed grades produced the lowest silver grades in the cyanide tailings.

- There does not appear to be any relationship between the flotation feed primary grind size and the silver assay in the cyanidation tailings. This comment applies only across the range of primary grind sizings generated in these tests (120 to 150 µm) and is a comment about global performance across the 25 samples tested.

- There was significant variation between silver assayed in the zinc rougher tailings and back calculated in the cyanidation feed. These variances could not be confirmed through check assays.

### 13.2.13 SGS Test Program – 50014-001 2009

These metallurgical studies were conducted on multiple samples to provide information on comminution design, flotation and cyanide leaching metallurgy, solid-liquid separation, and environmental stability of flotation tailings. The flotation testing was primarily performed to prepare products for the solid-liquid separation and environmental tests. Cyanide leaching tests were also performed on flotation tailings from flotation tests of an oxide sample.

Environmental testing included standard static and kinetic testing of flotation tailings samples.

#### 13.2.13.1 Comminution Results

A summary of the results from the Bond ball mill work index testing is presented in Table 13-6.

**Table 13-6: Summary of Bond Ball Mill Work Indices**

| Basal Conglomerate | | Environmental | | Sediments | |
|---|---|---|---|---|---|
| **Sample ID** | **Bond $W_i$** | **Sample ID** | **Bond $W_i$** | **Sample ID** | **Bond $W_i$** |
| PD – 157 | 14.7 | PD – 157 Oxide | 13.0 | PD – 157 | 21.0 |
| PD – 176 | 15.8 | PD – 176 Oxide | 21.5 | PD – 176 | 18.9 |
| PD – 181 | 19.7 | PD – 186 Sulphide | 13.0 | PD – 181 | 17.1 |
| PD – 186 | 19.7 | PD – 198 Sulphide | 16.3 | PD – 186 | 22.8 |
| PD – 194 | 14.7 | PD – 157 Horizon | 17.3 | PD – 196 | 21.9 |
| PD – 196 | 17.2 | PD – 176 Rhyolite | 17.9 | PD – 198 | 21.7 |
| PD – 198 | 14.6 | PD – 186 Rhyolite | 17.6 | | |
| | | PD – Andesite | 19.8 | | |

Note: Bond $W_i$- Bond Ball Mill Work Index, metric system

Coarse size samples of Sediments and Basal Conglomerate rock types were composited into two samples for JK drop weight testing. The JK drop test compares rock fragmentation in a standardised test to model the comminution circuit. The test evaluation results in the determination of characteristic parameters which are used in a proprietary software package to predict comminution requirements. The JK drop test modeling indicated a SAG mill-ball mill circuit could be used to grind the rock to flotation feed size.

#### 13.2.13.2 Cyanidation Testing

Ten cyanidation tests were conducted on flotation tailings from the oxide samples. Each of the tests was performed under the same test conditions to generate a suitable amount of tailings material for solid-liquid separation testing. Silver extraction averaged 36% for these tests. The metallurgical response of the oxide material was not investigated in detail in this testwork because processing of the oxide material is considered to be an opportunity for future consideration.

#### 13.2.13.3 Solid-Liquid Separation Testing

Flocculant screening, conventional (static) and dynamic (high-capacity) thickening, pulp rheology, pressure filtration and vacuum filtration tests were conducted on flotation tailings and cyanide residue samples.

Flocculant screening procedures indicated that flotation tailings could be flocculated with approximately 15 g/t to 30 g/t of a medium to high molecular weight, 15% charge density, anionic polyacrylamide flocculant. Leach residue could be flocculated with approximately 25 g/t to 30 g/t of a medium to high molecular weight 7% charge density anionic polyacrylamide flocculant.

Results of static thickening tests for conventional type thickener design gave recommended minimum unit area requirements as shown in Table 13-7.

**Table 13-7: Conventional Thickener Sizing Data**

| Material Tested | Minimum area requirements m²/tpd |
|---|---|
| Basal Conglomerate 100 µm | 0.15 - 0.25 |
| Basal Conglomerate 200 µm | 0.125 - 0.15 |
| Composite 100 µm | 0.125 - 0.15 |
| Composite 200 µm | 0.125 |
| Sediments 100 µm | 0.25 - 0.35 |
| Sediments 200 µm | 0.15 - 0.25 |
| Cyanide Leach Tailing | 0.15 - 0.20 |

Note: $m^2$/tpd- square metre of thickener surface area per tonnes per day of solids to be settled

Minimum unit area requirements assumed feed solids concentrations in the optimum range of 15% to 20% by weight for all materials, and with flocculant dosed in the most effective range and concentration as presented inTable 13-7 above.

Results of dynamic thickening tests for high rate type thickener design gave the following recommended maximum hydraulic feed loading rates as shown in Table 13-8.

**Table 13-8: High Rate Thickener Feed Load Rates**

| Material Tested | Maximum hydraulic feed load rates |
|---|---|
| | $m^3/m^2$ hr |
| Basal Conglomerate 100 µm | 3.0 -4.0 |
| Basal Conglomerate 200 µm n | 3.5 - 4.5 |
| Composite 100 µm | 4.0 - 5.0 |
| Composite 200 µm | 4.5 - 5.5 |
| Sediments 100 µm | 2.5 - 3.5 |
| Sediments 200 µm | 3.0 -4.0 |
| Cyanide Leach Tailing | 3.5 - 4.5 |

Note: $m^3/m^2$hr- cubic metres per hour of clarified solution per square metre of thickener surface area.

Minimum unit area requirements assumed feed solids concentrations in the optimum range of 15% to 20% by weight for all materials, and with flocculant dosed in the most effective range and concentration as presented in Table 13-8 above. It should be noted that high rate thickener sizing for the 200 µm materials was estimated based on static thickening test results.

Results of thickening and pulp rheology tests gave maximum, recommended thickener underflow solids concentrations as shown in the Table 13-9 for conventional or high rate thickener design.

**Table 13-9: Thickener Underflow Density**

| Material Tested | Maximum solids density |
|---|---|
| | % by weight |
| Basal Conglomerate 100 µm | 58 - 62 |
| Basal Conglomerate 200 µm | 60 - 64 |
| Composite 100 µm | 60 - 65 |
| Composite 200 µm | 65 - 69 |
| Sediments 100 µm | 54 - 58 |
| Sediments 200 µm | 58 -62 |
| Cyanide Leach Tailing | 60 - 65 |

Pressure filtration test results indicated the sizing basis for horizontal recess plate filter press design as shown in Table 13-10.

**Table 13-10: Plate Filter Press Parameters**

| Material Tested | Design Parameter |
|---|---|
| | $m^3/t$ |
| Basal Conglomerate 100 µm | 0.688 |
| Basal Conglomerate 200 µm | 0.669 |
| Composite 100 µm | 0.764 |
| Composite 200 µm | 0.744 |
| Sediments 100 µm | 0.756 |
| Sediments 200 µm | 0.729 |
| Cyanide Leach Tailing | 0.799 |

Note: $m^3/t$- cubic metres of pressure filter volume per tonne per day of dry solids

Minimum possible pressure filter cake moisture contents for all materials were in the range of 10.8% to 15.7% by weight, and normal design cake moisture contents were in the range of 12% to 16.8% by weight for all materials.

Vacuum filtration test results indicated achievable horizontal belt production rates. All reported production rates are for 10 mm cake thickness, and maximum dischargeable cake moistures based on a minimum 0.5 minute dry time. Test results are shown in Table 13-11.

**Table 13-11: Horizontal Belt Filter Design**

| Material Tested | With flocculant | Without flocculant |
|---|---|---|
| | $kg/m^2$ | $kg/m^2$ |
| Basal Conglomerate 100 µm | 269 | 795 |
| Basal Conglomerate 200 µm | 202 | 1,056 |
| Composite 100 µm | 729 | 1,144 |
| Composite 200 µm | 708 | 1,221 |
| Sediments 100 µm | 271 | 608 |
| Sediments 200 µm n | 221 | 712 |
| Cyanide Leach Tailings at pH10.5 | 316 | 862 |
| • without wash | 53 | 492 |
| • with 2.0 wash ratio | | |
| Cyanide Leach Tailing at pH 11.5 | 578 | 887 |
| • without wash | 145 | 480 |
| • with 2.0 wash ratio | | |

Note: $kg/m^2$- kilograms per hour of dry solids per square metre of filter area

Cake moisture contents for the maximum vacuum filter production rates were in the range of 17.4% to 27.0% by weight for all materials with no flocculant addition, and in the range of 24.4% to 31.7% by weight with flocculant addition. Vacuum filter cakes with flocculant addition were higher in moisture content, but discharged more easily than cakes without flocculant addition, and were considered stackable (whereas cakes without flocculant addition were not considered stackable, in most cases, at maximum dischargeable moisture contents).

## 13.3 INTERPRETATION OF TESTWORK

The testwork has covered most of the possible process options, but until now, it was difficult to predict metallurgical performance based on material type and location. The historic representation of a simple oxide and sulphide deposit has become better defined as an ore body with a method to locally identify the rock oxidation state.

Laboratory and pilot scale testing on sulphide composite samples demonstrated that the sulphide mineralisation was readily amenable to flotation process treatment. A conventional lead-zinc sequential flotation separation flow sheet can be the basis of the process design. The variability flotation testwork indicated that the sulphide mineralised zones are relatively similar in terms of rock grindability, chemical and mineral compositions, and flotation response. Galena and most of the copper sulphide minerals can be recovered in a lead flotation concentrate that will also contain the majority of the silver in the rock. The tailings from the lead-copper flotation circuit can then be processed by flotation to recover most of the sphalerite mineral in a zinc flotation concentrate.

Laboratory testing on oxide composite samples demonstrated that the oxide mineralisation was amenable to the cyanide leach process for the extraction of silver. A conventional cyanide leach circuit flow sheet can be the basis of the process design. The variability leaching testwork indicated that the oxide mineralised zones are relatively similar in terms of rock grindability, chemical and mineral compositions, and cyanide leaching response.

Laboratory testing on transitional composite samples demonstrated that the transition mineralisation was amenable to flotation process treatment and the flotation tailings were amenable to the cyanide leach process for the extraction of silver. The circuit proposed for the sulphide mineral flotation process would perform acceptably for the transition material and the cyanide leach circuit proposed for the oxide leaching circuit would also perform acceptably for the transition material. The variability testwork indicated that the transition mineralised zones are relatively similar in terms of rock grindability, chemical and mineral compositions, and leach response.

Identifying the mineralised material by oxidation code has allowed the metallurgical test results to be understood. The results were categorised to develop a predictive model of metallurgical performance for each material type. The models for sulphide material treated by the flotation process are conventional metal head grade to recovery relationships. For the transition material that will be processed by flotation and cyanide leaching, the sulphide flotation models can be used. The predicted performance from the sulphide model can then be reduced by increasing amounts based upon the oxidation code for a particular block of material. The flotation model cannot be used for material with an oxidation code above 3.5 (i.e. more oxidised). The models for cyanide leaching of the flotation tailings and the oxide material are based on a grade recovery relationships indicated from the test results.

The overall modeling logic for flotation includes three, separate mathematical units:

- Firstly, for each metal, a basic head grade to rougher recovery relationship;
- Secondly, an adjustment factor to this recovery to account for degree of oxidation; and
- Thirdly, a cleaning stage recovery applied to the oxidation adjusted rougher recovery.

The flotation tests results were combined into one larger data set for all rock types on the basis that the sulphide mineralogy is consistent across the rock types. The drillhole and sample intervals used to generate each metallurgically tested sample or composite were identified. For each interval, the geological oxidation code was recorded against the sample or composite and therefore each flotation test can be identified by an oxidation code value. All tests with particle sizes significantly finer or coarser than the plant design grind size distribution of 150 $P_{80}$ µm have not been included. The results of pilot plant tests have been included.

The combined data set for oxidation codes 0 to 2 (i.e. sulphide material) contains the results of some 130 individual rougher tests, 113 tests with cleaning stages, plus the four pilot plant campaigns. The raw data was sorted or "binned" into short grade ranges of metal values (i.e. silver, lead, zinc and copper) and then averaged. The binned averages were then analyzed by making scatter plots of comparative data, for example "percent lead head grade" versus "recovery of lead in lead rougher flotation". A "best-fit" three-term polynomial curve was fitted to each scatter plot. The apogee of a curve fitting the "percent lead head grade" and the "recovery of lead in lead rougher flotation" data points defines the value above which recovery is fixed at a maximum value. The data for lead, silver, copper, and zinc in the lead rougher flotation concentrate, the equations describing the recovery values, and the maximum recovery values are show in Figure 13-12, Figure 13-13, Figure 13-14 and Figure 13-15 below.

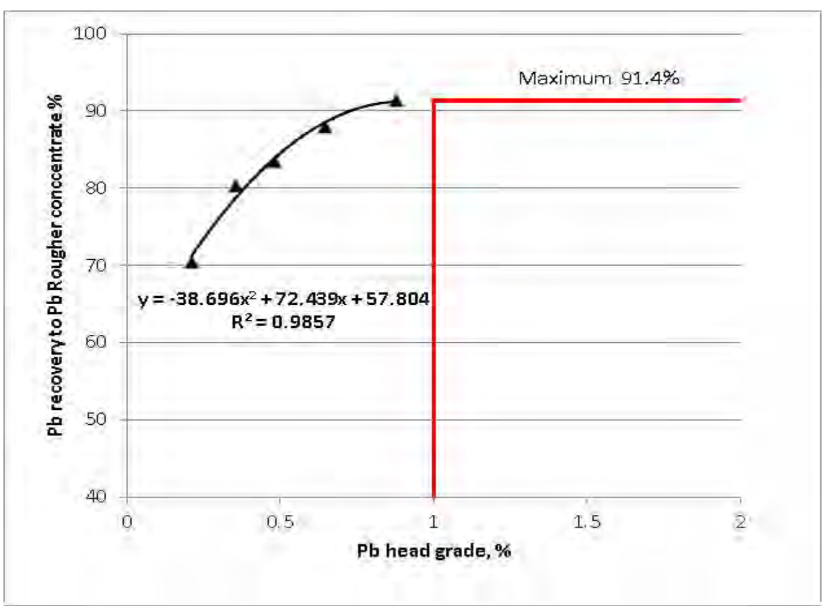



**Figure 13-12: Percent Lead (Pb) Head Grade versus Percent Recovery of Pb in Pb Rougher Flotation (Maximum 91.4% Recovery)**

Source: M3, 2012

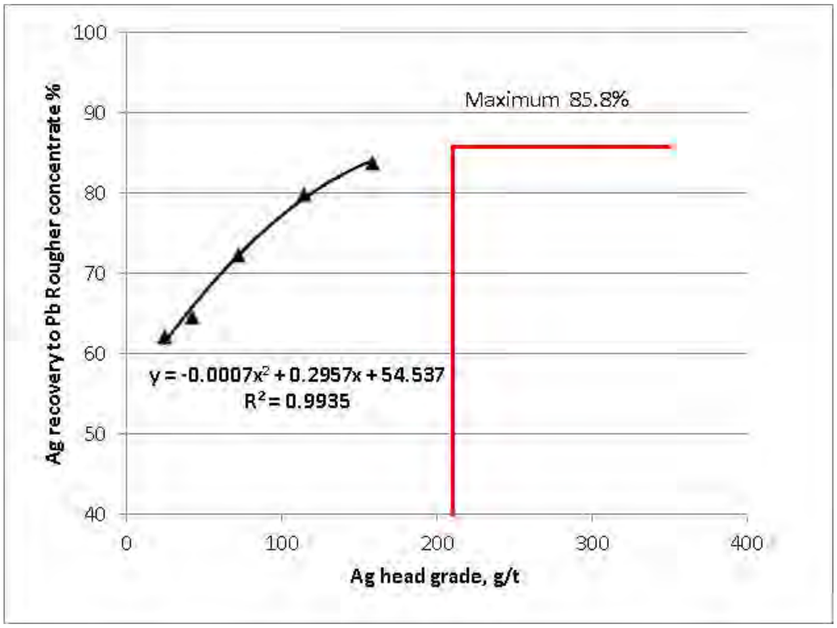



**Figure 13-13: Silver (Ag) Head Grade versus Percent Recovery of Ag in Pb Rougher Flotation (Maximum 85.8% Recovery)**

Source: M3, 2012

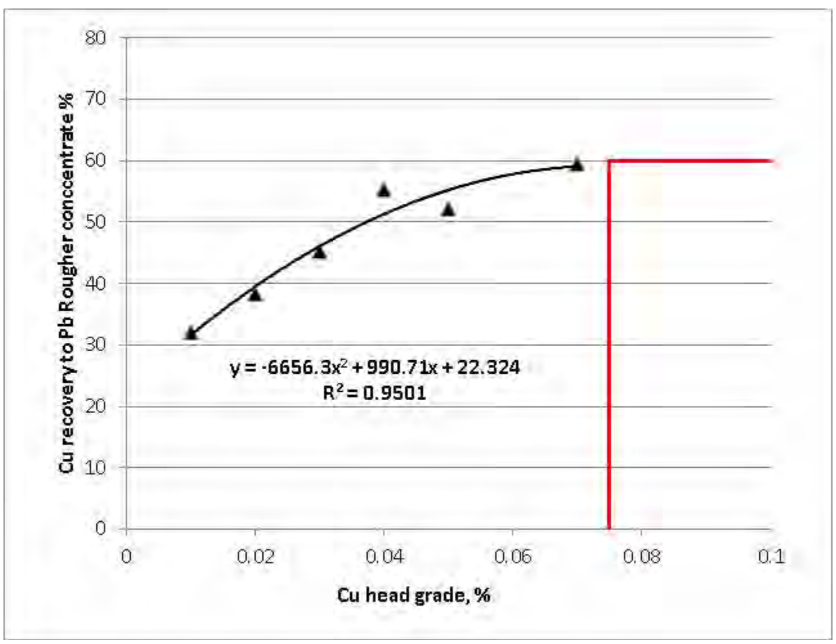



**Figure 13-14: Percent Copper (Cu) Head Grade versus Percent Recovery of Cu in Pb Rougher Flotation (Maximum 60% Recovery)**

Source: M3, 2012

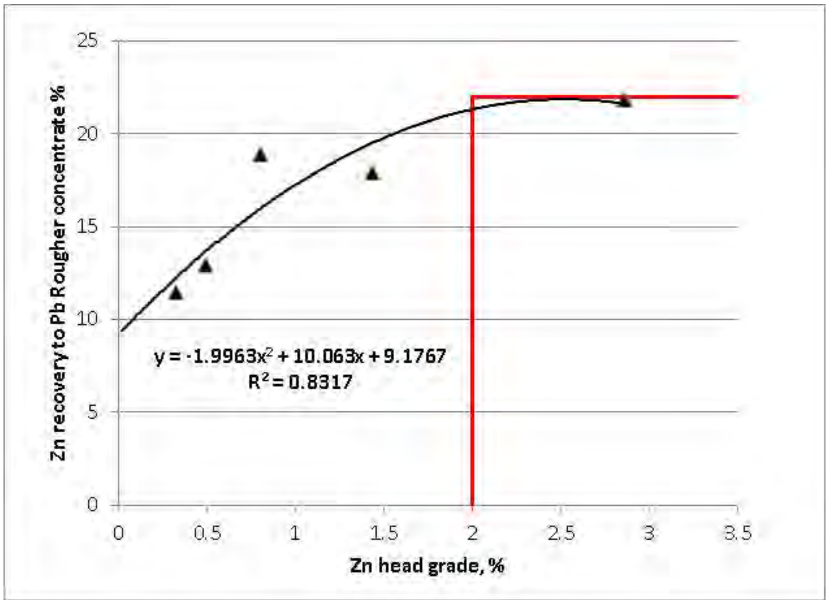



**Figure 13-15: Percent Zinc (Zn) Head Grade versus Percent Recovery of Zn in Pb Rougher Flotation (Maximum 22% Recovery)**
Source: M3, 2012

The data for lead, silver, copper, and zinc in the zinc rougher flotation concentrate, the equations describing the recovery values, and the maximum recovery values are shown in Figure 13-16, Figure 13-17, Figure 13-18 and Figure 13-19 below.

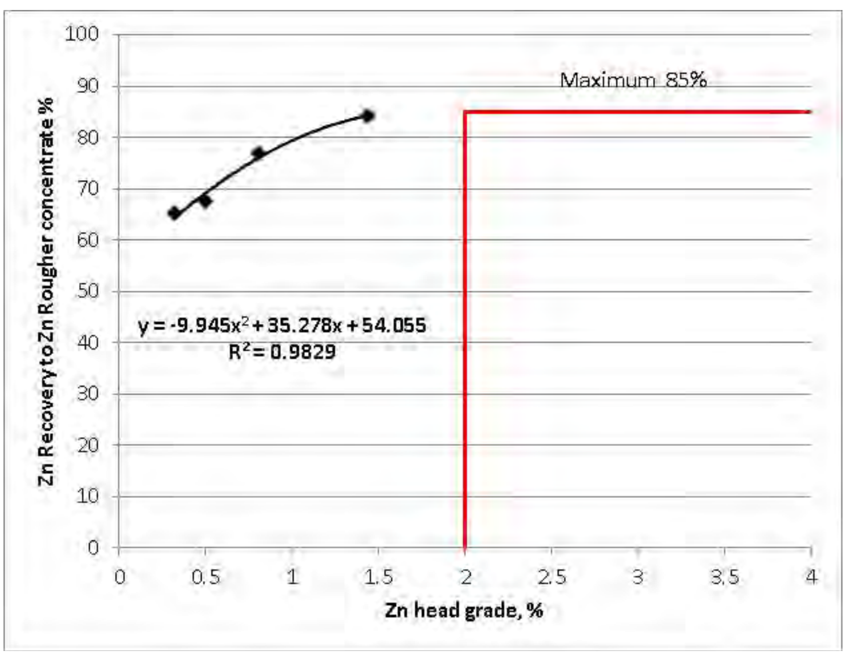



**Figure 13-16: Percent Zinc (Zn) Head Grade versus Percent Recovery of Zn in Zn Rougher Flotation (Maximum 85% Recovery)**
Source: M3, 2012

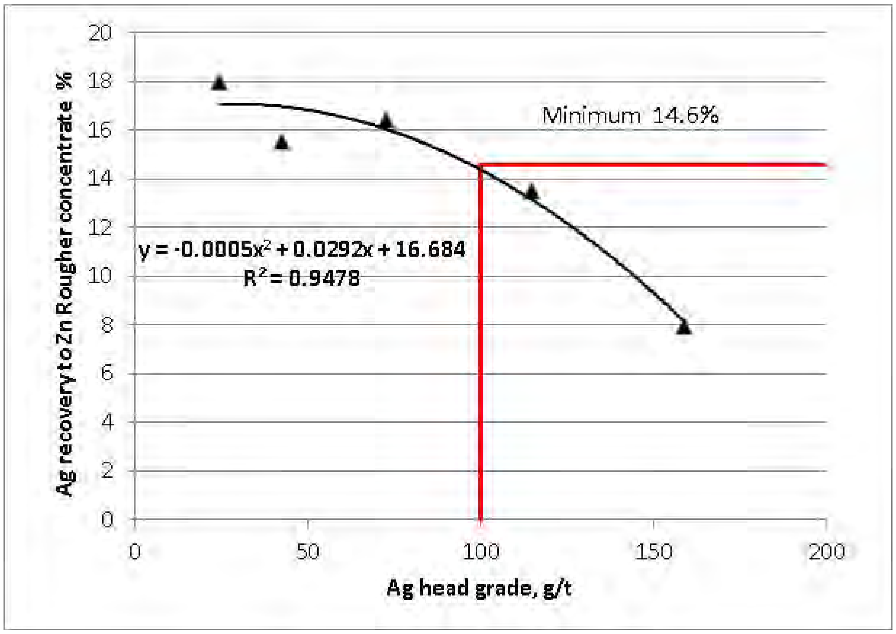



**Figure 13-17: Silver (Ag) Head Grade versus Percent Recovery of Ag in Zn Rougher Flotation (Minimum 14.6% Recovery)**

Source: M3, 2012

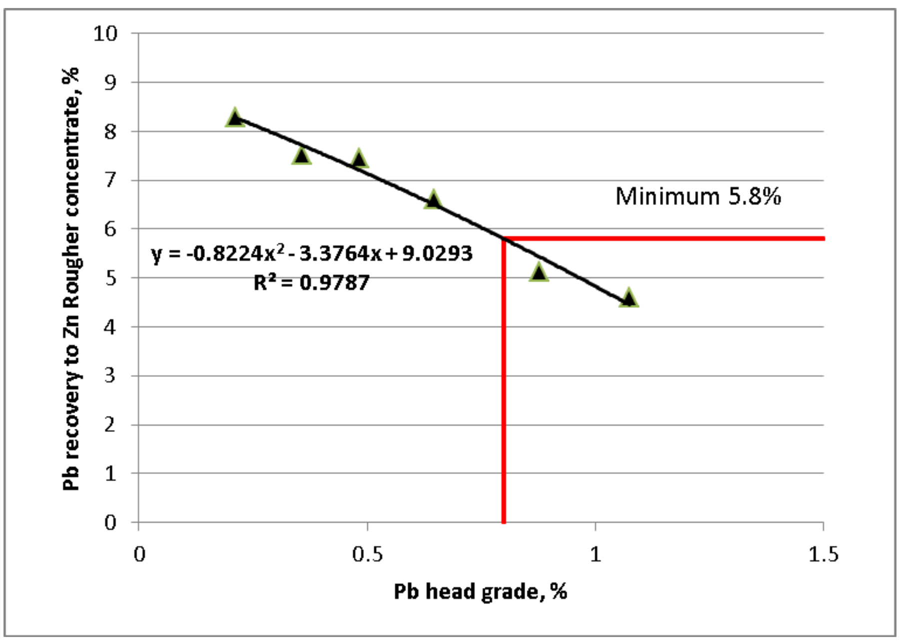



**Figure 13-18: Percent Lead Head Grade versus Percent Recovery of Pb in Zn Rougher Flotation (Minimum 5.8% Recovery)**

Source: M3, 2012

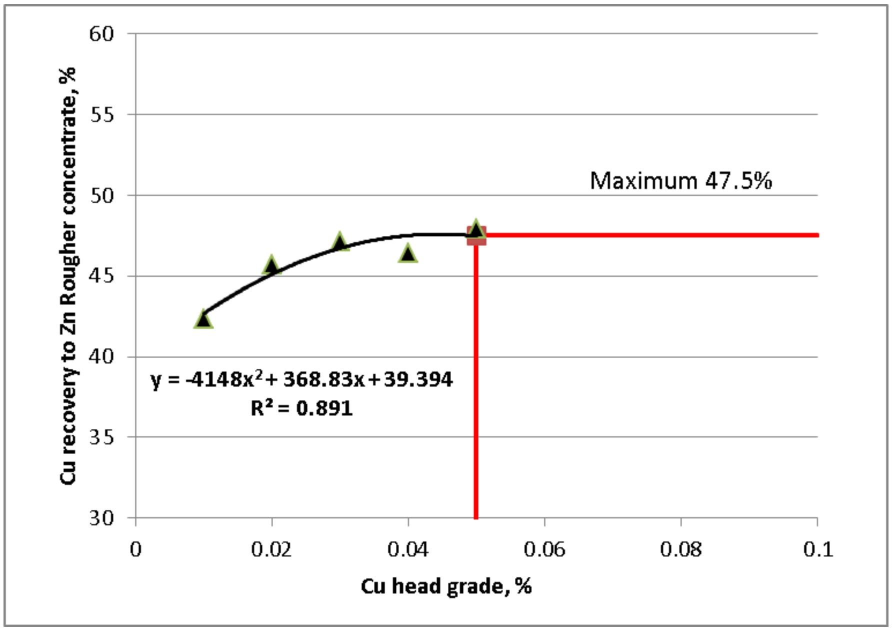



**Figure 13-19: Percent Copper (Cu) Head Grade versus Percent Recovery of Cu in Zn Rougher Flotation (Maximum 47.5% Recovery)**

Source: M3, 2012

Tonnage and grade data by rock type from the Mineral Reserve estimate (block model) were used to plot a metal head grade versus the cumulative frequency of that head grade for lead, zinc, and silver.

The graphs of cumulative frequency of occurrence versus metal head grade are presented in Figure 13-20, Figure 13-21 and Figure 13-22 . Also displayed as a vertical line, is the rougher recovery modeling apogee grade. Any grades above this value are assigned a fixed maximum recovery value. For the deposit, 2.77% of lead grade assays are at a value greater than 1% lead, 6.2 % of zinc grade assays are at a value greater than 2.0% zinc, and 2.35% of the silver grade assays are at a value greater than 210 g/t silver. It is considered that the small amounts of blocks limited by the maximum recovery are not significant to affecting the final estimated average recovery.

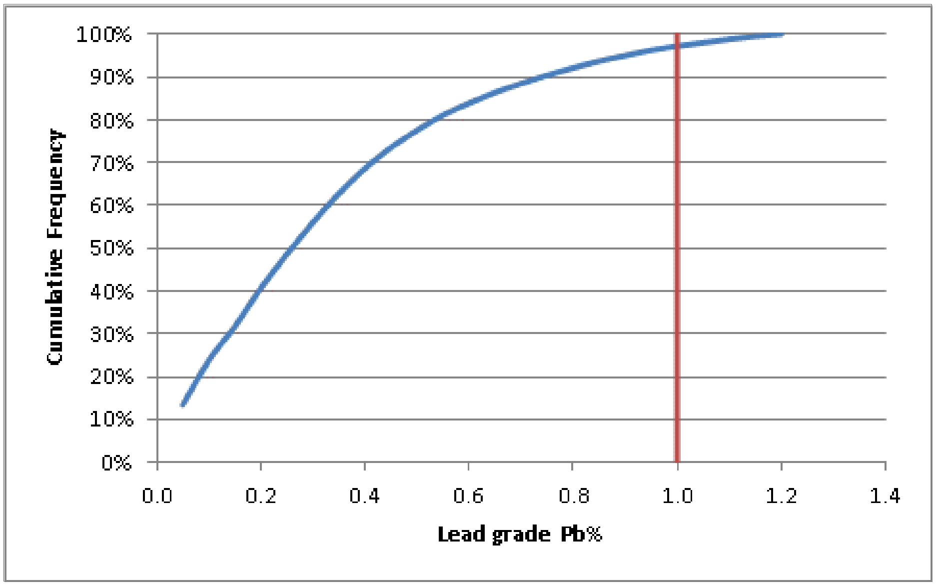



**Figure 13-20: Percent Lead (Pb) Head Grade versus Cumulative Frequency of Occurrence of Pb Grade Value in the Deposit**

Source: M3, 2012

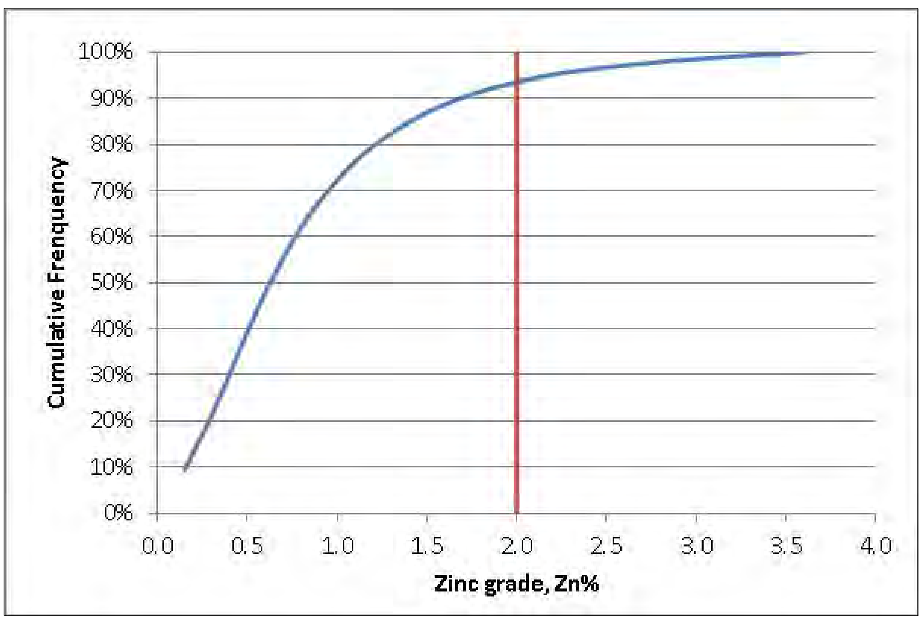



**Figure 13-21: Percent Zinc (Zn) Head Grade versus Cumulative Frequency of Occurrence of Zn Grade Value in the Deposit**

Source: M3, 2012

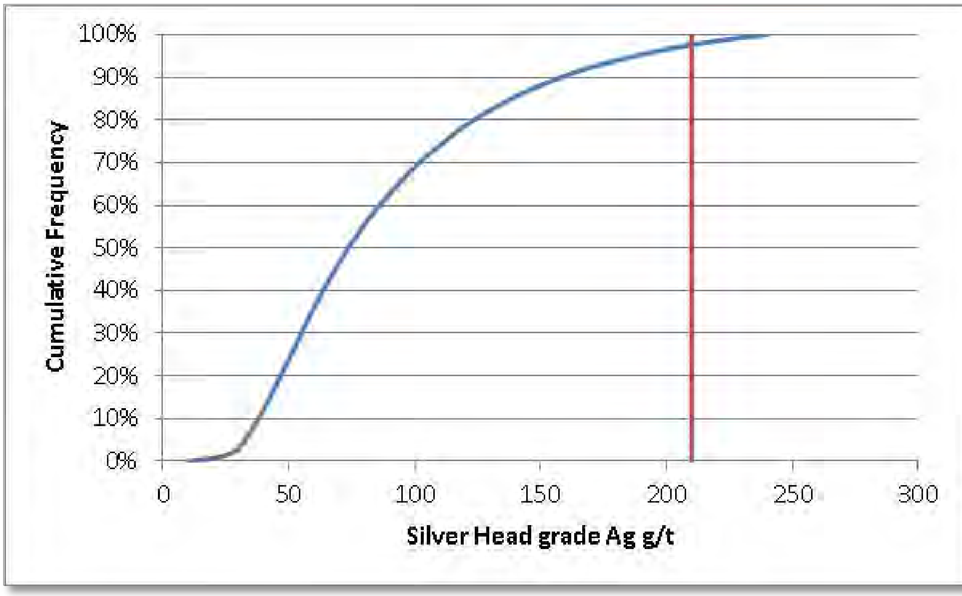



**Figure 13-22: Silver (Ag) Head Grade versus Cumulative Frequency of Occurrence of Ag Grade Value in the Deposit**

Source: M3, 2012

Testwork data sets were used to develop cleaner flotation stage recovery values based on mass pull values. Mass pull is defined as the percentage of plant feed tonnage that reports to the respective concentrate. For either of the two flotation concentrates, lead or zinc, the principle driver of concentrate mass is either the lead or zinc head grade. The head grades versus mass pull values were plotted to obtain a curve and an equation that can be used to predict concentrate production based on head grade. The graphs are presented in Figure 13-23 and Figure 13-24 for lead and zinc, respectively. Two stages of lead cleaner flotation and three stages of zinc cleaner flotation correspond to the proposed plant design criteria.

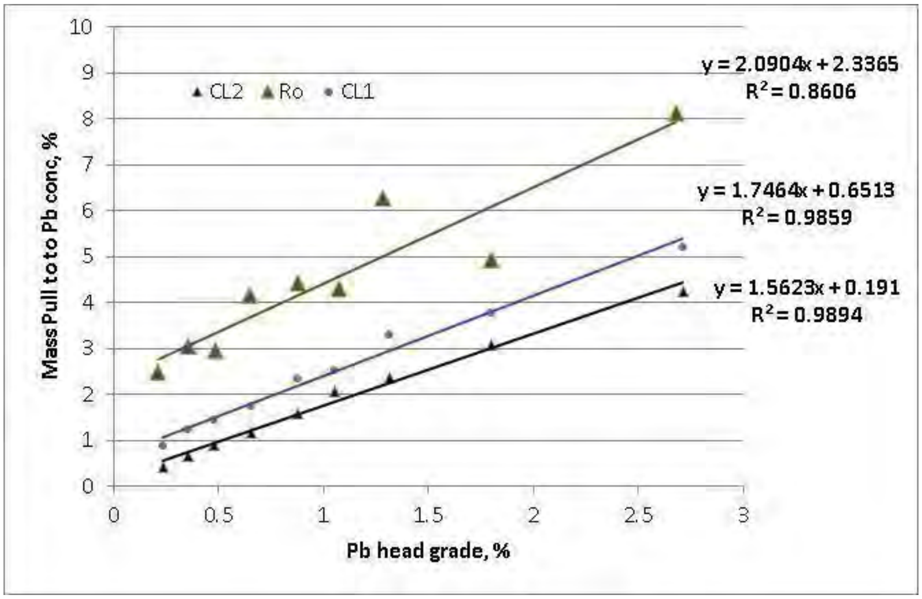



**Figure 13-23: Percent Lead (Pb) Head Grade versus Pb Concentrate Mass Pull**
Source: M3, 2012

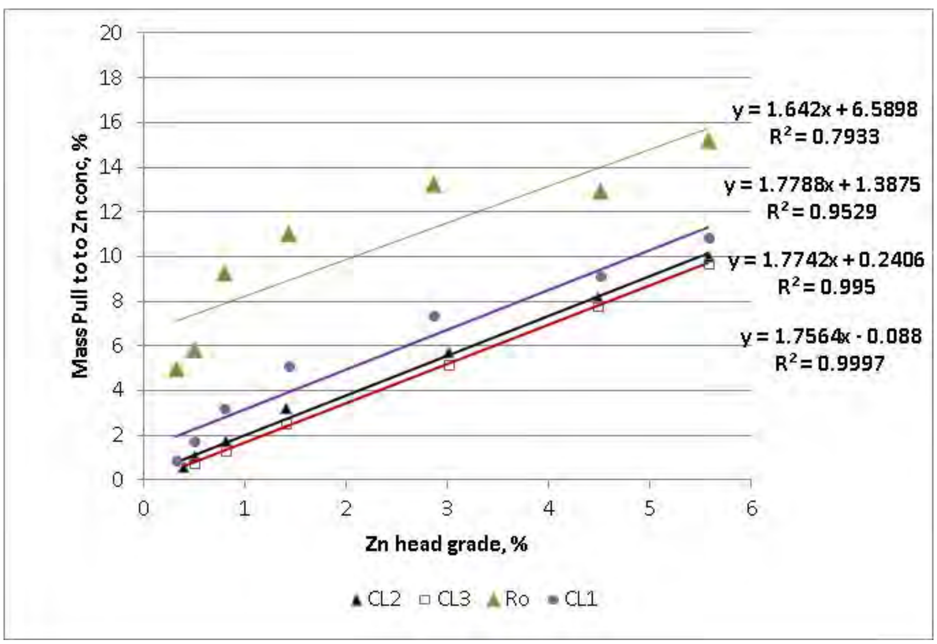



**Figure 13-24: Percent Zinc (Zn) Head Grade versus Zn Concentrate Mass Pull**
Source: M3, 2012

The flotation test results obtained from testing composites with varying, identified oxidation states has been previously described. Using the head grade to rougher recovery relationship, the predicted rougher recovery, by metal, was calculated for lead and zinc concentrate of each composite. Each individual test result was compared to the predicted value to obtain an "Oxidation Recovery Factor" which is given by:

$$\frac{\text{Rougher Recovery Value Determined by Test}}{\text{Rougher Recovery Value Determined by Formula}}$$

The resulting data was binned by oxidation code and plotted as a series of scatter plots of Oxidation Recovery Factor versus oxidation code. A fitted linear line was then generated to describe how the Oxidation Recovery Factor varied with oxidation code. Overall, the greater the oxidation code (indicating the material is more oxidised), the lower the resulting Oxidation Recovery Factor value. This analysis is presented in Figure 13-25, Figure 13-26, Figure 13-27, Figure 13-28, Figure 13-29, Figure 13-30, Figure 13-31 and Figure 13-32.

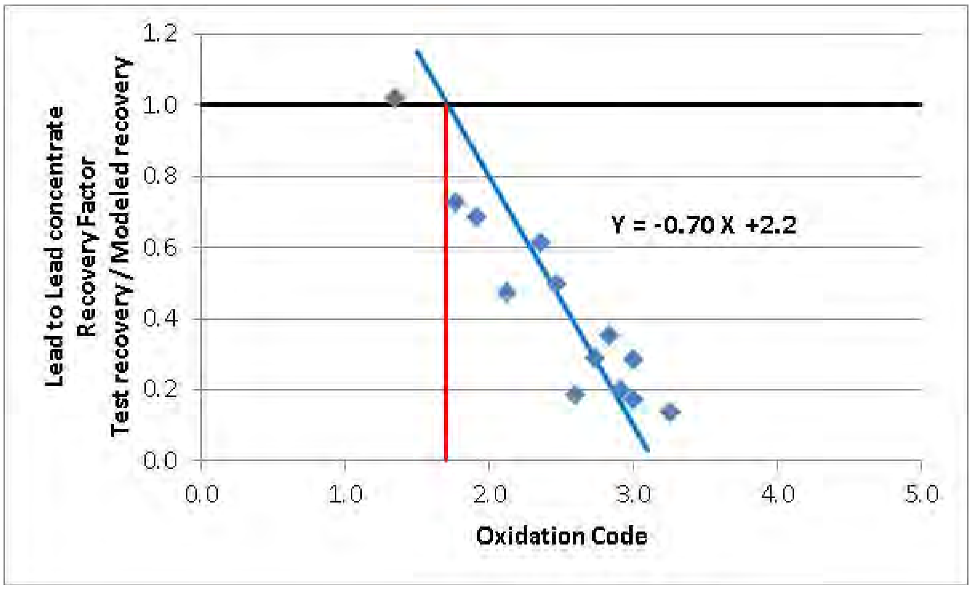



**Figure 13-25: Oxidation Code versus Lead to Lead Rougher Concentrate Oxidation Recovery Factor (Applied to Oxidation Codes 1.7 and Higher)**
Source: M3, 2012

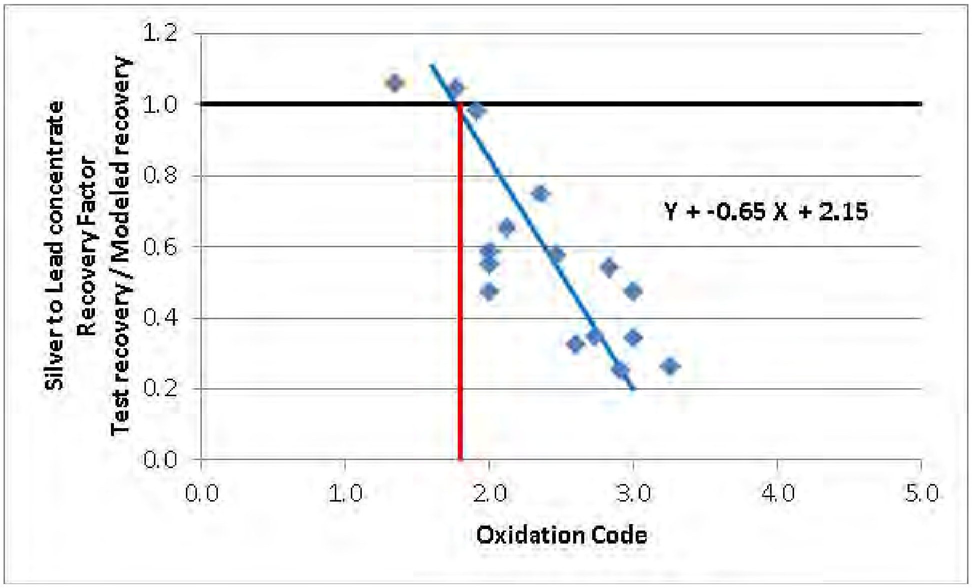



**Figure 13-26: Oxidation Code versus Silver to Lead Rougher Concentrate Oxidation Recovery Factor (Applied to Oxidation Codes 1.8 and Higher)**

Source: M3, 2012

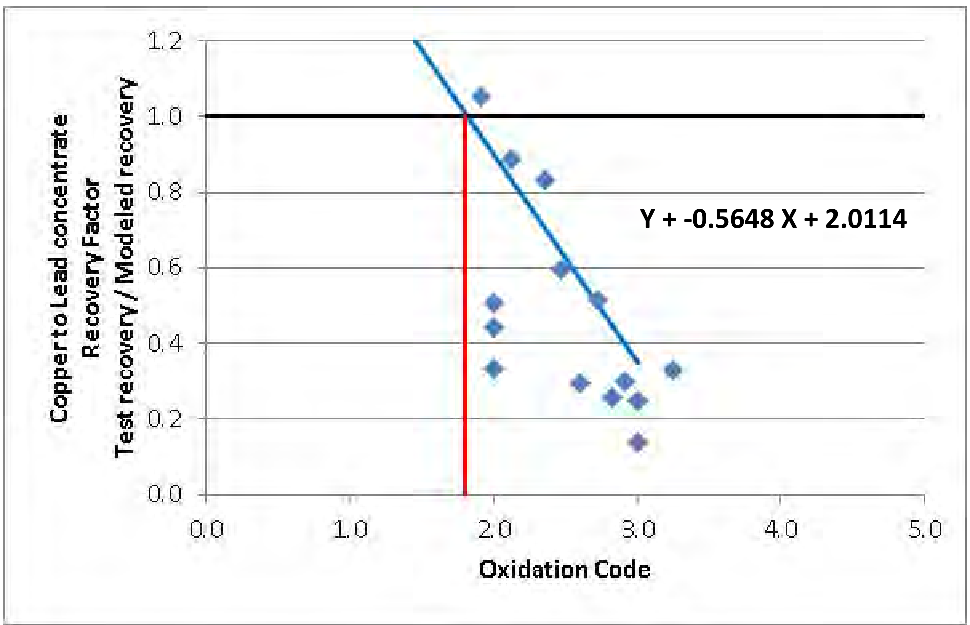



**Figure 13-27: Oxidation Code versus Copper to Lead Rougher Concentrate Oxidation Recovery Factor (Applied to Oxidation Codes 1.8 and Higher)**

Source: M3, 2012

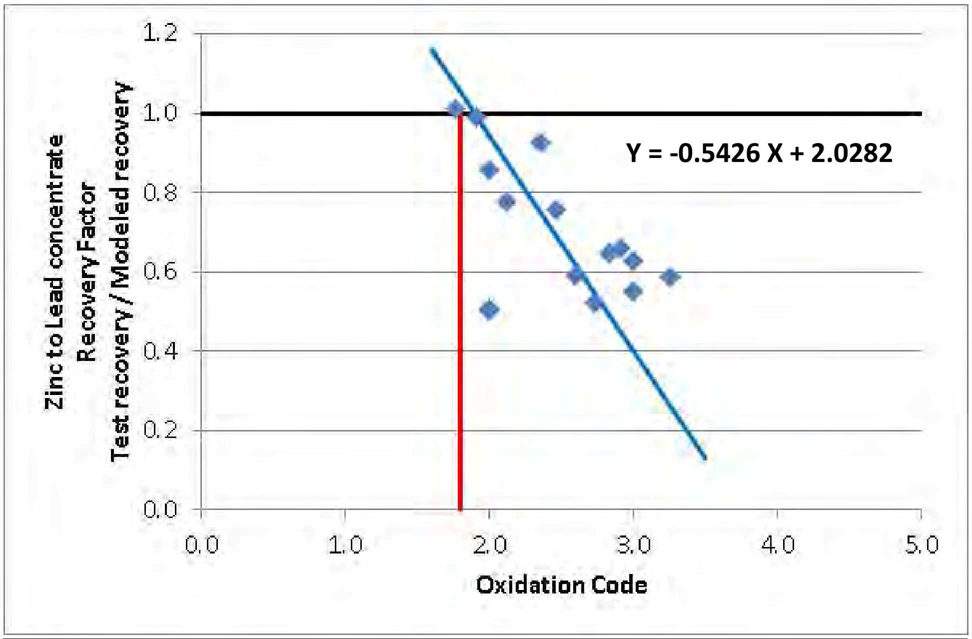



**Figure 13-28: Oxidation Code versus Zinc to Lead Rougher Concentrate Oxidation Recovery Factor (Applied to Oxidation Codes 1.8 and Higher)**
Source: M3, 2012

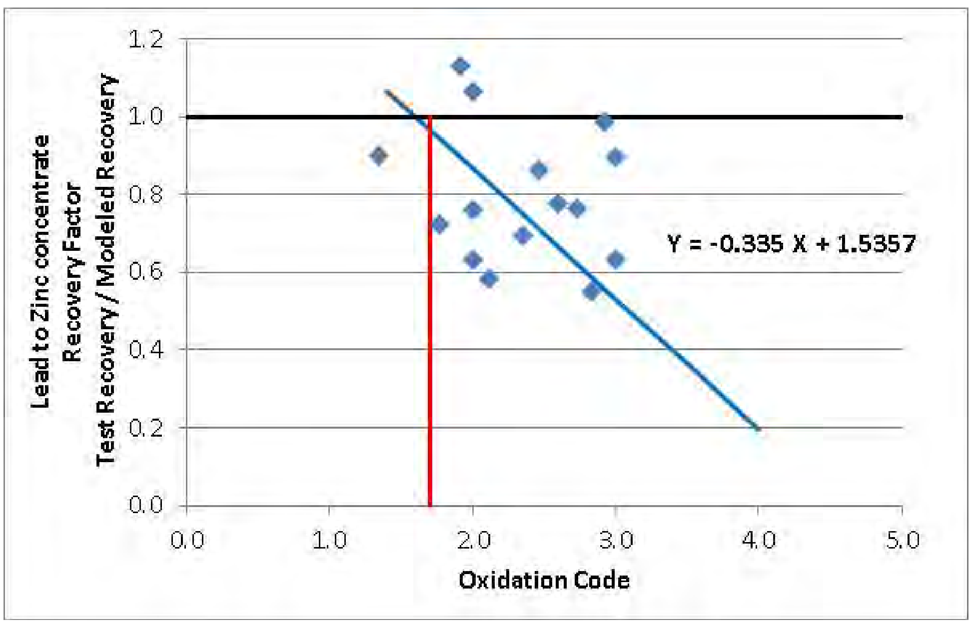



**Figure 13-29: Oxidation Code versus Lead to Zinc Rougher Concentrate Oxidation Recovery Factor (Applied to Oxidation Codes 1.7 and Higher)**
Source: M3, 2012

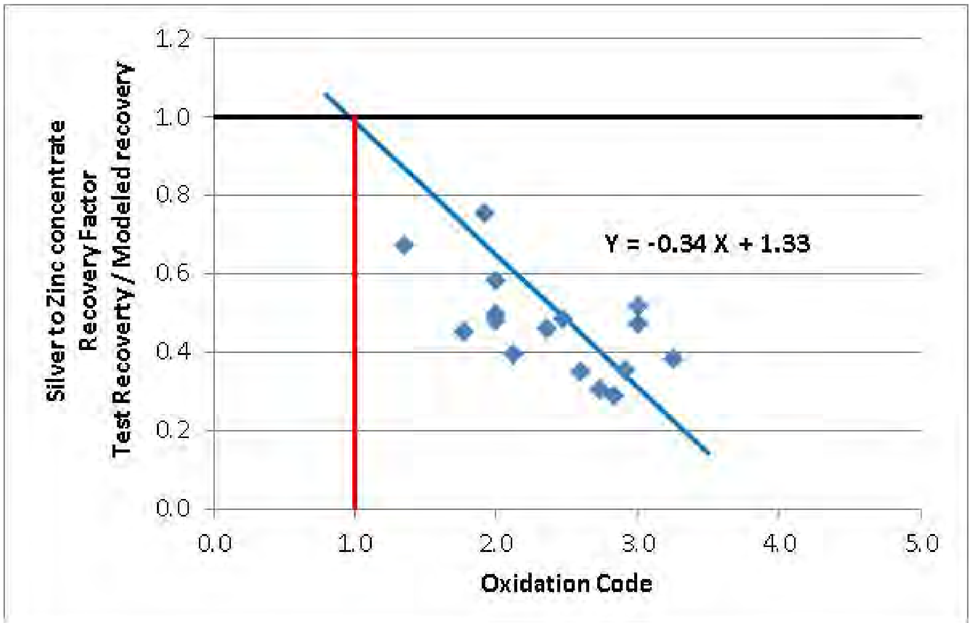



**Figure 13-30: Oxidation Code versus Silver to Zinc Rougher Concentrate Oxidation Recovery Factor (Applied to Oxidation Codes 1.0 and Higher)**

Source: M3, 2012

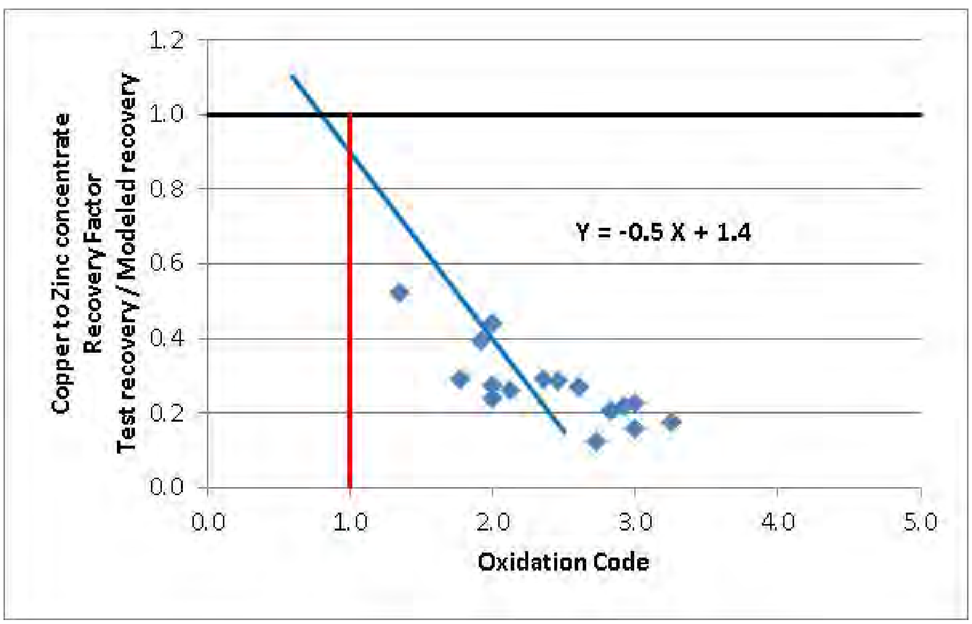



**Figure 13-31: Oxidation Code versus Copper to Zinc Rougher Concentrate Oxidation Recovery Factor (Applied to Oxidation Codes 1.0 and Higher)**

Source: M3, 2012

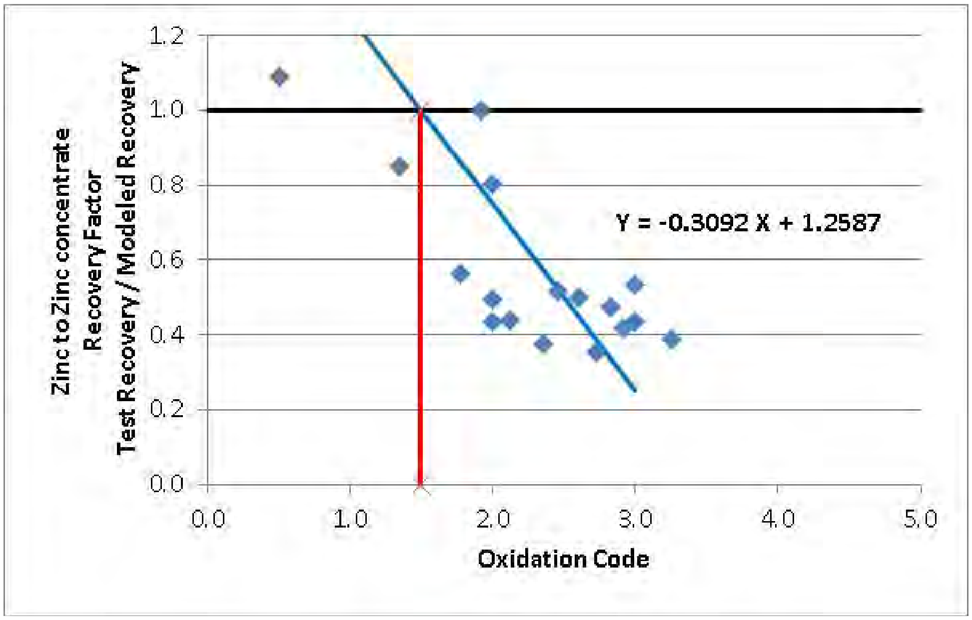



**Figure 13-32: Oxidation Code versus Zinc to Zinc Rougher Concentrate Oxidation Recovery Factor (Applied to Oxidation Codes 1.5 and Higher)**
Source: M3, 2012

The predicted metal recovery value for the rougher flotation process can be obtained by applying the following equation:

(Rougher Recovery Value Based on Head Grade) x (Oxidation Recovery Factor)

= (Predicted Rougher Recovery Value)

To obtain a final prediction for the metal recovery value, the Rougher Recovery Value must be adjusted by the expected performance of the rougher flotation concentrate cleaner flotation treatment stages. The lead rougher concentrate will be reground and cleaned by two stages of flotation. These cleaning stages incur a recovery loss of valuable metals to cleaner tailings that can be quantified by predicting a cleaner flotation process stage recovery. The cleaning stage recovery values from the testwork were used to generate the stage recovery for each metal for both lead and zinc concentrate.

In general terms, the greater the metal feed grade, the greater the cleaning stage recovery to cleaner concentrate. This is logical as it is easier to upgrade higher grade material to final concentrate grade than lower grade material. Similarly, in the cleaning of lead rougher concentrate, the cleaner stage recovery value for zinc is low. The zinc recovered in the lead rougher flotation concentrate is often locked with lead mineral particles. The regrinding process for the lead rougher concentrate liberates the zinc particles and they can be rejected to the lead cleaner flotation tailings.

The combined data set for Oxidation Codes 0 to 2 (i.e. sulphide material) contains the results of 113 tests with cleaning stages. The raw data was sorted or “binned” into short grade ranges of metal values (i.e. silver, lead, zinc and copper) and then averaged. The binned averages were then analyzed by making scatter plots of comparative data, for example “percent lead head grade” versus “recovery of lead in lead cleaner flotation”. A “best-fit” three-term polynomial curve was fitted to each scatter plot. The associated lead head grade above which the maximum is applied is the same value as was determined for the rougher flotation recovery maximum. The data for lead, silver, copper, and zinc in the lead cleaner flotation concentrate, the equations describing the recovery values, and the maximum recovery values are shown in Figure 13-33, Figure 13-34, Figure 13-35, and Figure 13-36. The data for zinc, silver, lead, and copper in the zinc cleaner flotation concentrate, the equations describing the recovery values, and the maximum recovery values are shown in Figure 13-37, Figure 13-38, Figure 13-39 and Figure 13-40.

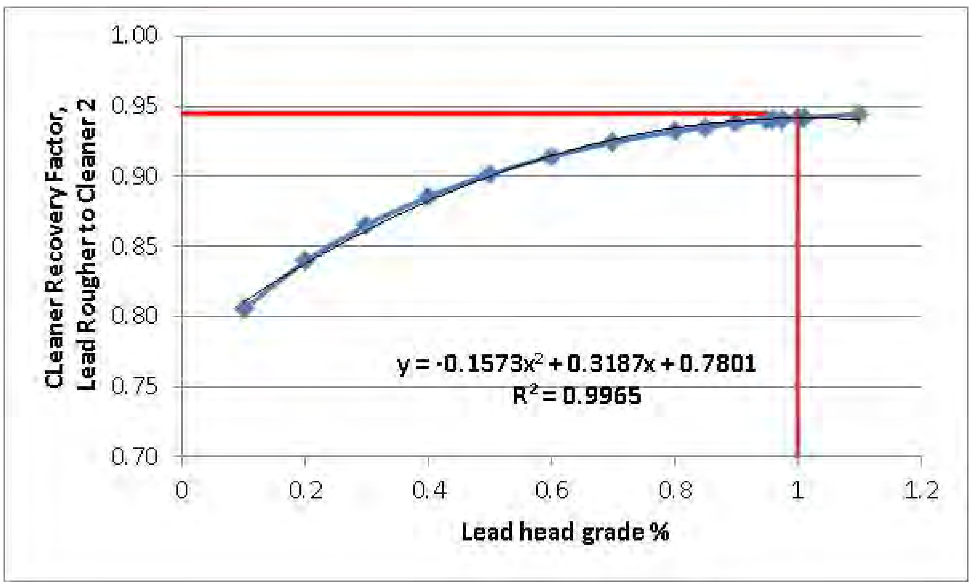



**Figure 13-33: Percent Lead Head Grade versus Cleaner Flotation Recovery Factor for Lead in Lead Cleaner Flotation (Maximum Recovery 94.5%)**

Source: M3, 2012

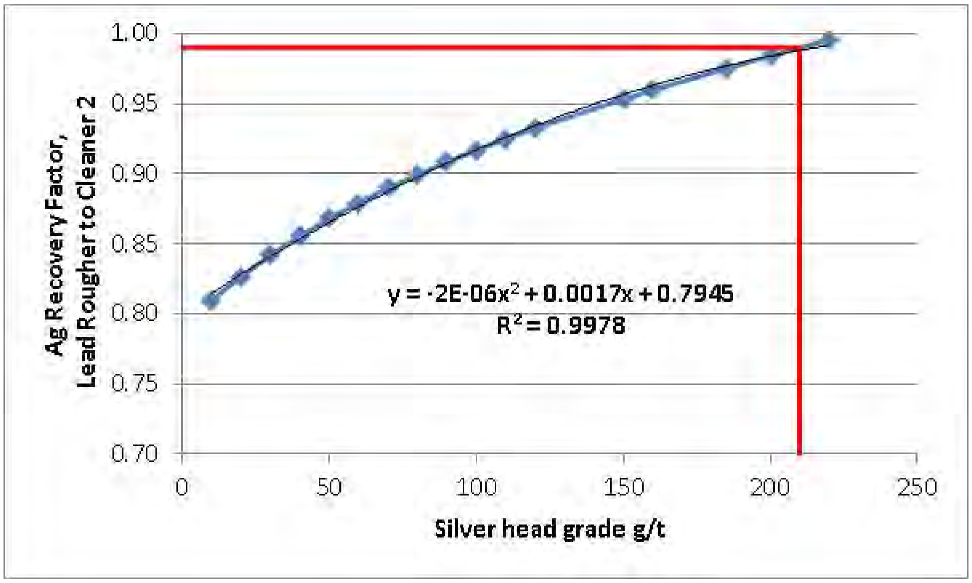



**Figure 13-34: Percent Silver Head Grade versus Cleaner Flotation Recovery Factor for Silver in Lead Cleaner Flotation (Maximum Recovery 97.5%)**

Source: M3, 2012

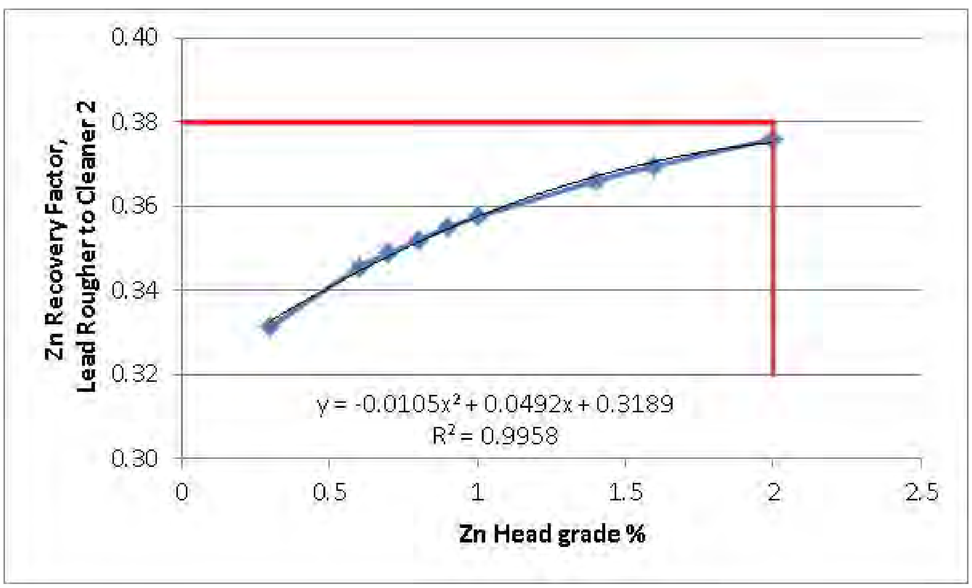



**Figure 13-35: Percent Zinc Head Grade versus Cleaner Flotation Recovery Factor for Zinc in Lead Cleaner Flotation (Maximum Recovery 38.0%**

Source: M3, 2012

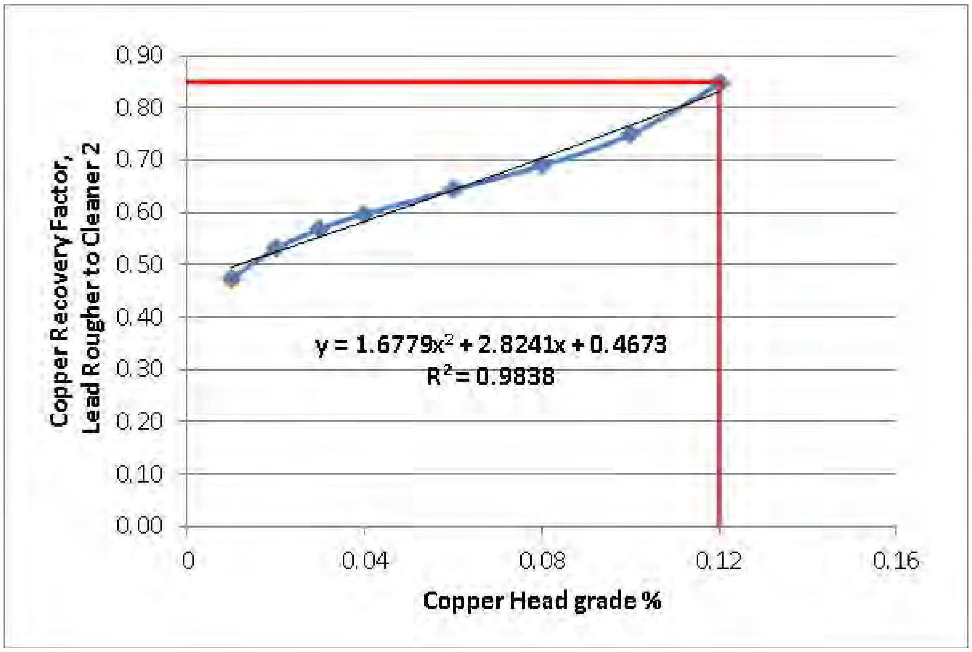



**Figure 13-36: Percent Copper Head Grade versus Cleaner Flotation Recovery Factor for Copper in Lead Cleaner Flotation (Maximum Recovery 85.0%)**

Source: M3, 2012

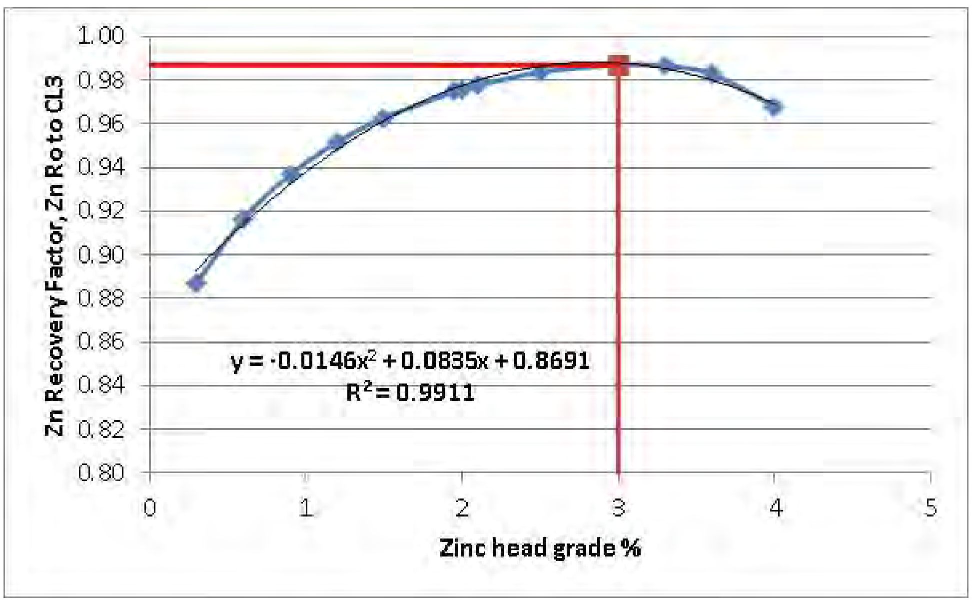



**Figure 13-37: Percent Zinc Head Grade versus Cleaner Flotation Recovery Factor for Zinc in Zinc Cleaner Flotation (Maximum Recovery 98.7%)**

Source: M3, 2012

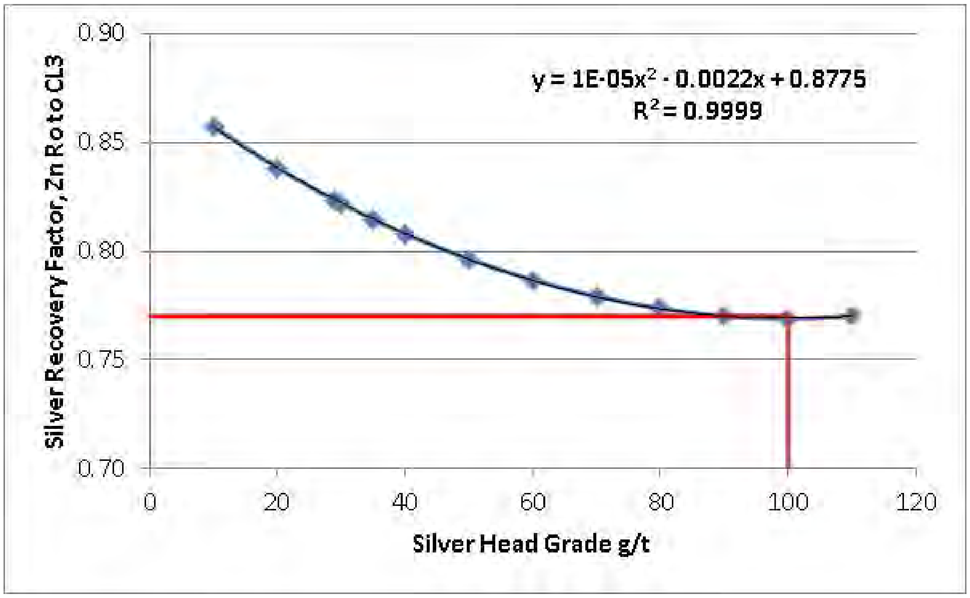



**Figure 13-38: Percent Silver Head Grade versus Cleaner Flotation Recovery Factor for Silver in Zinc Cleaner Flotation (Minimum Recovery 77.0%)**
Source: M3, 2012

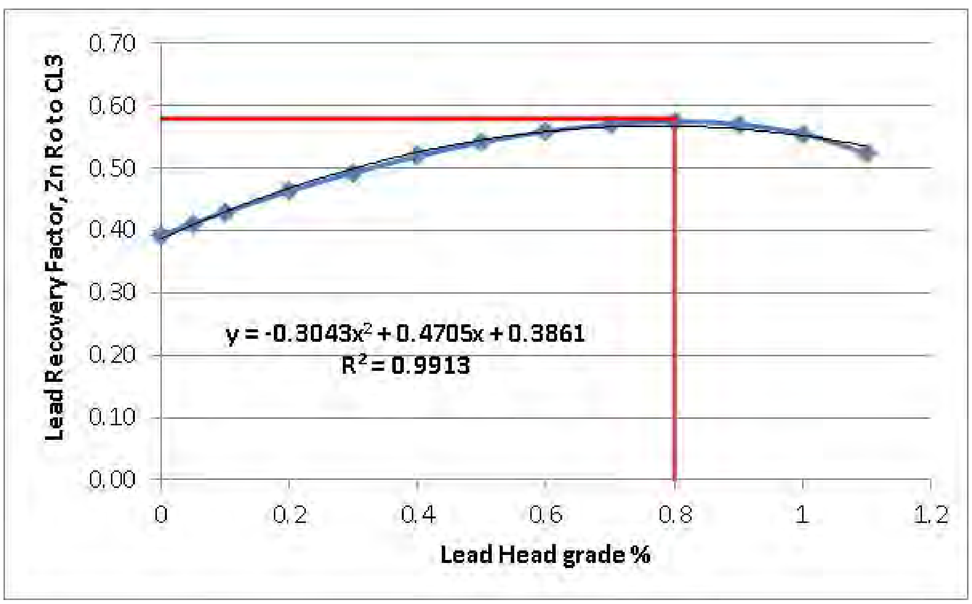



**Figure 13-39: Percent Lead Head Grade versus Cleaner Flotation Recovery Factor for Lead in Zinc Cleaner Flotation (Maximum Recovery 58.0%)**
Source: M3, 2012

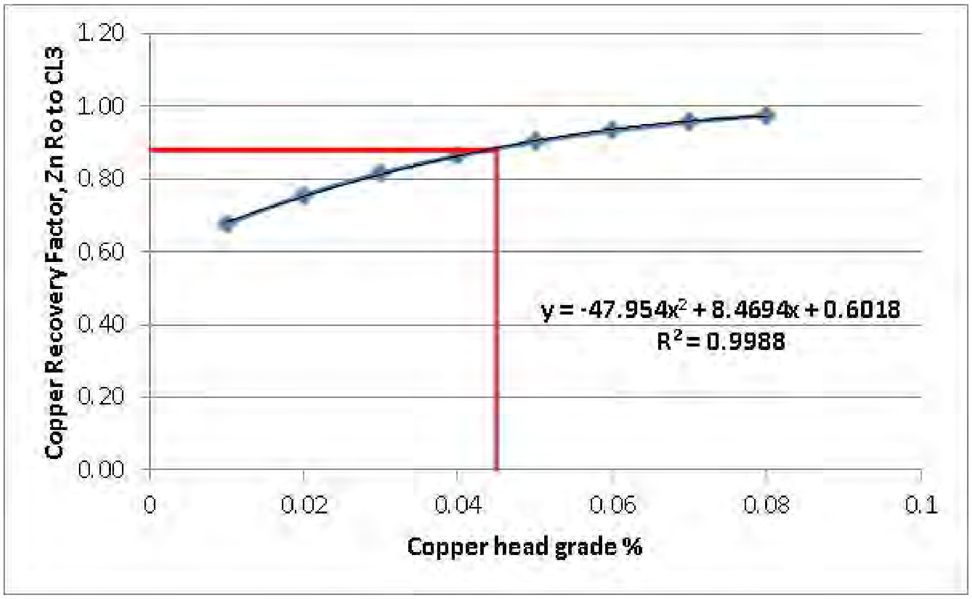



**Figure 13-40: Percent Copper Head Grade versus Cleaner Flotation Recovery Factor for Copper in Zinc Cleaner Flotation (Maximum Recovery 86.5%)**
Source: M3, 2012

The “Transitional’ rock type is a mixture of sulphide and oxide rock types. To predict the metal recovery value requires the consideration of the recovery values predicted for both sulphide and oxide materials. The overall silver recovery for flotation is first applied; thus the calculated silver grade in the flotation tailings can be used to predict the additional silver recovery, by cyanide leaching of these flotation tailings.

The extraction factor for silver from the flotation tailings has two forms. The first is an extraction factor based on testwork results obtained from cyanide leach tests of flotation tailings. The second extraction factor is based on the results of the hot cyanide silver assay (Ag-AA13HY) procedure of the feed to the flotation-cyanidation process. Two predictions of the silver metal extraction can be calculated, and the lowest value is used in the economic analysis.

A plot of data points from the flotation tailings leach tests of the values of flotation tailings assay versus silver extracted during the test was performed. The best fit line from the data plotted indicates that an extraction factor of 35% can be used to predict the silver extraction value from leaching of flotation tailings. The plot of the data and the best fit line are presented in Figure 13-41.

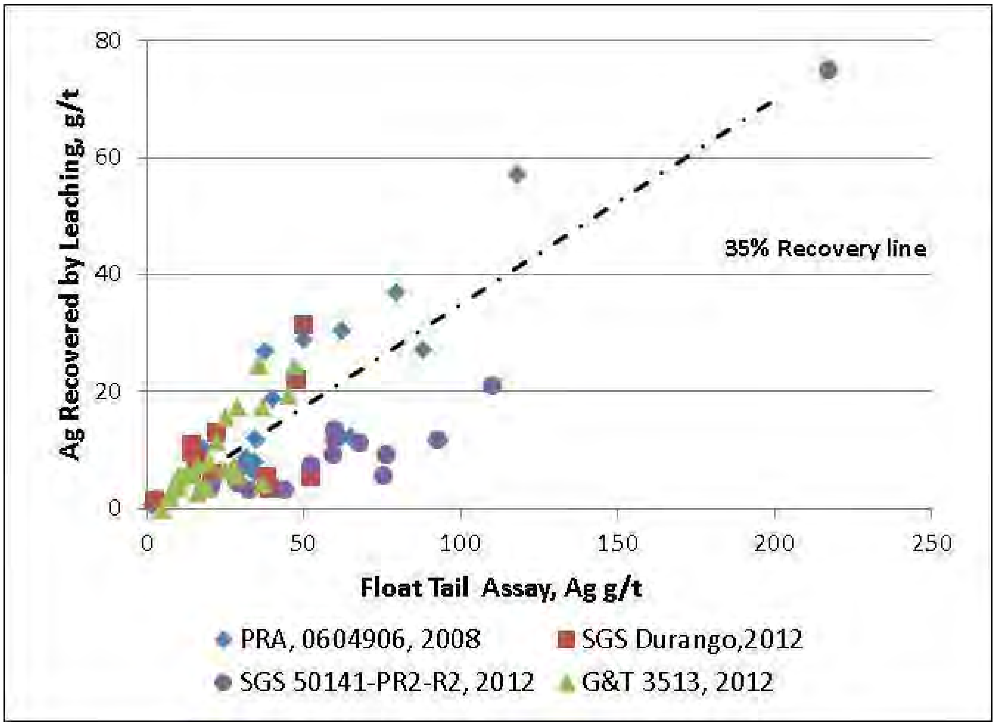



**Figure 13-41: Flotation Tailings Ag Assay (g/t) versus Ag Recovered by Leaching (g/t)**
Source: M3, 2012

The second extraction value determination is done by multiplying the result of bottle roll leach tests for the rock type processed by a leach extraction factor. The leach extraction factor was determined by plotting leach test data leach time (in hours) versus leach extraction factors (bottle roll leach test recovery divided by the hot cyanide extraction value) from the test data. The factor to apply is selected from the point indicated at the design process circuit leach time. The leach extraction factor for flotation leach tailings is presented with the plotted data in Figure 13-42.

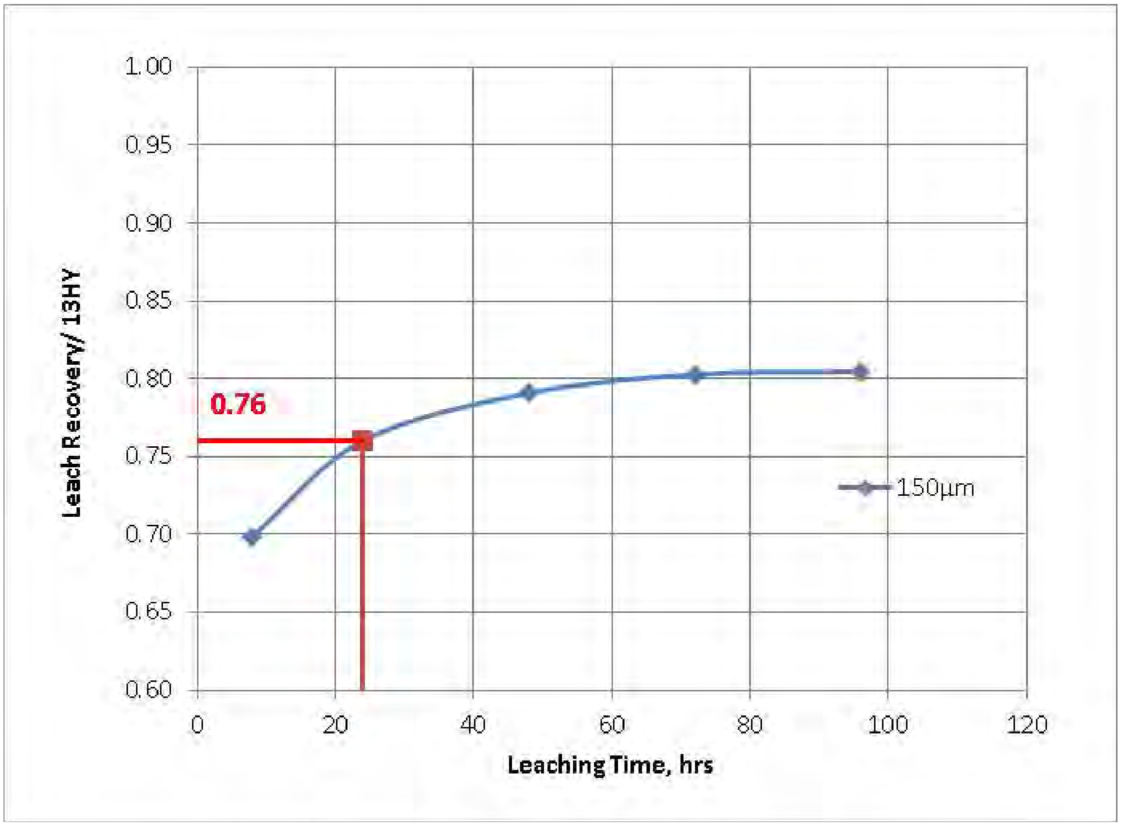



**Figure 13-42: Flotation Tailings Leach Time (hours) versus Leach Recovery Factor**
Source: M3, 2012

To predict the metal recovery value for oxide material, bottle roll test data, hot cyanide assay values, and leach time have been used to determine a leach extraction factor. The extraction value determination is done by multiplying extraction values from bottle roll leach tests performed on the rock type processed by the leach extraction factor. The leach extraction factor was determined by plotting leach time (in hours) versus leach extraction factors (bottle roll leach test recovery divided by the hot cyanide extraction value) from test data. The factor to apply is selected from the point indicated at the design process circuit leach time. The leach extraction factor for oxide material is presented with the plotted data in Figure 13-43.

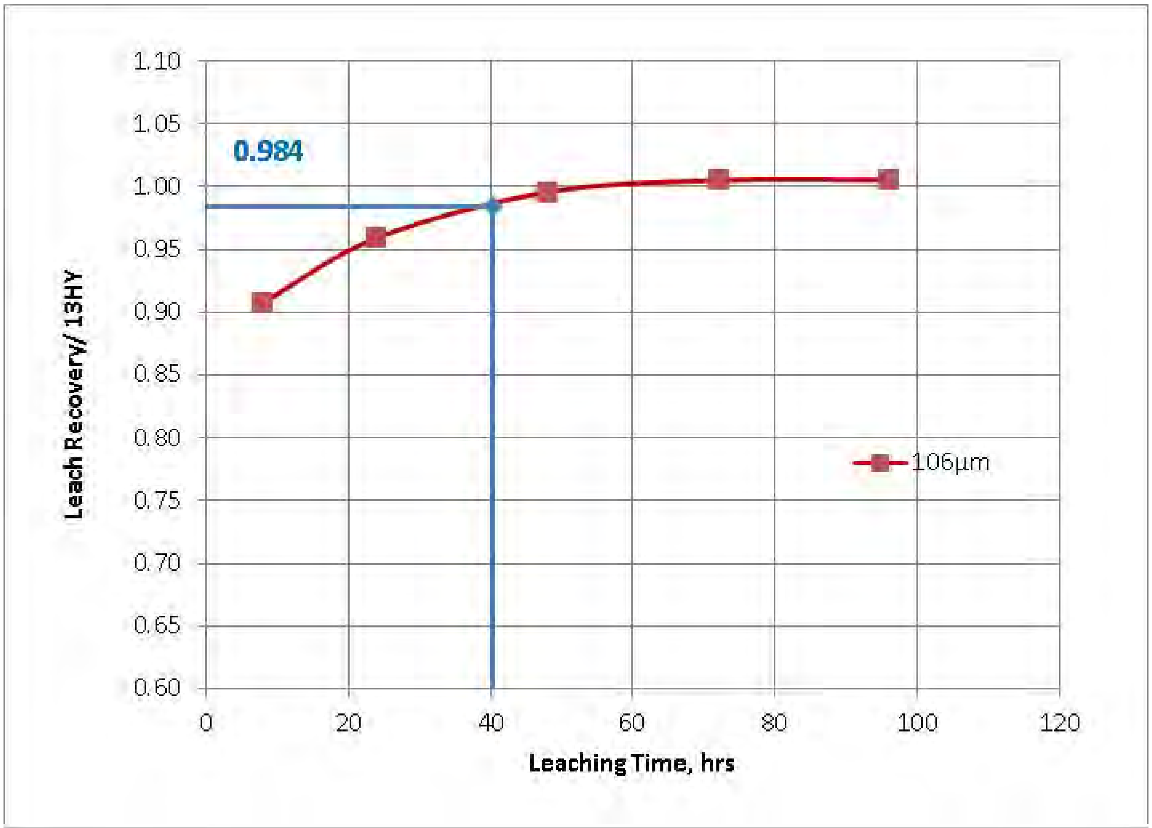



**Figure 13-43: Oxide Material Leach Time (hours) versus Leach Extraction Factor**
Source: M3, 2012

## 13.4 REAGENT CONSUMPTION & CONSUMABLES

Reagent consumption rates for the full scale plant operation have been estimated from the results of all testwork used for plant design.

### 13.4.1 Flotation Circuits

The reagents used are standard additives for the flotation of lead and zinc minerals. The general practice in treatment of lead-zinc mineralisation is to use reagents to float the lead concentrate first, while depressing the zinc minerals. After lead flotation, the zinc minerals are reactivated with copper sulphate and the zinc concentrate is floated. The selection of reagents for treating the Pitarrilla rock has been made based on laboratory testing and work reasonably well. Other reagent, reagent combinations, reagent dosage rates, or the order of addition may be investigated by additional testing or during plant trials to improve selectivity in the lead and zinc separation or to improve recovery.

In the lead mineral flotation circuit, flotation of the lead concentrate will be done at natural slurry alkalinity (estimated to be in the range of pH 7 to 8). Slurry alkalinity in the zinc flotation circuit will be adjusted to pH 10.5 and maintained by the addition of lime. Sodium cyanide will be added to the lead circuit to depress zinc minerals and pyrite while the lead minerals float to the concentrate. The flotation mineral collecting reagent will be the promoter Aerophine 3418A in the lead circuit and SIPX in the zinc circuits. Copper sulphate will be added after lead flotation and to activate the zinc minerals before zinc mineral flotation.

During plant operation there will be a carry back of lime from the tailing pond in the recycled process water which will reduce the consumption rate indicated in the testwork. Recent testwork indicates that Sediments and Basal Conglomerate rock types, which are the dominant rock types, required 1,480 g/t and 1,600 g/t of lime respectively for pH adjustment in flotation. Therefore 1,500 g/t dosage rate is recommended for the operating budget.

Early testwork used a xanthate reagent for the mineral collecting reagent. All flotation testwork after 2008 has indicated that using Cytec Aerophine 3418A improves the lead concentrate grade without a detrimental effect on lead recovery. A dosage rate of 50 g/t is recommended for the operating budget.

The dosage rate of 30 g/t for sodium cyanide has not been investigated for optimization. In addition to zinc and iron mineral depression, cyanide may be beneficial in activating the lead mineral (galena) due to cleaning action on the galena particle surfaces. The dosage rate of 30 g/t is recommended for the operating budget.

Copper sulphate dosage is based on the general rule of thumb that a dosage of 75 g/t of copper sulphate is appropriate for each percent of zinc content in the rock. For the estimated average zinc or grade of 1.31% zinc, the dosage rate of 98.5 g/t is recommended for the operating budget.

During all flotation testwork, SIPX was used as the zinc collector. A dosage rate of 30 g/t is recommended for the operating budget.

During all the flotation testwork, Methyl Isobutyl Carbinol (MIBC) was used as the primary frother. A high addition rate of 80 g/t has been used for design. A secondary frother (F-549) was used in some testwork as a stronger frother, this is no longer commercially available; an alternate Dowfroth 1012 is recommended at a dosage rate of 50 g/t.

Estimating the plant dosage rate of frother from laboratory test procedure data is not done with reasonable accuracy. A dosage rate of 80 g/t (which should be sufficient based on a manufacturer's historical information from operating plants) is recommended for the operating budget.

The flotation circuits estimated reagent consumption rates are presented in Table 13-12

**Table 13-12: Estimated Flotation Reagent Consumption Rates**

| Item | Rate (g/t) |
|---|---|
| Promoter, 3418A (lead and silver) | 50 |
| Collector, SIPX (zinc) | 30 |
| Frother MIBC | 80 |
| Frother Dowfroth 1012 | 50 |
| pH Modifier, Lime | 1,500 |
| Depressant, Sodium Cyanide | 30 |
| Activator, Copper Sulphate | 98.5 |

### 13.4.2 Cyanide Leaching

The reagents used are standard additives for the cyanide leaching and Merrill-Crowe processes for the recovery of silver.

The leaching circuit will be employed in two manners: firstly to cyanide leach Direct Leach rock, and secondly to cyanide leach the tailings from the flotation process.

There has been considerable cyanide leaching testwork to optimise the combination of particle size, leaching time, and cyanide strength. The recommended cyanide strength is one gram per litre sodium cyanide in the process slurry. At this cyanide strength, the consumption of sodium cyanide is 1,000 g/t for Direct Leach rock, and 700 g/t for flotation tailings. The estimated reagent consumption rates in cyanide leaching circuit are listed in Table 13-13.

**Table 13-13: Estimated Leaching Reagent Consumption Rates**

| Item | Rate (g/t) |
|---|---|
| Lime (leaching) | 2,000 |
| Lime (cyanide destruction) | 1,000 |
| Sodium cyanide (direct leach) | 1,000 |
| Sodium cyanide (flotation tailings) | 700 |
| Flocculant | 180 |
| Sulphur (cyanide destruction) | 570 |

## 13.5 OTHER METALLURGICAL RESEARCH

In addition to the testwork described in this Section, other metallurgical research and testwork were performed that did not yield satisfactory results and were therefore terminated.

The first testing of the Property, in 2005, gave typically poor recoveries. The samples were from the top of the deposit, which is now identified as oxide rock. An extensive series of both conventional and non-conventional process and pre-treatment steps were generally unsuccessfully attempted. These methods included ultra-fine grinding, microwaving, various chemical treatments to modify clays and manganese minerals, all which were applied prior to conventional cyanide leaching. Flotation of iron-oxides to produce an iron-oxide and silver concentrate, from the oxide rock was also unsuccessful.

The process that was the most successful was an adaption of the Bayer process, where a high pressure and temperature caustic dissolution of the silicate minerals was attempted, thus unlocking the silver minerals. Post-dissolution, two downstream processes are required after a common solid-liquid separation stage; the solution proceeds to a silicate precipitation with lime, whereas the silver-bearing solids proceed to cyanide leaching and metals recovery.

The consistent improvement in silver recovery (by cyanidation), by the application of the Bayer process, lead to a scoping level capital and operating costs study, by Hatch Ltd. (Hatch report H339910, dated March 28, 2012). The consistent improvement in silver recovery (by cyanidation), by the application of the Bayer process, lead to a scoping level capital and operating costs study, by Hatch Ltd. (Hatch report H339910, dated March 28, 2012). The battery limits for this study were receiving the ground rock from the upstream grinding circuit; and generating two products, the final residue solids (to feed into cyanide leaching), and the precipitated silica as a waste stream The throughput of the process was set at 4,000 tpd. The operating conditions and resulting silver recovery improvements were extracted from testing (Process Research Associates, 2007).

The scoping level study for the Bayer process yielded a capital cost estimate of $546 million, with an operating cost of $190 /tonne. The Bayer testwork demonstrated that the silver recovery would increase from 20% to 40%to 80% to 90%, varying with tested composite.

### 13.6 TEST SAMPLE REPRESENTIVITY

In order to develop the metallurgical understanding of the deposits spatially representative samples were collected from all rock types, the samples represented the surface oxide rock, then the deep sulphide rock, and the intermediate transitional rock.

As such the tested samples are considered to represent the various styles and types of mineralisation and the mineral deposit as a whole.

### 13.7 DELETERIOUS ELEMENTS

The flotation testwork has produced both lead and zinc concentrates. These were analysed during the test programs for any deleterious elements that could affect marketability. Typical results are shown, above, in Table 13-1, Table 13-2 and Table 13-3.

These values of deleterious elements were considered during the Marketing study (Section 19) and are not considered to have a significant impact on the marketability of the lead and zinc concentrates.

# 14 MINERAL RESOURCE MODEL

## 14.1 INTRODUCTION

Silver Standard prepared an updated in-house Mineral Resource estimate for the Pitarrilla mineral deposit located in Durango State, Mexico. SSR used the Mineral Resource estimate as the basis for the Pitarrilla Feasibility Study (M3, 2012) undertaken by Silver Standard, which focused on the open-pit mining potential of the Breccia Ridge Oxide and Sulphide Zones, and the surrounding oxide deposits, Peña Dyke, Cordon Colorado, and Javelina Creek. At the request of Silver Standard, Xstract Mining Consultants Pty Ltd (Xstract) conducted an independent due diligence technical review of Silver Standard's Mineral Resources estimate (Section 14.10). Xstract's scope of work for the technical review focused on a detailed review of the data compiled, the estimation methodology, and classification criteria selected.

The updated Mineral Resource is based on all available data for the Pitarrilla deposit. Relative to the August 2008 Mineral Resource estimate completed by P&E (2008), the updated Mineral Resource estimate includes revised geological and mineralised domain interpretations, an updated bulk density database and bulk density model, an oxidation model, as well as a change of support using Localised Uniform Conditioning ("LUC") to estimate recoverable resources.

Jeremy D Vincent, B.Sc. (Hons), P.Geo., Senior Geologist at SSR, prepared the December 4, 2012 Mineral Resource estimate and accompanying documentation. Stefan Mujdrica, M.Sc., MAusIMM (CPGeo), General Manager – Geology and Technologies at Xstract, completed the independent technical review of the Mineral Resource model and estimate (Section 14.10).

## 14.2 DATA PREPARATION

All available data for the Pitarrilla Project were collated by SSR from various sources into a series of databases. Following compilation of these files, a final stage of review was completed prior to mineral resource modelling. The files containing the drillhole collar, downhole survey, assay, lithology, oxidation, and dry bulk density data from recent drilling programs were reviewed against field notes, logs, and assay certificates. Minor irregularities and discrepancies were noted, and then investigated and iteratively corrected.

Final compilation of the assay data was conducted in Microsoft Excel in preparation for mineral resource estimation:

- Ag, Pb, Zn, Cu, AgCNFinal, As, S, and Ca data in the database were reviewed and validated for below detection limit and overlimit samples

- Final Ag data were compiled using the ME-ICP61 (four acid digestion with an ICP-AES finish; see Section 11) Ag data for grades below 100 ppm; and Ag-GRA21 (30 g fire assay with a gravimetric finish) for high grade Ag above 100 ppm. Values higher than the upper level of detection for the fire assay technique (>10,000 ppm) were set to 10,000 ppm. Values less than the lower detection limit of the ME-ICP61 method (<0.5 ppm) were set to 0.25 ppm.

- Final Zn grade data (in ppm) were compiled using the ME-ICP61a (four acid digestion with an ICP-AES finish) Zn data for grades below 10,000 ppm (1%); and the Zn-OG62 (four acid digestion with an AAS finish) data for Zn grades above 10,000 ppm. Values higher than the upper level of detection for the latter technique (>30%) were set to 300,000 ppm. Values less than the lower detection limit of the ME-ICP61 method (<2 ppm) were set to 1 ppm.

- Final Pb grade data (in ppm) were compiled using the ME-ICP61a (four acid digestion with an ICP-AES finish) Pb data for grades below 10,000 ppm (1%); and the Pb-OG62 (four acid digestion with an AAS finish) data for Pb grades above 10,000 ppm. Values higher than the upper level of detection for the latter technique (>20%) were set to 200,000 ppm. Values less than the lower detection limit of the ME-ICP61 method (<2 ppm) were set to 1 ppm.

- Final Cu grade data (in ppm) were compiled using the ME-ICP61a (four acid digestion with an ICP-AES finish) Cu data for grades below 10,000 ppm (1%); and the Cu-OG62 (four acid digestion with an AAS finish) data for Cu grades above 10,000 ppm. No values reached the upper detection limit of 400,000 ppm (40%). Values less than the lower detection limit of the ME-ICP61 method (<1 ppm) were set to 0.5 ppm.

- Final AgCNFinal (hot cyanide soluble silver) data were compiled using the Ag-AA13hy (hot 2% NaCN leach with an AAS finish). AgCNFinal data for grades below 100 ppm; and Ag-AA13hO (hot 2% NaCN leach with an AAS finish) for high grade AgCNFinal above 100 ppm. Values higher than the upper level of detection for the Ag-AA13hO technique (>1,500 ppm) were set to 1,500 ppm. Values less than the lower detection limit of the Ag-AA13hy method (<0.03 ppm) were set to 0.015 ppm.

- There were no overlimit techniques used for As (in ppm), S (in %), and Ca (in %) samples. Values below the analytical detection limit for As, S, and Ca were respectively set to 2.5 ppm, 0.005%, and 0.005%. Samples that reached the upper detection limits for As (>10,000 ppm), S (>10%), and Ca (>50%) were respectively set to 10,000 ppm, 10%, and 50%. The upper detection limit for Ca was not reached.

The December 4, 2012 Mineral Resource estimate database contains assay data derived from diamond core drillholes and RC drillholes. Based on a series of validation checks, SSR considers the two assay datasets as comparable and can be combined for use in Mineral Resource estimation. The final, validated header (drillhole collar file; 835 records), survey (drillhole downhole survey data; 3,509 records), assay (132,573 records), lithology (4,242 records), oxidation (12,001 records), and density (8,538 records) files used as input for the December 4, 2012 Mineral Resource model.

## 14.3 GEOLOGICAL AND MINERALOGICAL DOMAIN INTERPRETATION AND SOLID GENERATION

### 14.3.1 Lithology

In 2012, SSR prepared an updated geological interpretation representing the rock formations at Pitarrilla, as summarised in Section 7. Cross-sectional interpretations were first generated on paper and used to assist the modelling process using Aranz Leapfrog software version 2.4.5.17 (Leapfrog). SSR undertook modelling of fault surfaces as a part of study to further understand structural controls within the deposit (Section 7.3). Fault surfaces that exhibited the most displacement were chosen for incorporation into the lithological model. These were the Peña, Peña2, Peña5, PeñaWest2, and Regional West 3 faults.

Following the work of Somers *et al*. (2010), individual lithologies were grouped into geological units within the five fault-bounded blocks for geological modelling. The modeled lithologies define the LITHWF field in the block model and are summarised in Table 14-1.

**Table 14-1: Block model LITHWF Coding (from Oldest to Youngest)**

| Lithology Unit | LITHWF Code |
|---|---|
| Peña Ranch Formation | 1000 |
| Manto Rico Member | 2000 |
| Pitarrilla Formation | 3000 |
| Cardenas Formation | 4000 |
| Casas Blancas Formation | 5000 |
| Encino Formation | 6000 |
| Upper andesite sill | 7000 |
| Lower andesite sill | 8000 |
| Felsic intrusive | 9000 |

### 14.3.2 State of Oxidation

Due to a demonstrated relationship of increasing cyanide-leach and flotation metallurgical recovery with increasing depth (most likely associated with the transition from oxidised to primary sulphide material), and testwork relating oxidation state of test samples and metallurgical performance, SSR elected to re-log a total of 135.8km of core (from drillhole core photographs) into a six point qualitative oxide logging code ("OXCODE") scheme) to better understand the nature of the oxidation profile (Section 7.6.1), and to provide a model basis for developing metallurgical performance for the entire drill tested block model volume.

These data were composited to a 7.5 m length down from the drillhole collar with no additional constraints to improve point data representivity for estimation. Data were composited as integers, allowing these to become real numbers with decimals – to take combinations of integers as a function of the compositing into account, thereby retaining the resolution of the initial logging, as far as possible. The following OXCODE data bins therefore resulted, and were used as equivalent to the 0-5 classification scheme:

- = 0.00-0.49
- = 0.50-1.49
- = 1.50-2.49
- = 2.50-3.49
- = 3.50-4.49
- = 4.50-5.49

### 14.3.3 Mineralisation Domains

In 2010 a detailed review was conducted by SSR on the August 2008 sulphide domain solids (domains 20 through 110) completed by P&E (2008) using an updated drillhole database and an updated geological interpretation that had been completed in-house that year. Based on this review SSR decided to reinterpret the grade domain boundaries within the constraints of the geological interpretation, using the previous oxide and sulphide domains as a base.

A series of low grade (<20 g/t Ag), medium grade (20-100 g/t Ag) and high grade (>100 g/t Ag) mineralisation domains were interpreted by SSR within the constraints of the revised geological interpretation, and incorporating drillhole information not available at the time of the August 2008 Mineral Resource update, using Gemcom GEMS software. Interpretations were conducted in a series of vertical cross-sections striking 065°, and spaced at 25 m intervals from southeast to northwest along a bearing of 335°. Polylines were generated on each vertical cross-section for each domain that met the lithological, structural and grade criteria. Mutually exclusive wireframe solids were then generated from the polylines. Clipping was conducted based on the interpreted mineralisation model, with domains at successively higher structural positions in the deposit profile taking precedence over deeper domains. Domain codes were assigned to the wireframe solids, and the wireframes verified.

A detailed statistical review of the Ag data within the various interim mineralisation domains did not support generation of sub-domains for the 'low grade' mineralisation based on grade ranges. A total of ten 'low grade' mineralisation domains were therefore generated by SSR and named following the same naming convention used for the underground Pre-Feasibility Study completed in 2009 (P&E, 2008): BR_AB-SU, BR_C, BR_C-1, BR_C-2, BR_D, BR_E, BR_F, BR_G, BR_H, and BR_I (Table 14-2).

A detailed visual inspection of domain-coded and composited grade data (Ag, and to a lesser extent Pb and Zn) during the exploratory data analysis of these ten domains indicated that sub-domains (based on geographic clustering of high or low grades, and/or based on apparent differences in grade continuity geometry) were possible in the BR_AB-SU, BR_C-2, and BR_E domains. A lower grade and generally sub-horizontal northwestern sub-domain (20) could be differentiated from a higher-grade and generally moderately southeast dipping southeastern sub-domain (21) in the BR_AB-SU domain. Similarly a lower grade northwestern sub-domain (50) could be differentiated from a higher grade southeastern sub-domain (51) in the BR_C-2 domain. Elevated grades in the northwestern part (71) of domain BR_E could be differentiated from lower grades in the south-eastern part (70) of that domain.

Additionally, SSR noted that Zn grades extended beyond the boundaries of the Ag, Pb, and Cu mineralisation in the basal conglomerate (BR_D) horizon. A 'high grade Zn' extension domain (65) was created to capture the high grade Zn intersections outside of the BR_D mineralisation domain.

Diamond drilling in 2011 and 2012 primarily focused on the delineation of shallow oxide-associated mineralisation in the South Ridge, Cordon Colorado, and Peña Dyke Zones, as well as shallow oxide- and sulphide-associated mineralisation in the Breccia Ridge Zone. Based on the latest information after the completion of the 2012 drilling campaign and the updated geological interpretation completed in 2012, a detailed review was conducted by SSR on the Oxide domain wireframes of the South Ridge, Cordon Colorado, and Peña Dyke Zones from McCrea (2006); the Breccia Ridge Oxide (BR-AB_OX, Domain 10) as defined by P&E (2008); and the BR-AB_SU (domains 20 and 21) Sulphide domain wireframes revised by SSR in 2010. Based on this review SSR decided to reinterpret the grade domain boundaries using Leapfrog within the constraints of the geological interpretation, using the previous Oxide and Sulphide domains as a base. Domains within and below the BR_C horizon (i.e., domains 30 through 110) were left unchanged from the 2010 review. Figure 14-1 illustrates the December 4, 2012 mineralisation domain solids.

To facilitate mineralisation domain definition, several filters were created in Leapfrog to constrain the selection of samples based on lithology, oxidation, and grade. Lithological units were restricted to the Pitarrilla Formation and those units stratigraphically higher (i.e., LITHWF code 3000 through 9000). An OXCODE threshold of 2.5 was set to distinguish oxide material from transitional and sulphide material. Grade selection was based on a 20 g/t total silver cut-off over a minimum length of 7.5 m. A second grade filter, based on a 100 g/t total silver cut-off over a minimum length of 7.5 m, was combined with the lithology and oxidation filters and was used to identify zones of continuous high-grade silver mineralisation within the Breccia Ridge and South Ridge Zones. Table 14-2 summarizes the updated mineralisation domains (3 through 21) and the remaining domains (30 through 110) used in the December 4, 2012 Mineral Resources estimate.

## Table 14-2: Pitarrilla December 4, 2012 Mineralisation Domain Details

| Domain Code | Domain Name | Details |
|---|---|---|
| 3 | BR-AB_OX | Breccia Ridge/South Ridge Zone: High grade (Ag>100 g/t) Oxide mineralisation hosted primarily by the felsic intrusive and the volcaniclastic tuff units (Cardenas Fm), wholly contained within Domain 10. Domain also defined by OXCODE>2.5 from oxidation logging. |
| 4 | PD_OX | Peña Dyke Zone: Oxide mineralisation hosted by the felsic intrusive. |
| 6 | CC_OX | Cordon Colorado Zone: Oxide mineralisation hosted by the felsic intrusive. |
| 8 | JC_OX | Javelina Creek Zone: Disseminated Oxide mineralisation hosted by volcaniclastic tuff units (Pitarrilla Fm). |
| 10 | BR_AB-OX | Breccia Ridge and South Ridge Zone: Oxide mineralisation hosted primarily by the felsic intrusive and volcaniclastic tuff units (Cardenas Fm). Domain also defined by OXCODE>2.5 from oxidation logging. |
| 20 | BR_AB-TR | AB domain: Disseminated Transitional domain hosted by rhyodacitic volcaniclastics (Pitarrilla Fm). Domain defined by OXCODE < 2.5 from oxidation logging. |
| 21 | BR_AB-TR | AB domain: Disseminated high grade Transitional domain (Ag>100 g/t) hosted by rhyodacitic volcaniclastics (Pitarrilla Fm), wholly contained within Domain 20. Domain defined by OXCODE<2.5 from oxidation logging. |
| 30 | BR_C | C domain: Horizontal Sulphide mineralisation in lower andesite sill |
| 40 | BR_C-1 | C-1 domain: Horizontal Sulphide mineralisation in lower andesite sill. |
| 50 | BR_C-2 | C-2 domain (NW): Horizontal Sulphide mineralisation in lower andesite sill (low grade NW part). |
| 51 | BR_C-2 | C-2 domain (SE): Horizontal Sulphide mineralisation in lower andesite sill (higher grade SE part). |
| 60 | BR_D | D domain: Basal Conglomerate Zone (highest grade Sulphide mineralisation; Manto Rico Member). |
| 65 | BR_D | D domain: Basal Conglomerate Zone extensions (Sulphide mineralisation; high in Zn, low in Ag, Cu and Pb; Manto Rico Member). |
| 70 | BR_E | E domain (SE): hydrothermal breccia, fracture and vein-hosted Sulphide mineralisation adjacent to and inside felsic dykes intruded in shales and siltstones (low grade SE part) (Peña Ranch Fm) |
| 71 | BR_E | E domain (NW): hydrothermal breccia, fracture and vein-hosted Sulphide mineralisation adjacent to and inside felsic dykes intruded in shales and siltstones (higher grade NW part) |
| 80 | BR_F | F domain: hydrothermal breccia, fracture and vein-hosted Sulphide mineralisation adjacent to and inside felsic dykes intruded in shales and siltstones (Peña Ranch Fm) |
| 90 | BR_G | G domain: hydrothermal breccia, fracture and vein-hosted Sulphide mineralisation adjacent to and inside felsic dykes intruded in shales and siltstones (Peña Ranch Fm) |
| 100 | BR_H | H domain: hydrothermal breccia, fracture and vein-hosted Sulphide mineralisation adjacent to and inside felsic dykes intruded in shales and siltstones (Peña Ranch Fm) |
| 110 | BR_I | I domain: hydrothermal breccia, fracture and vein-hosted Sulphide mineralisation adjacent to and inside felsic dykes intruded in shales and siltstones (Peña Ranch Fm) |
| 99999 | - | Waste domain encapsulating the ultimate pit created to assist with mine planning. |

Note: 'BR_' prefix stands for Breccia Ridge Zone; 'PD' stands for Peña Dyke Zone; 'CC' stands for Cordon Colorado Zone; 'JC' stands for Javelina Creek Zone. Increasing depth is generally reflected in increasing alphanumeric values of domain code and name.

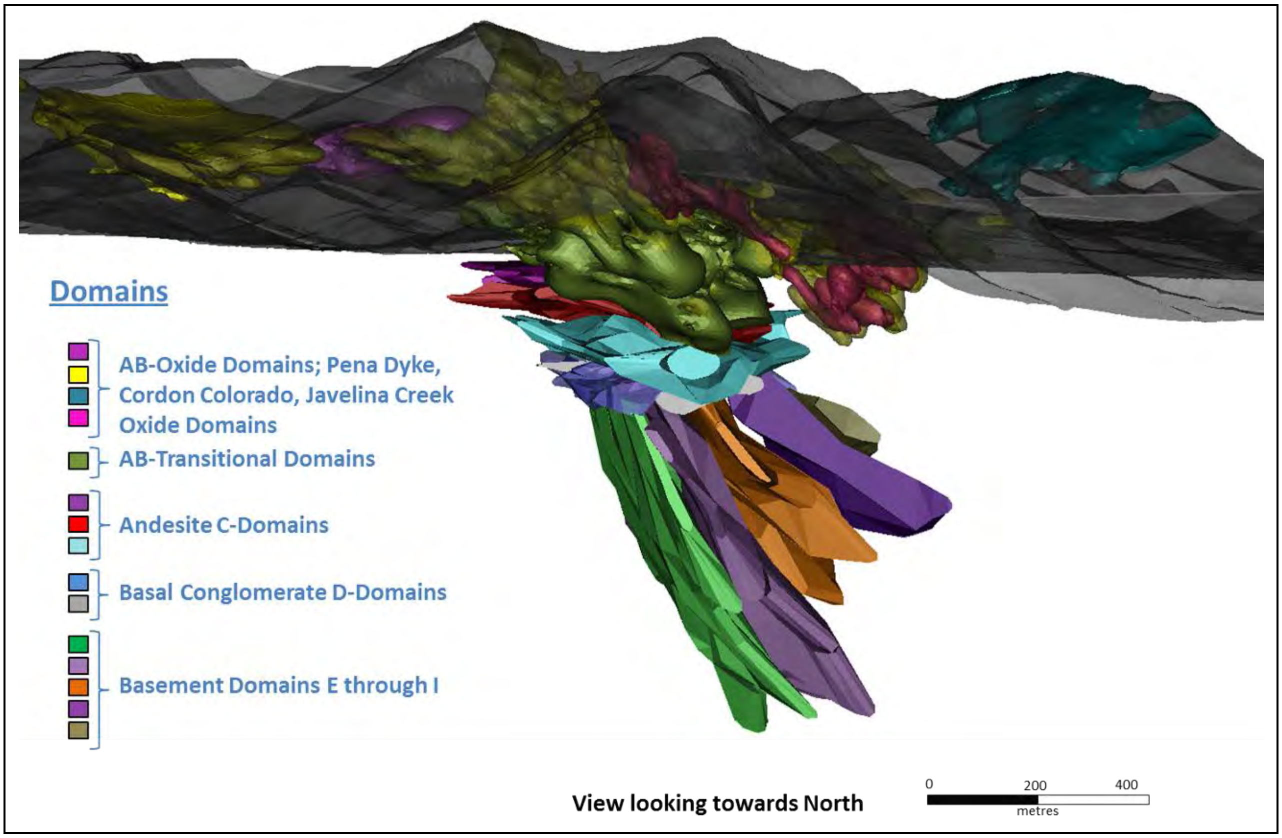



**Figure 14-1: Pitarrilla December 4, 2012 Mineralisation Domain Solids**

Isometric view of the Pitarrilla Ag resource domains .Source: M3, 2012.

## 14.4 EXPLORATORY DATA ANALYSIS

### 14.4.1 Overview

Following finalization of the 19 domain solids, the domain codes were back-coded to the drillhole data. Domain-coded drillhole data were then extracted and exported for exploratory data analysis using Snowden Mining Industry Consultants' ("Snowden") Supervisor software (version 7.11.10), summarised in Table 14-3.

**Table 14-3: Elements Evaluated as a part of the December 4, 2012 Mineral Resource Model**

| Element | Details |
|---|---|
| Ag | Prime economic contributor to the Pitarrilla project, comprising approximately 88% of payable metal value. |
| Pb | Minor economic contributor to the Pitarrilla project, comprising approximately 3% of payable metal value. |
| Zn | Minor economic contributor to the Pitarrilla project, comprising approximately 9% of payable metal value. |
| Cu | Present in low concentrations and unimportant to project economics. Payable metal is valued at 0%. Estimated grade data are not reported publically. |
| AgCNF | Hot-cyanide soluble silver grade values, which provide a rough approximation of Ag recovery in oxide and transitional mineralisation domains. These data are used for internal purposes only as the metal recovery function is determined through the metallurgical studies (Section 6). Estimated grade data are not reported publically. |
| AgREC% | Silver recovery, a calculated value by taking the ratio of AgCNF to Ag and expressing it in percent. These data are used for internal purposes only as the metal recovery function is determined through the metallurgical studies (Section 13 and Section 17). AgREC% data are not reported publically. |
| As | Deleterious element. These data are used for internal purposes only. Estimated grades are not reported publically. |
| S | Deleterious element estimated to assist with the definition of acid generating rock. These data are used for internal purposes only. Estimated grades are not reported publically. |
| Ca | Estimated to assist with location of acid-neutralizing rock. These data are used for internal purposes only. Estimated grades are not reported publically. |

### 14.4.2 Visual Analysis and Basic Statistics

Visual analyses of the colour-coded assay (and later, composite) data (both in Gemcom and in Supervisor) provided a good understanding of the mineralisation geometry, which guided the selection of horizontal, across-strike and dip-plane directions during the fan-based continuity analyses. The following observations were made by SSR:

- Ag mineralisation generally appears to be sub-horizontal in PD_OX and CC_OX (respectively domains 4 and 6). The potential to segregate high-grade Ag mineralisation within sub-domains exists and needs to be further investigated. Contents of Pb, Cu, S, and As are relatively low, thus it is difficult to discern their orientations and anisotropy.

- Ag mineralisation in the JC_OX (domain 8) follows the moderately west dipping stratigraphy and appears to be moderately anisotropic. Pb, Cu, As, and S in this domain are also present in low concentrations, making it difficult to determine orientations and anisotropies.

- Ag mineralisation in the domains above the unconformity and into the lower andesite sill unit (domains 30 through 65) generally appears to be sub-horizontal, while mineralisation in the transitional and oxide domains dips moderately to the northeast. Higher grade sub-domains were observed in the South Ridge Zone of the BR_AB-OX, the central part of the BR_AB-TR and BR_C-2 domains. Ag mineralisation in domain 60 dips at a shallow angle to the west. Ag mineralisation appears to be moderately anisotropic in all domains except 3, 21, 40, and 50, which appear to be isotropic.

- Below the unconformity, Ag grades appear to display generally anisotropic orientations down the dip of the steeply northeast dipping intrusion surrounding hydrothermal breccia and fracture zones. Overall there appear to be two 'conduits' of elevated Ag mineralisation - a more dominant one in the north-western parts of the steeply oriented zones, and a more diffuse zone further southeast, separated by lower grade mineralisation. The elevated Ag grades noted in the north-western parts of the BR_E zone lead to the generation of sub-domain 71.

- Elevated Ag mineralisation is most strongly developed in domains 3, 21, 51, 60 and 71, with reasonably strong mineralisation present in domains 40, 50, and 80.

- Elevated Pb mineralisation is more restricted in geographic extent relative to Ag, but displays similar geometric characteristics to the Ag mineralisation in domains 60, and 70 through 110. The strongest Pb mineralisation is developed in domains 60, 80, 100, and 110. Pb mineralisation displays similar orientation and anisotropy to the Ag mineralisation. In the oxide domains, Pb is present in low concentrations and does not exhibit a strong anisotropy.

- Elevated Zn mineralisation is more geographically extensive than either Ag or Pb, and extends beyond the extent of this mineralisation in the basal conglomerate (BR_D domain). This has resulted in the generation of the domain 65 extension haloes to the BR_D domain. Elevated Zn mineralisation is most strongly associated with the basal conglomerate (domain 60) and extension zones (domain 65), although elevated Zn mineralisation is also present in domains 3, 10, 20, 51, 71, 80, 100, and 110. Geometric orientation and anisotropy developed in the Zn mineralisation is broadly similar to that displayed by Ag, with differences observed in domains 60, 65, and 90. Zn is present in very low concentrations in the PD-OX, CC-OX, and JC-OX domains (domains 4, 6, and 8 respectively), thus making it difficult to discern orientations and anisotropy.

- Elevated Cu mineralisation is the most geographically restricted of the four economic variables. Relatively strong Cu mineralisation is limited to domains 20, 21, 30, 50, 60, and 90. The two mineralisation 'conduits' noted in the Ag, Pb, and Zn mineralisation below the unconformity are also developed in the Cu mineralisation. Although mineralisation orientations are generally similar to those of the abovementioned variables, the Cu mineralisation does display different anisotropy relative to these variables.

- In the Transitional and Sulphide domains, elevated As mineralisation broadly displays a similar geographic spread to the Ag mineralisation, with differences in geometric orientation and anisotropy in almost all domains. Elevated As mineralisation is most strongly developed in domains 40, 50, 60, 71, 80, 100, and 110. The two mineralisation 'conduits' discussed above are also evident in As mineralisation in the steeply northeast dipping domains (70 through 110) below the unconformity. In the Oxide domains, it has strongly contrasting spread to the Ag mineralisation, as it is focused strongly with the Breccia Ridge Zone of the BR-AB_OX domain, exhibiting the upper part of the mineralisation 'conduit'.

- Relatively elevated S contents are most strongly associated with domains 60 and 65. Other domains that display slightly, to moderately elevated S contents include domains 30, 40, 50, 51, 100, and 110. S contents display different orientations and anisotropy relative to the other variables in most domains except domains 100 and 110. S contents are low in the Oxide domains (domains 3 through 10) and do not display specific orientations or anisotropy.

- Ag, Pb, Zn, Cu, and S variables display similar orientations and anisotropy in domains 100 and 110.

- Ca contents are depleted throughout the mineralised domains except locally within domains 60 and 65, within the calcareous basal conglomerate unit. No specific orientations or anisotropy are exhibited in any of the mineralisation domains.

The above observations support the current conceptual geological and mineralisation model for the deposit.

The basic statistics of the uncomposited domain-coded data indicated that several of the domains were characterised by mixed populations (due to the incorporation of low grade internal waste) and/or strongly skewed distributions (due to the presence of extreme elevated values), which was reflected in their elevated coefficients of variation ("CV"). A detailed visual inspection of the various domains indicated that they could not simply be further sub-domained at the current drillhole spacing without significantly breaking up the resource into numerous discontinuous zones. This indicated that either a non-parametric approach (e.g., indicator kriging) or the use of grade capping (i.e., top cuts) with standard interpolation techniques (e.g., ordinary kriging, inverse distance, etc.) was required for meaningful grade estimation.

### 14.4.3 Regression Analysis

Regression analysis was conducted by SSR on the composited, domain-coded Ag, Pb, Zn, Cu, AgCNF, S, As, and Ca variables in all 19 domains to assess correlations between the variables to guide subsequent top cut analyses and estimation parameter selection. The following observations were made based on the results of the regression analysis:

- Ag and Pb tend to display moderate to good correlation in the Transitional and Sulphide domains. Consequently preservation of this relationship, as far as possible, in the interpolation process requires grade capping at the same percentile and the same continuity (semi-variogram) parameters to be used for the two components.

- As expected, Ag and AgCNF display strong correlations in the Oxide and Transitional domains. Relationship preservation during interpolation requires the same approach to grade capping and continuity parameters as described in the first bullet point.

- Ag, Pb, Zn, Cu, and S tend to display moderate to good correlation in domains 100 and 110. Relationship preservation during interpolation will also follow grade capping and continuity parameter selection procedures described above.

- Zn, Cu, As, S, and Ca variables appear to display sufficiently weak correlations to Ag and Pb, and to each other, that the assumption of independence for interpolation is considered reasonable. Scatter plots for these regressions indicate that outlier values may be responsible for some of the weak to moderate correlations developed between some of these variables.

The results of the regression analysis also appear to fit with the conceptual geological and multi-phase mineralisation model. The relatively good correlation between Ag, Pb, Zn, Cu, and S in domains 100 and 110 could indicate that these domains are more proximal to a primary source. Poorer correlations at higher structural levels are considered to reflect the multi-stage mineralizing process. The very weak negative correlations between Ag and Ca indicate that some remobilization of Ca is occurring during the mineralisation process.

### 14.4.4 Compositing

Results of the sample length analysis indicated that approximately 92% of the sample data was collected over sample lengths of 1.50 m or less, with mean sample lengths ranging between 1.05 m (domain 6) and 1.46 m (domain 110). Outside of domains 4, 6, and 8, which show the primary sample length to be 1.0 m from RC drilling programmes, the proportion of sample lengths of exactly 1.50 m generally range between 50% (domain 60) and 82% (domain 110), with the remaining domains in the 66% to 80% range. These results support the selection of a 1.50 m composite length (shorter than 1.50 m, given the proportion of exactly 1.50 m sample lengths, would result in an artificially suppressed nugget; larger would be acceptable, but not necessary given the proportion of sample lengths at or below 1.50 m).

Domain-coded drillhole data (all 19 domains) were composited in Gemcom to a length of 1.5 m, with residual lengths retained, and visually validated against domain wireframes. Basic ‘metal’ content validation checks (i.e., comparison of the sum product of sample length and variable grade between the raw and the composited datasets) were precluded due to the presence of a distinct negative correlation between grade and length (for the variables of interest) in the raw data, with elevated grades being associated with short sample intervals in zones with variable sample lengths. Additional composite validation was conducted by comparing the average full-width composite grade for a given domain generated from the 1.50 m composited data to that generated from the raw data. Good comparisons were obtained validating the compositing process.

#### 14.4.5 Top Cut Analysis

Grade capping or top cut analysis was conducted on the domain-coded and composited grade variable data to limit the influence of extreme values during grade interpolation. Top cut values were chosen through analyses of histogram, log-histogram, and log-probability plots to identify population breakdowns in the sample grade populations. Correlations between variables were taken into consideration in the generation of the top cut values (where variables were significantly correlated, the top cut of secondary variables was set to the exact percentile used for grade capping the primary variable).

### 14.5 CONTINUITY ANALYSIS

Three dimensional continuity analyses (variography) were conducted on the top cut, domain-coded, composite data using Snowden's Supervisor geostatistical software (version 7.11.10). Results of the visual exploratory data analysis (Section 14.4) were used to guide the selection of horizontal, across-strike, and dip-plane directions during semi-variogram fan analysis. In general, horizontal and across-strike directions were forced according to the modelled geology, with the dip-plane direction aligned to the direction of maximum continuity. If the direction of maximum continuity was unclear, the direction was set to either along strike or down dip. Semi-variogram fans were viewed as traditional and as normal-score transformed to check the selected directions. Four experimental semi-variograms were generated for each variable in each domain, a downhole semi-variogram and three directional semi-variograms, with one along each of the three principal directions of the modelled continuity ellipse from the semi-variogram fans.

The downhole semi-variogram was viewed at a 1.50 m lag (equivalent to the sample spacing) as a traditional semi-variogram and as a normal-scores transformed semi-variogram to assess the nugget variance. In general, a good match was found between the nugget for the two approaches (the normal-scores nugget was back-transformed prior to comparison).

Experimental semi-variograms for each of the three principal directions were generated with an angular tolerance of generally 20° (wider where necessary – i.e., where model directions 'see' few data due to their orientation relative to the selected drillholes). The best experimental semi-variograms were developed in normal-score transformed space, and consequently semi-variogram modelling was done using this transformation. Nugget, one-, two-, and occasionally three-structure standardised spherical models were generally used to model the experimental semi-variograms in normal-score transformed space. Semi-variogram models, once generated, were checked across a range of lags to assess robustness (starting lag selected at 50 m, close to the drillhole spacing). Semi-variogram model ranges were checked, and iteratively refined where necessary, for each model by viewing relative to traditional (untransformed) experimental semi-variograms. Hermite polynomials were then used to model the cumulative distribution functions for each variable in each domain, prior to estimating the back-transformed variance (nugget and sill) contributions, to generate the back-transformed semi-variogram models. Back-transformed variance contributions (nugget and sills) were checked relative to the apparent variance contributions for the given model in traditional (untransformed) space, where possible. Semi-variogram quality (Table 14-4), which was assessed by checking the robustness of the directional semi-variograms over a variety of lag distances, the quality of the definition of the nugget and short range structures, and the number of data pairs used to establish the experimental semi-variogram points, was noted for each domain and taken into consideration during resource classification

**Table 14-4: Experimental Semi-Variogram Quality**

| Domain | Ag | Pb† | Zn |
|---|---|---|---|
| 3 | Good | Good | Good |
| 4 | Good | Mod-Good | Good |
| 6 | Good | Moderate | Mod-Good |
| 8 | Mod-Good | Poor-Moderate | Poor-Moderate |
| 10 | V.Good | Mod-Good | Mod-Good |
| 20 | Mod-Good | Good | V.Good |
| 21 | Poor | Poor-Mod | Moderate |
| 30 | Poor-Mod | Moderate | Mod-Good |
| 40 | Poor-Mod | Moderate | Good |
| 50 | Poor | Moderate | Mod-Good |
| 51 | Poor-Mod | Poor-Mod | Poor-Mod |
| 60 | Moderate | Mod-Good | Mod-Good |
| 65‡ | N/A | N/A | N/A |
| 70 | Moderate | Poor-Mod | Mod-Good |
| 71 | Moderate | Poor-Mod | Moderate |
| 80 | Moderate | Poor-Mod | Moderate |
| 90 | Moderate | Moderate | Mod-Good |
| 100 | Poor | Poor | Poor |
| 110 | Poor | Poor | Poor |

Notes: Very good = robust in all three principal directions with good definition of nugget and short range structures; Good = semi-variogram definition in all three principal directions, but robustness across lags may be inconsistent; Moderate = semi-variogram definition in all three principal directions, but with moderate to poor definition of short range structures, and not robust across lags; Poor = weak to no definition of semi-variograms in each direction; not robust across lags.

†From Section 14.4.3 Ag semi-variogram models applied to Pb experimental semi-variograms for estimation purposes. This assumption is validated by the generally good fit of the Ag semi-variogram models over the Pb experimental semi-variograms.

‡Meaningful semi-variograms could not be generated for domain 65 due to its relatively small size and spatial distribution (it is represented as discrete pods of high grade Zn mineralisation peripheral to domain 60). Semi-variogram parameters were borrowed from Domain 60 for resource estimation.

A continuity ellipse, based on the back-transformed semi-variogram models, was generated for each variable in each domain for validation purposes. Compatibility was set to the Gemcom ZXZ rotation convention (with X set to principal direction D1). All continuity ellipses for each variable in each domain were generated as search ellipses in Gemcom to visually validate ellipse orientations.

To retain the correlations between various variables (e.g., Ag and AgCNF in domains 3 through 21, Ag and Pb in domains 20 through 110, and Ag, Pb, Zn, Cu, S in domains 100 and 110), continuity parameters for Ag were assumed and applied for the relevant variables in the appropriate domains. This assumption was checked by generating experimental semi-variograms for the related variables in the relevant domains, using the same semi-variogram fan directions as for Ag in that domain. Ag semi-variogram models were then superimposed on the resulting experimental semi-variograms in the three principal continuity directions. In general, the Ag continuity models were found to be reasonable-to-good models of the continuity of the related variables, thereby justifying the use of Ag continuity data in approximating the continuity of these variables.

## 14.6 BLOCK MODELLING AND GRADE ESTIMATION

### 14.6.1 Block Model Generation and Coding

A non-rotated block model was generated by SSR as the primary model for the December 4, 2012 estimate. A block size of 15mE x 15mN x 7.5mElevation was selected as the selective mining unit ("SMU"). Bench height studies testing various heights indicated that a 7.5mElevation height was an optimal value. Model parameters are shown in Table 14-5.

**Table 14-5: Parameters for the Pitarrilla December 4, 2012 Block Model**

| Model Parameter | Parameter Details |
|---|---|
| Block Model Origin* | X: 502800 m<br>Y: 2809800 m<br>Z†: 2120 m |
| Block Model Rotation | None |
| Block Size | Columns: 15 mE<br>Rows: 15 mN<br>Levels: 7.5 mElevation |
| Number of Blocks | Column size: 156<br>Row size: 162<br>Level size: 177 |

*Gemcom block model origin convention uses the upper, northwest corner of block model.
†Block model Z-coordinate values after block modelling and resource estimation were translated vertically upward by 20.9 m to account for the new topographic surface, which is orthometric and is relative to mean sea level.

An updated digital elevation model ("DEM") was utilised to constrain mineralised and waste blocks below the topographic surface for the December 4, 2012 resource estimate. The 2012 DEM is orthometric and is relative to mean sea level, which is in contrast to the WGS84 ellipsoid referenced by the topographic surface used in the 2008 resource estimate (P&E, 2008). The difference between these two surfaces over the footprint of the Pitarrilla deposit is corrected by applying a 20.9 m translation to the z-coordinate values. The 20.9 m vertical elevation translation is consistent across the deposit. Due to time considerations, it was decided that the 20.9 m shift would be most readily handled by translating the DEM vertically downward to the level of the previous topography (thereby precluding the need to translate all existing drillhole intercepts, wireframe solids, and the existing block model upward to match the elevation of the new DEM surface). Block modelling and resource estimation were then conducted under the translated, updated DEM surface. Before releasing the December 4, 2012 resource model for mining planning work, the block model z-coordinates were shifted upward to the original surveyed DEM surface elevation (i.e., upward by 20.9 m) and validated in Gemcom. In future model iterations, all relevant drilling data, wireframe solids, and the block model will be translated so that they are correctly located spatially with respect to the updated DEM surface.

Primary block model attributes created in the block model for coding and estimation are summarised in Table 14-6.

**Table 14-6: Block Model Fields**

| Field | Unit | Range | Detail |
|---|---|---|---|
| | | | **Primary Fields** |
| Rock Type | Integer; unitless | Min = -99<br>Max = 99999 | Initialised to 0, domain codes (3-110) updated from mineralisation solids; waste domain code (99999) updated from waste solid; air blocks (-99) updated above topographic surface (as per Table 14-2). |
| Density | $t/m^3$ | Min = -99<br>Max = 3.32 | Initialised to -99; estimated and assigned dry bulk density values in mineralisation and waste domains; air density = 0. |
| LITHWF | Integer; unitless | Min = -99<br>Max = 9,000 | Initialised to -99; lithological code updated from lithological solids (as per Table 14-1). |
| LITHECODE | Integer; unitless | Min = 0<br>Max = 990064 | Initialised to 0; density domain code for estimation or assignment (as per Table 14-7). |
| OXROCK | Integer; unitless | Min = 0<br>Max = 1 | Initialised to 0; oxidation domain code set to 1 inside waste and mineralisation solids. |
| OXCODE2 | Integer; unitless | Min = -99<br>Max = 4.49 | Initialised to -99; oxidation code; estimated where OXROCK = 1 (i.e., within waste and mineralisation solids); no values estimated where OXROCK = 0. |
| AGLUC | ppm | Min = 0.00<br>Max = 1,397.76 | Initialised to 0; Ag grade data estimated by Localised Uniform Conditioning (LUC) in domains 3 through 110; no values in other domains. |
| AGPPM | ppm | Min = 0.00<br>Max = 1,156.08 | Initialised to 0; Ag grade data estimated by Ordinary Kriging in domains 3 through 110 and by inverse distance squared for domain 99999. |
| AGPPMFINAL | ppm | Min = 0.00<br>Max = 1,397.76 | Initialised to 0; Ag grade data; comprises AGLUC values from domains 3 through 60 and 70 through 110, AGPPM values from domain 65 and 99999; Used for Mineral Resource reporting and mine planning. |
| AGCNFINAL | ppm | Min = 0.00<br>Max = 904.46 | Initialised to 0; Hot cyanide soluble Ag data estimated by Ordinary Kriging in domains 3 through 21. |
| AGREC% | % | Min = 0.00<br>Max = 100.00 | Initialised to 0; calculated as the ratio of AGCNFINAL/AGPPM, expressed as percent in domains 3 through 21. |
| PBPPM | ppm | Min = 0<br>Max = 44,321 | Initialised to 0; Pb grade data estimated by Ordinary Kriging in domains 3 through 110 and by inverse distance squared for domain 99999. |
| ZNPPM | ppm | Min = 0<br>Max = 149,870 | Initialised to 0; Zn grade data estimated by Ordinary Kriging in domains 3 through 110 and by inverse distance squared for domain 99999. Used for Mineral Resource reporting. |

| Field | Unit | Range | Detail |
|---|---|---|---|
| CUPPM | ppm | Min = 0<br>Max = 12,490 | Initialised to 0; Cu grade data estimated by Ordinary Kriging in domains 3 through 110 and by inverse distance squared for domain 99999. Used for Mineral Resource reporting. |
| ASPPM | ppm | Min = 0.00<br>Max = 8,311 | Initialised to 0; As grade data estimated by Ordinary Kriging in domains 3 through 110 and by inverse distance squared for domain 99999. |
| S_% | % | Min = 0.00<br>Max = 11.34 | Initialised to 0; S grade data estimated by Ordinary Kriging in domains 3 through 110 and by inverse distance squared for domain 99999. |
| CA% | % | Min = 0.00<br>Max = 20.89 | Initialised to 0; Ca grade data estimated by Ordinary Kriging in domains 3 through 110 and by inverse distance squared for domain 99999. |
| CLASS | Integer; unitless | Min = 0<br>Max = 4 | Initialised to 0; Assigned values of 1 (Measured), 2 (Indicated), 3 (Inferred) as per Section 4.9. Values of 4 are not reported and comprise a "Mineral Inventory", which is mineralised material not contained within mineralised domains 3 through 110 that can be used to assist with future exploration efforts. |
| **Fields to Assess Estimation Quality** | | | |
| AGNN | ppm | Min = 0<br>Max = 1,850.00 | Initialised to 0; Ag grade estimated by Nearest Neighbour for domains 3 through 110. |
| PBNN | ppm | Min = 0<br>Max = 87,002 | Initialised to 0; Pb grade estimated by Nearest Neighbour for domains 3 through 110. |
| ZNNN | ppm | Min = 0<br>Max = 220,826 | Initialised to 0; Zn grade estimated by Nearest Neighbour for domains 3 through 110. |
| AGBV | unitless | Min = 0<br>Max = 0.85 | Initialised to 0; Block variance, Ag variable; domains 3 through 110. |
| AGKV | $ppm^2$ | Min = 0.00<br>Max = 1.40 | Initialised to 0; Kriging variance, Ag variable; domains 3 through 110. |
| AGLAGRANGE | unitless | Min = -0.73<br>Max = 0.02 | Initialised to 0; LaGrange multiplier, Ag variable; domains 3 through 110. |
| AGDISTANCE | m | Min = 0.00<br>Max = 158.27 | Initialised to 0; Average distance of samples used to estimate block, Ag variable; domains 3 through 110. |
| AGPOINTS | Integer; unitless | Min = 0<br>Max = 20 | Initialised to 0; Average number of points used to estimate block, Ag variable; domains 3 through 110. |
| AGSLOPE | unitless | Min = -27807.8<br>Max = 841.6 | Initialised to 0; Ordinary Kriged estimation regression slope, Ag variable; domains 3 through 110. |

Block coding within the 19 mineralisation domain solids was undertaken in Gemcom, which utilizes "needling", a mathematical approach used to identify the proportion of a block within a solid. The mineralisation domain code was assigned to blocks in domains above the unconformity (i.e., the sub-horizontally oriented domains) to the Rock Type attribute using a 5x5 "needling" grid in an XY plane (i.e., block coding in a vertical profile). Blocks were designated as "in" if they were at least 50% within the mineralisation domain solid. Coding of blocks located below the unconformity (i.e., domains 70 through 110) was undertaken by applying the 5x5 "needling" grid in two horizontal profiles (in both XZ and YZ planes) using the same 50% rule to designate a block as either "in" or "out" of a domain solid. During the coding process, structurally higher domains (i.e., those with lower Rock Code values – Section 14.3.2) were given priority where blocks overlapped. A visual check block coding was performed to ensure correct coding. Additionally, the volumes of the blocks filled within domain solids were checked against the volume of the domain solids.

### 14.6.2 Oxidation Modelling

The validated, composited oxidation data were extracted as points from Gemcom and imported into Supervisor for variography analysis. Analysis was conducted on the waste (domain 99999) and the mineralised domains (3-110) with no domain-related constraint on the data. In general, horizontal and across-strike directions were forced according to the modelled geology, with the dip-plane direction aligned to the direction of maximum continuity. Interpolation of the composited oxide codes into the OXCODE2 block model attribute was conducted into the relevant blocks using an iteratively refined Ordinary Kriging technique.

Model validation included a visual inspection of the estimated oxide codes against the composited oxide coded input data on a series of 25 m spaced cross-sections and level plans through the model. Input oxide codes were honoured by the block estimates with the expected level of smoothing observed in the ordinary kriged estimates. In addition, a comparison of the estimated OXCODE2 values to the composited input drillhole file along northing, easting, and, elevation swath plots showed that the ordinary kriged model trends followed the input trends with expected amounts of smoothing.

### 14.6.3 Dry Bulk Density Modelling

The 2008 resource model interpolated 172 SG measurements using a single search to estimate density for the open pit and underground resources (P&E, 2008). These had been previously termed as bulk density samples within the report, but were SG samples based on a review of the analytical procedures followed. These samples were converted to dry bulk density values (DBD) by applying a void index function, which took into account the void index as a function of the depth of the bulk density sample down the hole, and used as a part of the December 4, 2012 DBD dataset.

In comparison to the 2008 resource model (P&E, 2008), a more sophisticated approach was utilised to represent density variability throughout the Pitarrilla deposit. A total of 8,538 dry bulk density measurements ("DBD") were used to generate a density model for the December 4, 2012 Mineral Resources estimate. DBD measurements were selected to be spatially and geologically representative (i.e., representative of geology, lithology, structure, mineralisation, alteration). The data were coded according to domain (Rock Type; 3 through 110, and 99999 for waste), lithology (LITHWF; 1,000 through 9,000), and oxidation code (OXCODE; 0 through 5) in Gemcom. This was exported to a Microsoft Excel spreadsheet for further coding, density averaging, and preparation of file subsets for variography in Supervisor and estimation in Gemcom.

#### 14.6.3.1 Dry Bulk Density in Mineralised Domains

An initial analysis indicated that the relationship between bulk density and oxidation code *sensu strictu* is tenuous, although *sensu lato* one can see that the higher up in the profile, the lower the density data. Consequently, SSR decided upon a hybrid approach of applying density data averages in domains where there were few conditioning data, taking oxidation code into account where possible, and estimating data using Ordinary Kriging by lithology and mineralisation domain, where sufficient conditioning data so allowed. DBD coding into the LITHECODE model attribute used a '1' prefix for a mineralised zone, followed by the lithology code, and, where necessary, the domain code. As an example, it was decided that all mineralisation in domains 70 through 110 was essentially the same style (fractures and hydrothermal breccias adjacent to, and partially inside of the felsic intrusive dykes), therefore they were allocated to a single domain for estimation – 11000 (mineralised, Cretaceous Sediments-hosted). Another example is that of codes 13010, 13020, and 13021 – it was necessary, due to mineralisation (oxide versus disseminated transitional material) and orientation differences, to estimate these in individual domains, despite being hosted in dacite.

Where blocks were coded, but no input data was available, a detailed review of adjacent domains was conducted, taking oxidation into account, to assign average density values to the relevant domains. Iterative validation was conducted during the estimation process to improving local estimates. Efforts were made to prevent over representation of elevated DBD values by restricting the range of their influence during estimation. The majority of the estimated bulk density domains have mean block bulk density values that display minor differences at the third decimal to the input conditioning data. Those domains that did not, had means within 5% and the differences were clearly due to the 'block effect' – i.e., 50 input data, clustered in one side of the domain, but many blocks on the other side getting estimates from a few data at the edge of the cluster. This was mitigated by using range limitations on elevated values, and by increasing the level of smoothing, where necessary.

#### 14.6.3.2 Dry Bulk Density in Unmineralised Waste Domains

Given the relatively poor spatial representivity of bulk density in the waste rock relative to the size of the waste domain (there are 3,427 DBD samples in the waste domain), SSR used average DBD data, per lithological unit and by oxidation code. The data were coded into the LITHECODE field as a concatenation of the domain, lithology, and oxidation fields. For example, a '9' prefix indicating the waste domain (Rock Type 99999), was followed by the lithology code (LITHWF 1000 through 9000), and finally the oxidation code (OXCODE 0 through 5). For example, the unmineralised lower andesite with an oxidation code of 3 would have a code of 980003.

Where possible, SSR averaged the DBD data within each oxidation code. Sometimes this was not possible due to limited data, in which case the data were averaged from two or more oxidation codes, always cognizant of the geology of the deposit. Being the waste domain, SSR considered this approach to be sufficient in achieving a model of the broad density variations in the waste, given the current conditioning data.

**Table 14-7** summarizes the LITHECODE dry bulk density domains estimated by Ordinary Kriging and those assigned average DBD values.

**Table 14-7: Summary of Estimated and Assigned Mean Dry Bulk Density Domains**

| Dry Bulk Density Mineralised Domain (LITHECODE) | Estimated, or Average DBD Value Assigned (t/m3) | Dry Bulk Density Waste Domain (LITHECODE) | Average DBD Value Assigned (t/m$^3$) |
|---|---|---|---|
| 11000 | Estimated | 910000, 910001 | 2.58 |
| 12000 | Estimated | 910002, 910003 | 2.54 |
| 13000 | 2.59 | 920000 | 2.62 |
| 13004 | 2.09 | 920001,920002 | 2.58 |
| 13006 | 2.40 | 920003 | 2.52 |
| 13010 | Estimated | 930000-930002 | 2.37 |
| 13020 | Estimated | 930004, 930005 | 2.27 |
| 13050 | 2.40 | 940000-94002 | 2.24 |
| 14006 | 2.23 | 940003 | 2.19 |
| 14008 | Estimated | 940004 | 2.14 |
| 14010 | Estimated | 940005 | 1.90 |
| 14020 | 2.23 | 950000-950003 | 2.15 |
| 15000 | 2.18 | 950004 | 2.14 |
| 16010 | 2.13 | 950005 | 1.97 |
| 17010 | 1.98 | 960000-960003 | 2.14 |
| 18000 | Estimated | 960004 | 2.13 |
| 19000 | Estimated | 960005 | 1.84 |
| 19004 | Estimated | 970000-970003 | 2.30 |
| 19006 | Estimated | 970004, 970005 | 1.98 |
| - | - | 980000 | 2.57 |
| - | - | 980001-980003 | 2.56 |
| - | - | 990000, 990001 | 2.47 |
| - | - | 990002 | 2.39 |
| - | - | 990003 | 2.28 |
| - | - | 990004 | 2.17 |
| - | - | 990060-990062 | 2.07 |
| - | - | 990063 | 2.01 |
| - | - | 990064 | 2.10 |

## 14.7 GRADE ESTIMATION

### 14.7.1 Ordinary Kriging

Ag, Pb, Zn, Cu, AgCNF, As, S, and Ca grades were estimated into the block model using Ordinary Kriging (AGPPM, PBPPM, ZNPPM, CUPPM, AGCNFINAL, ASPPM, S_%, and CA% model fields respectively) as the primary interpolation technique. A three-fold expanding search was used (in order of decreasing confidence), with geometry, ranges and number of samples used in each search pass based on the results of the variography analyses. Numbers of samples required to estimate each block were selected taking the modelled nugget (i.e., low nugget less samples, high nugget more samples; see Vann *et al*., 2003) and the change in support from 1.5m drillhole samples to 15mN by 15mE by 7.5mElevation blocks (i.e., selecting sufficient local drillhole samples to provide an average sample grade that is likely to reasonably approximate that expected for the selected block size) into account. Ag estimation parameters were iteratively refined through a series of runs to optimise the search parameters. A nearest neighbour (of Ag, Pb, and Zn) interpolation technique was used to validate the Ordinary Kriging ("OK") estimates. Semi-variogram, search ellipse and interpolation parameters for each variable were created in Gemcom, validated, and used for grade estimation.

A Nearest Neighbour ("NN") estimation technique was used for model validation purposes to assess the estimation for global bias for Ag, Pb, and Zn (AGNN, PBNN, and ZNNN model fields respectively). The NN estimation technique declusters the input drillhole data and allows for a mean grade comparison of the NN model to the OK model. Additional fields were incorporated to assess estimation quality. A summary of the various fields set up in the block model are detailed in Table 14-6.

SSR created a waste domain solid (domain 99999; <20 g/t Ag) that broadly encompasses the ultimate pit-shell to treat low-grade Ag, Pb, Zn, Cu, As, S, and Ca data outside of the mineralisation domains in order to assist with mine planning. Inverse distance weighting ("IDW") squared was selected as the estimation method for waste grades.

Preliminary model validation steps included checking for empty blocks and blocks with negative grades (in the few areas where low grade samples shielded extreme values during the kriging run). Additional runs with slightly varied estimation and search parameters were conducted to correct both cases (to estimate all blocks in the former, to reduce and/or remove the negative grades in the latter). In those few cases where negative grades persisted following several iterations, such grades were set to the minimum block grade of the relevant domain. Validation checks on the pre- and post-corrected data showed no effect on the mean grade of the domain, with the corrected (low) grade blocks spatially associated with low grade drillhole data.

### 14.7.2 Localised Uniform Conditioning

#### 14.7.2.1 Theory and Background

Non-linear techniques (e.g., Uniform Conditioning; UC) are employed in the estimation of recoverable mineralisation at a given mining selectivity; that is, a given selective mining unit, SMU (15mE by 15mN by 7.5mElevation). Whilst the UC method estimates tonnage and grade of mineralisation in small SMU blocks within larger panels, the grade of which has been determined by Ordinary Kriging (OK), the UC estimates only provide the proportions of recoverable mineralisation in each panel without specifying the actual locations of the economically extractable material. This is the major disadvantage of the UC technique, as it complicates mine planning. Abzalov (2006) developed a UC post-processing technique, LUC, which allows for the prediction of the spatial location of the recoverable mineralisation in a given panel. This method applies the appropriate grade-tonnage relationships modelled by UC to the spatial grade distribution patterns approximated by direct kriging of the SMUs from the sparse data grid (i.e., drillholes), to provide Localised SMU grade estimates.

As at the end of the 2012 drilling campaign at Pitarrilla, drillhole spacing was generally 50-60 m in plan, except in the area of the geostatistical drilling cross, in which 33 drillholes were spaced approximately 15 m apart in a large hatch pattern (Figure 14-2). Due to mining considerations, a SMU block size of 15mE by 15mN by 7.5mElevation was selected. For resource estimation purposes, the ideal block size should be at least one-third (and ideally half) of the drillhole spacing to avoid excessive conditional bias and oversmoothing of estimated grades (e.g., Vann *et al*., 2003). The selected Pitarrilla SMU is between one-third and one-quarter of the drillhole spacing, which is sub-optimal for grade estimation. This means estimated grades are susceptible to increased conditional bias due to undersampling and to insufficient treatment of the volume-variance effect (change of support) based on the selected SMU size. To address this potential outcome, an estimation technique that takes into account the change of support in order to estimate a recoverable resource should be utilised (e.g., Conditional Simulation, Uniform Conditioning, or Multiple Indicator Kriging with an Affine, Log-normal, or uniform conditioning correction). It is noted that OK does take into account the change of support through the use of discretization points within the block, but validations through a global change of support may indicate that it is insufficiently accounted for; in other words, the resource estimate can be “too” smoothed relative to the input conditioning data at the selected block size.

During initial model validation after Ag estimation by Ordinary Kriging, a global change of support (“GCOS”) was undertaken to assess for the level of smoothing of Ag grades over a range of Ag cut-off grades. The analysis showed that December 4, 2012 Ag OK-estimated grades were generally oversmoothed above Ag cut-offs of 40-50 g/t. In light of this, SSR elected to undertake a change of support and estimate a recoverable resource for Ag by using LUC. This would more appropriately represent the grade variability at the 15mE by 15mN by 7.5mElevation (SMU) block support given the current input conditioning data. SSR notes that the estimates of the other elements (Pb, Zn, etc.) by OK will also exhibit some degree of oversmoothing, but since they are minor contributors to project economics (Ag accounts for approximately 88% of payable metal), risk during Mineral Resource reporting will be mitigated at the Mineral Resource classification stage.

Fundamental to the OK process, and to any change of support technique such as UC/LUC, is the confidence in the quality of the supporting semi-variogram. Particular importance is given to the appropriate modelling of the nugget and the short range structure(s) of the semi-variogram, as they generally account for the majority of the total variance contributing to block estimates.

As mentioned above, SSR completed a geostatistical drilling cross as a part of the 2012 drilling campaign (Figure 14-2) comprising 33 drillholes. Drillholes within the cross are spaced 15 m apart in order to provide additional short-range sample pairs during continuity analysis, yielding higher confidence in the contributions of the nugget and short range structure to the total variance.

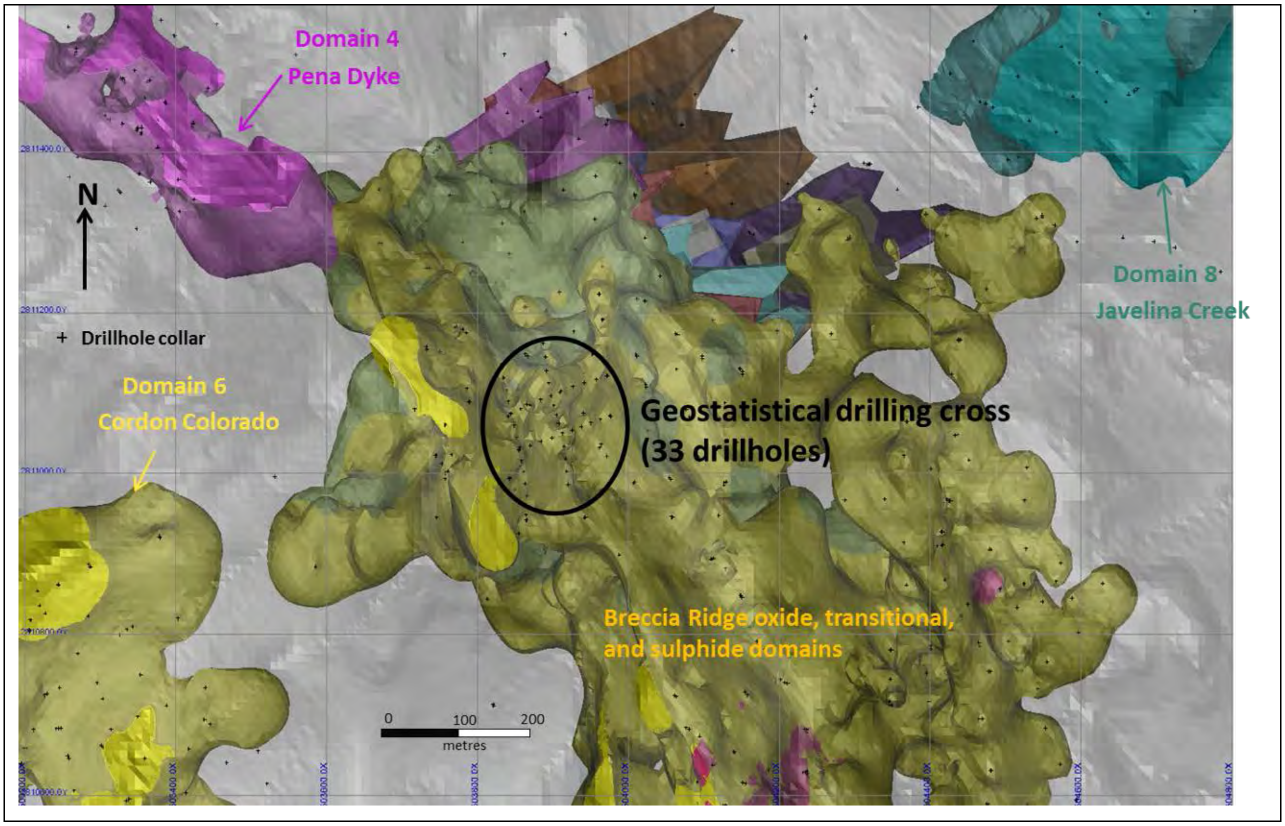



**Figure 14-2: Location of Geostatistical Drilling Cross**
Source: M3, 2012

14.7.2.2 Process Overview

14.7.2.2.1 Drillhole Preparation

Composited, domain coded drillhole data were exported from Gemcom, top cuts were applied, and then the data were imported into Geovariances Isatis (Isatis) software (version 12.00). Selection variables were created in Isatis to allocate grade data into the appropriate domains. The data were then declustered by domain; with the selection of the appropriate decluster grid size taking into account the drillhole spacing, and any evidence of clustering of drillholes in low- or high-grade areas. Edge effects were taken into account by choosing the grid size at the first plateau in the output graphs. Iterative review of the decluster cell size selection was compared to the mean grades from the OK model to ensure an appropriate cell decluster grid size was selected. Some domains did not require cell declustering.

14.7.2.2.2 Change of Support

Point anamorphoses (an anamorphosis is the process of transforming one function to another) were conducted on the decluster-weighted Ag grades for each domain to obtain theoretical grade-tonnage curves for point support data. Interactive fitting of the Hermite polynomials was used to improve the modelling of the CDF. Up to 100 Hermite polynomials were used to model the CDF. Validation included review of the CDF fit, the histogram fit, and the graphical fits of the Quantity (i.e., metal content; Q), Mean grade (M), and Tonnage (T) variables in Isatis. Grade-tonnage curves were reported above a range of cut-offs from 0-100 g/t Ag in increments of 10 g/t Ag.

Variogram parameters were transferred from Supervisor into Isatis format. Standardised variances from Supervisor were converted into real variance values for each domain, as this was the required input for the anamorphoses.

Block anamorphoses, taking the Information Effect ("IE") into account, were conducted on each domain. Each block anamorphosis used the corresponding point anamorphosis, the declustered input drillhole data, and the variogram parameters of that domain. A discretisation of 5mE by 5mN by 3mElevation was chosen for the SMU. The IE was generally taken into account using a grade control spacing 7.5mE by 7.5mN by 7.5mElevation. The kriged block variance was compared to the kriged block – real block covariance to ensure they were similar in value, with the kriged block variance being greater. Grade tonnage curves were reported above a range of cut-offs from 0-100 g/t Ag at increments of 10 g/t Ag. Validation included a comparison of the modelled CDF, histogram, and QMT variables of the blocks against the point distributions as provided in Isatis.

14.7.2.2.3 Estimation by OK into SMUs and Panels

Decluster-weighted, domain coded drillhole data was used as the input for OK estimation in Isatis. Iterative OK estimation was undertaken of the blocks to create a conditionally unbiased AGPPM variable at each level of sample support. Other estimated variables, for estimation quality assessment purposes, included standard deviation, number of neighbours, mean sample distance, weight of the mean, estimation variance, and slope of regression.

The estimation quality was deemed sufficient if the weight of the mean was generally less than 0.2 and the slope of regression was generally greater than 0.9 (i.e., conditionally unbiased estimates, which is key for the input to the UC process). Because the panel and SMU volumes extended beyond the modelled mineralisation wireframes (as was made necessary in order to have coincident panel and SMU volumes – a critical aspect of the UC/LUC process), slope of regression and the weight of mean statistics decreased in quality (e.g., the slope of regression generally ranged between 0.7-1.0 and the weight of the mean generally ranged between 0.0-0.45) in the distal areas outside of the true mineralisation wireframes. This was not considered to be an issue as was proven to be the case during model validation.

Checks of the spatial distribution were conducted on the weight of mean and slope of regression values relative to the input wireframes and drillhole data. Where poor quality kriging statistics were noted within the mineralised wireframe domain, additional estimation iterations were undertaken.

14.7.2.2.4 Uniform Conditioning of Panels

A UC change of support was undertaken on the kriged panel estimates using the block anamorphosis and the dispersion variance of the estimated panel grades. To accurately define the resulting grade tonnage curve for recoverable resource at SMU support, a selection of grade cut-offs ranging from 0 g/t Ag to the maximum grade, in increments of 10 g/t Ag were selected. Validation of the UC process was undertaken by investigating any Error Code (as defined in Isatis) occurrences that may have been generated due to a tonnage correction factor being applied during the UC process. For each domain, an Error Code of 0 was generated, indicating that no tonnage correction was applied and that all grades were consistent with the model.

14.7.2.2.5 Localised Uniform Conditioning of SMU Grades into Panels

Localised uniform conditioning post processing of UC involved assigning the recoverable resource UC grade-tonnage data at SMU support to spatial locations within the panel following the method proposed by Abzalov (2006). This is accomplished by the following steps (this is conducted in Isatis):

- Construct a set of grade-tonnage curves for each panel (using the UC results) by applying a series of cut-off grades, which are used as the input to the LUC process

- A set of grade classes are created based on the cut-off grades in each panel at the correct SMU support where the grade class is the proportion of the panel whose grade is higher than a given cut-off, but lower than the next defined cut-off

- Rank the directly kriged small block grades (from smallest to largest)

- Mean grades of the grade classes, as deduced from the UC model, are assigned to the SMU blocks whose rank (based on the direct OK estimates) matches the grade class

14.7.2.2.6 Incorporation of LUC Estimates into the Final Gemcom Model

The block centroid coordinates, LUC estimated Ag grade, slope of regression, and weight of the mean attributes were exported by domain from Isatis and imported into Gemcom. Since the block centroids in each domain covered a larger area, care was taken to ensure that only blocks within the mineralisation domain solids were assigned with the LUC estimated Ag grades, and the slope of regression and weight of the mean statistics during the importation process.

14.7.2.2.7 Validation of LUC Estimates

Standard validation checks for the OK estimates of the SMUs and panels were made including visual inspection of grades and mean grade comparison to declustered input data. All results showed that the OK estimates were honouring the input drillhole data. Weight of mean and slope of regression estimation quality variables, as imported from Isatis, were viewed on a domain basis and were generally found to be less than 0.2 for weight of the mean and greater than 0.9 for slope of regression, which were of sufficient quality to verify that the OK estimates (upon which the LUC estimates were based) demonstrated minimal conditional bias.

Uniform Conditioning relies on the assumption that the grade data follow a multi-Gaussian distribution. In practice, a check that it does not violate the condition of bivariate Gaussianity is performed. There are several methods available to test this assumption, but the method chosen compared the madogram (similar to a semi-variogram, but the difference between sample values is not squared) to the square root of the average variogram over a series of lag distances. For the bivariate Gaussianity assumption to be valid, their ratio should equal the square root of pi over the selected lag distances. Checks were conducted using a lag distance of 45-60 m along the major and semi-major axes and at 1.5-6.0 m along the minor axis of the semi-variogram for each domain. Based on this review, bivariate Gaussianity was considered to be a valid assumption for all domains.

SSR conducted two additional tests to gauge the quality of the local estimates using the close-spaced drilling information gathered from the geostatistical drilling cross within the BR-AB_OX domain (10) (Figure 14-2). For these tests the geostatistical cross drillholes were removed, thereby creating a wide-spaced conditioning data grid (approximately 50-60 m drillhole spacing) to serve as a proxy for the other domains in the deposit. The results of the tests were then compared to the December 4, 2012 model containing the close-spaced drilling data.

First, SSR checked the impact of the modelled nugget and short-range structures of the semi-variograms between the wide- and close-spaced drilling data. The modelled semi-variograms exhibited identical standardised nugget values of 0.06 and exhibited very similar short-range characteristics. Modelled ranges were also comparable, and thus provided a strong indication the semi-variograms utilised for the LUC estimations in the domains with drillhole spacing of 50-60 m were appropriate.

A perimeter 25 m wider than the extents of the drilling cross was created to undertake the second LUC validation test. It compared the reported tonnage and grade of material through estimation by LUC using both the wide- and close-spaced conditioning data (Table 14-8).

**Table 14-8: Comparison of LUC Estimates Using Wide- and Close-Spaced Conditioning Data Grids**

| Cut-off (g/t Ag) | Wide-spaced* Tonnage (Mt) | Close-spaced* Tonnage (Mt) | Wide-spaced* Grade (Ag g/t) | Close-spaced* Grade (Ag g/t) | Difference Tonnes (%) | Difference Ag grade (%) |
|---|---|---|---|---|---|---|
| 20 | 8.81 | 8.53 | 57.60 | 64.42 | -3.3 | 10.6 |
| 30 | 8.27 | 7.66 | 59.70 | 68.91 | -8.0 | 13.4 |
| 40 | 6.57 | 6.26 | 66.11 | 76.53 | -4.9 | 13.6 |
| 50 | 4.60 | 4.74 | 75.29 | 86.79 | 3.0 | 13.3 |
| 60 | 2.06 | 3.50 | 85.62 | 98.29 | 12.4 | 12.9 |

*Note: Wide-spaced grid does not contain the drillholes from the geostatistical cross, whereas the close-spaced grid does.

The comparison shows that the estimated tonnage and grade are within 15% of each other between cut-off grades of 20-60 g/t Ag, indicating the LUC technique is providing an adequate estimation of tonnage and grade supported by wide-spaced drilling data (i.e., 50-60 m, the average drillhole spacing, as at December 4, 2012).

It is critical to note that LUC should not be confused as a technique that provides an accurate prediction of tonnage and grade on a block by block basis (Abzalov, 2006). It was not designed to do this. Techniques such as LUC take into account the change of support and generally provide a better estimate of the recoverable resource in comparison to other estimation techniques (e.g., inverse distance weighting, nearest neighbour, ordinary kriging) when the estimating into a block that is smaller than one half the size of the drillhole spacing grid. SSR stresses that "good" local estimates can only be achieved through close-spaced drilling (e.g., grade-control drilling) whose grid pattern is based on the SMU. The results of the validation tests comparing the semi-variograms and the estimated tonnage and grade using both close- (20 m) and wide-spaced (50-60 m) drilling indicate that the LUC technique employed for the December 4, 2012 resource is of sufficient quality to estimate recoverable tonnage and grade over an annual production timeframe, and thus supports the Indicated Mineral Resource classification assigned to this material (Section 14.9).

## 14.8 MODEL VALIDATION

The December 4, 2012 Pitarrilla Mineral Resource model was validated by SSR in the following way:

- Visual comparison of block grades for the various variables against the input top cut, composite data for each domain on a series of 25 m spaced cross-sections and level plans through the model. Input grades were honoured by the block estimates with the expected level of smoothing observed in the ordinary kriged estimates (Ag, Pb, and Zn), and the expected relative reduction in smoothing for the LUC estimate (Ag).

- Comparison of global and domain average grades for the various components in the ordinary kriged model to the cell-declustered mean input grade data. In addition, the nearest neighbour models, which also provide a reasonable approximation of the declustered mean grade, were compared against the respective OK and LUC models to assess estimation bias. All comparisons were favourable and did not indicate the presence of global grade bias.

- Comparison of the domain mean grades for the various components in the model to those in the relevant nearest neighbour model and the declustered, top cut, composited input drillhole file along northing, easting, and elevation swath plots to assess potential spatial bias in the model. The results showed that ordinary kriged model trends followed the declustered input trends and the Nearest Neighbour trends with expected amounts of smoothing. Swath plots comparing LUC estimated grades to the OK estimated grades were favourable. The LUC model trends showed less smoothing (higher selectivity) than the OK model trends, as expected, and showed no evidence of spatial bias.

- Grade-tonnage curves were generated for the point and block anamorphoses, the OK and NN estimates, and the LUC estimate, and compared and evaluated for expected amounts of smoothing/selectivity. Factoring of the point and block anamorphosis grade-tonnage curves was undertaken to give all grade-tonnage curves of a given domain a common grade above a zero cut-off to facilitate a comparison between curves. As the grade above a zero cut-off is highly sensitive to the decluster grid size chosen, not factoring these curves would have a significant impact on their vertical locations in grade-tonnage space. This factoring does not affect the metal and tonnage proportions above a cut-off, nor does it affect the shape of the curve. Silver Standard tested this using the data from Domain 60 and found that the curve did not change shape, nor did the cut-off grades change location along the curve.

- Grade-tonnage data were reported using several software packages to ensure no errors in the reporting stage.

Based on the results of the detailed model validation described above, and the numerous validation checks conducted during the drillhole and model preparation steps, SSR believes that the modelled grades by LUC and OK estimation techniques honour the input drillhole data and provide a reasonable approximation of the geology and mineralisation of the Pitarrilla deposit, as presently understood within the limitations of the current drillhole spacing.

### 14.9 MINERAL RESOURCE CLASSIFICATION

Model classification uses standard terminology as defined by CIM (2010). A Mineral Resource is a concentration or occurrence of (in this case) base and precious metals in or on the Earth's crust in such a form and quantity and of such a grade and 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 (adapted from CIM, 2010).

- A **Measured Mineral Resource** (CIM, 2010) is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and 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 drillholes that are spaced closely enough to confirm both geological and grade continuity. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit. Measured Resources can be converted, considering appropriate modifying parameters (e.g., mining, metallurgical, economic, marketing, legal, environmental, social, and governmental factors), to Proven Mineral Reserves and, in cases of lower confidence in some or all of the modifying factors, to Probable Mineral Reserves as per CIM Definition Standards (2010).

- An **Indicated Mineral Resource** (CIM, 2010) 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 drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed. An Indicated Mineral Resource is of sufficient quality to support studies forming the basis for major development decisions, and can be converted to Probable Mineral Reserve as per CIM Definition Standards (CIM, 2010), taking the abovementioned modifying factors into account.

- An **Inferred Mineral Resource** (CIM, 2010) 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 drillholes. An Inferred Mineral Resource must be excluded from estimates forming the basis of feasibility or other economic studies as confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure.

Mineral Resources classified as Measured, Indicated, and Inferred are defined as codes 1, 2, and 3 respectively in the model field, CLASS. SSR took into account the following factors to derive the resource classification:

- Quality of the grade data and bulk density samples from analysis of QC samples (Section 11). The overall quality of the QAQC programs utilised by Silver Standard concluded that the data were of sufficient quality for use in Mineral Resource estimation.

- Confidence in the geological interpretation and domain interpretation wireframes, where grade continuity between drillholes and drillhole sections can be reasonably inferred and followed. All material inside the mineralisation wireframes is considered to be at least Inferred in confidence (as the boundaries of these domains were defined by the drillhole data).

- The number and spatial representivity of dry bulk density data.

- Semi-variogram quality, which takes into account the robustness of the modelled semi-variogram across a variety of lag distances, and the confidence in the nugget and the short range structure.

- Estimation statistics including kriging variance, slope of regression, weight of the mean, and number of samples used to inform a block. For Measured material, slope of regression and weight of the mean values greater than 0.9 and less than 0.2 respectively are used. For Indicated material, the slope of regression must generally be greater than 0.8 and the weight of the mean less than 0.3.

- Average distance to the nearest sample for Measured material is set at approximately 40-45 m for Measured material and 60-65 m for Indicated material based on the semi-variogram.

A Measured classification of Ag grades has been retained based on precedence set in the August 2008 resource model (P&E, 2008). The material is located in the deeper sulphide domains, BR_E, BR_F, and BR_G (70-90), and although drilling information is relatively dense in this area (average distance of sample points used to estimate the blocks within the Measured zone was approximately 50 m), an average drillhole spacing of approximately 40-45 m would be more appropriate based on modelled semi-variograms. This indicates these domains remain undersampled. A risk analysis (e.g., conditional simulation study) should be conducted on these blocks to assess whether the error is within 15% on a quarterly production volume basis to support the Measured classification. The slope of regression and weight of the mean of this material were in the range of 0.8 and 0.3 respectively, indicating that the quality of the kriged estimates in the Measured zone is reasonably good. Since this material will not be reached during the initial years of mining, there is minimal risk to leaving this material classified as Measured. Additional drilling can be undertaken in the future to increase confidence in these blocks before they are extracted by mining.

The LUC estimation technique takes into account the volume-variance effect (change of support) from point support (drillhole sample volume) to the SMU block support (15mE by 15mN by 7.

Elevation block volume). This has provided additional certainty in the quality of the estimated Ag values above higher Ag cut-off grades. Estimates of Ag, Pb, and Zn through OK also take into account the volume-variance effect, but validation tests indicate oversmoothing of grades above higher Ag cut-offs, most likely due to the relationship between the SMU block size and the spacing of the conditioning drillhole data. Grade estimation of Pb and Zn by LUC has not been undertaken due to their significantly smaller contributions to project economics (Ag, Pb, and Zn account respectively for approximately 88%, 3%, and 9% of payable metal revenues at Pitarrilla) and due to the time intensive nature of the estimation technique. To account for added uncertainty due to the volume-variance effect when reporting elements estimated by different techniques (Ag by LUC and Pb and Zn by OK), Pb and Zn Mineral Resources previously classified as Measured (P&E, 2008), have been reallocated to the Indicated category. In addition, the correlation between Ag and Pb, which is strong in the domains comprising the Measured region of the deposit (domains 70-90), is affected when estimating these elements using different techniques. This added risk is mitigated through the reallocation of the Pb and Zn Measured resources to the Indicated category.

Blocks informed by at least ten samples (i.e., those in the first two search volumes) were found to have sufficient estimation quality (based on Ag kriging variance, and slope of regression considerations, and visual comparisons to the drillhole data) to be classified as at least Indicated. An effort was made to assign an Indicated classification only those blocks satisfying the above requirements that formed a coherent group that could be used in a mine plan. This is to help prevent discontinuous, isolated blocks of Indicated material around single drillholes.

Additional dry bulk density sampling is required to estimate resource blocks situated within the geostatistical drilling cross, which otherwise have sufficient assay data to be moved from the Indicated to the Measured category. Once completed SSR estimates this would convert approximately 10 million tonnes of material with an average grade of 72 g/t Ag, 0.37% Pb, and 0.68% Zn above a 30 g/t Ag cut-off from Indicated to Measured. At the time of writing, SSR is undertaking the additional dry bulk density sampling.

### 14.10 INDEPENDENT REVIEW CONDUCTED BY XSTRACT MINING CONSULTANTS

Xstract Mining Consultants Pty Ltd ("Xstract") was engaged by Silver Standard to carry out a high-level critical review of the resource evaluation process associated with the December 4, 2012 update of the Pitarrilla Mineral Resource. The high-level review examined the domaining (geological/geochemical controls) and estimation techniques used to establish the Mineral Resource. Silver Standard did not require of Xstract to conduct any verification, validation and/or review of the sampling techniques, sample security, data collection, sample recovery, sample analytical techniques, sample quality assurance/quality control, and/or database management associated with the Pitarrilla Project.

Xstract carried out a visit to the Pitarrilla Project site in June 2012. The site visit included a review of the local geology and drillcore within the area planned for the first five years of production. Xstract confirmed that the current exploration practices observed during the site visit are adequate for Mineral Resource evaluation purposes.

Xtract determined that the Mineral Resource methodology carried out by Silver Standard to establish the reported Mineral Resource is technically robust. The high-level review by Xstract did not encounter any critical issues that would affect the reporting of the Mineral Resource or the application of the Mineral Resource for open pit mining engineering studies.

Xstract provided several recommendations to improve the development of future resource estimates and reporting, however these items are not considered by Xstract to be critical to the December 4, 2012 Mineral Resource.

### 14.11 MINERAL RESOURCE REPORTING

Global Mineral Resource sensitivities over a range of Ag cut-offs for Measured and Indicated Inferred resources are tabulated in Table 14-9. Table 14-10 summarises global Mineral Resources reported by mineralisation type, as Oxide, Transitional, or Sulphide above a 30 g/t Ag cut-off grade. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

**Table 14-9: Pitarrilla December 4, 2012 Global Measured and Indicated Mineral Resource Sensitivity**

| Classification | Cut-off Ag (g/t) | Tonnes (Mt) | Ag (g/t) | Pb (%) | Zn (%) | Ag (Moz) | Pb (Mlbs) | Zn (Mlbs) |
|---|---|---|---|---|---|---|---|---|
| Measured | 20.00 | 23.62 | 85.56 | - | - | 65 | - | - |
| | **30.00** | **20.31** | **95.42** | **-** | **-** | **62** | **-** | **-** |
| | 40.00 | 16.90 | 107.62 | - | - | 58 | - | - |
| Indicated | 20.00 | 268.73 | 75.89 | - | - | 656 | - | - |
| | **30.00** | **240.00** | **81.94** | **-** | **-** | **632** | **-** | **-** |
| | 40.00 | 199.61 | 91.41 | - | - | 587 | - | - |
| | 20.00 | 292.35 | - | 0.31 | 0.71 | - | 2,009 | 4,581 |
| | **30.00** | **260.31** | **-** | **0.32** | **0.72** | **-** | **1,815** | **4,146** |
| | 40.00 | 216.51 | - | 0.33 | 0.75 | - | 1,574 | 3,590 |
| Measured + Indicated | 20.00 | 292.35 | 76.67 | 0.31 | 0.71 | 721 | 2,009 | 4,581 |
| | **30.00** | **260.31** | **82.99** | **0.32** | **0.72** | **695** | **1,815** | **4,146** |
| | 40.00 | 216.51 | 92.68 | 0.33 | 0.75 | 645 | 1,574 | 3,590 |
| Inferred | 20.00 | 26.48 | 55.98 | 0.21 | 0.48 | 48 | 123 | 281 |
| | **30.00** | **22.08** | **62.12** | **0.21** | **0.49** | **44** | **101** | **236** |
| | 40.00 | 17.09 | 70.00 | 0.21 | 0.49 | 38 | 79 | 186 |

Notes:

1. Jeremy D. Vincent, B.Sc. (Hons), P.Geo., is the Qualified Person for the reported Mineral Resources estimate.
2. All Mineral Resource estimates have been classified in accordance with current Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definition standards.
3. Ag was estimated using Localised Uniform Conditioning (LUC). Pb and Zn were estimated using Ordinary Kriging (OK).
4. Mineral Resource estimates of Pb and Zn are not classified as Measured to account for the added uncertainty introduced by the volume-variance effect when using different estimation techniques (Ag by LUC; Pb and Zn by OK).
5. A silver cut-off grade of 30 g/t Ag is considered at this time to be the most likely economic cut-off grade for large-scale open-pit mining of the Pitarrilla deposit.
6. The reported Measured and Indicated Mineral Resources are regarded as sufficient for medium to long term production open pit planning and mine scheduling on a quarterly basis. Grade control drilling and a mine blending strategy to control grade variations are recommended for short-term mine planning.
7. Mineral Resources situated below the current open-pit shell design are considered potentially economically viable in an underground mining scenario, and are therefore included in the total reported Pitarrilla Mineral Resources. A Preliminary Economic Assessment (PEA) or higher level study validating the economics of the underground mining scenario has not been undertaken at this time.
8. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
9. The reported tonnes, grade, and metal content may not tally precisely due to rounding.

**Table 14-10: Pitarrilla December 4, 2012 Global Measured and Indicated Mineral Resource by Mineralisation Style**

| Material Type | Classification | Ag Cut-off (g/t) | Tonnes (Mt) | Ag (g/t) | Pb (%) | Zn (%) | Ag (Moz) | Pb (Mlbs) | Zn (Mlbs) |
|---|---|---|---|---|---|---|---|---|---|
| Oxide | Measured | 30.00 | - | - | - | - | - | - | - |
| | Indicated | 30.00 | 118.19 | 80.45 | 0.10 | 0.34 | 306 | 268 | 891 |
| | **Measured + Indicated** | **30.00** | **118.19** | **80.45** | **0.10** | **0.34** | **306** | **268** | **891** |
| | Inferred | 30.00 | 12.97 | 59.96 | 0.06 | 0.19 | 25 | 17 | 56 |
| Transitional | Measured | 30.00 | - | - | - | - | - | - | - |
| | Indicated | 30.00 | 57.57 | 74.13 | 0.28 | 0.60 | 137 | 351 | 763 |
| | **Measured + Indicated** | **30.00** | **57.57** | **74.13** | **0.28** | **0.60** | **137** | **351** | **763** |
| | Inferred | 30.00 | 4.92 | 67.28 | 0.15 | 0.60 | 11 | 16 | 65 |
| Sulphide | Measured | 30.00 | 20.31 | 95.42 | - | - | 62 | - | - |
| | Indicated | 30.00 | 64.24 | 91.68 | - | - | 189 | - | - |
| | Indicated | 30.00 | 84.55 | - | 0.64 | 1.34 | - | 1,196 | 2,492 |
| | **Measured + Indicated** | **30.00** | **84.55** | **92.58** | **0.64** | **1.34** | **252** | **1,196** | **2,492** |
| | Inferred | 30.00 | 4.19 | 62.73 | 0.73 | 1.25 | 8 | 67 | 116 |

Notes:
1. Jeremy D. Vincent, B.Sc. (Hons), P.Geo., is the Qualified Person for the reported Mineral Resources estimate.
2. All Mineral Resource estimates have been classified in accordance with current Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definition standards.
3. Ag was estimated using Localised Uniform Conditioning (LUC). Pb and Zn were estimated using Ordinary Kriging (OK).
4. Mineral Resource estimates of Pb and Zn are not classified as Measured to account for the added uncertainty introduced by the volume-variance effect when using different estimation techniques (Ag by LUC; Pb and Zn by OK).
5. A silver cut-off grade of 30 g/t Ag is considered at this time to be the most likely economic cut-off grade for large-scale open-pit mining of the Pitarrilla deposit.
6. The reported Measured and Indicated Mineral Resources are regarded as sufficient for medium to long term production open pit planning and mine scheduling on a quarterly basis. Grade control drilling and a mine blending strategy to control grade variations are recommended for short-term mine planning.
7. Mineral Resources situated below the current open-pit shell design are considered potentially economically viable in an underground mining scenario, and are therefore included in the total reported Pitarrilla Mineral Resources. A Preliminary Economic Assessment (PEA) or higher level study validating the economics of the underground mining scenario has not been undertaken at this time.
8. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
9. The reported tonnes, grade, and metal content may not tally precisely due to rounding.

## 14.12 DISCUSSION OF MATERIAL EFFECTS ON THE MINERAL RESOURCE

Silver Standard is unaware of any current environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the December 4, 2012 Mineral Resource estimate presented in Section 14.

# 15 MINERAL RESERVE ESTIMATE

## 15.1 SUMMARY

In June 2009, SSR announced the results of a Pre-Feasibility Study for the underground component of Breccia Ridge completed by Wardrop. The Wardrop Mineral Reserve estimate contained probable silver reserves of 91.7 Moz, using a silver price of US$11.00/oz, a zinc price of US$0.70/lb, a lead price of US$0.50/lb and a US$50.00/tonne net smelter returns (NSR) cut-off. As presented in the Pre-feasibility Study (news release dated June 24, 2009), the underground project envisaged had a 12-year mine life (plus two years of pre-production), mining 4,000 tpd via a combination of long-hole and room-and-pillar mining methods, and producing approximately seven million ounces of silver per year. Capital costs were projected at US$277 million with average operating costs of US$33.81/tonne. The reserve was based on a $50/tonne NSR. Ore extraction was planned at 89% for room-and-pillar extraction and both mining methods assumed 95% stope recovery. Metallurgical recoveries via a conventional two-product flotation plant were considered to vary by rock type. The expected weighted average recoveries were 88.4% for silver, 93.2% for zinc and 89.6% for lead.

**Table 15-1: Pitarrilla Mineral Reserve Estimate, June 2009**

| Reserve Category | Tonnage (Mt) | Mined Grade | | | Contained Metal |
|---|---|---|---|---|---|
| | | Ag (g/t) | Pb (%) | Zn (%) | Ag (Mozs) |
| Probable | 16.7 | 171.0 | 1.120 | 2.570 | 91.7 |

SSR subsequently investigated the underground component of the Breccia Ridge Zone using a combination of sub-level caving and open-hole stope extraction. The results of this study were not announced as the project evolved to examine the opportunity of a staged construction with the mining of the near-surface Peña Dyke, Cordon Colorado, and South Ridge oxidised silver deposits ahead of an underground development.

When the shallow resources were examined in more detail, the value of the transitional ore was highlighted and the metallurgical analysis of this ore type was advanced. As the shallower ore increased in value, a large open pit extraction method was selected in preference to either an underground or combined open pit/underground mining method.

Other major studies were conducted in support of the 2012 open cut extraction method analysis. These include but are not limited to the following:

- Close-spaced drilling information was obtained to qualify the SMU geostatistical control parameters;
- Geomechanical drilling and analysis;
- Geotechnical drilling and test pits were completed to enhance understanding of the mechanical performance of waste dumps; the waste dump locations were optimised for cost and available location;

- Acid-based accounting and metal leaching testwork has been completed on drillcore to help guide a waste dump construction plan;
- A new resource model was developed;
- A pit design (and sub-phasing) was completed on the new resource using cost and pricing knowledge to feasibility level understanding;
- A mine plan was developed from a range of alternatives that results in the best return on capital invested; and
- Metal prices were altered from the 2009 basis and marketing studies of the concentrates and doré were completed.

The 2012 Mineral Reserve replaces the 2009 Mineral Reserve. The 2012 Mineral Reserve does not preclude future underground development, but the 2009 Mineral Reserve cannot in any way be considered additional to the 2012 Mineral Reserve.

SSR has not previously conducted mining of ore on the Property.

The Mineral Reserve estimate for the Pitarrilla Project was calculated using the following assumptions and parameters:

- The reserve classification converts Measured and Indicated Mineral Resources to Probable Mineral Reserves within the pit design;
- Inferred Mineral Resources occur within the design, but these were given no value, and their existence does not in any way influence the design developed;
- The mining recovery was taken as 100% within the pit design;
- The Mineral Resources were not diluted beyond the selection of the SMU;
- The Mineral Reserve assumes that mining uses open cut mining methods as described in this Technical Report;
- The cut-off grade assigned was \$16.38/t NSR for direct leach ore and \$16.40/t for flotation/leach ore and is detailed in Table 15-2 and Table 15-3; and
- The NSR value uses non-linear grade-weathering-recovery relationships outlined in Section 15.3.1.

## 15.2 MINERAL RESOURCE BASIS

The starting point of calculations for the Mineral Reserve utilised for the Pitarrilla Feasibility Study (M3, 2012) was the Feasibility Mineral Resource model effective December 4, 2012. The tonnages and grades within the model were checked against reports of the resource estimate prior to commencing optimization and mine design.

The Mineral Reserve statement includes no provision of mineable recoverability of Mineral Resources below the designed pit.

## 15.3 CUT-OFF GRADE

The Mineral reserve statement is based on a Value per Tonne ("VPT") evaluation. Table 15-2 and Table 15-3 summarise the cost parameters used in the cut-off grade calculations for both direct leach and flotation/leach processes. The cut-off value for direct leach ore was $16.38/tonne and for flotation/leach ore was $16.40/tonne. The waste mining cost differential between ore and waste is estimated at -$0.04/tonne; that is, waste mining is slightly more expensive than ore mining; this is mostly due to the inclusion of limestone mining for ARD mitigation.

**Table 15-2: Direct Leach NSR**

| Concept | Plant Cost ($/t) |
|---|---|
| Crushing, grinding, tailings pumping and overheads | 6.94 |
| Leaching, Merrill-Crowe and detox | 7.55 |
| Administration | 1.42 |
| Re-handle | 0.01 |
| Tailings disposal | 0.50 |
| Ore mining differential | -0.04 |
| Total | 16.38 |

**Table 15-3: Flotation/Leach NSR**

| Concept | Plant Cost ($/t) |
|---|---|
| Crushing, grinding, tails pumping & overheads | 6.94 |
| Lead flotation | 1.67 |
| Zinc flotation | 1.43 |
| Sulphide tailings leach, Merrill-Crowe & detox | 4.83 |
| Administration | 1.06 |
| Re-handle | 0.01 |
| Tailings disposal | 0.50 |
| Ore mining differential | -0.04 |
| Total | 16.40 |

## 15.4 NSR GRADE CALCULATION

Metallurgical testwork has demonstrated that lithology plays no part in flotation metallurgical recoveries. Instead, a combination of oxidation and head grade controls flotation recoveries. Additionally, the concentrate grade produced varies with in-situ grade. To account for these complexities, a FORTRAN subroutine was developed within Minesight reserve programming to calculate NSR grade value in US$ per tonne for each block in the block model. This methodology was rigorously checked with individual blocks and compared to spreadsheet analysis of single data points and groups of data. The NSR methodology followed the steps listed below:

- Rougher recoveries for both lead and zinc concentrates were applied based on recovery curves that varied with metal grades for Ag, Pb, Zn and Cu;

- Rougher recoveries were trimmed to a maximum or minimum value;
- An oxidation effect factor was determined for Ag, Pb, Zn and Cu for both concentrates;
- Cleaner recovery factors were estimated;
- Final recoveries were a multiplication of the Rougher Recovery x Oxide Factor x Cleaner Factor;
- Concentrate mass pull was estimated;
- Lead concentrate grades were calculated based on the recovery of lead, the lead head grade, the oxidation effect, and the individual metal grades;
- Gross concentrate values were calculated using the project metal prices;
- Smelting and refining charges were applied to each of the two concentrates;
- Treatment terms and price participation costs were applied;
- Penalty element charges for each concentrate were applied;
- Transportation related costs were calculated;
- Costs to produce each concentrate were estimated;
- The tailings grade of the flotation concentrates was estimated;
- The recovery of the tailings via leaching was estimated as the lesser of 76% of the direct leach estimated recovery or 35%;
- The cost to leach the tailings was added;
- The NSR values of the individual blocks were calculated via flotation and leaching of the tailings;
- The cost to direct leach the ore without flotation was estimated;
- The value of doré from direct leach only was estimated;
- The NSR values of the individual blocks were calculated via direct leach;
- The NSR of the two alternatives was compared and then the process of highest value was assigned to the block; and
- The highest value process was flagged to the block and used to accumulate the reserve type as either flotation with leaching the tailings or as direct leach.

The recovery parameters used for the Mineral Reserve are fully described in Section 16.5.4.4.

In summary, the NSR calculation method varies for the two ore types. For the two ore types combined, the overall average process recovery within the overall Mineral Reserves of silver, lead and zinc are 69.6%, 57.4% and 61.3% respectively.

For direct leach ore, the NSR is estimated from the silver head grade, the cyanide leach silver recovery and the applicable doré sales costs and refining costs. The cyanide leach silver recovery is directly estimated in the model from assays and metallurgical testing. The average direct leach process recovery for silver, lead and zinc is 53.7%, 0% and 0% respectively.

For flotation/leach ore, the NSR is estimated based on recoveries that vary by head grade for Ag, Pb and Zn and are also reduced in performance depending on the amount of oxidation present. Concentrate grades also vary by oxidation and head grades. NSR estimates are inclusive of transport costs, penalties and refinery charges. The NSR of this ore type is augmented by the addition of cyanide leach of the flotation tail net of leach process costs and doré sales and refining costs. The average flotation/leach process recovery for silver, lead and zinc is 74.8%, 68.3% and 72.0% respectively.

## 15.5 METAL PRICES

The metal prices used to calculate the NSR grade values are shown in Table 15-4.

**Table 15-4: NSR Grade Metal Prices used in the Reserve**

| Metal | Price |
|---|---|
| Silver (US$/oz) | $25.00 |
| Zinc (US$/lb) | $0.95 |
| Lead (US$/lb) | $0.90 |

Concentrate metal cost recommendations were supplied in a report prepared by Base Metals Marketing Services Ltd. ("BMMS") in 2012; the values used in the NSR calculation are included in Table 15-5.

**Table 15-5: Reserve Metal Costs**

| | Doré | Lead Concentrate ($/dmt) | Zinc Concentrate ($/dmt) |
|---|---|---|---|
| Payable Base Metal | | 95% (3% min) | 85% (8% min) |
| Payable Silver | 98.5% | 95% (50 g/t min) | 75% (109 g/t min) |
| Treatment | Incl. in RC | $291.04 | $277.70 |
| Silver Refining Charge | $0.60/oz | $1.25/oz (5% of price) | $0.75/oz |
| Penalties | none | $15.00 | $15.00 |
| Shipping (land and sea) | Incl. in RC | $132.00 | $132.00 |

## 15.6 DILUTION

No mining dilution was applied to the grade of the blocks. It was considered that the SMU of 15m x 15m x 7.5m was sufficient to define the mineable characteristics of the ore using the equipment selected. The SMU was selected such that it was larger than the minimum SMU attributable to the size of the equipment proposed in order to accommodate all envisioned mining dilution.

## 15.7 MINING RECOVERY

Mining recovery was taken to be 100% of the Measured and Indicated Resources. Inferred Resource was assigned as waste.

## 15.8 MINE DESIGN SURFACES USED

The reserve is quoted within the natural topography and the ultimate mine design BR4_20121004, which is described in Section 16.3.5.

## 15.9 MINERAL RESERVE ESTIMATE

According to the Definitions Standards on Mineral Resources and Reserves adopted by the CIM Council on November 27, 2010 and incorporated into NI 43-101, the definitions of Proven Mineral Reserves and Probable Mineral Reserves are outlined in Section 15.3.7 and 15.3.8, respectively.

### 15.9.1 Proven Mineral Reserve

"A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. The study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified." (CIM, 2010).

The Mineral Reserves quoted in this Technical Report adhere to these standards. Only Measured and Indicated Mineral Resources have been used to establish the Probable Mineral Reserves. Inferred Resources were considered to be waste in this Technical Report.

There is no stated Proven Mineral Reserve within the pit design.

### 15.9.2 Probable Mineral Reserve

"A 'Probable Mineral Reserve' is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. The study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified." (CIM, 2010)

The Mineral Reserve as prepared for this Technical Report is considered 100% probable.

## 15.10 MINERAL RESERVE STATEMENT

The Mineral Reserve estimate is summarised in Table 15-6.

**Table 15-6: Pitarrilla Mineral Reserve Estimate (Effective as of December 4, 2012)**

| Category | Process Type | Tonnage | Mined Grade | | | Contained Metal | | |
|---|---|---|---|---|---|---|---|---|
| | | (Mt) | Ag (g/t) | Pb (%) | Zn (%) | Ag (Mozs) | Pb (Mlbs) | Zn (Mlbs) |
| Probable | Direct Leach | 43.4 | 91.5 | 0.17 | 0.42 | 127.5 | 161.6 | 403.9 |
| Probable | Flotation/Leach | 113.2 | 96.5 | 0.34 | 0.93 | 351.2 | 851.8 | 2,317.7 |
| Total Probable | All | 156.6 | 95.1 | 0.29 | 0.79 | 478.7 | 1,013.5 | 2,721.5 |

Notes to Mineral Reserves Table 15-6:

1. Mineral Reserves are contained within Measured and Indicated pit designs using metal prices for silver, lead and zinc of US$25/oz, US$0.90/lb, and US$0.95/lb, respectively.
2. The pit designs are generated from appropriate mining costs, processing costs, metal recoveries and inter ramp pit slope angles (varying from 36° to 48°)
3. The Mineral Reserve uses a net smelter return (NSR) calculation to determine the cut-off. The Mineral Reserve contains two ore types: direct leach ore and flotation/leach ore. The constant cut-off value for direct leach ore is $16.38 /tonne and for flotation/leach ore is $16.40/tonne.
4. Average process recovery within the total Mineral Reserves of silver, lead and zinc are 69.6%, 57.4% and 61.3% respectively.
5. No mining dilution is applied to the grade of the resource. Dilution intrinsic to the resource model is considered sufficient to represent the mining selectivity considered.
6. The life of mine strip ratio is 5.96.
7. Tonnage and grade measurements are in metric units. Contained silver ounces are reported as millions of troy ounces (Mozs). Contained lead and zinc are reported as millions of imperial pounds (Mlbs).
8. The reserve is 100% in-situ; no mining of the ore has occurred.
9. Table may not sum due to rounding.

## 15.11 COMMENT ON MINERAL RESERVE

The Pitarrilla Project is not an operating entity and as such all infrastructure projects are yet to be constructed. The Qualified Person for this section is of the opinion that the Mineral Reserves for the Project have been prepared to industry best practices and conforms to the requirements of CIM (2010). The Mineral Reserves are adequate to support mine planning.

Mineral Reserve by definition has taken into account environmental, permitting, legal, title, taxation, mining, metallurgical, infrastructure, socio-economic, marketing and political factors and other constraints, as discussed in various Sections (4, 13, 14, 16, 17, 18 and 20) of this Technical Report.

The results of the economic analysis to support Mineral Reserves represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Areas of uncertainty that may materially impact mineral reserve estimation include but are not limited to:

- Commodity price and exchange rate assumptions;
- Capital and operating cost estimates; and
- Geotechnical slope designs for pit walls.

# 16 MINING METHODS

SSR completed most of the mine planning activities for the Pitarrilla Project open cut feasibility assessment.

SSR also retained the services of Independent Mining Consultants Tucson ("IMC") to verify the process undertaken by SSR by making independent process calculations of each major step of the estimation process.

Knight Piésold Limited ("KPL") was retained to complete the geomechanical study, to investigate pit wall stability, to supply pit wall design angles and pit design criteria. KPL was also retained to complete a waste dump geotechnical analysis including acid rock drainage estimation.

## 16.1 MINING BACKGROUND

SSR has assessed the Pitarrilla Project using various mining methods and sizes of operation to various levels of confidence of results. Not limited to the following are some of the alternatives considered:

- Small open pit extraction of oxide reserves
- Block caving
- Sub-level caving and open stoping combined
- Open stoping
- Combination of small oxide pits and open stoping
- Large scale open cut extraction of oxide, transitional and sulphide mineralisation

In June 2009, SSR announced the results of a Pre-feasibility Study for the underground component of Breccia Ridge completed by Wardrop (2009). The new November 2012 Reserve replaces the 2009 Reserve that was included in the 2009 underground pre-feasibility report.

It has been the advent of resolution of combination recovery methods of leaching and flotation as applied to the transitional ore that has resulted in focussing best economic results to a large open cut scenario that gains value from the upper ore profile, as well as the deeper richer ores. This mining method of extraction is the one discussed in this report, which summarises the open cut Pitarrilla Feasibility Study (M3, 2012). The results of the mining methods evaluation support the conclusions of the Pitarrilla Feasibility Study (M3, 2012).

## 16.2 GENERAL MINING DESCRIPTION

The Pitarrilla Feasibility Study (M3, 2012) calls for the mining of 157 Mt of silver ore and 933 Mt of waste via open cut mining methods from a single, large, semi-conical pit. The total waste tonnes include additional waste for potentially acid generating ("PAG") neutralizing material to be sourced outside the main pit. The ore and some of the waste is planned to be mined on 7.5m high benches and where there are suitable areas without ore, these are planned to be mined on 15m high benches (47% of tonnage).

The prime earth moving fleet is planned to comprise of four 21 $m^3$ shovels, two 19 $m^3$ loaders, up to 28 x 150 tonne trucks and ten drill rigs. The fleet is planned to move 190 ktpd to 210 ktpd of total material during peak production years.

The mining fleet is planned to supply two major ore types to the plant: a direct leach ore and a flotation/leach ore, which is planned to be processed at 12 ktpd for direct leach ore or 16 ktpd for flotation/leach ore. The plant is planned to either process one ore type or the other at any one time. Ore mining is planned to be in excess of plant capacity by 40%.

The mine excavation is planned to be completed in 20 years (including three pre-production years), leaving 12 years of processing the low grade stockpiles. A total of approximately 60 Mt of low grade ore is planned to be stockpiled and processed after pit mining ceases.

The waste dumps are planned to surround the pit and completed in a formation that are amenable to rehabilitation to 3H:1V (horizontal: vertical) slopes. Some control of PAG rock may be required through the mining of carbonate rich rocks from the Manto Rico formation and mixing this within the waste, as it is produced.

The flotation/leach ore is planned to recover two concentrates, lead and zinc, and the tailings of the flotation process are leached to produce a doré. The direct leach process is planned to recover only silver and produce a doré. It can be described that the direct leach ore is the most oxidised ore, and that the flotation/leach ore is a mix of partly oxidised (transitional) ore through to fresh, sulphide ore.

## 16.3 PIT DESIGN

The geological setting of the ore body is important for open pit slope design. The current Pitarrilla Project mine plan calls for the ore body to be mined using a single, large, open pit. The depth of the open pit is expected to be up to 600 m. KPL was engaged by SSR in late 2011 to complete the geomechanical work needed to support the pre-feasibility slope design for the proposed open pit (termed the “Oxide-Sulphide” Pit in KPL reports). KPL’s scope included: geomechanical and hydrogeological site investigations, a review of the available geological and structural information, characterization of the engineering properties of the encountered rock masses, laboratory strength testing, slope stability analyses and slope design for each sector of the proposed open pit.

SSR completed cashflow pit optimisations using Overall Slope Angles (“OSA”) determined from KPL’s pre-feasibility recommendations. The optimisations were used as a guide for Computer Aided Design (“CAD”) pit design combined with the detailed recommendations of bench and berm arrangements also supplied by KPL. The final CAD designs were then re-assessed by KPL using the pre-feasibility recommendations, thus assuring close design adherence to the recommendations.

It was identified that KPL did not require more raw data to conclude the design recommendations to geomechanical feasibility standard, but could deliver higher confidence results through adding further data analysis. KPL so continued the design process to geomechanical feasibility level in September to November 2012, by concentrating on a select set of studies. The results of the final feasibility analysis indicated that although most sectors remained unchanged, some design sectors required modification to some of the wall design parameters. SSR completed optimisation and cost checks of the stated changes and found that the net effect to project cashflow was less than -0.8% and as such the required changes are not material to the study outcomes. SSR plans to incorporate the design changes in the detailed engineering period.

KPL also supplied pit construction notes for attending to areas of geomechanical concern during construction, in order to advance risk management and mitigation strategies for risk control excellence.

### 16.3.1 Data Collection

The prefeasibility field program was completed between December 2011 and March 2012 and consisted of eight oriented and triple-tubed diamond drillholes with associated detailed geomechanical logging and hydraulic conductivity testing. This information was supplemented by three borehole tele-viewer surveys, surface mapping of select outcrops, and the installation of three multi-point vibrating wire piezometers. The drilling was carried out by Major Drilling Group International Inc. under the direct supervision of KPL and SSR. The drillholes associated with the geomechanical site investigation program are summarised in Table 16-1. The prefeasibility geomechanical drill program builds upon two earlier geomechanical drill programs completed in 2010 and 2011.

## Table 16-1: Geomechanical Drillhole Details

OXIDE-SULPHIDE OPEN PIT SLOPE DESIGN
GEOMECHANICAL DRILLHOLE DETAILS

| Drillhole Name | Drillhole Details | | | | | | Comments | Target |
|---|---|---|---|---|---|---|---|---|
| | Collar Coordinates | | | Azimuth (°) | Dip (°) | Length (m) | | |
| | Northing | Easting | Elevation (masl) | | | | | |
| 2010 GEOMECHANICAL SITE INVESTIGATION PROGRAM (201-227/14) | | | | | | | | |
| BPG-001 | 2,811,289 | 503,673 | 2,027 | 147 | -76 | 575 | | Breccia Ridge |
| BPG-002 | 2,811,296 | 504,290 | 1,957 | 236 | -64 | 950 | | |
| BPG-003 | 2,811,449 | 503,875 | 2,016 | 164 | -74 | 930 | | |
| BPG-004 | 2,811,476 | 504,245 | 1,904 | 249 | -63 | 842 | | |
| BPG-005-A | 2,811,378 | 504,104 | 1,976 | 208 | -54 | 850 | Re-drill of BPG-005 | |
| BPG-006 | 2,811,121 | 504,435 | 1,900 | 289 | -56 | 963 | | |
| BPG-007 | 2,811,150 | 503,950 | 2,063 | 26 | -80 | 757 | | |
| 2011 GEOMECHANICAL SITE INVESTIGATION PROGRAM (201-227/17) | | | | | | | | |
| BPG-008 | 2,810,834 | 503,180 | 1,970 | 330 | -45 | 85 | | Cordon Colorado |
| BPG-009 | 2,810,725 | 503,119 | 1,962 | 330 | -73 | 120 | | |
| BPG-010 | 2,810,644 | 503,194 | 1,929 | 194 | -60 | 100 | | |
| BPG-011 | 2,810,673 | 503,318 | 1,905 | 080 | -60 | 90 | | |
| BPG-012 | 2,811,432 | 503,342 | 1,924 | 030 | -60 | 130 | | Pena Dyke |
| BPG-013 | 2,810,905 | 504,415 | 1,876 | 280 | -60 | 260 | | SouthRidge |
| BPG-014 | 2,810,788 | 504,400 | 1,894 | 122 | -60 | 253 | | |
| BPG-015 | 2,810,476 | 504,200 | 1,953 | 065 | -65 | 127 | | |
| BPG-015A | 2,810,476 | 504,188 | 1,894 | 065 | -60 | 122 | Re-drill of BPG-015 | |
| BPG-016 | 2,810,558 | 504,193 | 1,985 | 220 | -60 | 275 | | |
| BPG-017 | 2,810,717 | 504,170 | 2,024 | 005 | -60 | 180 | | |
| BPG-017A | 2,810,717 | 504,170 | 2,024 | 005 | -60 | 240 | Re-drill of BPG-017 | |
| BPG-018 | 2,810,695 | 503,085 | 1,972 | 195 | -60 | 93 | | Cordon Colorado |
| BPG-019 | 2,810,844 | 503,220 | 1,935 | 020 | -55 | 114 | | |
| BPG-020 | 2,811,365 | 503,402 | 1,926 | 140 | -60 | 100 | | Pena Dyke |
| BPG-021 | 2,811,432 | 503,342 | 1,925 | 207 | -45 | 110 | | |
| 2011/2012 GEOMECHANICAL SITE INVESTIGATION PROGRAM (201-227/18) | | | | | | | | |
| BPG-022 | 2,811,469 | 504,241 | 1,902 | 097 | -60 | 502 | | Oxide-Sulphide Pit |
| BPG-023 | 2,811,450 | 503,880 | 2,017 | 308 | -71 | 633 | Piezometer installed in drillhole | |
| BPG-024 | 2,811,130 | 504,148 | 2,065 | 120 | -69 | 210 | | |
| BPG-024A | 2,811,131 | 504,147 | 2,065 | 120 | -69 | 739 | Re-drill of BPG-024 | |
| BPG-025 | 2,811,032 | 503,961 | 2,044 | 214 | -63 | 701 | | |
| BPG-026 | 2,811,467 | 503,948 | 2,052 | 045 | -71 | 731 | Piezometer installed in drillhole | |
| BPG-027 | 2,811,013 | 503,753 | 2,094 | 286 | -70 | 526 | Piezometer installed in drillhole and BPG-027A | |
| BPG-027A | 2,811,013 | 503,753 | 2,094 | 286 | -70 | 700 | Re-drill of lower portion of BPG-027 | |

NOTES:
1. COLLAR COORDINATES WERE SURVEYED BY SSR.
2. AZIMUTH ADJUSTED TO TRUE NORTH USING A DECLINATION OF 9° EAST.
3. DRILLHOLE DETAILS FROM 2010, 2011 AND 2012 GEOMECHANICAL PROGRAMS.
4. BRECCIA RIDGE AND PENA DYKE ORE BODIES TARGETED IN PREVIOUS GEOMECHANICAL PROGRAMS.

Drillholes were strategically planned to supplement existing information from the 2010 and 2011 geomechanical programs. All holes were collared in PQ3 and reduced to HQ3 with depth. Core orientation was undertaken on each geomechanical drillhole to determine discontinuity orientations within all encountered rock units. Core orientation was completed using the electronic Reflex ACT II tool, which generally allows for improved accuracy and drilling productivities relative to mechanical tools. KPL and SSR supervised core orientation at the drill and collected core orientation parameters to assess the quality of the collected data on an on-going basis. Core samples were also collected for laboratory strength testing.

Detailed geomechanical logging was completed on site at a centralised location by KPL. Rock mass parameters were input directly into KPL's electronic logging spreadsheet. Standard logging procedures were modified to meet the specific project conditions and requirements.

Detailed logging parameters were collected to characterize downhole variations in the rock mass quality.

Field estimates of unconfined compressive strength were made by KPL using a low-impact Schmidt Hammer. The Schmidt Hammer readings were collected using procedures adapted from ASTM standards ("ASTM D5873").

Tele-viewer surveys were undertaken to increase the amount of structural data collected. Three borehole tele-viewer surveys were completed as part of the site investigation program.

Surface mapping was also conducted at various outcrops and road cuts within the project area to collect discontinuity orientations and better characterize the engineering properties of the near-surface rock masses. Three different approaches were utilised depending on the position and characteristics of the exposure. These approaches consisted of Orientation Spot Mapping (Level 1), RMR89 / GSI Window Mapping (Level 2) and Detailed RMR89 Line Mapping (Level 3). Core samples were also collected from select drillholes to allow for the following laboratory testing: Uniaxial Compressive Strength ("UCS"), Triaxial Compressive Strength, Brazilian Indirect Tensile Strength, Direct Shear ("DS") Strength and Mineralogy.

Laboratory samples were collected for each significant rock unit, alteration type and observed joint-set family within each drillhole.

The final sample selection process was completed in North Bay, Ontario by KPL. The laboratory strength testing was carried out by the accredited rock mechanics lab at Queen's University in Kingston, Ontario.

The hydrogeological component of the program was intended to characterize the hydraulic conductivity of the encountered rock masses and to determine the elevation of the groundwater in the vicinity of the deposit. This information was needed to estimate mine inflow rates and help assess the impact of water pressure on slope performance. The hydrogeological site investigations consisted of borehole packer testing and the installation of three multi-point vibrating wire piezometer installations.

### 16.3.2 Discussion of Results

The geomechanical site investigations and laboratory strength testing results allowed several geomechanical domains to be defined using a combination of lithology and oxidation intensity. A summary of the rock mass characteristics for each domain is listed below in Table 16-2. The distribution of the geological formations with respect to the final stage pit is shown in Figure 16-1.

**Table 16-2: Geomechanical Drillhole Details**

| Domain | Oxidation States | UCS Value (MPa) | $m_i$ Value | $RMR_{89}$ | General Characteristics |
|---|---|---|---|---|---|
| Cardenas Formation | 2, 3, 4 & 5 | 55 | 10 | 50 | POOR to GOOD quality rock |
| Pitarrilla Formation - Main Pit | 0, 1 & 2 | 45 | 9 | 60 | GOOD quality rock |
| Pitarrilla Formation - Cordon Colorado | 2 & 3 | 25 | 9 | 55 | POOR to GOOD quality rock |
| Felsic Intrusive | 2 & 3 | 60 | 12 | 45 | POOR to GOOD quality rock |
| Lower Andesite Sill | 0, 1 & 2 | 90 | 8 | 65 | GOOD quality rock |
| Manto Rico Member | 0, 1 & 2 | 55 | 8 | 65 | GOOD quality rock |
| Pena Ranch Formation | 0 & 1 | 60 | 10 | 60 | GOOD quality rock |

For ease of reference, the pit was divided into three main regions: Main Pit, South Ridge and Cordon Colorado. Each region was further sub-divided into a series of design sectors presented on Figure 16-1. Based on the location and characteristics of the geomechanical domains and the pit shell provided to KPL, 18 design sectors were identified. Slope stability analyses were undertaken on each sector to define achievable slope configurations. These analyses included kinematic and Limit-Equilibrium analyses. The results from these analyses provided guidance on achievable bench face, inter-ramp and overall slope angles for each design sector. A summary of the recommended slope geometry can be found in Table 16-3, Table 16-4 and Table 16-5. The Inter Ramp Angle ("IRA") recommendations are also presented in Figure 16-1. Smaller bench widths and steeper slope angles are thought to be achievable for the South Ridge (SR label on Figure 16-1) and Cordon Colorado (CC label on Figure 16-1) regions of the pit based on the expected shorter stand-up times and shallower slopes. In all cases, a 15 m effective bench height was utilised.

# Table 16-3: Summary of Recommended Pit Slope Design Angles – Main Pit

**OXIDE-SULPHIDE OPEN PIT SLOPE DESIGN**
**SUMMARY OF RECOMMENDED PIT SLOPE DESIGN ANGLES - MAIN PIT**

| Pit Design Sector | | Nominal Pit Wall Dip Direction (°) | Total Slope Height (At 38° OSA) (m) | Dominant Final Wall Domains (At 38° OSA) | Dominant Potential Kinematic Failure Mode | Bench Configurations: Bench Face Angle (BFA) (°) | Bench Configurations: Bench Height (m) | Bench Configurations: Bench Width (m) | Inter-Ramp Slope Configurations: Inter-Ramp Angle (IRA): From Bench Configuration (°) | Inter-Ramp Angle (IRA): Achievable Based on Kinematics | Inter-Ramp Angle (IRA): Achievable Based on LE | Inter-Ramp Slope Configurations: Max. Slope Height (m) | Overall Slope Configurations: Overall Slope Angle (OSA): Estimated From IRA (°) | Overall Slope Angle (OSA): Achievable Based on LE | Overall Slope Angle (OSA): Achievable Based on Precedent Practice | Comments |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Northeast | Excluding Pena Ranch | 181 | 490 | Pitarrilla, Lower Andesite, Manto Rico | Wedge, Planar | 70 | 15 | 8 | 48 | Yes | Yes | 200 | 43 (3 stepouts) | Yes | Yes | - None |
| | Pena Ranch | | | Pena Ranch | None | 70 | 15 | 8 | 48 | Yes | Yes | 100 | | | | - None |
| East (North) | Excluding Pena Ranch | 240 | 605 | Cardenas, Manto Rico | Toppling | 70 | 15 | 10 | 44 | Yes | Yes | 200 | 40 (3 stepouts) | Yes | Yes | - Toppling failure on a major joint set limits IRA to 45° or less.<br>- Bench width increased to accommodate toppling failure. |
| | Pena Ranch | | | Pena Ranch | Toppling | 70 | 15 | 12 | 40 | Yes | Yes | 200 | | | | - Toppling failure on a major joint set limits IRA to 40° or less.<br>- Bench width increased to accommodate toppling failure.<br>- The 290 Fault intersects the slope in this sector. The reduced rock mass quality associated with the fault will likely result in bench-scale failures. |
| East (South) | Excluding Pena Ranch | 240 | 485 | Cardenas, Pitarrilla | Toppling | 70 | 15 | 10 | 44 | Yes | Yes | 100 | 37 (4 stepouts) | Yes | Yes | - The 290 Fault intersects the slope in this sector. The reduced rock mass quality associated with the fault will likely result in bench-scale failures. |
| | Pena Ranch | | | Pena Ranch | Toppling | 70 | 15 | 12 | 40 | Yes | Yes | 200 | | | | - None |
| Southeast (East) | Excluding Pena Ranch | 295 | 241* | Pitarrilla | Wedge | 70 | 15 | 8 | 48 | Yes | No (46°) | 100 | 40 (2 stepouts) | Yes | Yes | - None |
| | Pena Ranch | | | Pena Ranch | Toppling | 70 | 15 | 8 | 48 | Yes | Yes | 100 | | | | - None |
| Southeast (West) | Excluding Pena Ranch | 340 | 270* | Pitarrilla, Lower Andesite | Wedge | 70 | 15 | 8 | 48 | Yes | Yes | 200 | 42 (2 stepouts) | Yes | Yes | - Potential for localized toppling failures. |
| | Pena Ranch | | | Pena Ranch | Planar | 70 | 15 | 8 | 48 | Yes | Yes | 100 | | | | - None |
| Southwest | Excluding Pena Ranch | 21 | 489 | Pitarrilla, Lower Andesite, Manto Rico | Wedge, Planar | 65 | 15 | 10 | 41 | Yes | Yes | 200 | 37 (3 stepouts) | Yes | Yes | - Wedge failure on a major joint set and a fault limits IRA to 45° or less.<br>- Bench width increased to accommodate wedge failure.<br>- The Pena Fault runs parallel to the slope in this sector. The reduced rock mass quality associated with the fault will likely result in bench-scale failures. A step-out may be required below the fault. |
| | Pena Ranch | | | Pena Ranch | Wedge, Planar | 70 | 15 | 10 | 44 | Yes | Yes | 200 | | | | - Bench width increased to accommodate wedge and planar failure. |
| West | Excluding Pena Ranch | 67 | 551 | Pitarrilla, Lower Andesite | Wedge, Planar | 60 | 15 | 10 | 39 | Yes | Yes | 200 | 35 (3 stepouts) | Yes | Yes | - Bench width increased to accommodate planar failure and localized wedge failures along faults.<br>- The Pena Fault runs parallel to the slope in this sector. The reduced rock mass quality associated with the fault will likely result in bench-scale failures. A step-out may be required below the fault. |
| | Pena Ranch | | | Pena Ranch | Wedge, Planar | 60 | 15 | 12 | 36 | Yes | Yes | 200 | | | | - Planar failure on a major joint set limits IRA to 40° or less.<br>- Bench width increased to accommodate wedge and planar failure. |
| Northwest | Excluding Pena Ranch | 117 | 412 | Pitarrilla, Lower Andesite | Wedge | 70 | 15 | 10 | 44 | Yes | Yes | 200 | 39 (3 stepouts) | Yes | Yes | - Possible wedge failure between major joint set and a fault could limit IRA to 35°.<br>- Bench width increased to accommodate wedge failure.<br>- Potential for localized wedge failures along faults.<br>- The 290 Fault intersects the slope in this sector. The reduced rock mass quality associated with the fault will likely result in bench-scale failures. |
| | Pena Ranch | | | Pena Ranch | None | 70 | 15 | 8 | 48 | Yes | Yes | 100 | | | | - Potential for localized wedge failures along faults. |

**NOTES:**

1. ALL THE STABILITY ANALYSES ARE FOR STATIC CONDITIONS ONLY.
2. ACHIEVABLE SLOPE ANGLES WILL VARY BASED ON ENCOUNTERED PIT SLOPE GEOLOGY.
3. TOTAL SLOPE HEIGHTS FOR SOUTHEAST (EAST) AND SOUTHEAST (WEST) SECTORS DO NOT INCLUDE PORTIONS OF THE SLOPE WITHIN THE SOUTH RIDGE LOBE.
4. THE ESTIMATED OSAs FOR THE NORTHEAST AND NORTHWEST SECTORS INCLUDE AN ADDITIONAL STEPOUT TO PROVIDE THE OPERATIONAL FLEXIBILITY NEEDED TO ACCOMMODATE POOR QUALITY ROCK ASSOCIATED WITH ALTERATION AND FAULTS.

# Table 16-4: Summary of Recommended Pit Slope Design Angles – South Ridge

OXIDE-SULPHIDE OPEN PIT SLOPE DESIGN
SUMMARY OF RECOMMENDED PIT SLOPE DESIGN ANGLES - SOUTH RIDGE

| Pit Design Sector | Nominal Pit Wall Dip Direction | Total Slope Height (At 44° OSA) | Dominant Final Wall Domains (At 44° OSA) | Dominant Potential Kinematic Failure Mode | Bench Configurations | | | Inter-Ramp Slope Configurations | | | | Overall Slope Configurations | | | Comments |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Bench Face Angle (BFA) | Bench Height | Bench Width | Inter-Ramp Angle (IRA) | | | Max. Slope Height | Overall Slope Angle (OSA) | | | |
| | | | | | | | | From Bench Configuration | Achievable Based on Kinematics | Achievable Based on LE | | Estimated From IRA | Achievable Based on LE | Achievable Based on Precedent Practice | |
| | (°) | (m) | | | (°) | (m) | (m) | (°) | | | (m) | (°) | | | |
| North | 232 | 186 | Cardenas, Pitarrilla | Planar, Toppling | 60 | 15 | 8 | 42 | Yes | Yes | 186 | 42 (No Stepout) | Yes | Yes | - Toppling failures on major joint sets may impact IRA slope performance.<br>- Potential for localized wedge failures along faults.<br>- The 290 Fault intersects the slope in this sector. The reduced rock mass quality associated with the fault will likely result in bench-scale failures. |
| East | 265 | 159 | Cardenas, Pitarrilla | Planar, Toppling | 60 | 15 | 8 | 42 | Yes | Yes | 159 | 42 (No Stepout) | Yes | Yes | - Toppling failures on major joint sets may impact IRA slope performance. |
| South | 357 | 177 | Cardenas, Pitarrilla | Wedge | 70 | 15 | 10 | 44 | Yes | Yes | 177 | 40 (1 Stepout) | Yes | Yes | - Pit wall dip direction variable; localized wedge failures likely.<br>- Bench width increased to accommodate wedge failure.<br>- The Pena Fault runs parallel to the slope in this sector. The reduced rock mass quality associated with the fault will likely result in bench-scale failures. A step-out has been incorporated into the OSA configuration to help manage any failures. |
| West | 83 | 277 | Pitarrilla | Wedge, Planar, Toppling | 70 | 15 | 12 | 41 | No (40°) | Yes | 200 | 38 (1 Stepout) | Yes | Yes | - Toppling failures on major joint set may impact IRA slope performance.<br>-Planar and wedge failures on faults limit IRA to 40°.<br>- Bench width increased to accommodate planar, toppling and wedge failure.<br>- Potential for localized wedge failures along faults.<br>- The Pena Fault runs parallel to the slope in this sector. The reduced rock mass quality associated with the fault will likely result in bench-scale failures. The increased bench width will help manage any failures. |

NOTES:
1. ALL THE STABILITY ANALYSES ARE FOR STATIC CONDITIONS ONLY.
2. ACHIEVABLE SLOPE ANGLES WILL VARY BASED ON ENCOUNTERED PIT SLOPE GEOLOGY.
3. MAXIMUM INTER-RAMP SLOPE HEIGHT LIMITED BY TOTAL SLOPE HEIGHT.

# Table 16-5: Summary of Recommended Pit Slope Design Angles – Cordon Colorado

OXIDE-SULPHIDE OPEN PIT SLOPE DESIGN
SUMMARY OF RECOMMENDED PIT SLOPE DESIGN ANGLES - CORDON COLORADO

| Pit Design Sector | Nominal Pit Wall Dip Direction (°) | Total Slope Height (At 44° OSA) (m) | Dominant Final Wall Domains (At 44° OSA) | Dominant Potential Kinematic Failure Mode | Bench Configurations | | | Inter-Ramp Slope Configurations | | | | Overall Slope Configurations | | | Comments |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Bench Face Angle (BFA) (°) | Bench Height (m) | Bench Width (m) | Inter-Ramp Angle (IRA) | | | Max. Slope Height (m) | Overall Slope Angle (OSA) | | | |
| | | | | | | | | From Bench Configuration (°) | Achievable Based on Kinematics | Achievable Based on LE | | Estimated From IRA (°) | Achievable Based on LE | Achievable Based on Precedent Practice | |
| Northeast | 280 | 53 | Felsic | Wedge | 70 | 15 | 8 | 48 | Yes | Yes | 53 | 48 (No Stepout) | Yes | Yes | - Potential for localized wedge failures along faults. |
| East | 200 | 78 | Felsic | Toppling | 70 | 15 | 8 | 48 | Yes | Yes | 78 | 48 (No Stepout) | Yes | Yes | - None |
| Southeast | 305 | 81 | Pitarrilla | Wedge, Toppling | 65 | 15 | 8 | 45 | Yes | Yes | 81 | 45 (No Stepout) | Yes | Yes | - None |
| South | 20 | 98 | Pitarrilla, Felsic | Wedge, Toppling | 60 | 15 | 8 | 42 | Yes | Yes | 98 | 42 (No Stepout) | Yes | Yes | - The Pena West Regional Fault intersects the slope in this sector. The reduced rock mass quality associated with the fault will likely result in bench-scale failures. |
| West | 82 | 113 | Pitarrilla, Felsic | Wedge, Toppling | 60 | 15 | 8 | 42 | Yes | Yes | 113 | 42 (No Stepout) | Yes | Yes | - Wedge and toppling failures on major joint sets may impact IRA slope performance.<br>- The Pena West Regional Fault intersects the slope in this sector. The reduced rock mass quality associated with the fault will likely result in bench-scale failures. |
| Northwest | 125 | 102 | Felsic | Planar, Wedge | 65 | 15 | 8 | 45 | Yes | Yes | 102 | 45 (No Stepout) | Yes | Yes | - Planar failures on major joint sets may impact IRA slope performance.<br>- Toppling failure associated with faults limits IRA to 45°.<br>- The Pena West Regional Fault intersects the slope in this sector. The reduced rock mass quality associated with the fault will likely result in bench-scale failures. |

NOTES:
1. ALL THE STABILITY ANALYSES ARE FOR STATIC CONDITIONS ONLY.
2. ACHIEVABLE SLOPE ANGLES WILL VARY BASED ON ENCOUNTERED PIT SLOPE GEOLOGY.
3. MAXIMUM INTER-RAMP SLOPE HEIGHT LIMITED BY TOTAL SLOPE HEIGHT.

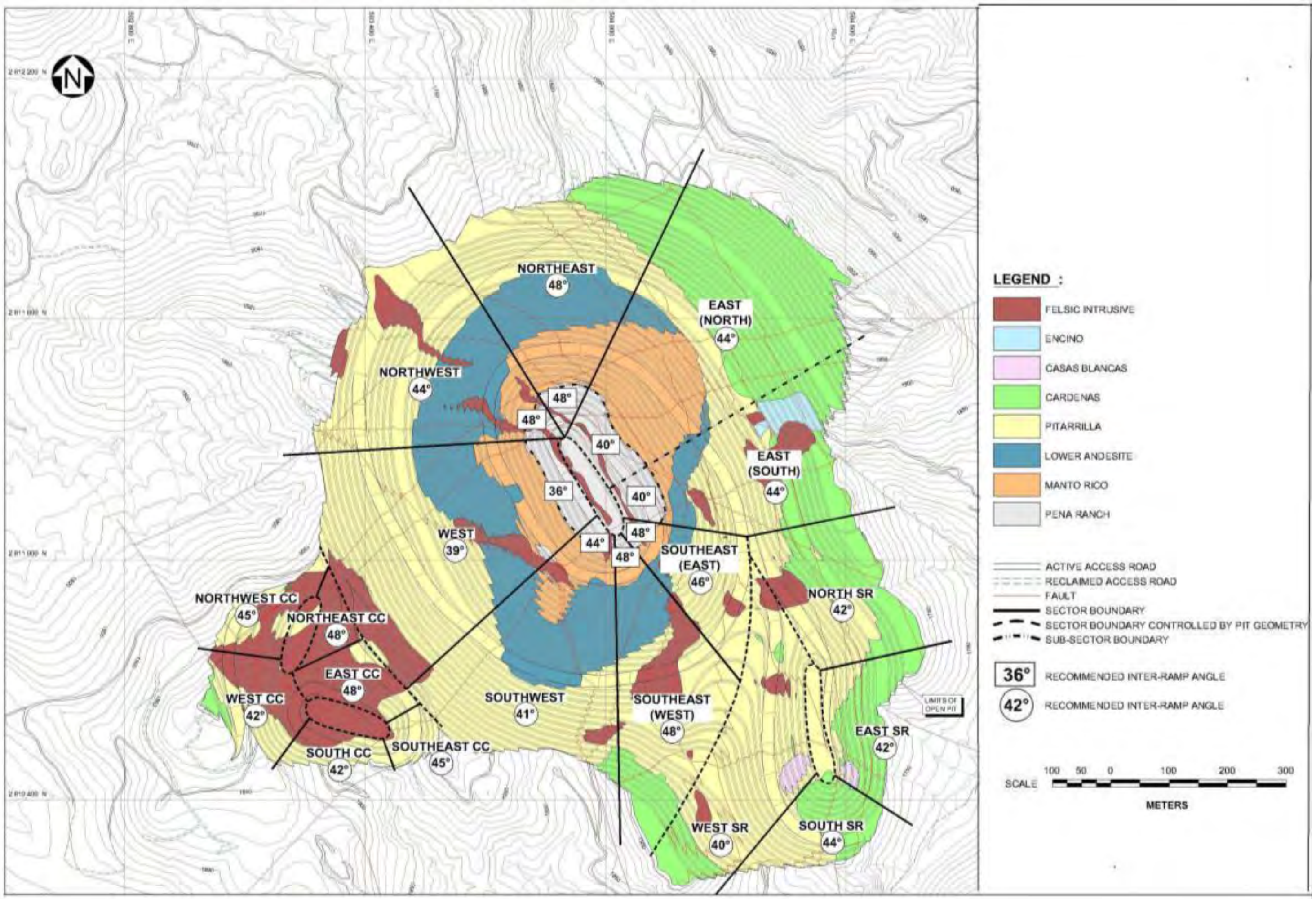



**Figure 16-1: Recommended Inter-Ramp Angle by Design Sector**
NAD 27 UTM Zone 13N Source: KPL, 2012b

The pit slope design recommendations were based upon the available geomechanical, hydrogeological and geological data. The completed stability analyses and a review of the precedence practice plot suggested that the recommended geometries are reasonable and appropriate.

To achieve these angles, the design assumes that controlled blasting and proactive geotechnical monitoring is planned to be undertaken along with an on-going commitment to geomechanical data collection and analyses. The analyses suggested that groundwater depressurization will not strongly influence overall slope stability, however the phreatic surface that develops behind the pit walls should be monitored and depressurization implemented on an as-needed basis.

### 16.3.3 Feasibility Geomechanical Work

The pit designs included in the Pitarrilla Feasibility Study (M3, 2012) were completed using the design recommendations of the pre-feasibility geomechanical study completed by KPL in August 2012 (KPL, 2012b).

The pre-feasibility geomechanical study concluded that to supply recommendations to a feasibility level, some additional analysis would be beneficial. The geomechanical pre-feasibility study also stated that no further drilling data was required to support the designs or analysis to feasibility level standard. KPL completed the additional feasibility level analysis between September and November 2012.

The timing of the completion of the additional geomechanical analysis to feasibility level did not allow full integration of the results to the feasibility pit designs. However, SSR was able to complete optimisation and design analysis with the slightly modified wall angles of the feasibility geomechanical study and found that pre-tax cashflow reduced by a maximum of 0.8%. This small, pre-tax cashflow adjustment is not material to the findings of the feasibility report. SSR plans to incorporate the adjustments to the feasibility designs in the detailed engineering period as part of the on-going commitment to project adjustments as data improves in quality and refinement commensurate with project advancement.

The results of the feasibility geomechanical study concluded with updated pit slope recommendations, by sector, which were then modified, to include the results of the analysis completed as at November 2012.

Maintaining flexibility in the mine plan is an important feature in accommodating potential slope stability issues.

### 16.3.4 Future Geomechanical Work for Detailed Engineering

Future work will include a review of the updated pit design, when it becomes available, as well as the work required to support detailed design and construction. The future work is expected to include more detailed analyses based on additional or updated data for the deposit.

### 16.3.5 Pit CAD Designs

The recommended OSA's in Section 16.3 were the major guide to optimisation inputs. The optimisation shells and previous pit design findings for best road access layouts were combined with the IRA and bench design recommendations to develop a set of feasibility pit CAD designs. The designs were completed by SSR and delivered to KPL for final verification in the feasibility study geomechanical. Discussion of the optimisation controls are discussed in Section 16.5.4.

All designs include the following considerations not mentioned in the geomechanical design consideration:

- Roads are 30 m in width for dual lane access with slight upside protection for increasing equipment size;
- 10% ramp grade (inside curve radius controlled);
- All berms have ramp access;
- All major catch berms have haul road access to the waste dumps, to allow removal of fallen debris;
- In some cases, for the last four benches of a phase into minor extraction volumes, roads are reduced to single lane access of 18 m width;
- Only one switch-back is included in all the designs (Phase 2); it is recommended that this be reviewed in the detailed engineering period for cost effective removal;
- Pit bases are not less than 25m in width;
- Pit design internal curve radii are not less than 12.5m;
- Pit design external curve radii are not less than 75m;
- Near to surface, the narrowest working area is not less than 75m;
- All cutbacks are more than 100 m in working width;
- Ramps in each phase are designed, in large part, such that they minimize traffic flow issues;
- A number of sectors require changing slope face angles sector to sector. The designs for feasibility work respect this consideration, but each bench treatment is unique. It is recommended that during detailed design, a consistent manner of application be adopted that can translate to a consistent field operating method; and
- Road intersections are wide for good visibility; exact road intersections are not designed and final safety considerations are to be completed in the detailed engineering phase.

The designs are named in time sequence of development:

- Breccia Ridge Phase 1 pit ("BR1East_20120924" – Figure 16-2)
- Cordon Colorado pit ("CordonColorado_20120924" - Figure 16-2)
- Breccia Ridge Phase 1A pit ("BR1A_20120925" - Figure 16-3)
- Breccia Ridge Phase 2 pit ("BR2_20121004" - Figure 16-4)
- Breccia Ridge Phase 3 pit ("BR3_20121004" - Figure 16-5)
- Breccia Ridge Phase 4 pit ("BR4_20121004" - Figure 16-6)

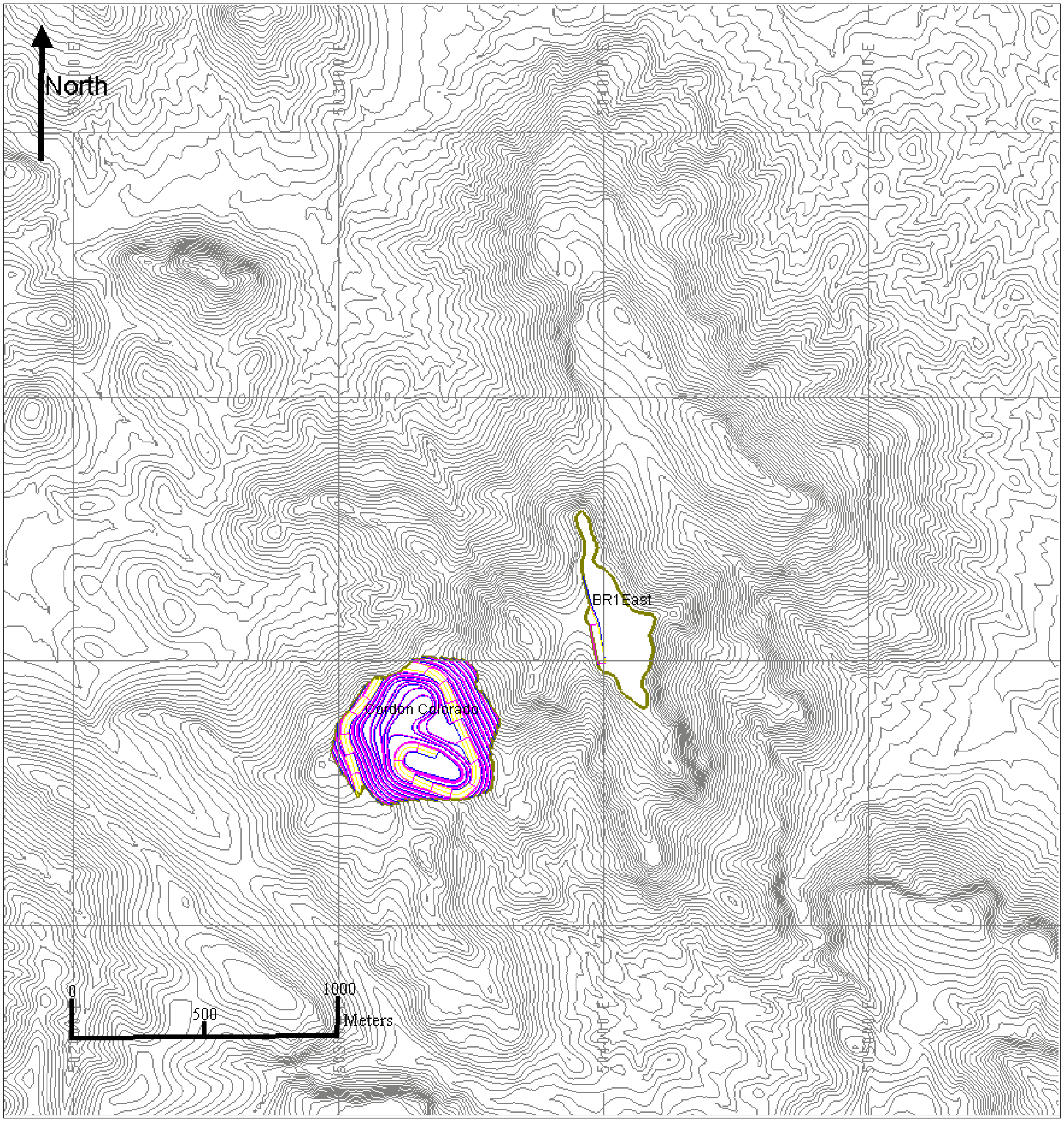



**Figure 16-2: Cordon Colorado Pit (Cordon Colorado_20120924) and Breccia Ridge Phase 1 East Pit (BR1East_20120924))**

NAD 27 UTM Zone 13N Source: M3, 2012

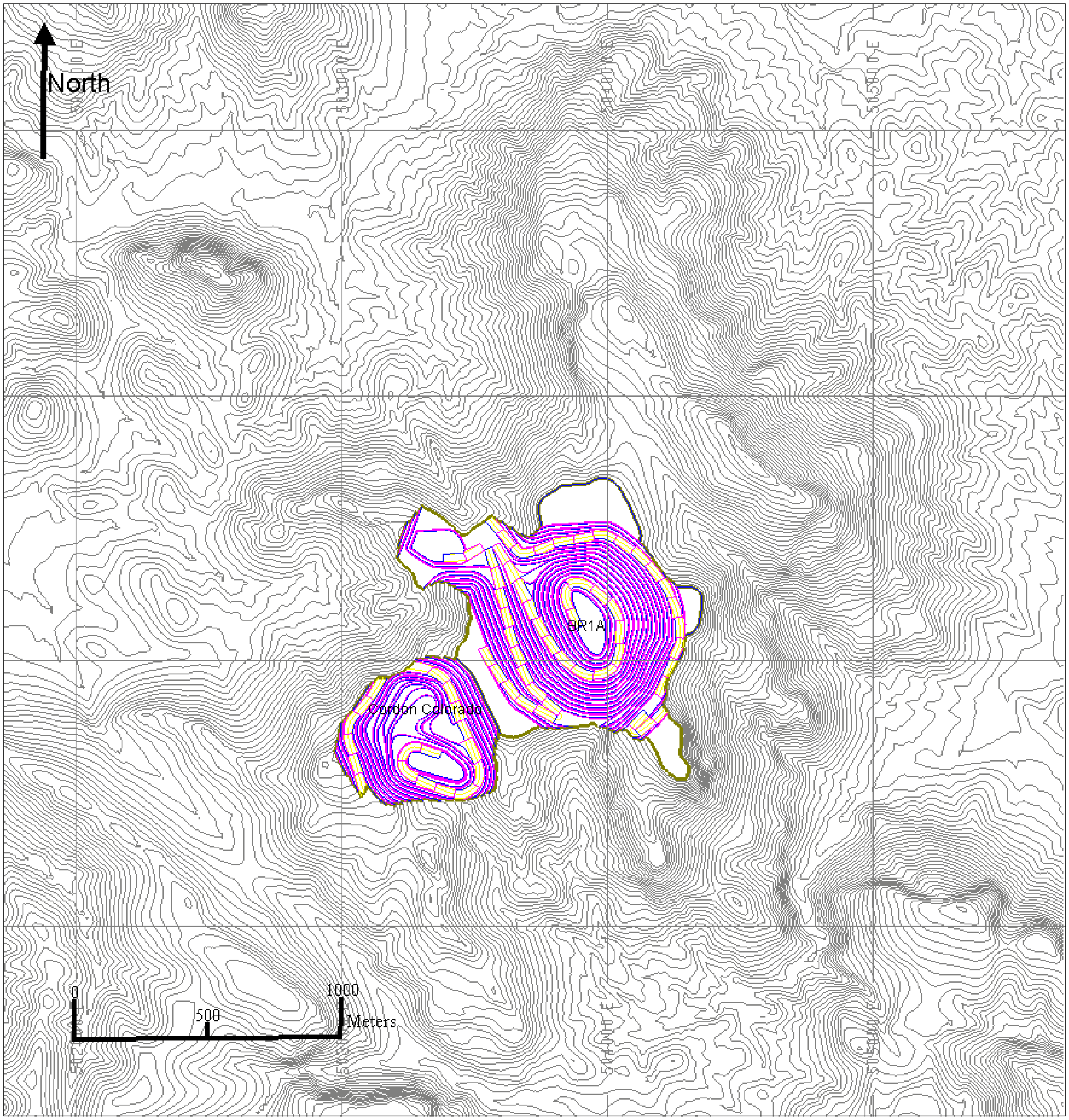



**Figure 16-3: Breccia Ridge Phase 1A Pit (BR1A_20120925)**

NAD 27 UTM Zone 13N Source: M3, 2012

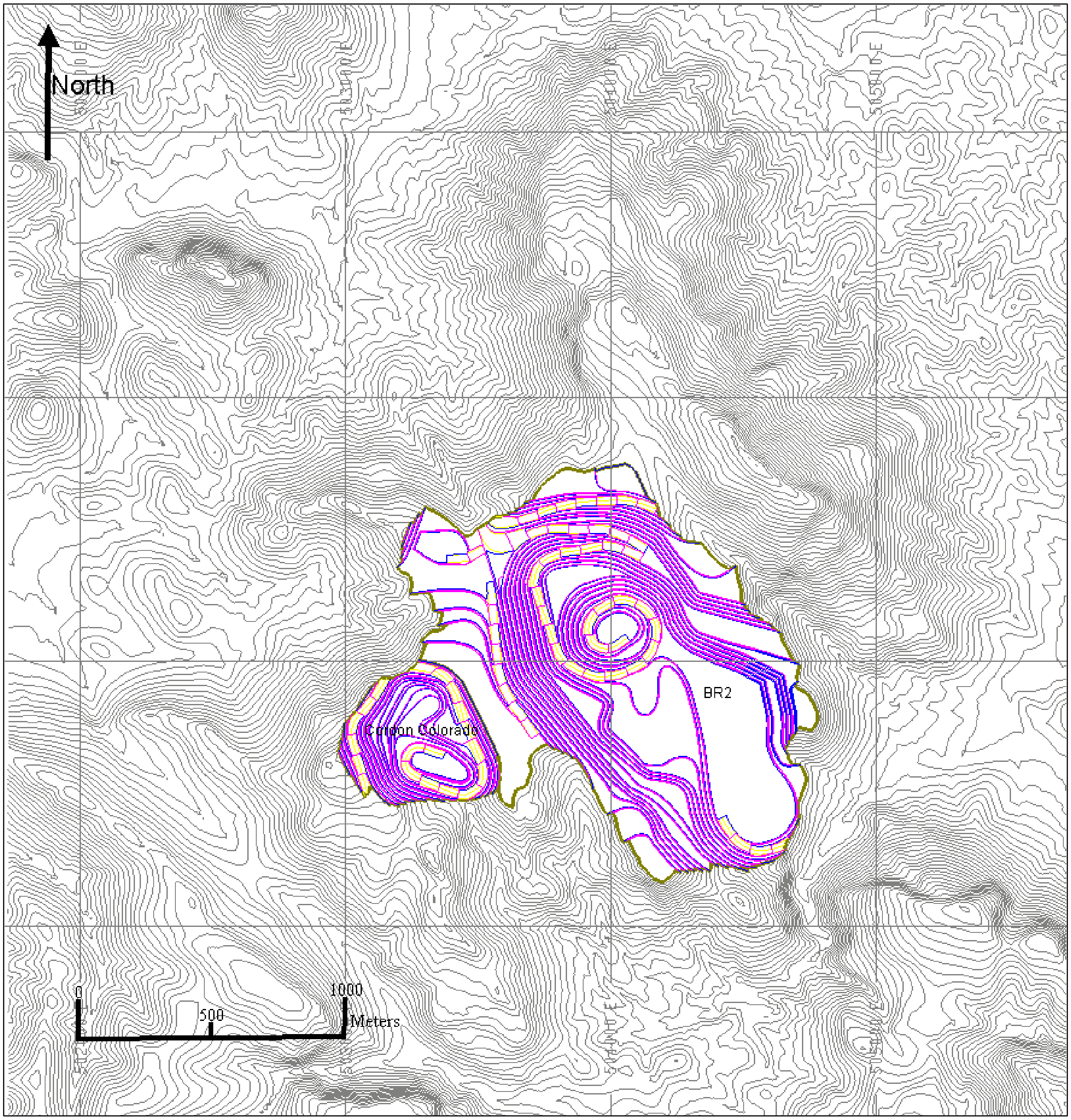



**Figure 16-4: Breccia Ridge Phase 2 Pit (BR2_20121004)**

NAD 27 UTM Zone 13N Source: M3, 2012

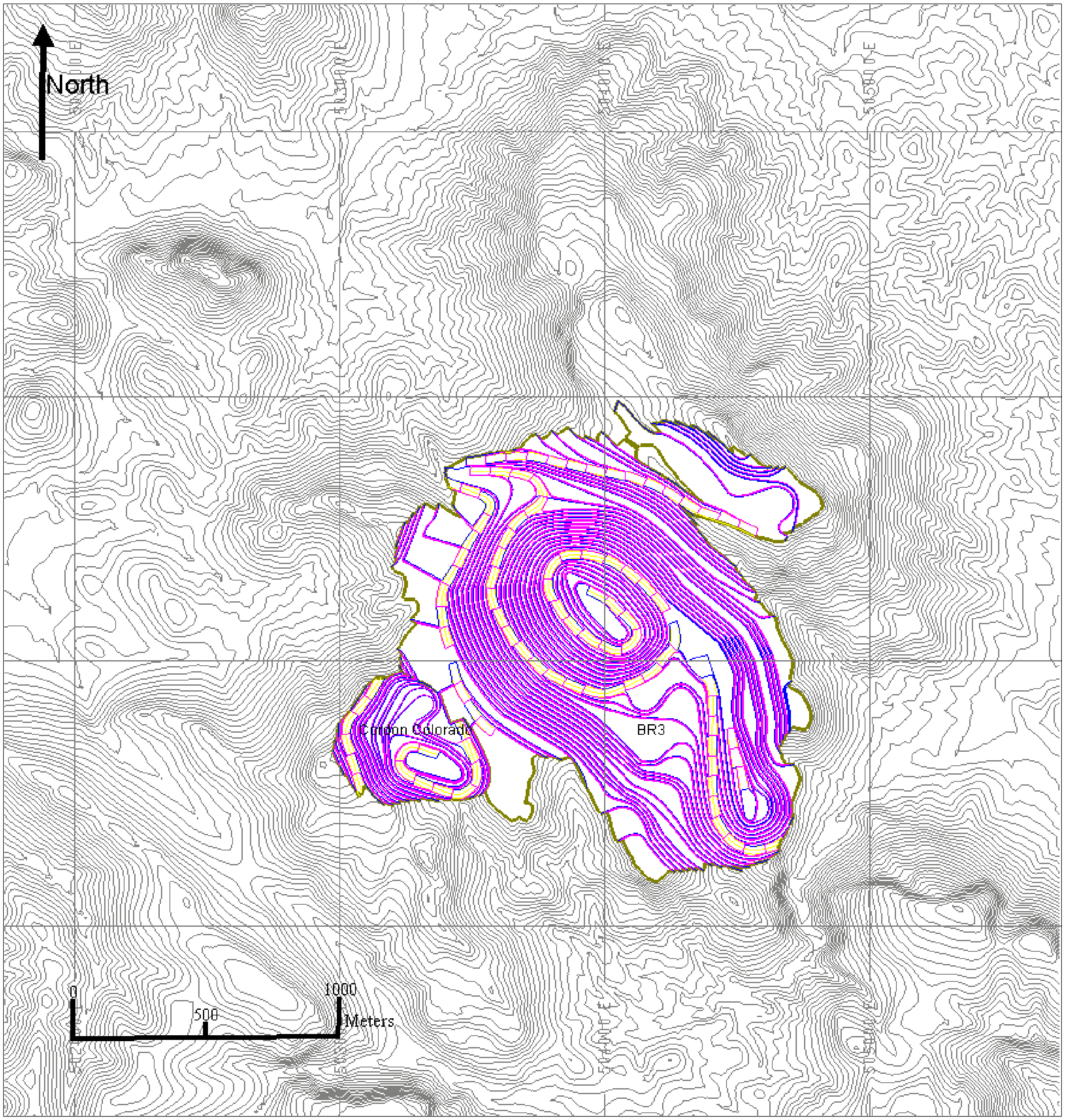



**Figure 16-5: Breccia Ridge Phase 3 Pit (BR3_20121004)**

NAD 27 UTM Zone 13N Source: M3, 2012

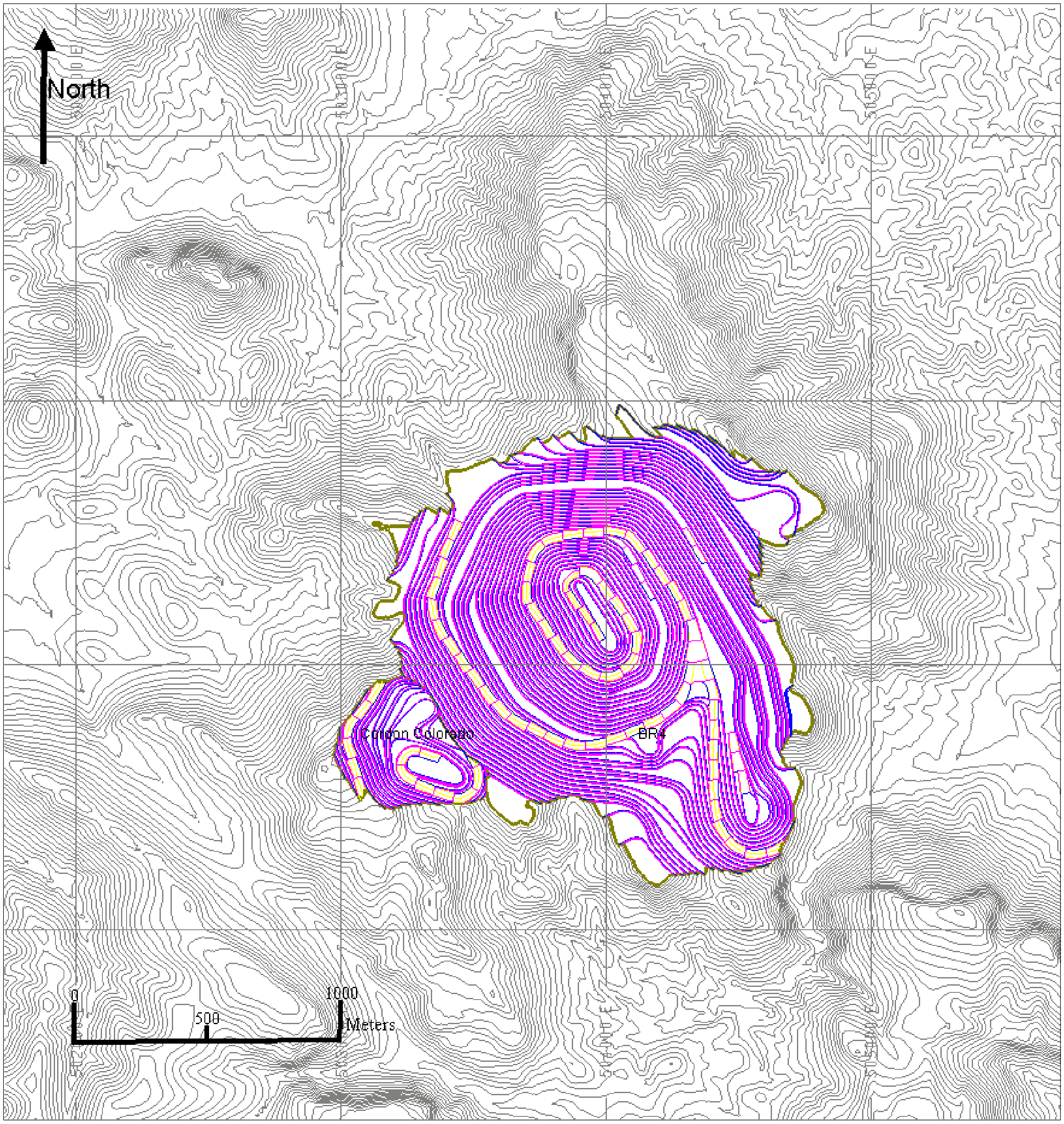



**Figure 16-6: Breccia Ridge Phase 4 Pit (BR4_20121004)**
NAD 27 UTM Zone 13N Source: M3, 2012

Designs were also completed for detailed engineering of the initial pit access operation and NPAG source pit mining (shown in Figure 16-7 and Figure 16-8).

Access roads are designed assuming:

- The roads will have an initial 5m deep cut and fill road developed by dozer and some drill and blast, except where the rock has been identified as too steep or hard to cut. In these steeper or harder locations, the roads will be built with 100% fill;
- The roads are a total of 26 m in width, where fill is only available from a borrow pit, and 30 m otherwise, and may be widened over time; and
- Road cuts are made at 65˚; fill is placed at 36˚.

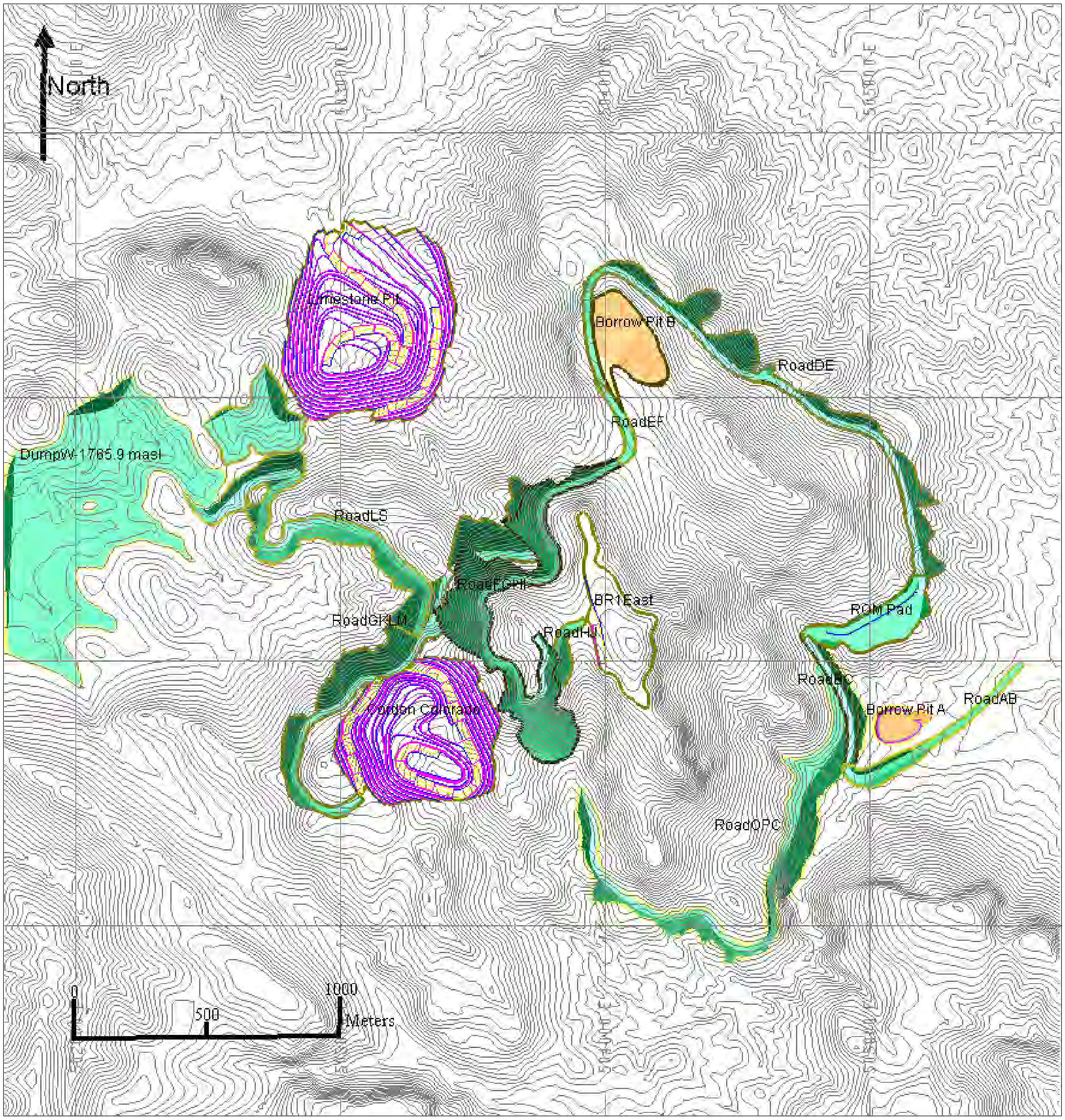



**Figure 16-7: Initial Pit Access Roads**

NAD 27 UTM Zone 13N Source: M3, 2012

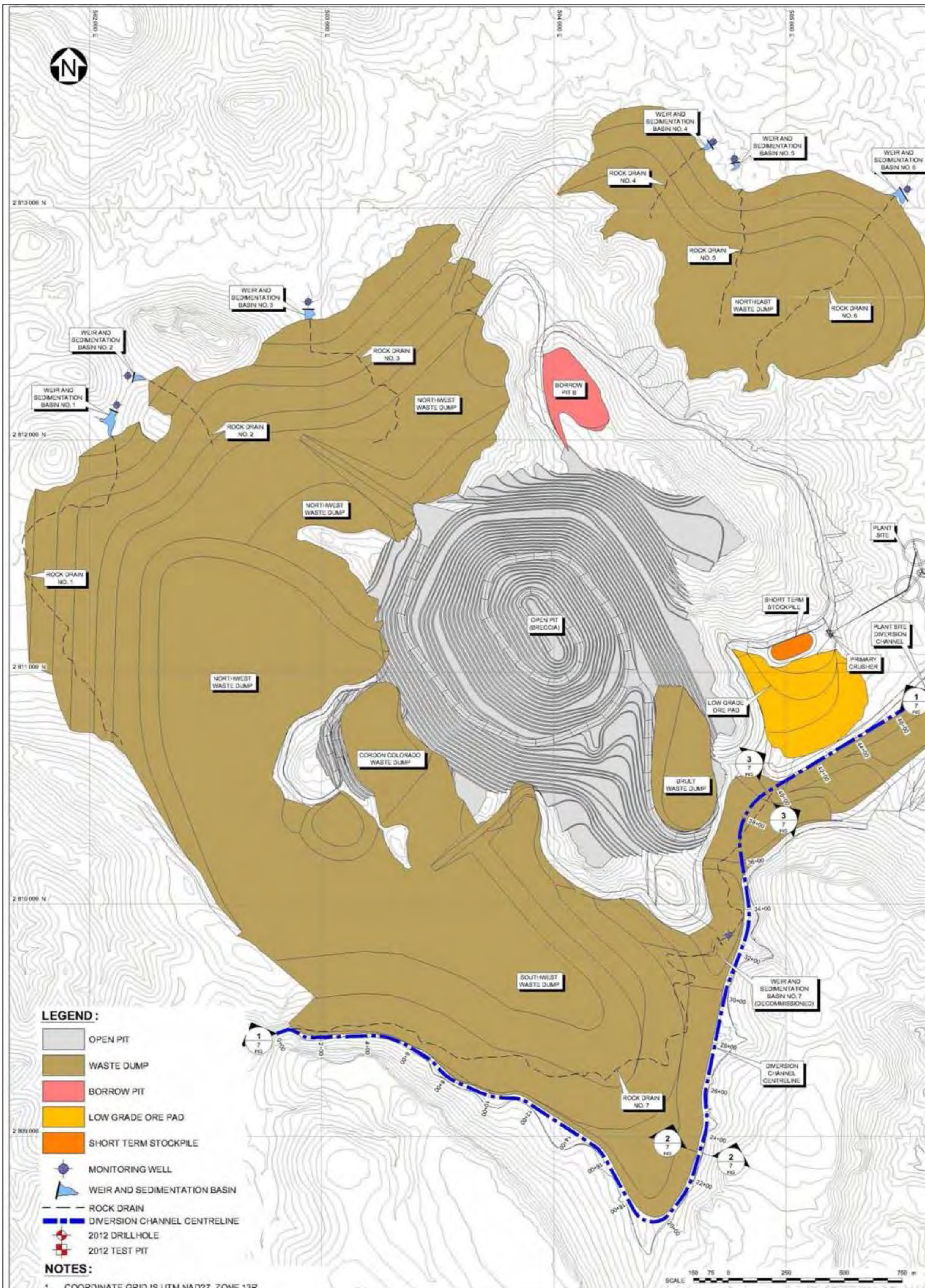



**Figure 16-8: Waste Dump Design Final Arrangement**

NAD 27 UTM Zone 13N. Source: KPL 2012c

## 16.4 WASTE DUMP DESIGNS

The waste dump designs were developed in a series of iterations. The feasibility designs are matched in total volume to the waste dump volume of rock assuming a 1.25 swell factor and that some waste (a total of approximately 5%) will be used in the tailings dam wall when the dam is expanded in future years (as saddle dams in Stage 2 and beyond). The waste rock development plan as defined by KPL was adopted, where rock of highest acid neutralizing potential ("ANP") value (i.e. rocks richer in carbonate such as the Manto Rico member and the limestone clast bearing conglomerate in the upper part of the Pitarrilla Formation) will be used to counteract the effects of PAG rocks, especially material such as the sulphur-rich felsic dyke close to the ore zone. A pit was designed specifically to source the Manto Rico horizon and to extract sufficient material centred on an outcrop of this in the Casas Blancas valley. The pit design is 100% within the footprint of the final waste dump and scheduling of the material being mined and stockpiled for later use has been completed. It is suggested, that an opportunity exists during detailed engineering, that this strategy (although it meets all recommendations by KPL) should be reviewed to find the most optimal cost plan for the equivalent environmental outcome.

### 16.4.1 Waste Rock Dump Optimisation and Design

The waste dump locations were initially developed through Lerches Grossman optimisation for minimization of cost with the constraints of final wall angle and available footprint, while leaving 50 m free around the pit edge and leaving the plant site free of waste. The specific characteristics developed for the optimisation and later design were:

- Overall final slope of 3H:1V (i.e. 18˚ slope angle). This was to achieve 2.5H:1V final slope faces with drainage berms every 60 m of vertical height;
- 36˚ batters;
- 60 m or 30 m dumping levels, although due to timing of dump construction, some dumps will temporarily be up to 200 m in height. Dumping methodologies to ensure a safe workplace will be adopted based on world's best geotechnical control and operational practices and are detailed in KPL's waste dump report;
- 30 m wide ramps;
- 10% ramp grade;
- The final slope toes had to lie 25 m inside all required property boundaries;
- Final CAD dump designs also included water management features from the KPL waste dump and water management design report;
- The water diversion on the southern waste dump was designed at 1% gradient and cut into in-situ topography by 5 m; however, this could be built with fill material and in either case is planned to require installing a geosynthetic liner. This diversion is further detailed in the KPL waste dump design report.

### 16.4.2 Waste Rock Development Plan

KPL conducted a review of the Pitarrilla Mine waste rock development plan, while considering the acid generating potential ("AGP") of the rock mass to estimate the potential for acid rock drainage and metal leaching from the dumps. Specific aspects addressed include the following:

- Site characteristics including physiographic setting, climate, hydrometeorology and seismicity;
- Geotechnical conditions at the waste dumps;
- Waste characterization methodology and results;
- Surface and storm water management, including rock drains, diversion channels, water collection and sediment control measures;
- Layout and design of the waste dumps, including the incorporation of PAG waste within the dumps;
- Waste dump stability;
- Operation and monitoring considerations;
- Reclamation and closure; and
- Material, quantity and cost estimates for capital items;

A feasibility level design was presented for the storage and management of approximately 909 Mt of waste rock which will be deposited into three primary waste dumps over a projected 20 year mine life. KPL used an estimate of pit volumes calculated in May 2012 which yielded the above waste tonnes; final waste tonnage is 933 Mt including the PAG neutralizing material. The difference is not considered material to the ARD mitigation plan developed by KPL. The waste dumps will effectively surround the open pit from the south to the north and northeast. In addition to the three main waste dumps, the mine plan incorporates in-pit waste dumping in the Cordon Colorado and Breccia Ridge Phase four pits. These dumps are known respectively as Cordon Colorado and Brult dumps and are small in comparison to the three main waste dumps.

The final arrangement of the waste dumps at the end of the mining life is as shown on Figure 16-8 and also includes the locations of water and sediment control measures and the site investigation locations.

The analyses completed for the waste dump design included the following:

- Characterization of the waste rock with regards to acid neutralization potential, acid generation potential and metal leaching, and recommendations for additional test work.

- Development of a water management and sediment control program for the waste dumps. The 1 in 200 year storm was used to size rock drains in valley bottoms underneath the waste dumps and a diversion channel along the south side of the waste dumps. The 1 in 2 year storm event was used to design sedimentation basins and flow monitoring weirs downstream of the waste dump toes. Monitoring wells are also included in the design to measure the groundwater quality over the mine life and at closure.

- Static and seismic (pseudo-static) stability analyses of the waste dumps. These analyses included consideration of both dump face failures and deep seated failures in the foundation units. The waste dump stability meets all of the requirements for all analyzed conditions, based on relevant guidelines.

- Development of a disposal strategy for the potentially acid-generating ("PAG") waste. The proposed strategy utilizes the addition of mined and potentially processed calcium-rich rock fill at the dump face and encapsulation of the PAG waste with non-acid-generating ("NAG") waste.

- Development of an operations, closure and reclamation plan to ensure the chemical and physical stability of the waste dumps at closure.

The design incorporates the results from 2012 site investigations and provides the results of the hydrologic, geochemical and engineering analyses. The waste dump development plan begins with waste rock placement at high crest elevations adjacent to the open pit and ends with wrap-around lifts of waste rock at lower elevations to provide overall 3H:1V slopes at closure.

SSR arranged and managed Acid Base Accounting ("ABA") and Metal Leaching ("ML") testwork to characterize the waste rock. The testing was completed by ALS Laboratories in Monterrey, Mexico and certifications of the testwork can be obtained from the lab or from SSR.

A total of 50 ABA tests were completed in May 2012 and 34 tests were completed in July 2012. ML testing was completed on the 34 samples tested in July 2012.

### 16.4.3 Waste Classification Assumptions and Criteria

To classify the waste rock, sulphur concentrations from the assay results corresponding to the ABA testwork were assumed to be 100% sulphide. This is deemed to be conservative since only a portion of sulphur in the samples is sulphide. Insoluble sulphur does not assist in the generation of acid.

On that basis, seven criteria were set up as follows:

- Criterion 1: If the sulphur concentration is greater than 0.3% by weight, the suggested classification is PAG.
- Criterion 2: If the sulphur concentration is less than 0.3% by weight, the suggested classification is NAG.
- Criterion 3: If the Acid Neutralization Potential/Acid Generation Potential ("ANP/AGP") ratio is less than 1.0, the suggested classification is PAG.
- Criterion 4: If the ANP/AGP ratio is greater than 1.0 but less than 3.0, the suggested classification is Uncertain.
- Criterion 5: If the ANP/AGP ratio is greater than 3.0, the suggested classification is NAG.

- Criterion 6: If the Net Neutralization Potential ("NNP") is less than 20 kg CaCO3/t, the suggested classification is PAG.
- Criterion 7: If the NNP is greater than 20 kg CaCO3 /tonne, the suggested classification is NAG.

Criterion 1 was analysed first. Samples considered to be PAG under this criterion were classified as PAG. No comparisons were made with other criteria, because it is generally accepted that rock with sulphur content greater than 0.3% by weight is considered to be PAG. All samples with sulphur content less than 0.3% by weight were then evaluated by the remaining criteria for final classification.

#### 16.4.4 Acid Base Accounting Results

The ABA testwork was evaluated in order to recommend waste classifications for each rock type to be encountered in the open pit and to recommend additional testing for detailed design.

- The Manto Rico Altered Member (9 tests) is classified as PAG, based on the sulphide concentrations.

- The Manto Rico Member (11 tests) is classified as NAG, based on the criteria outlined above. This unit is expected to have significant buffering potential and is located at the bottom of the open pit, making it ideal for a cap rock.

- The Lower Andesite Sill (11 tests) is classified as PAG, based on the sulphide concentrations.

- The Upper Andesite Sill (2 tests) is classified as PAG, based on the sulphur concentrations. This is a very preliminary assessment because there are only two results for this rock unit.

- The Peña Ranch Formation (10 tests) is classified as Uncertain, based on the criteria outlined above. Five of the ten tests have sulphur concentrations that are greater than 0.3% by weight, which suggests that this rock type is more likely to be PAG than NAG.

- The Pitarrilla Formation (16 tests) is classified as PAG, based on the sulphur concentrations.

- The Casas Blancas Formation (13 tests) is classified as Uncertain, based on the criteria outlined above. Only one test has a sulphur concentration above 0.3% by weight, which suggests that this rock type is more likely to be NAG than PAG.

- The Felsic Intrusive (12 tests) is classified as Uncertain, based on the criteria outlined above. Seven of the 12 tests have sulphur concentrations that are greater than 0.3% by weight, which suggests that this rock type is more likely to be PAG than NAG.

The ABA test-work was evaluated in order to recommend waste classifications for each rock type to be encountered in the open pit and to recommend additional testing for detailed design.

As a consequent remedial action to neutralize PAG waste from the mining operations, ANP and AGP values obtained from the geological block model were used to estimate the quantity of calcareous material (limestone) required for acid waste rock neutralization. It was determined that approximately 6 Mt of limestone is required to neutralize the acid waste rock over the mine life (or more if the NPAG value is less effective than limestone). Based on existing SSR drill and mapping data, a preliminary calcium carbonate rich source was subsequently located near the property, and a source pit has been designed.

Detailed engineering will establish the precise limestone pit location and tonnage requirements.

The current dumping plan ensures a life of mine ANP/AGP ratio of 3.7:1. The target ratio for acid waste neutralization is 3.0 in any one dump location; the average is slightly higher due to some inefficiency in timing of materials. Detailed engineering will fine-tune the calcareous material scheduling.

#### 16.4.5 Metal Leaching Results

SRK completed an earlier metal leaching study of a composite waste sample that indicated no metal leaching (source: Pitarrilla SRK Mining Waste Characterization Memo, October 2012).

Results from 34 samples which were taken during the later KPL project show that 23 metal leachability values are below the Mexican Regulation 141-SEMARNANT-2003 and the EPA Regulation 6020A-2007. The remaining 11 samples were not tested because the assay grades were found to be below the limits for initiating metal leaching test-work.

#### 16.4.6 Recommended Additional Waste Dump Plan Work

Price and Errington (1997) suggest that static waste characterization testing be completed on ten samples for every 1 Mt of waste rock for small waste dumps. Price and Errington also recommend that 80 samples be tested for each 10 Mt of excavated rock (ore and waste).

KPL noted that there are no guidelines or recommendations to determine the required sampling frequencies for large waste dumps. Experience suggests that the sampling frequency can be curtailed significantly for large tonnages because the general character of the rock unit is not expected to change significantly over the entire rock mass. Therefore, judgment is to be used when selecting the required number of samples to adequately characterize the waste and it is important to sample large rock units spatially to confirm the general character. Upon reviewing the number of samples and the distribution of those samples within each lithology, as well as the variation of the suggested classifications (NAG, Uncertain or PAG), KPL has recommended that an additional 76 samples undergo static testwork to characterize the waste rock under an interpretation of the Price and Errington recommendations. The program is currently under initiation with SSR.

Kinetic (barrel and humidity cell) testing may follow the additional static results. Metal leaching results have been within the Mexican acceptable guidelines in static testwork to date.

### 16.4.7 Waste Dump Reshaping

As part of the Asset Retirement Obligation ("ARO"), a detailed analysis was completed to determine the cost of rehabilitation of the final waste dumps. To this end, the volume of material to be reformed was estimated and areas requiring surface treatment were estimated.

The waste dumps as described were placed within a 3D surface reshaping program that completes balanced cut and fill calculations for complex 3D surfaces (shown in Figure 16-9).

Pit areas were not re-shaped.

The re-shaping costs were included in the overall ARO estimate.

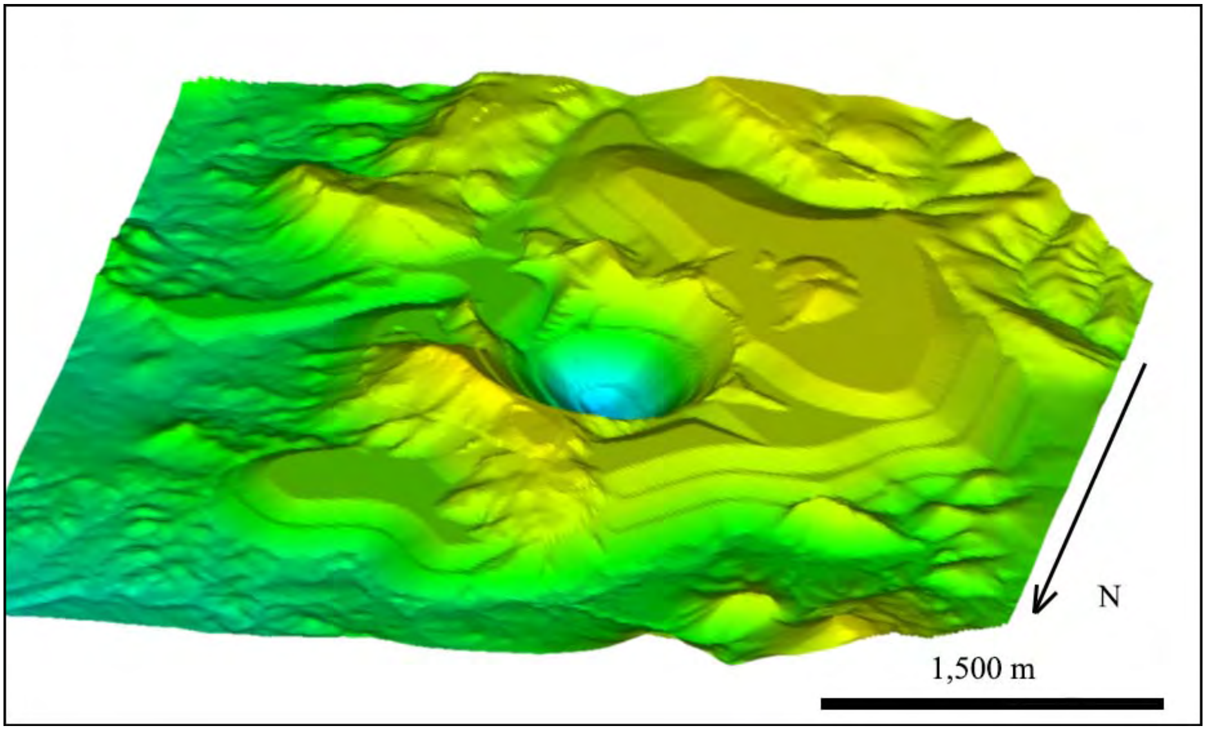



**Figure 16-9: Isometric View of the Final Reshaped Waste Dumps**
Notes: Assuming 2.5H:1V Slope Faces and 3H:1V Overall Final Slope Angles.
The Colors Represent Elevation (blue for deeper, yellow for higher). Source: SSR 2012.

## 16.5 MINING STUDIES

### 16.5.1 Bench Height Analysis

The Pitarrilla Feasibility Study (M3, 2012) uses a 15 m x 15 m x 7.5 m selectivity basis for selection of ore. The size of the SMU smallest mining unit ("SMU") is considered appropriate to the size of equipment considered and should accommodate all mining dilution characteristics.

The selection of the bench height was based upon a bench height optimisation study conducted by SSR in April 2012 and reviewed by IMC. The studies concluded some further optimization could occur but further improvements would be slight.

It was also concluded that the bench height would support the mining rate considered.

### 16.5.2 Milling/Mining Rate Analysis

The feasibility process began at 16,000 tpd under the consideration of milling rate based on the simple to apply Taylor's Rule. This was built upon the grade model available in January 2012 and PEA level costs and recoveries available at the time.

The optimum daily extraction rate = 5 × (expected reserves)$^{3/4}$ ÷ (production d/a), in which expected reserves are generally interpreted to mean proven plus probable reserves. In this case, the resource within the mineable shape was taken as the expected reserve.

The optimum Pitarrilla extraction rate = 5 × (114 Mt)$^{3/4}$ ÷ (350 dpa) = 15,760 tpd.

In January 2012, this moved the project to begin with a rounded throughput of 16,000 tpd.

During the feasibility process, however, many advances to reduce costs, increase recovery and increase tonnages in the resource, as delivered through more drilling and resource estimation method changes, have occurred. Repeating Taylor's Rule on the final feasibility reserve indicates that this method would suggest:

The optimum Pitarrilla extraction rate = 5 × (157 Mt)$^{3/4}$ ÷ (350 d/a) = 20,036 tpd.

Scheduling studies undertaken by SSR to optimize the cashflow and develop achievable mining schedules concluded that mining rates of 190 to 210 ktpd for a 16 ktpd plant were most appropriate. These results were supported from independent results from IMC.

Other schedules were developed by SSR for a 20k tpd plant (with scaled capital costs) and these concluded that the advantage to increasing the milling rate were a minimal advantage to the 16 ktpd option.

It was determined that the project is relatively insensitive to milling rate. It may be possible that increasing the project mill capacity to 20 ktpd represents a project opportunity to increase the

IRR by 0.5% and the NPV (5% discount) by around $60 million. The capital cost for the plant expansion would increase by around $40 million.

Further, increasing milling rates and plant size above 16 ktpd quickly approach the threshold capacity of the forecast 115VA powerline estimated in this study. Due to capacity limitations at the Canatlan substation, and issues which might be created by higher capacity, a 230 VA line would be required for larger plant options. Expansion of the powerline to 230 VA, potentially from the generating plant, would add further capital to the incremental cost of expansion.

### 16.5.3 Crusher Height Analysis

An analysis of the optimal delivery height for crusher feed to the plant was completed by SSR in January 2012. The crusher height range was optimised to minimise ore haulage costs. The selected crusher location sat within the optimal height range and also had appropriate geomechanical properties.

Concepts such as in-pit crushers, ore passes and other mass material movement methods were assessed and rejected as compared to truck haulage.

### 16.5.4 Pit Optimisation

Pit cashflow optimisations were performed on the December 4, 2012 feasibility resource model. The cashflow optimisations were performed using the Minesight Lerchs-Grossman algorithm on value per blocks calculated using a user-written reserve subroutine.

#### 16.5.4.1 Pit Optimisation Costs

IMC developed operating mining costs for a fleet of suitable equipment for the pit optimisation process. The mine operating cost was estimated based on a 7.5 m bench height. The cost estimate was based on earlier pit and waste dump designs and matching haul route profiles provided by SSR.

The operating costs were summarised on a bench-by-bench basis, and a simplified formula was derived to relate overall operating cost to bench height. The majority of the control to operating cost is haulage productivity.

Ore mining = 1.440 + 0.0039/m below 1,800 masl

Waste mining = 1.477 + 0.0039/m below 1,800 masl

Mine re-handle Cost = 1.04/tonne of trucked stockpile re-handle.

Mine re-handle costs were applied to all material identified as low grade. The category of low grade was identified by the VPT of the ore and defined as less than $6/tonne after all costs. This generated approximately 20 Mt of low grade material. It should be noted that the final schedule moved this value to $8/tonne.

The mining operating costs include camp costs, NPAG waste rock mining, and sustaining capital.

M3 Engineering prepared an administration operating cost estimate that was variable, per mill throughput selected. The estimate included sustaining administration capital.

The administration cost includes camp costs for the administration personnel only. Camp costs for the mine personnel are included in the mine operating cost and similarly camp costs have been distributed to the plant for plant personnel.

With no distributions, M3 Engineering has estimated the base administration and complete camp cost to total $2.46/tonne (regardless of type of ore).

For reserve estimation, the administration costs require some modifications to handle the intricacies of process speed and ore type differences. The administration operating cost for the set of mill throughputs is shown in the following Table 16-6:

**Table 16-6: Administration Operating Cost**

| Throughput (tpd) | Admin Cost ($/t) | Crusher Rehandle ($/t) | Tailings Disposal ($/t) | Total Cost ($/t) |
|---|---|---|---|---|
| 12,000 Oxide HG | $1.42 | $0.35 | $0.26 | $2.03 |
| 12,000 Oxide LG | $1.42 | $0.01 | $0.50 | $1.93 |
| 16,000 Sulphide HG | $1.06 | $0.35 | $0.26 | $1.67 |
| 16,000 Sulphide LG | $1.06 | $0.01 | $0.50 | $1.57 |

Adapted from source: Pitarrilla_Opex_12011_10.19.2012.xls

SSR, in combination with Tierra Group International, estimated the TSF costs. Year 1 and 2 were considered capital (Stage 1 construction) costs for the starter dam, while Stages 2 to 5 were considered sustaining capital. Based on a storage capacity of 108.6 Mt for Stages 2 to 5 and a cost of $27.94 m for these stages, the sustaining capital cost for the TSF is estimated as $0.26 /tonne. This cost is not separated for oxide or sulphide processed ore storage.

For tailings disposal above 112 Mt, a cost of $0.50 /tonne was considered based on increasing disposal costs due to increasing containment wall constructions with added height.

The mill operating costs were developed for each major applicable plant section inclusive of camp costs and sustaining capital and are shown in the following Table 16-7:

**Table 16-7: Modified Milling Operating Cost**

| Concept | Plant Cost ($/t) |
|---|---|
| Crushing, grinding, tailings and overheads | 6.94 |
| Oxide additive costs | 7.55 |
| Lead Float | 1.67 |
| Zinc Float | 1.43 |
| Sulphide tailings leach | 4.83 |

For the optimisation process, direct leach process costs are estimated as:

$6.94+$7.55 = $14.49/tonne

The flotation/leach process cost totals as:

$6.94+$1.67+$1.43+$4.83 = $14.87/tonne

16.5.4.2 Pit Optimisation Other Considerations

Rehabilitation costs were not included in the optimisation analysis. The cost of borrowing was not included in the analysis. Capital expenditures prior to commencing the project were not considered.

The optimisation included only measured and indicated ore for derivation of all costs and revenues. Inferred resources form no part of the economic analysis of ore. All inferred resource within the mine design is counted in the waste mass for costs.

16.5.4.3 Pit Optimisation Metal Prices and Costs

Base case metal prices used in the optimisation base case are demonstrated in Table 16-8.

**Table 16-8: Pitarrilla Economic Parameter Assumptions**

| Item | Price |
|---|---|
| Silver Price | $25/oz |
| Lead Price | $0.90/lb ($1,984/t) |
| Zinc Price | $0.95/lb ($2,095/t) |
| Peso:USD Exchange Rate | 12:1 |
| Fuel Price | $0.93/l |
| Inflation | nil |

Concentrate metal cost recommendations were supplied in a report prepared by BMMS in 2012 and the final values used in the optimisation are included in Table 16-9.

**Table 16-9: Scaling Study Metal Costs**

| Item | Doré | Lead Concentrate ($/dmt) | Zinc Concentrate ($/dmt) |
|---|---|---|---|
| Payable Base Metal | | 95% (3% min) | 85% (8% min) |
| Payable Silver | 98.5% | 95% (50g/t min) | 75% (109 g/t min) |
| Treatment | Incl. in RC | $291.04 | $277.70 |
| Silver Refining Charge | $0.60/oz | $1.25/oz (5% of price) | $0.75/oz |
| Penalties | none | $15.00 | $15.00 |
| Shipping (land and sea) | Incl. in RC | $132.00 | $132.00 |

The metal costs were applied within the model on a block-by-block basis. For each block, the concentrate grade was estimated and the precise formula applied for all parameters.

The lead concentrate treatment charge was based on:

$280 /dmt + 6% above a price of $1,800 tonne = 280 + (1984 – 1800) x 0.06 = $291.04

The zinc concentrate treatment charge was based on:

$260 /dmt + 6% above a price of $1,800 tonne = 260 + (2095 – 1800) x 0.06 = $277.70

Refining charges for silver in zinc concentrate were given in a range of $0.50 to $1.00 per ounce. A mid-range charge was chosen at $0.75 /oz.

It is assumed that the concentrate is sold to a market in Asia for a total logistic mine to smelter charge of $120 /wmt. (Assuming 10% moisture content, this yields $132 /dmt.)

Doré metals sales costs were documented in a memorandum completed by BMMS. The report documented several combinations of refining and payable proportions. The doré sales costs used in the optimisation study were for a refining and shipping cost for silver at $0.60 /oz and a 98.5% payable proportion, which in net effect were similar to the average values in the marketing study that averaged $0.40 /oz refining cost and 99.5% payable proportion.

16.5.4.4 Pit Optimisation Metallurgical Recovery

The current model assesses all blocks for both processes and then selects the most profitable process. This requires a three step process in determining the flotation characteristics for a block and then calculating the costs and metal sales values. This value is then placed against the same for the direct leach path.

The direct leach recovery, as determined by the Hot CN sample method, has been estimated into the model where assays exist.

Where Hot CN assays were not available, a formula was used to estimate the recovery based on analysis of values at various oxidation states of the existing assays.

$$\text{Ag Recovery Mean} = 0.0083\ x^3 - 0.0586\ x^2 - 0.0098\ x + 0.5993$$

Where x is the oxidation code in the range 0 to 5.

The Hot CN recovery was trimmed above and below a formula based on the actual recoveries from all raw data. Where the estimated Hot CN recovery was above the maximum, it was reduced to the maximum and where below the minimum was raised to the minimum.

$$\text{Ag recovery max} = 0.0054\ x^4 - 0.0394\ x^3 + 0.0412\ x^2 + 0.0283\ x + 0.7734$$

$$\text{Ag recovery min} = 0.0014\ x^3 + 0.0006\ x^2 - 0.0919\ x + 0.4279$$

The formula is finally modified by the Hot CN to plant recovery factor of 0.984.

In the case of leaching of sulphide tailings, the recovery was set at the minimum overall recovery of either 35% of the tailings grade entering the leach process (i.e. post floatation recovery) or the direct leach recovery of 0.76. The 0.76 factor takes into account the coarser grind (150 µm) and faster leach times.

For floatation recovery, a three step calculation is required for each metal in either of the lead or zinc concentrates.

1. Determine the Rougher floatation recovery (R)
2. Determine the effect of oxidation on rougher recovery (O)
3. Determine the rougher to cleaner recovery (Cl)

All formulas are in the generalised form:

R = A * X2 + B * X + C

O = D * Oxid12 + E * Oxid1 + F
Cl = G * X2 + H * X + J
Recovery = R * O * Cl

Where X is the block grade for Ag, Pb, Zn and Cu, respectively. The parameters A, B and C refer to recovery to rougher; D, E and F refer to oxidation rougher reduction; and G, H and J refer to rougher to cleaner recovery. Oxid1 is the oxidation code in the range 0 to 5.

All grades are trimmed to an appropriate apogee for the function (Maximum Rougher Limit grade). All recoveries or factors are trimmed to be in the range 0 to 1.

Mass pull to the concentrate was determined from the formulas in the same way, but simplified to the grade of the primary metal (lead for lead concentrate and zinc for zinc concentrate).

Rm = 0.01 * X
O = D * Oxid12 + E * Oxid1 +F (parameters taken from grade calculation)
Clm = N * X2 + P * X + Q
Mass Pull = X * Rm * O * Clm

The parameters N, P and Q refer to the rougher to cleaner mass pull. Parameters for lead and zinc concentrates are shown in Table 16-10 and Table 16-11, respectively.

## Table 16-10: Lead Concentrate Parameters

| | | | Parameters for Lead Concentrate | | | |
|---|---|---|---|---|---|---|
| **Item** | | | **Silver** | **Lead** | **Zinc** | **Copper** |
| **Recovery to Rougher** | **A** | $X^2$ | -0.000007 | -0.3869 | -0.02 | -66.563 |
| | **B** | X | 0.00296 | 0.72439 | 0.10063 | 9.9071 |
| | **C** | C | 0.54537 | 0.57804 | 0.09177 | 0.22423 |
| **Maximum Rougher Limit** | | | 185 | 1 | 2 | 0.075 |
| | | | | | | |
| **Oxidation Rougher Reduction** | **D** | $X^2$ | | | | |
| | **E** | X | -0.65 | -0.7 | -0.5426 | -0.5648 |
| | **F** | C | 2.15 | 2.2 | 2.0282 | 2.0114 |
| **Rougher to Cleaner Recovery** | **G** | $X^2$ | -0.000005 | -0.1573 | -0.0105 | 1.6779 |
| | **H** | X | 0.0017 | 0.3187 | 0.0492 | 2.8241 |
| | **J** | C | 0.7945 | 0.7801 | 0.3189 | 0.4673 |
| **Mass Pull to Rougher** | | $X^2$ | | | | |
| | | X | | | | |
| | | C | | 0.01 | | |
| **Mass Pull to Rougher to Cleaner** | **N** | $X^2$ | | | | |
| | **P** | X | | 1.4344 | | |
| | **Q** | C | | 0.0695 | | |

## Table 16-11: Zinc Concentrate Parameters

| | | | Parameters for Zinc Concentrate | | | |
|---|---|---|---|---|---|---|
| **Item** | | | **Silver** | **Lead** | **Zinc** | **Copper** |
| **Recovery to Rougher** | **A** | $X^2$ | -0.000005 | -0.000822 | -0.09945 | -41.48 |
| | **B** | X | 0.000292 | 0.033764 | 0.35278 | 3.6883 |
| | **C** | C | 0.16684 | 0.090293 | 0.54055 | 0.39394 |
| **Maximum Rougher Limit** | | | 100 | 0.8 | 2 | 0.05 |
| | | | | | | |
| **Oxidation Rougher Factor** | **D** | $X^2$ | | | | |
| | **E** | X | -0.34 | -0.335 | -0.3092 | -0.5 |
| | **F** | C | 1.33 | 1.5357 | 1.2587 | 1.4 |
| **Rougher to Cleaner Recovery** | **G** | $X^2$ | -0.00001 | -0.00082 | -0.0146 | -47.945 |
| | **H** | X | 0.0022 | 0.03376 | 0.0835 | 8.4694 |
| | **J** | C | 0.8775 | 0.090293 | 0.8691 | 0.6018 |
| **Mass Pull to Rougher** | | $X^2$ | | | | |
| | | X | | | | |
| | | C | | | 0.01 | |
| **Mass Pull to Rougher to Cleaner** | **N** | $X^2$ | | | | |
| | **P** | X | | | 1.7564 | |
| | **Q** | C | | | 0.12 | |

#### 16.5.4.5 Pit Optimisation and the Influence of Underground Mining

Optimisations were completed that analysed the influence of extraction of ore via underground methods. The base case selected pit shell was insensitive to the presence of underground mining as an economic alternative. This does not preclude the future mining of deeper resources by either open cut or underground mining.

#### 16.5.4.6 Pit Optimisation Design Wall Angles

KPL prefeasibility report recommendations were used for overall slope angles for optimisation.

#### 16.5.4.7 Pit Optimisation Discussion

Pit optimisations were run on the Pitarrilla feasibility block model to generate a series of discounted and undiscounted pit shells.

A series of sensitivity analyses were performed to select the optimal pit shell. Incremental strip ratio and cost per ore tonne profiles were additionally used as part of the optimal pit shell selection criteria. The data used in the following graphs is represented by revenue factor where a revenue factor of 1.0 is the base case price assumption and a revenue factor of 0.9 would be where all prices are multiplied by a factor of 0.9. The cashflow figures demonstrated are before tax and capital but include sustaining capital.

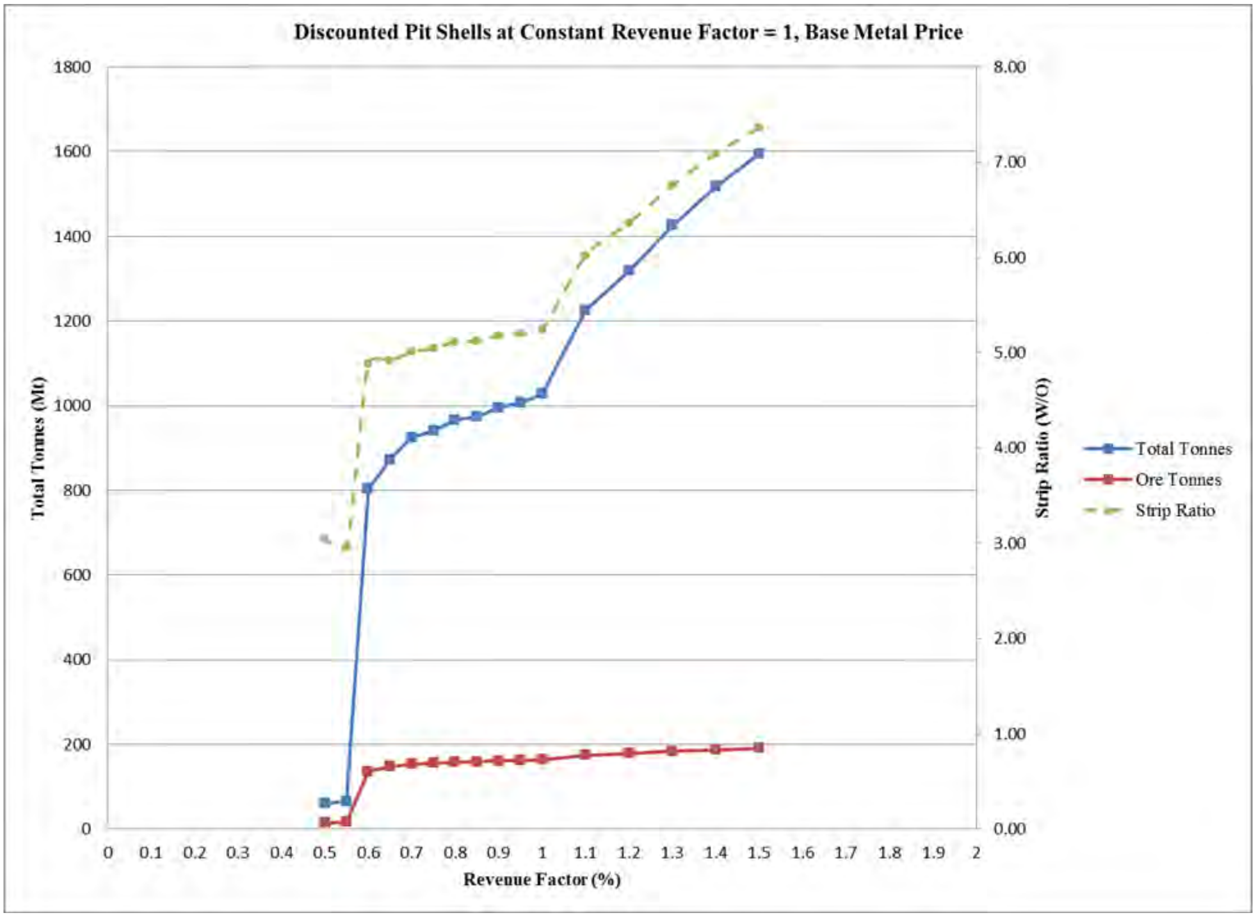



**Figure 16-10: Tonne-Revenue Curves**
Source: M3, 2012

16.5.4.8 Pit Optimisation Undiscounted Pit Shells

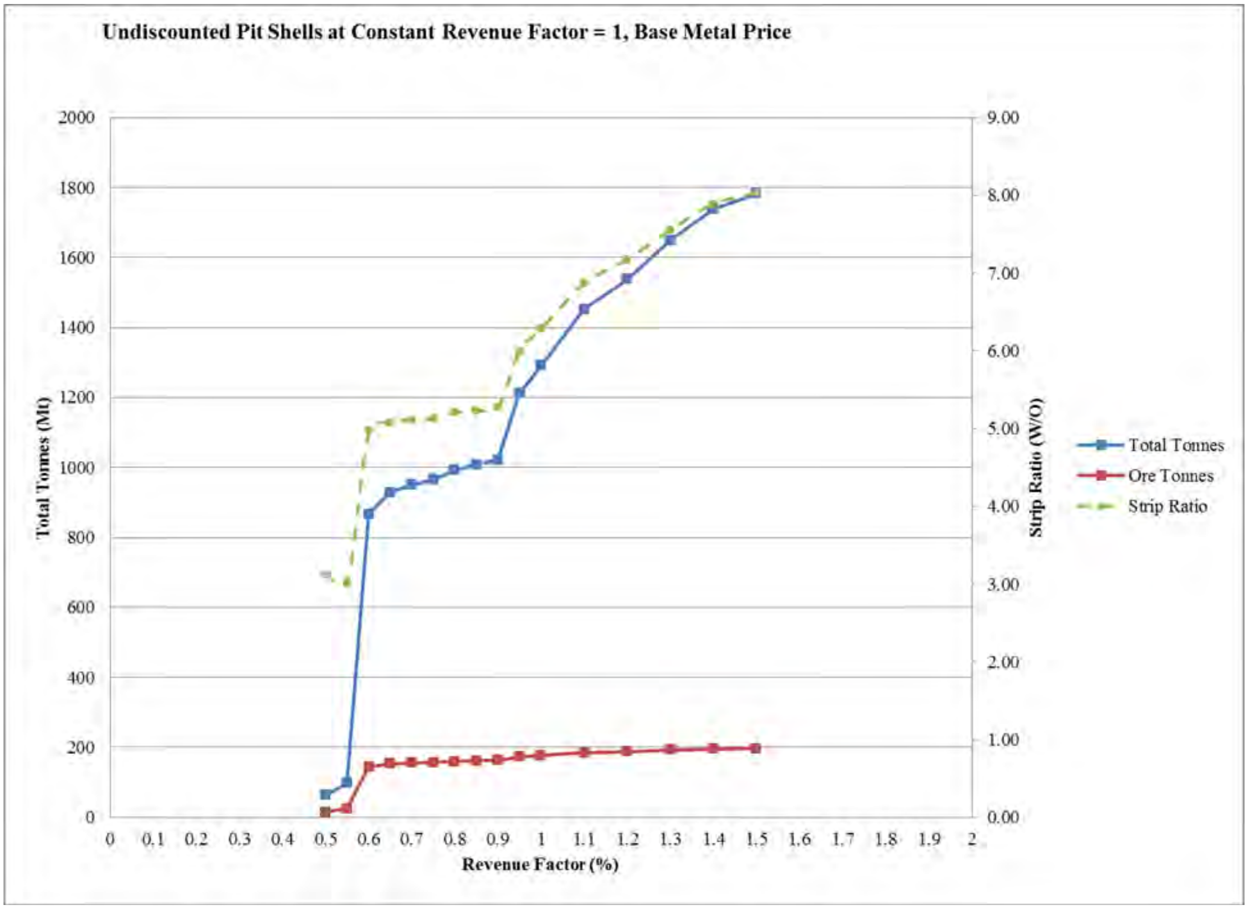



**Figure 16-11: Tonne-Revenue Curves**
Source: M3, 2012

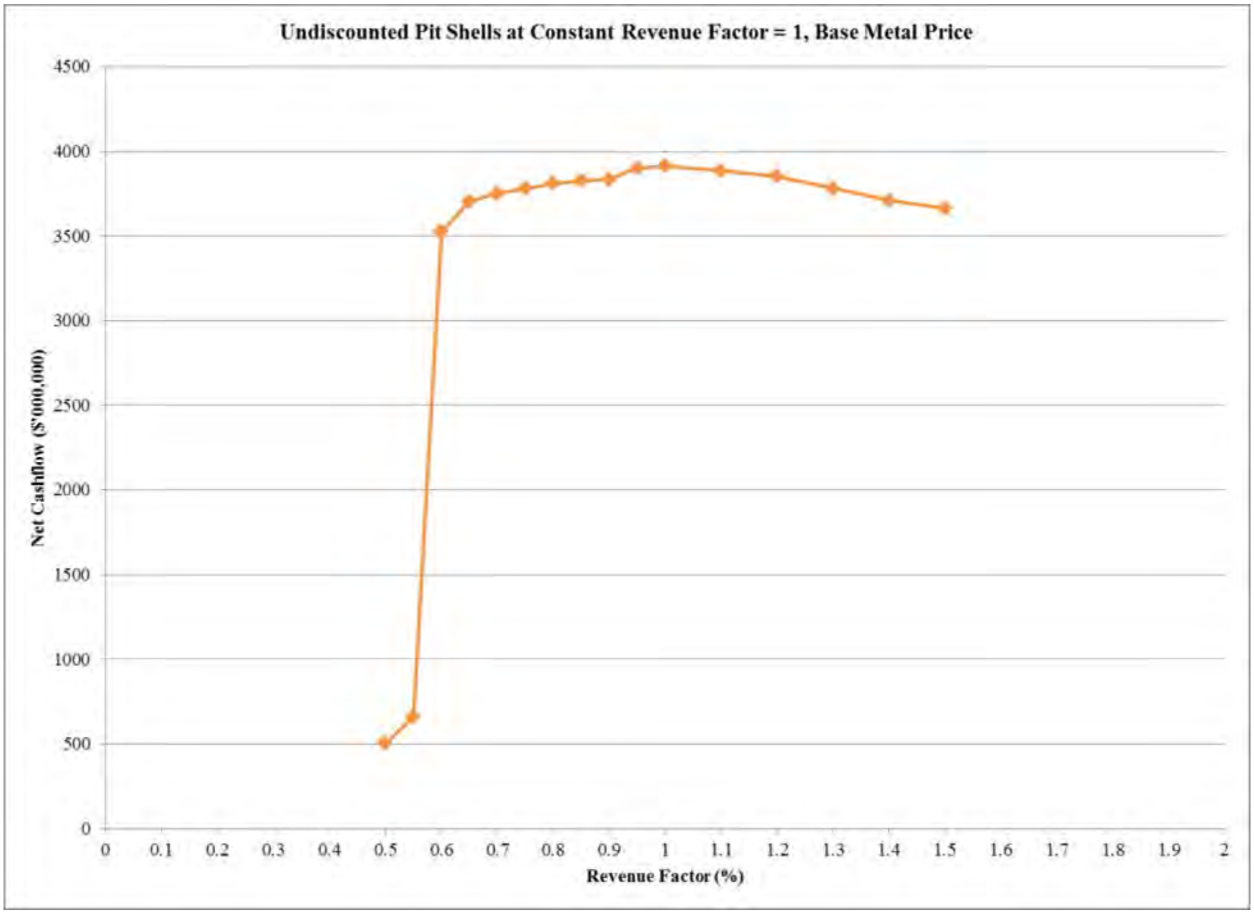



**Figure 16-12: Net Cashflow-Revenue Curve**
Source: M3, 2012

16.5.4.9 Pit Optimisation Sensitivity Analysis

**Table 16-10: Undiscounted Pit Shell Reserves @ 0.9 Revenue Factor**

| Pit Shell # | Total Tonnes | Ore Tonnes | Waste Tonnes | SR | Revenue | Net Cashflow |
|---|---|---|---|---|---|---|
| P19 (base case) | 0% | 0% | 0% | 0% | 0% | 0% |
| P21 (-10% metal price) | -3% | -12% | -1% | 13% | -16% | -31% |
| P17 (+10% metal price) | 26% | 18% | 28% | 8% | 27% | 35% |
| P38 (+10% mining cost) | -1% | 0% | -1% | 0% | 0% | -6% |
| P37 (-10% mining cost) | 19% | 6% | 21% | 14% | 8% | 7% |
| P40 (+10% milling cost) | -1% | -7% | 0% | 8% | -3% | -8% |
| P39 (-10% milling cost) | 19% | 14% | 19% | 5% | 10% | 9% |
| P42 (+10% metal cost) | -1% | -2% | -1% | 1% | -1% | -5% |
| P41 (-10% metal cost) | 18% | 8% | 20% | 12% | 8% | 5% |

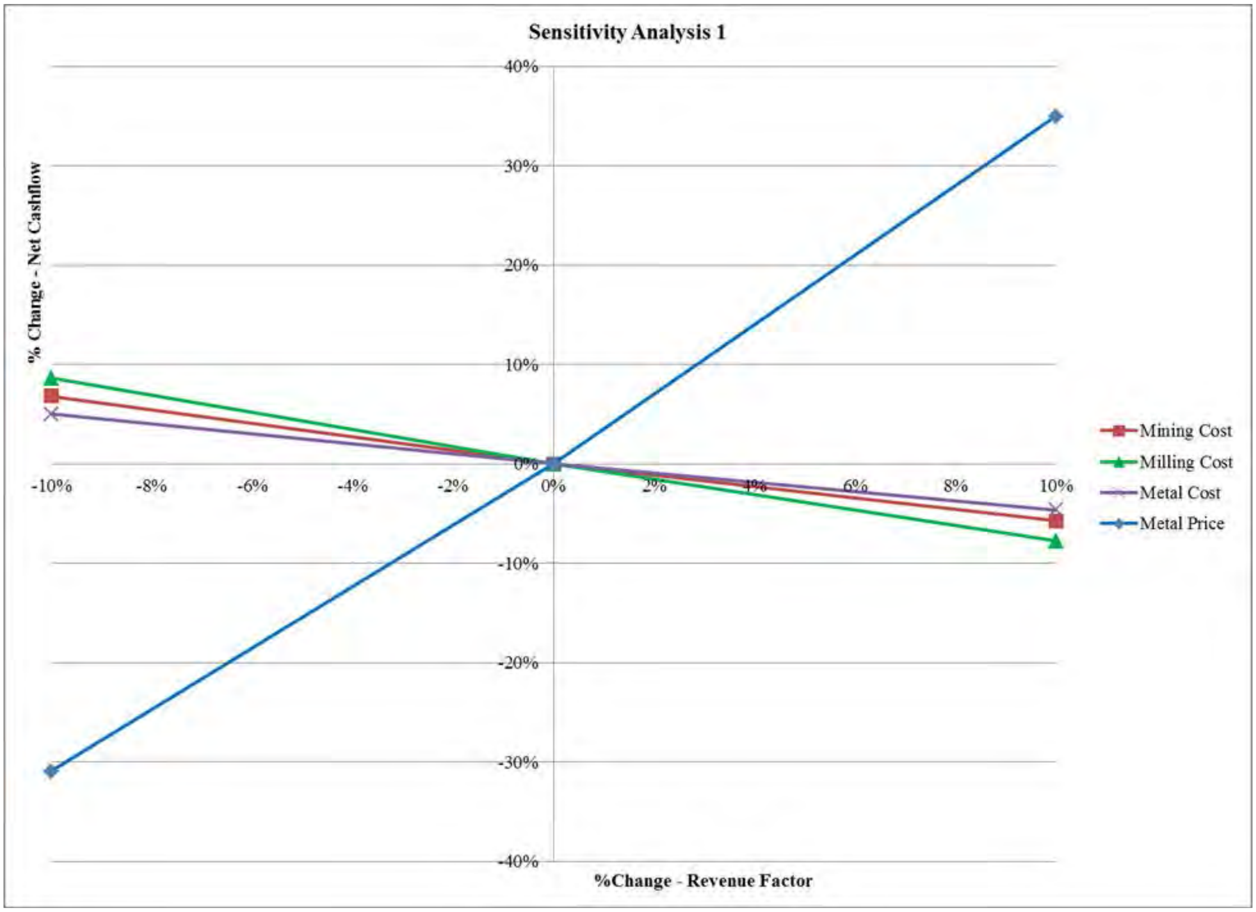



**Figure 16-13: Sensitivities – Metal Price, Mining, Milling and Metal Costs**

Source: M3, 2012

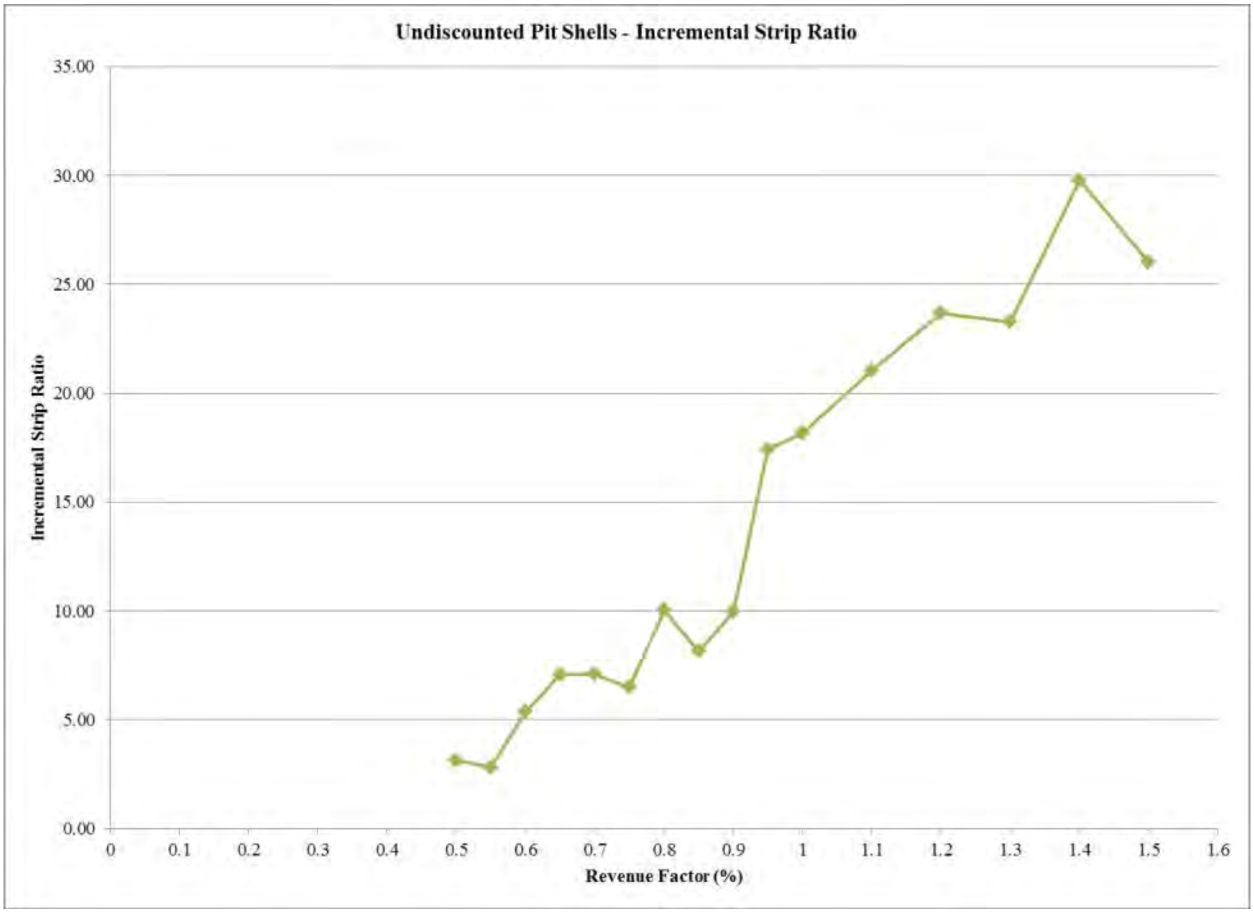



**Figure 16-14: Sensitivities – Incremental Strip Ratio**

Source: M3, 2012

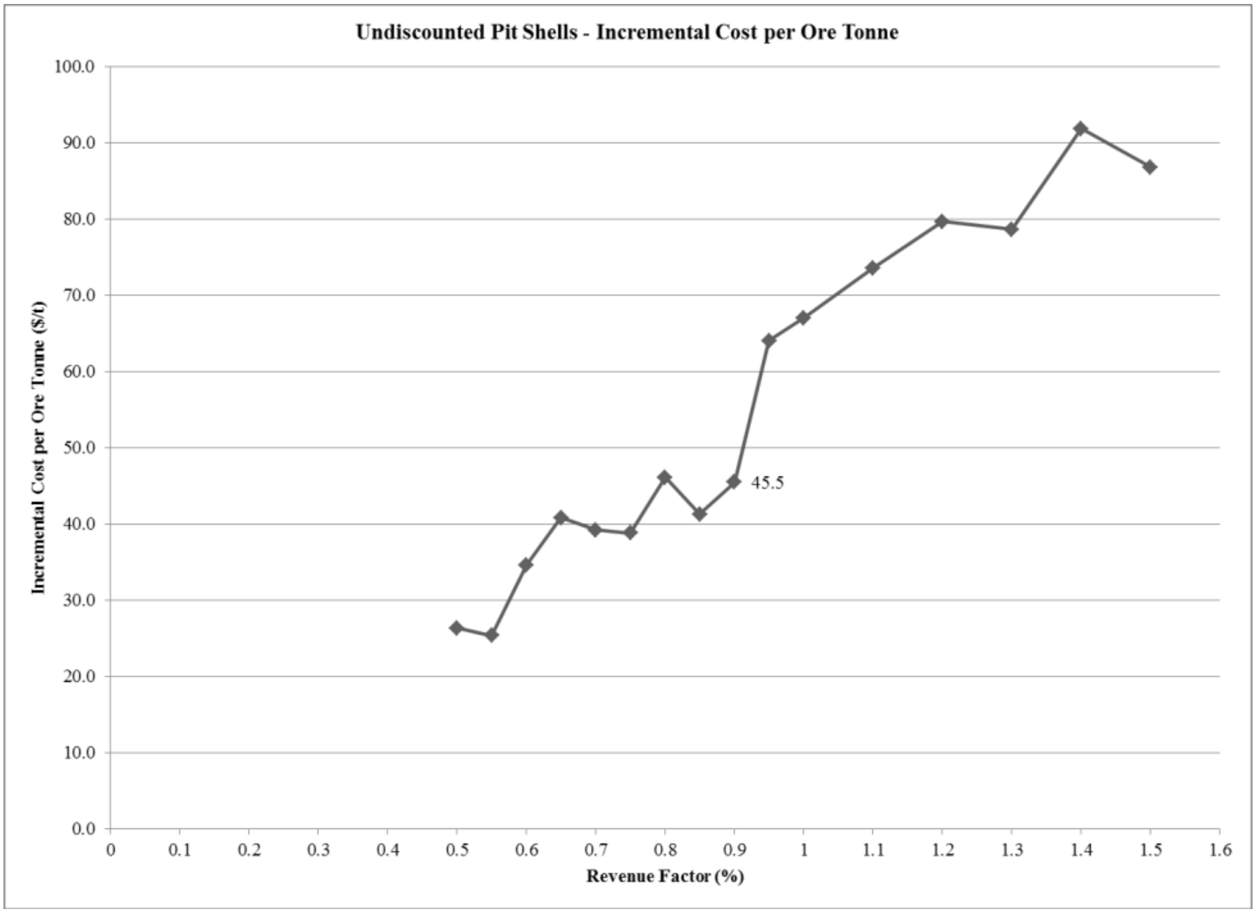



**Figure 16-15: Sensitivities – Incremental Cost per Ore Tonne**

Source: M3, 2012

16.5.4.10 Pit Optimisation Conclusion

Pit optimisations were run to generate both discounted (7.5% discount factor applied) and undiscounted pit shells. The effects of inferred category inclusion, as well as underground mining cost constraint, were considered in the analyses. Incremental strip ratio and cost per ore tonne profiles were additionally used as part of the optimal pit shell selection criteria.

The 0.9 revenue factor undiscounted pit shell and the 1.0 discounted (at 7.5% revenue factor) pit shell perform similarly. Shells larger than this selected point begin to add waste stripping costs above the cost to mine the same ore via underground methods. It is considered that at some future date, possibly more than a decade after commencement, consideration will be given to underground mining or extraction of this deeper ore by open cut methods. This deeper ore is not considered a reserve at this time, but should be considered as a mineable resource.

For the purpose of the Pitarrilla Feasibility Study (M3, 2012), the 0.9 revenue factor (undiscounted) pit shell was selected as the guide for the design of the "ultimate" reserve pit design. This pit shell develops a potential mineral reserve, within the shell, of 162 Mt. After designs were completed, this figure altered slightly to 157 Mt of proven plus probable reserve (delineated in Section 15).

For the interim pit phases, a number of optimisations were completed that were used as design guides. The process to develop the shells was iterative with constraints of minimum mining width, possible depth of development (ramp length), and value.

The optimisations are most sensitive to metal price change than any other factor. Metal price is twice as sensitive as compared to all other operating costs combined.

### 16.6 MINING DEVELOPMENT SCHEDULE

The main critical path item in the development schedule will involve the timely completion of full access to the top of the ore body. It is a key part of the plan to order the first fleet using more readily 100 tonne trucks, a roadbed water truck, a 12.3 $m^3$ bucket size loader, 66.5 tonne dozers and a 37.6 tonne hydraulic excavator. Two of the 100 tonne haul trucks will later be converted to water truck usage and the roadbed water truck will serve in the plant and main access road. The loader will later become the ROM Pad re-handle loader.

At nine months, longer lead time and larger, 150 tonne trucks will begin to arrive. The arrival schedule is designed such that equipment steadily arrives. This will aid the mobile equipment team to evenly distribute their workload.

Also critical will be the commencement date of mining at Cordon Colorado to assure early ore supply.

Declaration of commercial operations is forecast to occur 36 months after the start of the construction decision.

The key milestones and dates for the preproduction development are shown on Table 16-11. Table 16-12 and Table 16-13 show the projects equipment purchase schedule spread over the life of the project.

### Table 16-11: Pitarrilla Milestone Schedule

| Development Milestones | Months from Start of Construction |
|---|---|
| Order First Arrival Mine Fleet | 0 |
| 6 month ordering equipment arrives – 5 x 100 tonne haul trucks and 12.3 $m^3$ front end loader | 7 |
| Begin Borrow Pit A and construction of road to ROM Pad | 7 |
| Begin Crusher Pad mining and building road to Borrow Pit B | 9 |
| First 150 tonne haul truck and 19 $m^3$ front end loader arrives | 10 |
| Begin Borrow Pit B and road to BR1 East and Cordon Colorado | 11 |
| Convert 1 x 100 tonne haul truck to water truck | 12 |
| Begin BR1 East | 13 |
| Begin BR Phase 1 – First ore to crusher pad | 14 |
| Begin limestone mining | 14 |
| First shovel arrives | 15 |
| Begin Cordon Colorado | 15 |
| Limestone to Cordon access road completed | 19 |
| Convert 1 x 100 tonne haul truck to water truck | 20 |
| First ore in crusher | 27 |
| | **Years from Start of Commercial Operation** |
| End Cordon Colorado | 1 |
| Begin Phase 2 | 1 |
| Begin Phase 3 | 4 |
| End Phase 1 | 5 |
| Begin Phase 4 | 7 |
| End Phase 2 | 9 |
| End Phase 3 | 13 |
| End Phase 4 | 17 |
| End re-handle | 29 |

## Table 16-12: Pitarrilla Equipment Arrival Schedule (Months 1 to 36)

| | Description | Units | Year -3:7 | Year -3:8 | Year -3:9 | Year -3:10 | Year -3:11 | Year -3:12 | Year -2:1 | Year -2:2 | Year -2:3 | Year -2:4 | Year -2:5 | Year -2:6 | Year -2:7 | Year -2:8 | Year -2:9 | Year -2:10 | Year -2:11 | Year -2:12 | Year -1:1 | Year -1:2 | Year -1:3 | Year -1:4 | Year -1:5 | Year -1:6 | Year -1:7 | Year -1:8 | Year -1:9 | Year -1:10 | Year -1:11 | Year -1:12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | No.Days | days | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 |
| Number of Units Purchased | | pre-existing | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| | Production Drill rig | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 1 | 0 | 0 | 1 | 0 | 0 | 1 | 1 | 0 | 1 | 0 | 1 | 0 | 0 |
| | Pre-split Drill Rig | 0 | 0 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Pit Shovel | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 1 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Pit FEL | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | 150t Haul Trucks | 0 | 0 | 0 | 0 | 0 | 1 | 1 | 0 | 1 | 1 | 1 | 0 | 0 | 2 | 2 | 1 | 2 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 | 0 | 0 |
| | 100t Haul Trucks | 0 | 3 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | ROM FEL | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Scaler | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Water Truck | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Dozers | 1 | 1 | 1 | 1 | 0 | 0 | 1 | 1 | 1 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Graders | 0 | 1 | 1 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Operated Ancillary | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Non-Operated Ancillary | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |

## Table 16-13: Pitarrilla Equipment Arrival Schedule (Production Years 1 to 30)

| | Description | Units | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 | Year 11 | Year 12 | Year 13 | Year 14 | Year 15 | Year 16 | Year 17 | Year 18 | Year 19 | Year 20 | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | No.Days | days | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 |
| Number of Units Purchased | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| | Production Drill rig | | 0 | 2 | 0 | 0 | 0 | 1 | 1 | 4 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Pre-split Drill Rig | | 0 | 0 | 0 | 0 | 0 | 2 | 1 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Pit Shovel | | 0 | 0 | 0 | 0 | 0 | 0 | 2 | 2 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Pit FEL | | 0 | 0 | 0 | 0 | 0 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 |
| | 150tHaul Trucks | | 0 | 0 | 0 | 4 | 0 | 3 | 12 | 9 | 0 | 0 | 0 | 0 | 0 | 0 | 7 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 4 | 0 | 0 | 0 | 0 | 0 | 0 |
| | 100t Haul Trucks | | 0 | 0 | 0 | 0 | 0 | 3 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 3 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | ROM FEL | | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Scaler | | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Water Truck | | 0 | 0 | 0 | 0 | 0 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Dozers | | 0 | 0 | 0 | 0 | 1 | 4 | 4 | 0 | 0 | 0 | 0 | 0 | 0 | 2 | 2 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Graders | | 0 | 0 | 0 | 0 | 0 | 2 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 2 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Operated Ancillary | | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0.3 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Non-Operated Ancillary | | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0.3 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |

## 16.7 PRODUCTION SCHEDULE

The production schedule for the Pitarrilla Project was completed using spreadsheet analysis, searching for maximization of NPV on a production based reporting basis, through iteration of mining rate. In addition to primary pit operations, mining of a limestone quarry was completed to allow filling of the same locations later in the mine schedule. Material was directed on a whole of dump basis to achieve the best outcome for the total dump to a better than 3:1 ANP:AGP ratio. The annual production rate is presented in Figure 16-16.

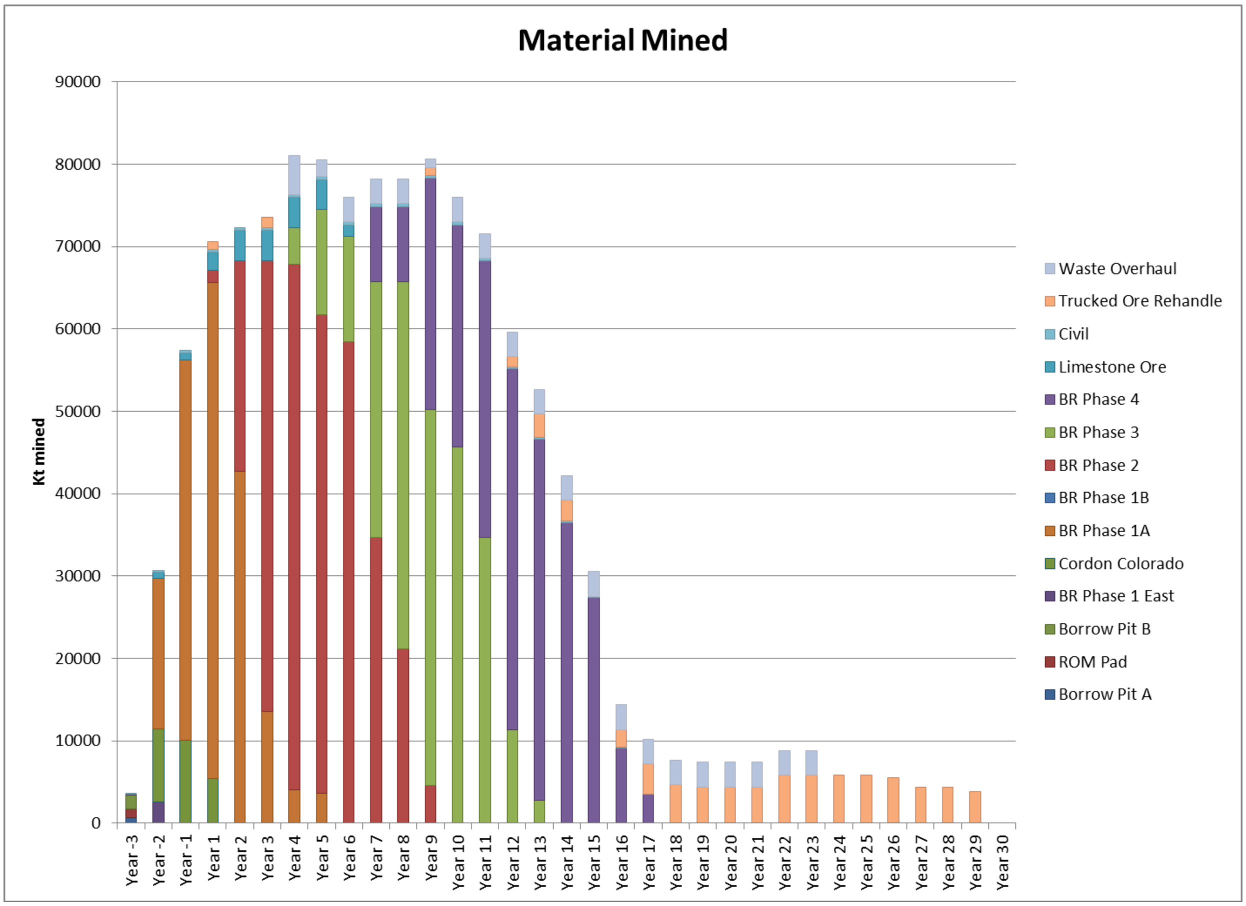



**Figure 16-16: Annual Total Production Rate**
Source: M3, 2012

## 16.8 DRILLING AND BLASTING

The current mine plan assumes that drilling and blasting will be performed on all material moved. Mining will occur in part on 15 m benches (100% waste zones) and 7.5 m benches (mixed ore and waste zones).

The input variables on the blast design include the explosive density, blast-hole diameter, material density, bench height, blast-hole diameter/burden ratio, spacing/burden ratio, sub-drill/burden ratio, and stemming/burden ratio. The target was to achieve 0.17 kg/t and 0.13 kg/t powder factor for the 7.5 m and 15 m blasting, respectively.

In selecting the drilling equipment to deploy for the blasthole drilling operations, the following considerations were made:

- Rock mass characteristics;
- Bench height; and
- Blast-hole pattern sizes (which are also dependent on rock mass characteristics).

Upon considering variable technical specifications on a number of blast-hole drilling rigs on the market, two rig types were selected for the Pitarrilla Feasibility Study (M3, 2012): a production rig with a 9 m rod and a smaller versatile rig for 89 mm to 140 mm diameter holes for pre-splits, special shots and as back-up to production drilling.

Pre-splitting of the high-wall will be performed on every 15 m bench. This will assure good wall conditions and minimize the potential of wall failures (this is a recommendation of the KPL prefeasibility geomechanical study, 2012). A new crest and catch berm will be formed every 15 m.

The drill penetration rates selected in the Pitarrilla Feasibility Study (M3, 2012) are 31.6 m/oph and 22.5 m/oph for 7.5 m and 15 m benches, respectively. These rates are based on rock conditions expected at the mine based on measurements of UCS. UCS testing results undertaken on the deposit indicated a well distributed range of values from 25 to 90 MPa. A conservative UCS value of 100 MPa was used in estimating the drill penetration rates for the Property.

Drill consumable consumption rates are based on moderate to low abrasion levels. Ai tests of the rocks at the Property indicate an Ai average of 0.27, which is between the lower and second lower quartile range of all deposits tested. The range indicates that some material at the Property varies from very low abrasion and upwards to moderate-hard. This will create some local scheduling variance worth investigating during detailed engineering.

Mechanical availability of the drilling fleet is estimated at 75% over the mine life.

The drilling and blasting operations would require 13 drill rigs during the peak of the mine life. This would comprise ten production rigs and three pre-split drill rigs. These estimates include a provision made for equipment rebuilds and replacements.

Drilling labour will include up to 28 drill operators at the peak of the operation with an average of 18 over the mine life. Supervision will be supplied through general mine administration and the blasting foreman.

## 16.9 BLASTING

For the bench heights selected, blasting parameters were determined based on industry experience values, major control parameters (such as UCS rock strength) and current information available for similar rock types in other operating mines in the region. It is recognised that further blast pattern adjustments will occur during detailed engineering and the initial production phases according to local rock mass characteristics encountered.

The groundwater conditions incorporated into the Liquid-Equilibrium analyses undertaken by KPL assume that all mining within the relatively shallow South Ridge and Cordon Colorado areas will be above the water table. The assessments were based on the data collected during KPL site investigations and were predicted using the MODFLOW pit inflow model (USGS, 2000) and confirmed using SEEP/W® (Geo-Slope International, 2004). The water table is estimated to be around 350 m deep below surface.

On the above basis, and with an estimated water table at 1691 masl, it is further estimated that approximately 15% of the total material to be mined could be wet. Aside from rainfall, potentially wet benches may occur during the mining of Breccia Ridge phases 3 and 4. Ground water ingress to the pit is estimated between 10 and 15 L/s (KPL, Draft Memo: VA12-01054 Update to Preliminary Estimate of Groundwater Inflow to Pit). In-pit pumping is included in the cost estimate and is expected to reduce the amount of wet-holes requiring emulsion to 5% of the total holes below 1691 masl (adapted from KPL, Oxide-Sulphide Open Pit Slope design, NB201-227/18-1 Rev 0).

Stemming material will be used to fill between the explosive charge and the collar of each blast-hole to confine the explosive gases. This material will consist of screened mined waste to be loaded by a contractor onto 20 tonne trucks and dumped at specified areas near the shots.

With regards to explosives, the mine plan estimates the following:

- 30 tpd of Ammonium Nitrate Fuel Oil (“ANFO”), or Slurry + ANFO combined;
- The mine operation will load and shoot between 300 and 350 holes per day; and
- The mine will receive one or two tanker loads of Ammonium Nitrate prills every day.

Annual explosive consumptions for all explosive components were completed.

On the above basis, a storage capacity of 240 tonnes on site has been estimated with matching ancillary magazines. Plans for the explosive storage facility were developed by M3 Engineering. The drawing consists of two structures for storing explosives and detonators separately. There will also be an ANFO storage tower.

## 16.10 LOADING OPERATIONS

Primary loading operations will be performed with a maximum of four 21 $m^3$ hydraulic shovels and two 19 m3 capacity bucket front end wheel loaders. A single 12.3 $m^3$ capacity bucket front end loader will be deployed at the ROM Pad for crusher feeding and for occasional use with the auxiliary 100 tonne trucks when they are not being loaded by excavator. There will be one 37.6 tonne hydraulic excavator and one 84.9 tonne hydraulic excavator to load up the auxiliary 100 tonne trucks.

Life of mine mechanical availabilities of the pit shovels and ROM loader are estimated at 82%, while that of the pit loaders are estimated at 85%.

Digging faces will be defined by ore control procedures and will be marked in the field and on maps that will be provided to the loading operators.

### 16.11 HAULING OPERATIONS

Primary hauling operations of ore and waste will be performed with a maximum of 28 x 150 tonne haul trucks, which will be loaded by 21 $m^3$ hydraulic shovels and 19 $m^3$ capacity bucket front end wheel loaders. There will also be a maximum of five auxiliary 100 tonne trucks, which will be loaded by one 37.6 tonne hydraulic excavator and one 84.9 tonne hydraulic excavator and occasionally by a 12.3 $m^3$ capacity bucket front end loader. The five 100 tonne trucks will be acquired at the start of the pre-production period. Two 100 tonne trucks will be converted to water trucks, after the 150 tonne trucks begin to arrive. The remaining three 100 tonne trucks will perform all the civil works as well as provide support in the pit haulage operations. The 100 tonne trucks are matched to the 84.9 tonne hydraulic excavator for pit wall clean-ups and will sometimes be matched to the 12.3 $m^3$ front end loader. The hauling equipment was carefully selected to match with the capacity of the loading units selected for efficient hauling cycles.

Haul profiles were generated and submitted to IMC for productivity simulations. Generation of these profiles was done using MineSight mine planning software (version 7.0-3). These profiles were then entered into a spreadsheet provided by IMC.

Using the haul profile data provided, IMC ran productivity simulations using IMC's proprietary software package (comparable to the methodologies used in vendor simulation packages such as the CAT Vsim), and presented the results to SSR. The resulting productivity/depth graph was used in the Pitarrilla Feasibility Study (M3, 2012), adjusting the values with each pit phase exit.

Approximately 1,170 Mt of material will be hauled by the trucks during the mine life inclusive of civil works materials and re-handle. The amount of material moved in civil works is estimated at 0.5% of total material movement based on experience gained from similar open cut operations.

An average of 48 personnel will be required per period over the life of mine for the haulage operations, rising to 98 during the peak periods and diminishing to four at the start and end of the mine life.

Life of mine mechanical availabilities of the haul trucks are estimated at 87% for both truck types.

As a remedial action to neutralize PAG waste from the mining operations, ANP and AGP values obtained from the geological block model were used to estimate the quantity of calcareous material (limestone) required for acid waste rock neutralization. It was determined that approximately 6 Mt of external calcareous material is required to neutralize the acid waste rock over the mine life. Based on existing SSR drill and mapping data, a preliminary calcium carbonate rich source was subsequently located near the property, and a source pit has been designed.

Productivities used are based on hauling the waste to designated waste dumps and hauling the calcareous material to the classified PAG waste dumps. Detailed engineering will fine-tune the above numbers by establishing the precise limestone pit location and tonnes.

The current dumping plan ensures a life of mine ANP/AGP ratio of 3.7. The target ratio for acid waste neutralization is 3.0 in any one dump location; the average is slightly higher due to some inefficiency in timing of materials. Detailed engineering will also fine-tune the calcareous material scheduling.

Dump progression stages of the above waste dumps at the end of Year 1, 5, 9, 13 and 17 are presented in Figure 16-17 to Figure 16-21. These were the dump progression plans used for the haulage optimisation in the Pitarrilla Feasibility Study (M3, 2012).

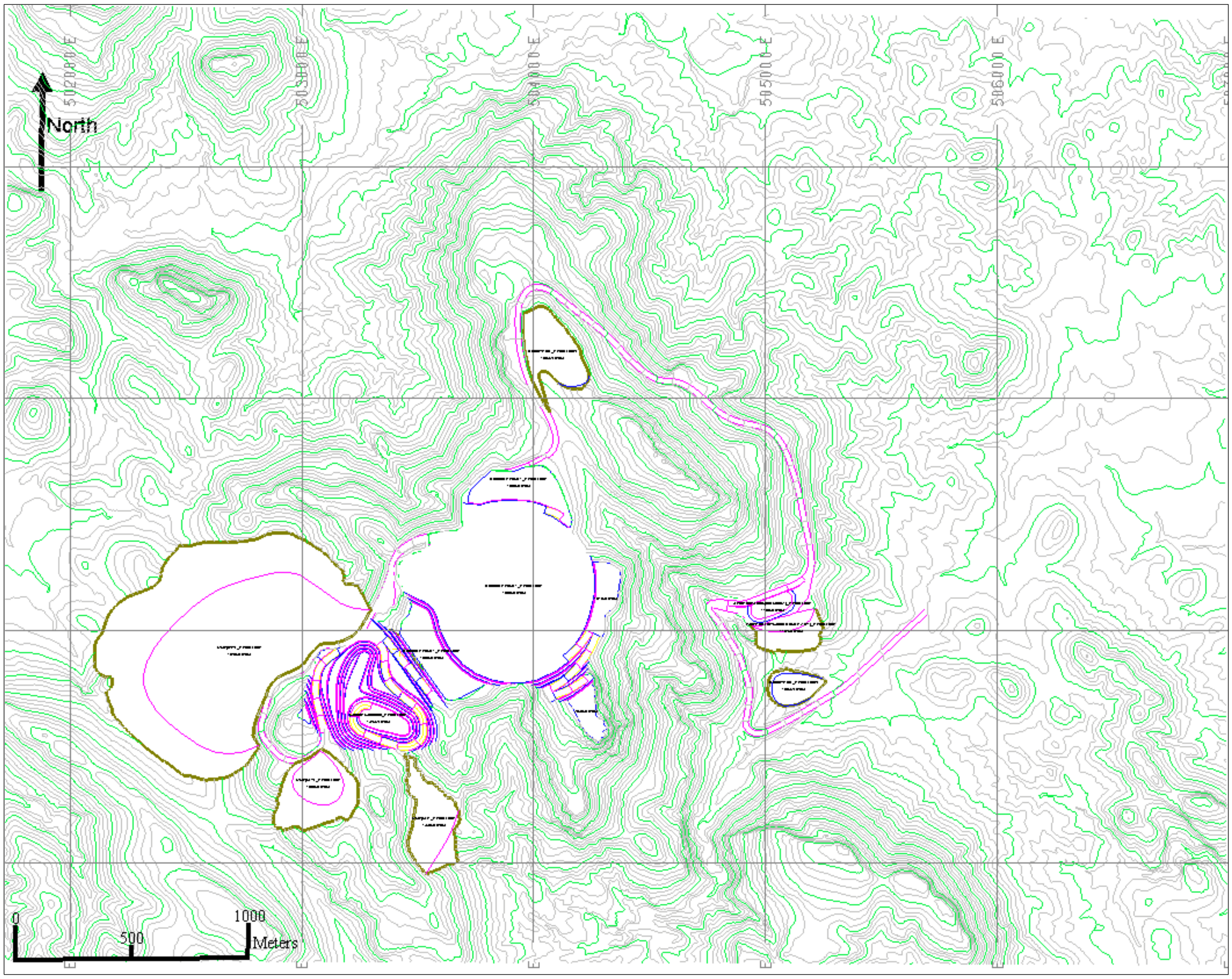



**Figure 16-17: End of Production Year 1 Dump Plan**
NAD 27 UTM Zone 13N Source: M3, 2012

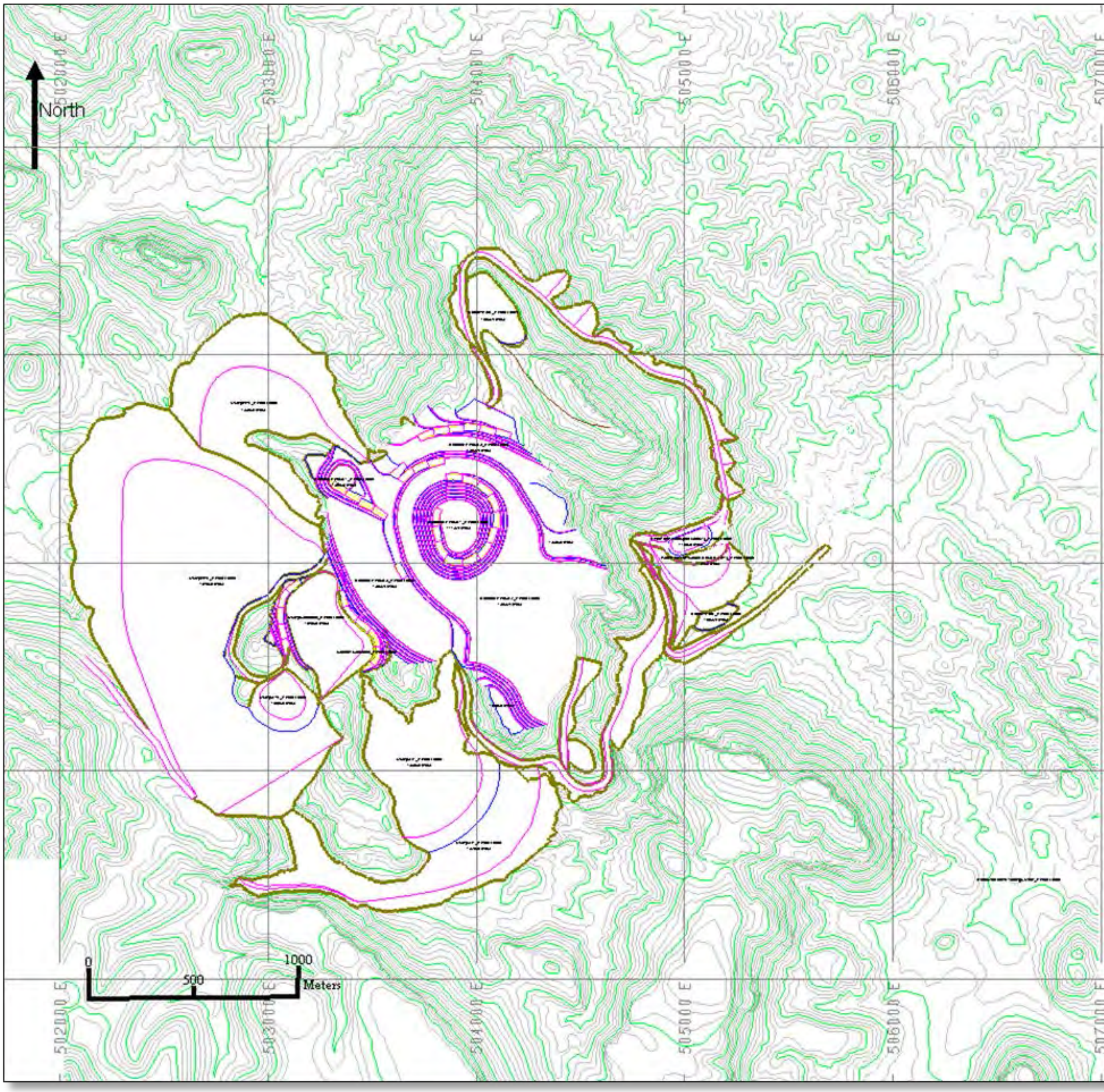



**Figure 16-18: End of Production Year 5 Dump Plan**
NAD 27 UTM Zone 13N Source: M3, 2012

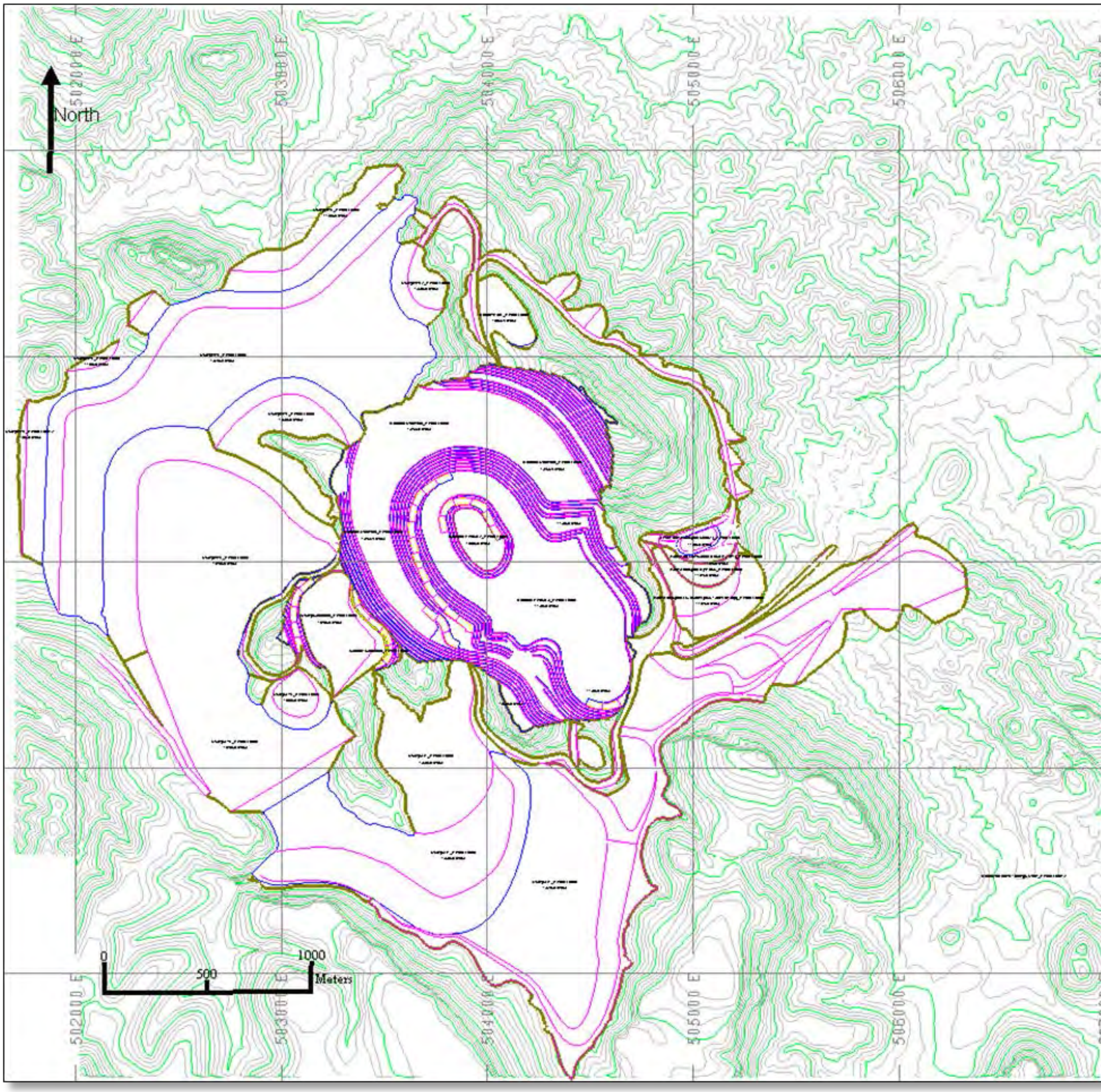



**Figure 16-19: End of Production Year 9 Dump Plan**
NAD 27 UTM Zone 13N Source: M32012

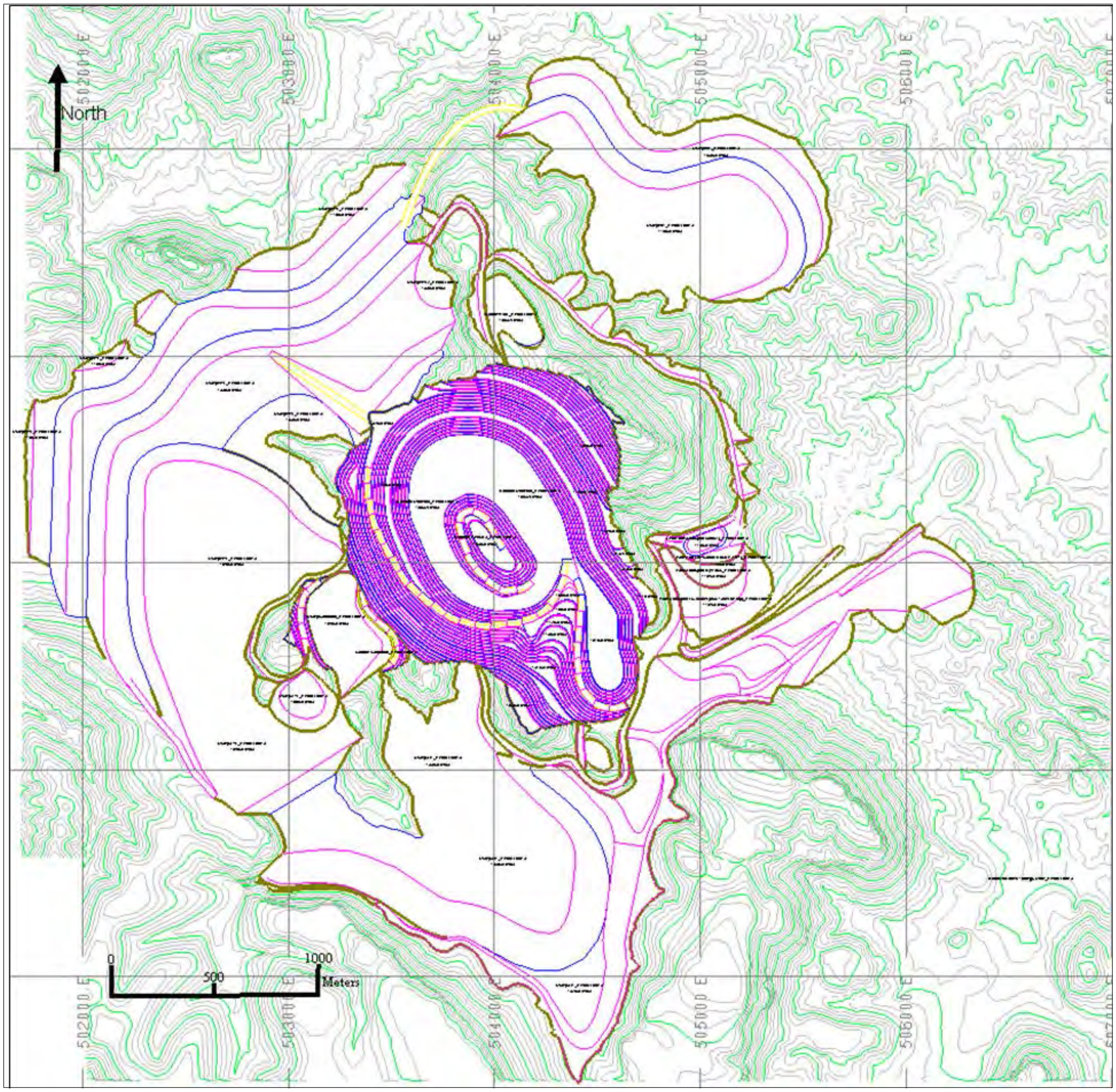



**Figure 16-20: End of Production Year 13 Dump Plan**
NAD 27 UTM Zone 13N Source: M3, 2012

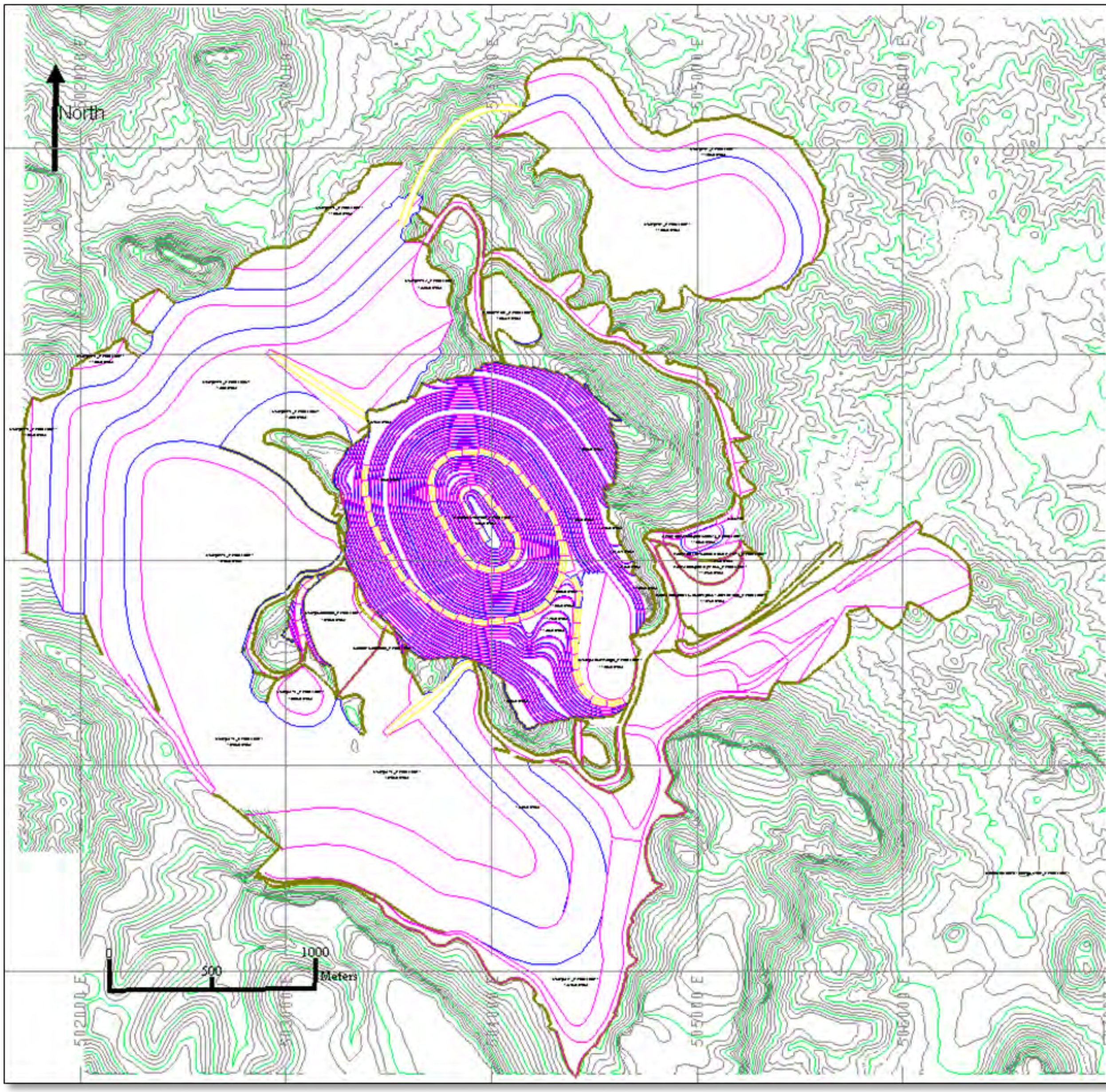



**Figure 16-21: End of Production Year 17 Dump Plan**
NAD 27 UTM Zone 13N Source:M3, 2012

## 16.12 MINE SUPPORT

Mine support functions will be performed with a diversified number and types of equipment during the entire life of the project. These will include a scaler, water trucks, dozers, graders as well as operated ancillary and non-operated ancillary equipment.

An average of 30 personnel will be required per period over the life of mine for the mine support operations, rising to 55 during the peak periods and diminishing to one at the end of the mine life.

The goal is to direct tip ROM ore to the crusher. However, at times, due to two types of ore being mined and processed plus crusher maintenance periods, cross conflicts of mining and processing of the ore types will dictate some material being directed to a temporary stockpile. A 12.3 $m^3$ front end loader has been allocated to ore re-handle on a 50% utilization basis. The ROM pad area is designed with capacity for a minimum of 0.3 Mt short term storage capacity representing three weeks full milling capacity. The mine plan also calls for some occasional long term stockpile re-handle which will be completed with either the 12.3 $m^3$ front end loader and 100 tonne auxiliary fleet or 19 $m^3$ front end loader and the 150 tonne main fleet, depending on availability of equipment at the time.

## 16.13 MINE MAINTENANCE

Mine maintenance is an integral function of the mining operations and relates to the day-to-day upkeep of the mining equipment. Activities such as preventive maintenance, equipment rebuilds and fixing equipment on breakdowns are all included in the mine maintenance function. The objective is to provide efficient maintenance of the mining fleet, thereby increasing reliability and availability of this equipment through effective strategies, planning and continuous improvement. High levels of equipment availability and reliability facilitate operational and delivery performance, resulting in asset intensity reduction and reduced direct operational and maintenance costs.

At peak production, the mine maintenance labour strength is expected to consist of 145 personnel.

### 16.13.1 Mobile Units

The mobile maintenance equipment fleet will comprise two 15,000 litres fuel/lube trucks, one (8 to 10 tonne) flatbed truck, one (8 to 10 tonne) crane, a tire handler to handle 150/100 tonne truck tires, four mechanics trucks, one welding truck, one workshop forklift, one RT forklift and a 75 tonne tractor and lowboy. The tractor and lowboy will have the ability to easily move to the mining equipment longer distances for maintenance work required. The fuel/lube truck will be used to service the drills, shovels and dozers at the working faces in or near the pit.

### 16.13.2 Main Shop

The main shop will be located next to the plant site. The shop is designed to hold six trucks at any one time for service or major repair. The shop will have three major maintenance bays (two trucks per bay) equipped with an eight to ten tonne crane, lube bay, wash bay, and tire bay. The maintenance bays are designed to provide ample height for equipment repair.

### 16.13.3 Communication System and Truck Management System

Communication and information systems will provide for high-quality voice and data transmission throughout the mine site.

The telephone utility, Telmex, will supply telephone services for the Pitarrilla Project mine site. The Pitarrilla Project telephone system will be tied into the Telmex system via a fiber optic cable communications path on the 115 kV transmission line's overhead ground wire from the CFE Nuevo Ideal substation to the Pitarrilla Project mine site's main substation. Cellular phones and radios (handheld/mobile/base stations) with dedicated frequencies and a repeater pair will also be provided for remote operations throughout the mine site.

A new, high-capacity, wide area network ("WAN") with either a satellite internet connection or the above high-speed fiber optic connection will be utilised for broadband data communications. A local area network ("LAN") at site will have sufficient capacity for use by operations, maintenance, engineering, and administration personnel. Computers connected to the LAN will be able to transfer information, use centralised server systems, and be linked for email, voice and data transfer internally via the LAN, and externally via a WAN connection (M3, Draft Feasibility Study – Volume II, March 2011).

The size of the mining fleet planned for the Pitarrilla Project mining operations calls for an incorporation of a fleet management system to ensure optimised mine control and quality of the development phase. The objective is to achieve best utilization of the mining fleet and enhance efficiencies with the shift plan. As part of the Pitarrilla Feasibility Study (M3, 2012), a preliminary assessment of the industry's fleet management systems on the market was undertaken. Detailed engineering will include a trade-off study on the three major vendor contenders to ensure the best and most optimal system is selected.

### 16.13.4 Fuel Storage and Distribution

Diesel supply to the crawler mounted equipment (e.g. drills, shovels, dozers) will normally be provided by two 15,000 litre fuel/lube trucks. All haul trucks and other wheeled equipment will normally receive fuel at the designated fuel bay. There will be two main fuel stations that will comprise one haul truck fuel station (designated for the haul trucks) and a light vehicle fuel station.

## 16.14 MINE GENERAL AND ADMINISTRATION

Mine General and Administration ("G&A") relates to all the day-to-day supervision and engineering support of mining operation activity. Expenses included in the mine G&A are mine labour charges, mine dewatering, mine office supplies, safety supplies, light vehicles, diesel consumption, laboratory charges, software upgrades and support contracts, consultants and external services, education/seminars/scholarships, travel expenses, distributed human resource charges (for all mining personnel) and miscellaneous pit costs.

At peak production, the total labour strength for mine G&A is expected to be 65 and reflects production levels at those periods.

All mine personnel will be transported from the mine camp to work areas using 4 x 4 light vehicles. All materials will be transported utilizing a 20 tonne flatbed truck with telescopic crane or by light vehicle.

### 16.14.1 Grade Control

Blast-hole sampling will be used to define ore zones. One sample will be taken per hole on a 4 m x 5 m pattern on the 7.5 m high benches and will exclude sub-drill sampling. It is estimated that 15% of the material produced during drilling will be captured using a cyclone splitter to return a sample mass of between 10 to 15 kg.

A maximum of twenty samplers will be required at peak production to carry out sampling of grade control drilled holes in the pit. This number reduces to two during the pre-production months and towards the end of the mine life.

The samples will be bagged, labeled, and taken to the on-site laboratory where they will be analysed for Ag, Pb, Zn and Cu using AAS analysis. Sampling for S and Ca will also be completed for PAG/NPAG control in waste on a less intense basis than for ore definition. Methodology for flotation performance is yet to be developed and is currently based on field logging of oxidation state. This will be the minimum requirement for operation performance. Other spectrographic responses for process automation are currently under review.

Results of the grade control drilling will be modeled using a mine planning software. This information will be used by the grade control geologists to define material type boundaries and for extraction planning. The resulting ore mark-up plans produced by the grade control Geological team will be available to the Mining Engineers and Operations personnel on a daily basis.

### 16.14.2 Mine Dewatering

To estimate the cost for mine dewatering, groundwater pit inflow was first determined. Preliminary calculations of groundwater inflows into the open pit were based on information from site investigations conducted by KPL in 2008, 2010 and 2012 and an existing hydrogeological conceptual model developed by KPL (2010a)

Considering the climate and context at the Pitarrilla Project, a lower bound of 10 L/s is more conservative for water balance calculations. An upper bound value of 15 L/s is more appropriate for design of pit dewatering systems at the final stage of pit development (KPL, 2012e)

The above submission from KPL served as guidance in determining the mine dewatering requirements. Calculations for sizing of pump capacity were completed in detail and include provision for rain water removal, with a maximum lowest level of pit disturbance of no more than four weeks. Pit pumps will be trailer-mounted diesel units, for flexibility.

The current plan assumes that mine dewatering will continue beyond actual pit operations through to the end of the stockpile re-handle years.

### 16.14.3 Mine Safety

The mine will have two ambulances and a fire fighting unit available. The mine will retain two doctors and two paramedics stationed on site.

There will be two mine rescue teams who will be trained to competently assess accident conditions and be able to properly fight a fire. The mine rescue teams will be trained to effectively participate in a recovery operation after a mine disaster and the paramedics will actively participate in these teams.

## 16.15 MINE PERSONNEL

Direct manpower requirements were determined using first principles-based productivity estimates for all activities.

Base labour rates were developed by SSR, using a combination of industry wage surveys and experience from other projects in Mexico.

Crew size and composition was developed specifically for the Pitarrilla Project mining operations to suit the project mine plan and development schedule. Indirect staffing requirements for management and technical personnel were based on an organization chart provided by SSR. Maintenance manpower requirements were determined using productivity factors based on the amounts of mobile equipment utilised. An allocation of an additional 8% was applied to direct hourly and maintenance manpower to allow for vacation, sickness, absenteeism, and training. The owner's maximum, on-site daily manpower required for mining is 432 in Year 5. Mining manpower does not cover civil construction in pre-production years. Manpower requirements per year for mining are shown in Table 16-14.

**Table 16-14: Annual Mining Manpower Requirement**

| | | Capital Period in Years | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Description** | **Units** | **Year -3** | **Year -2** | **Year -1** | **Year 1** | **Year 2** | **Year 3** | **Year 4** | **Year 5** | **Year 6** | **Year 7** | **Year 8** | **Year 9** | **Year 10** | **Year 11** | **Year 12** | **Year 13** | **Year 14** | **Year 15** | **Year 16** | **Year 17** | **Year 18** | **Year 19** | **Year 20** | **Year 21** | **Year 22** | **Year 23** | **Year 24** | **Year 25** | **Year 26** | **Year 27** | **Year 28** | **Year 29** | **Year 30** |
| **No.Days** | **days** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** |
| **Total Labor** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Drilling | ea | 7 | 19 | 33 | 32 | 37 | 35 | 38 | 39 | 38 | 36 | 33 | 33 | 33 | 33 | 24 | 19 | 14 | 15 | 4 | 2 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Blasting | ea | 3 | 7 | 11 | 11 | 11 | 11 | 12 | 12 | 11 | 12 | 11 | 12 | 11 | 11 | 9 | 7 | 6 | 4 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Mine Loading | ea | 1 | 12 | 18 | 18 | 17 | 17 | 18 | 18 | 17 | 18 | 18 | 19 | 17 | 16 | 14 | 13 | 11 | 9 | 5 | 3 | 2 | 1 | 1 | 1 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 0 |
| Mine Haulage | ea | 15 | 54 | 84 | 83 | 80 | 82 | 95 | 97 | 97 | 96 | 95 | 97 | 98 | 98 | 84 | 81 | 73 | 60 | 29 | 19 | 12 | 11 | 11 | 11 | 13 | 13 | 7 | 7 | 7 | 5 | 5 | 5 | 0 |
| Mine Support | ea | 16 | 42 | 55 | 54 | 53 | 54 | 55 | 56 | 49 | 49 | 54 | 57 | 54 | 52 | 47 | 44 | 43 | 43 | 30 | 21 | 5 | 4 | 4 | 4 | 5 | 7 | 7 | 7 | 7 | 5 | 5 | 1 | 0 |
| Open Cut Mine Maintenance | ea | 28 | 92 | 134 | 130 | 130 | 130 | 143 | 145 | 139 | 138 | 139 | 142 | 139 | 138 | 119 | 111 | 101 | 88 | 48 | 30 | 12 | 12 | 12 | 12 | 13 | 14 | 11 | 11 | 11 | 8 | 8 | 5 | 0 |
| Open Cut Mine General and Administration | ea | 18 | 38 | 57 | 54 | 58 | 57 | 63 | 65 | 61 | 59 | 56 | 56 | 58 | 61 | 46 | 43 | 37 | 42 | 17 | 14 | 6 | 6 | 6 | 6 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 4 | 0 |
| **Total** | **ea** | **88** | **264** | **392** | **382** | **386** | **387** | **424** | **432** | **411** | **407** | **406** | **414** | **410** | **410** | **343** | **318** | **285** | **260** | **133** | **89** | **36** | **35** | **35** | **35** | **39** | **41** | **32** | **32** | **31** | **25** | **25** | **16** | **0** |

# 17 RECOVERY METHODS

## 17.1 GENERAL

The Pitarrilla Project will consist of an open pit mine, ore processing facility, and miscellaneous infrastructure and support facilities.

After crushing and milling, one of two processing types will be applied to the ore:

- Highly-oxidised ore will be direct leached and then silver will be extracted via the Merrill-Crowe process to produce doré. For the purposes of this report, this ore will be referred to as either "oxide" ore or "direct leach" ore.

- Less-oxidised ore (transitional) through to un-weathered sulphide ore will be processed by sequential flotation to extract lead and zinc minerals into separate lead and zinc mineral concentrates. The tailings from these flotation processes will then proceed to the cyanide leach circuit to produce doré. For the purposes of this report, this ore will be referred to as either "sulphide" ore or "flotation/leach" ore.

The design basis for the ore processing facility is 16,000 tpd of sulphide ore (equivalent to 5,840,000 t/a) and 12,000 tpd of oxide ore (equivalent to 4,380,000 t/a), using the same crushing and grinding plant circuit. Sufficient ore is available for 30 years of milling at these rates.

A summary diagram of the overall process flowsheet is presented in Figure 17-1. Process unit operations that will be used include:

- All Ore
  - Primary crushing
  - SAG mill grinding
  - Ball mill grinding
  - Pebble crushing

- Oxide or Direct Leach Ore:
  - Pre-leach thickening
  - Leaching
  - CCD thickening
  - Cyanide recovery
  - Cyanide destruction
  - Merrill-Crowe
  - Refinery

- Sulphide or Flotation/Leach Ore:
  - Lead rougher flotation
  - Lead rougher concentrate regrinding
  - Lead 1$^{st}$ and 2$^{nd}$ cleaner flotation
  - Zinc rougher flotation
  - Zinc rougher concentrate regrinding

  - Zinc 1st, 2nd, and 3rd cleaner flotation
  - Lead concentrate dewatering
  - Zinc concentrate dewatering

- followed by processing the flotation tailings by:
  - Leaching
  - CCD thickening
  - Cyanide recovery
  - Cyanide destruction
  - Merrill-Crowe
  - Refinery

The process plant will be located approximately 1,250 m from the ultimate pit limits. An overall site plan of the plant and pit is presented in Figure 17-2. A summary diagram of the process plant facilities site plan is shown in Figure 17-3.

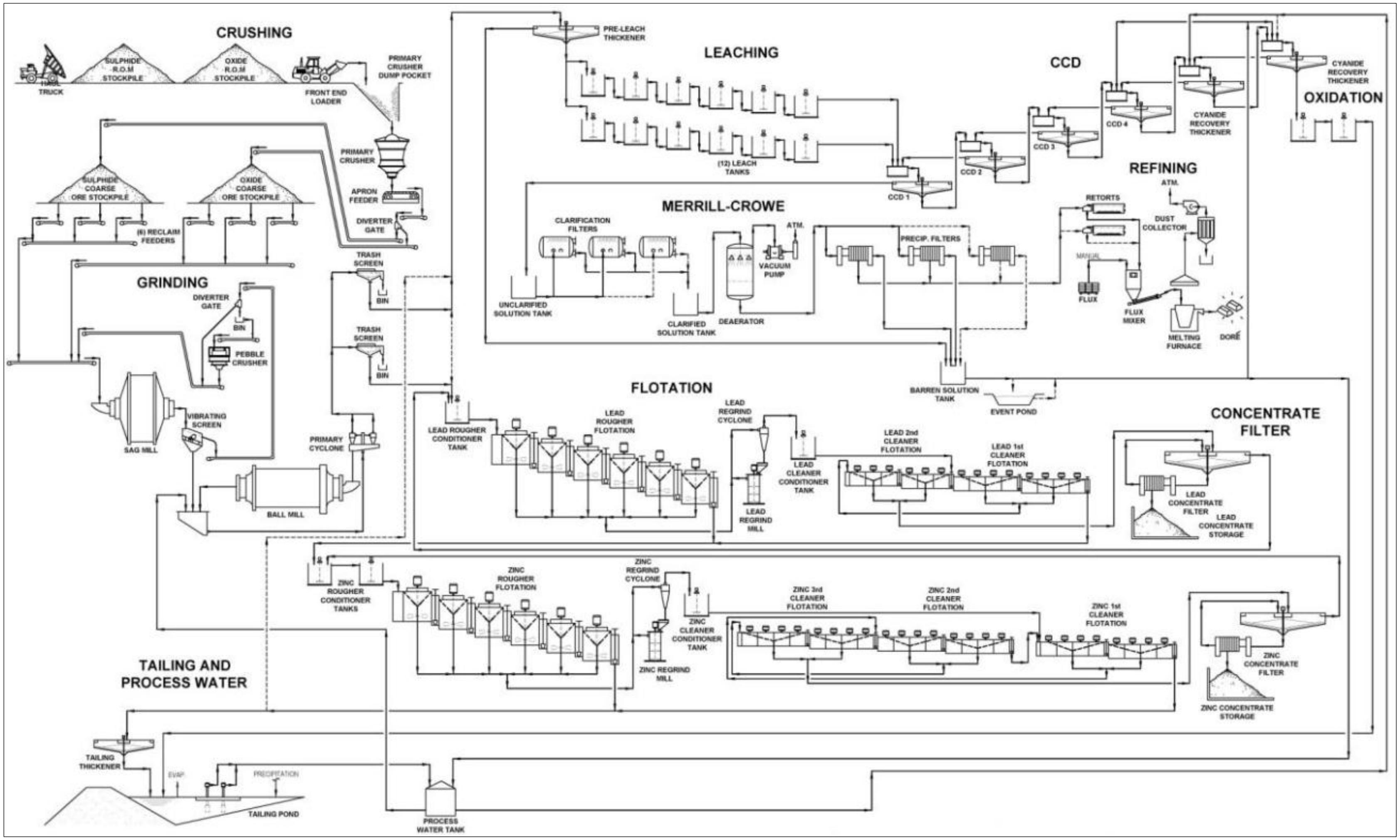



**Figure 17-1: Overall Process Flow Sheet**
Source: M3, 2012

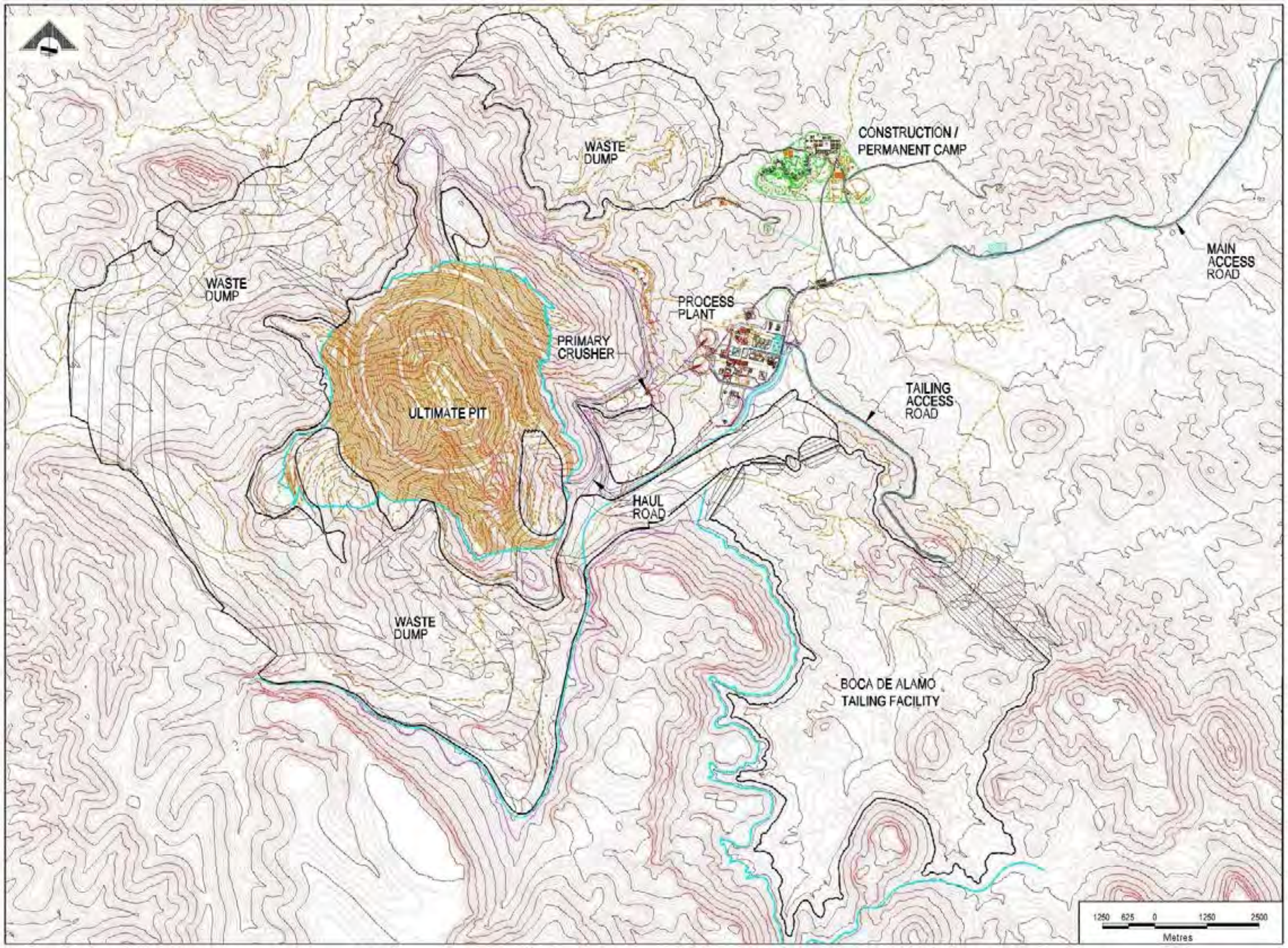



**Figure 17-2: Overall Site Plan**
Source: M3, 2012

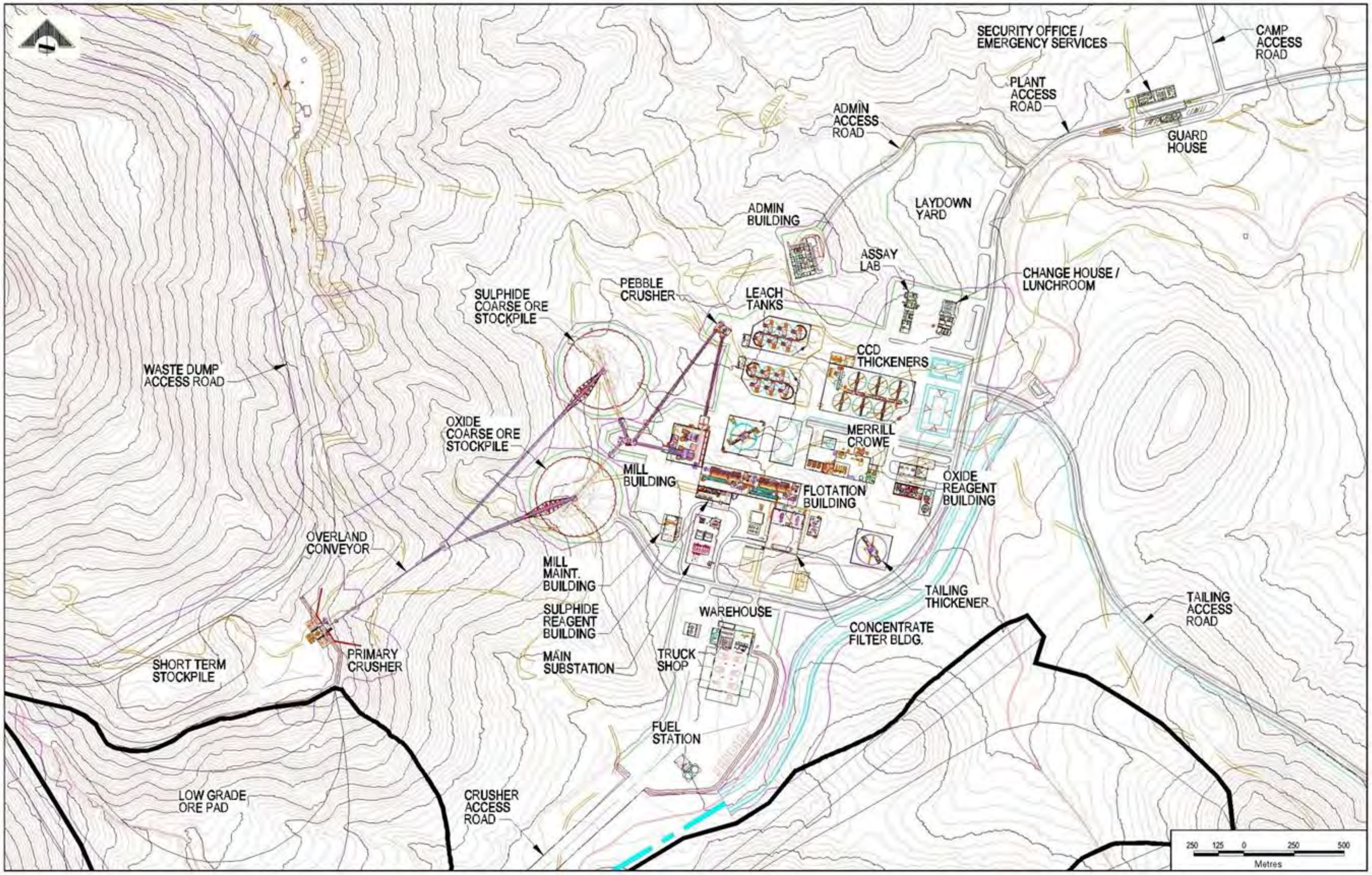



**Figure** 17-3**: Process Facilities**

Source: M3, 2012

## 17.2 PROCESS FACILITY DESCRIPTION

### 17.2.1 Process Direct Leach Overview

The following items summarize the process operations required to extract silver from the direct leach ore:

- Size reduction of the mined ore by a primary gyratory crusher to reduce the ore from run-of-mine 900 mm (maximum size) to $P_{80}$ minus 150 mm.
- Storage of the primary crushed ore in a coarse ore stockpile, which can then be reclaimed to the SAG mill by feeders and a conveyor belt.
- Grinding of ore in a SAG mill, ball mill and pebble crusher. The SAG mill will operate in closed circuit with a vibrating screen and a pebble crushing circuit. The ball mill will operate in closed circuit with hydro-cyclones, with a product size of $P_{80}$106um.
- Primary cyclone overflow will be thickened to approximately 40% solids.
- Agitation leaching with cyanide followed by four-stage counter-current decantation ("CCD") thickening.
- Recovery of silver from the pregnant leach solution in a Merrill-Crowe zinc precipitation/cementation plant.
- Melting the zinc precipitate with fluxes to produce a silver doré bar, the final product of the direct leach processing facility.
- Two stages of cyanide recovery with wash thickening, followed by slurry cyanide neutralization using sulphur dioxide and oxygen (with copper sulphate as a catalyst), prior to disposal in a slurry to the TSF.
- Residual and precipitation water from the TSF will be returned for use in the process plant. Plant water stream types include: process water, freshwater, barren solution and potable water.
- Storing, preparing, and distributing reagents used in the process. Reagents include: sodium cyanide, pebble lime, flocculant, zinc dust, anti-scalant, diatomaceous earth, sulphur, copper sulphate, and several refinery fluxes.

### 17.2.2 Primary Crusher, Overland Conveyors and Storage

Run-of-mine ore will be discharged to the primary crusher dump and fed to the gyratory primary crusher to reduce rock from 900 mm to 150 mm sizing. Primary crushed ore product then discharges onto the apron feeder, which loads onto the stockpile feed conveyor where the crushed ore will be transported to either of the two coarse ore stockpiles (one for each of oxide or sulphide ore).

The total capacity of each coarse ore stockpile will be approximately 16,000 live tonnes. Three belt feeders (two operating, one standby) will reclaim the coarse ore from each stockpile. The oxide reclaim feeders will discharge onto the reclaim conveyor, which in turn discharges onto the SAG mill feed conveyor feeding the SAG mill.

Tramp iron will be removed using a self-cleaning magnet that will be located at the discharge of the apron feeder, and a metal detector that will be installed on the stockpile feed conveyor. The crushing production rate will be monitored by a belt scale located at the end of the stockpile feed conveyor.

#### 17.2.2.1 Grinding

The grinding circuit, consisting of a primary SAG mill and a secondary ball mill, will grind the crushed oxide ore to a final product size of 106µm. Primary grinding will be performed in a SAG mill operated in closed circuit with a SAG discharge screen (one operating, one standby) and a pebble crusher. SAG mill discharge screen oversize will report to the pebble crusher feed conveyor where a mounted belt scale will monitor the SAG mill recycle feed rate, and a magnet will remove any tramp metal before discharging into the diverter gate. Here, the oversised ore will be either fed to the pebble crusher, or bypass the pebble crusher and report to the pebble crusher discharge conveyor. The pebble crusher products will discharge back onto the SAG mill feed conveyor. Undersize from the SAG mill discharge screen will flow by gravity to the cyclone feed box, where it will be fed into the secondary grinding circuit.

The secondary grinding circuit will consist of a ball mill that operates in a closed circuit with a cyclone cluster. The cyclone cluster will feed the ball mill and the ball mill will discharge into the cyclone feed box combining with the underflow from the SAG discharge screen. Cyclone feed pumps (two operating, one standby), all variable speed horizontal centrifugal slurry pumps, will be used to pump the combined slurry in the cyclone feed box to the cyclone cluster for classification. The cyclone cluster underflow will report back to the ball mill, while the overflow will flow by gravity to the trash screens for removal of tramp material. The trash screen overflow will be collected in a tote bin for periodic disposal. The trash screen undersize will flow by gravity to the pre-leach thickener feed box in the oxide leach plant.

Cyclone overflow will be sampled and analyzed for metallurgical control prior to leaching.

Grinding balls will be added to the SAG and ball mill using ball buckets.

#### 17.2.2.2 Leach Circuit

The trash screen underflow will report to the high rate pre-leach thickener. Flocculant and dilution water will be added to the thickener feed to aid in settling. The withdrawal rate of the settled solids will be controlled by variable-speed, pre-leach thickener underflow pumps (one operating, one standby) to maintain either thickener underflow density or thickener solids loading. The thickener underflow will be pumped to the leach tank feed box. The thickener overflow will flow via pre-leach thickener overflow pumps (one operating, one standby) to the barren solution distribution tank. Thickener underflow will be sampled by a leach feed sampler before entering the leach tank feed box. Here, slurry will be distributed to two parallel circuits of six leach tanks that operate in series and are arranged such that the slurry can advance from leach tank to leach tank by gravity flow and finally be collected in the leach tank general discharge box. The leach tanks (16.7 m diameter, 16.7 m height) will provide 41 hours of total plug-flow retention time, at 40% solids with 3,570 $m^3$ total working volume.

Cyanide solution can be added to the first and third tanks of each tank line. Lime will be piped to the first and second tank of each tank line. Leach tank blowers will add air as needed into both agitated leach circuits. Lime and cyanide will be added as needed to both agitated leach circuits.

#### 17.2.2.3 CCD Thickeners

The leached slurry will gravity flow from the leach tank general discharge box to the CCD thickeners and then to the two cyanide recovery thickeners that all operate in series. The slurry advances upstream with variable speed horizontal centrifugal slurry pumps, CCD thickener underflow pumps (one operating, one standby per thickener) and cyanide recovery thickener underflow pumps (one operating, one standby per thickener), all operating at 50% solids. The final cyanide recovery thickener underflow pump will be sampled via a sampler before going to the oxidation feed box for the tailings detoxification circuit.

Thickener overflow streams will flow by gravity, in a counter-current direction, from the last cyanide recovery thickener to the first CCD thickener. Each thickener will have a dilution box to receive thickener feeds and a diverter box to bypass one thickener in the series. The final CCD thickener overflow stream associated with the first CCD thickener will gravity feed into the CCD overflow tank where it will be pumped to the pregnant solution tank by variable speed, horizontal centrifugal slurry pumps (one operating, one standby).

Barren solution, used as wash water, will be added to the dilution boxes of the thickeners. Process water will be added to the cyanide recover thickener, while milk of lime and flocculant are added to each thickener to aid in settling.

#### 17.2.2.4 Tailings Detoxification

In the tailings detoxification tanks, any residual weak acid dissociable cyanide will be oxidised to the relatively non-toxic form of cyanate using sulphur dioxide and oxygen, with copper sulphate as a catalyst. Milk of lime will also be added to maintain a slurry pH in the range of 8.0 to 8.5. The more stable iron cyanides will be removed from solution as insoluble Ferro-cyanide precipitates. The cyanide levels will thereby be reduced to an acceptable level for discharge to the TSF. The detoxification reactor tanks will provide a residence time of approximately 1.7 hours. Slurry discharged from the detoxification tanks will be sampled by the tailings sampler and will gravity flow into the tailings box. Tailings pumps (3 operating, 3 standbys) will pump the final oxide plant tailings slurry to the TSF.

#### 17.2.2.5 Merrill Crowe-Refinery

Silver will be recovered from pregnant solution by zinc precipitation of metal ions using zinc dust, and then smelting the collected silver precipitate into doré bars.

The process of recovering silver by the Merrill-Crowe process includes:

- Clarification and filtering of pregnant solution to remove suspended solids;
- De-aeration of pregnant solution to reduce dissolved oxygen;
- Precipitation of silver metal by addition of zinc dust;
- Filtering and drying of precipitate, and smelting the precious metal precipitate, in a crucible furnace to produce doré bars.

Pregnant solution from the pregnant un-clarified solution tank will be pumped using horizontal, centrifugal clarifier filter feed pumps (one operating, one standby) to two of three self-cleaning pressure leaf clarifier filters (two operating, one standby). The operating filters are pre-coated using diatomaceous earth ("DE") as a filter aid and also have a continuous body feed addition of DE, to assist filtering as needed. For these purposes, four pumps with a common tank will be provided with the filter units. The pre-coat pumps will be peristaltic-type pumps which will receive feed from the agitated DE tank and then pump filter aid into each filter feed stream. Pressure for filter operations will be provided by the clarifier filter feed pump (one operating, one standby). Filtrate will discharge directly to the de-aeration tower. Clarified solution will be passed through the de-aeration tower to remove dissolved oxygen to less than 0.5 ppm prior to zinc dust addition. The de-aeration tower will be connected through a barometric seal to vacuum pumps (one operating, one standby). The clarified, de-aerated, pregnant solution will be withdrawn from the bottom of the de-aeration tower by single-stage, horizontal, in-line, centrifugal precipitate filter feed pumps (one operating, one standby). The pump will discharge to the precipitation filter presses (two operating, one standby) where the precipitated precious metals will be collected. The plate and frame press will be manually opened and cleaned with the zinc precipitate being collected in carts.

Zinc precipitate will be placed in the drying ovens prior to fluxing. The drying ovens will be equipped with mercury retorts for mercury recovery. The dried filter cake will be mixed in a mixer feeder with fluxing materials and charged to an electric furnace. The melted charge will be poured into conical molds where the doré will sink to the bottom of the mold and slag, containing fused fluxes and impurities, will float to the top. Doré will be sampled using vacuum tube samplers during melting.

After cooling and solidifying, the molds will be dumped and the slag will be knocked off the doré bars by hand. The bars will be cleaned under a water stream using a needle gun, weighed and stamped with an I.D. number and weight. Doré bars weighing approximately 20 to 30 kilograms will be the final product of the operation and will be stored in a safe, until secure shipment.

Slag will be crushed and screened to recover high grade prills, which will be returned to the melting furnace. Remaining slag will be collected for proper disposal.

Fumes from the melting furnace will be collected through ductwork and cleaned in a bag house dust collector system before discharging to the atmosphere.

#### 17.2.2.6 Oxide Reagents

Reagents requiring handling, mixing, and distribution include:

- Lime
- Sodium Cyanide
- Copper Sulphate
- Sulphur (S)
- Flocculant

17.2.2.6.1 Lime

Lime will be added to the pre-leach thickener feed box, leach tank feed box, leach tanks, CCD thickener dilution box and oxidation feed box.

| | |
|---|---|
| Delivered form | bulk 35 tonne truck |
| Method of storage | in silo |
| Mix tank content | 15% |
| Day tank content | 15% |
| Usage rate, g/t in cyanide leaching | 2,000 |
| Usage rate, g/t in detoxification | 1,000 |
| Total usage g/t | 3,000 |

17.2.2.6.2 Sodium Cyanide (Leaching)

Sodium cyanide will be added to the leach tank feed box and third leach tank of each line.

| | |
|---|---|
| Delivered form | Flo-Bin solid |
| Method of storage | Flo-Bin storage |
| Mix tank content | 24% |
| Day tank content | 24% |
| Usage rate, g/t mill feed | 1,000 |

17.2.2.6.3 Copper Sulphate

Copper sulphate will be added to the oxidation feed box and oxidation tanks.

| | |
|---|---|
| Delivered form | dry powder in 25 kg bags |
| Method of storage | bags on pallets |
| Mix tank content | 20% |
| Day tank content | 20% |
| Usage rate, g/t mill feed | 85.5 |

17.2.2.6.4 Sulphur

Sulphur will be added to the detoxification tanks in gas form.

| | |
|---|---|
| Delivered form | dry powder in 25 kg bags |
| Method of storage | bags on pallets |
| Mix tank content | 20% |
| Day tank content | 20% |
| Usage rate, g/t mill feed | 570 |

17.2.2.6.5 Flocculant

Flocculant will be added to the pre-leach thickener feed box, in the cyanide recovery dilution box of the two stages of cyanide recovery, and in the dilution box of the four CCD thickeners.

| | |
|---|---|
| Delivered form | dry powder in 25 kg bags |
| Method of storage | bags on pallets |
| Solution storage content | 1% (Mix Tank), 0.1% (Feed to Thickener) |
| Usage rate, g/t mill feed | 180 |

### 17.2.3 Sulphide or Flotation - Leach Ore Process Overview

The following items summarize the process operations required to extract lead, silver and zinc from the sulphide or flotation/leach ore:

- Size reduction of the mined ore by a primary gyratory crusher to reduce the ore size from run-of-mine 900 mm maximum sizing to $P_{80}$ minus 150 mm.
- Storage of primary crushed ore in a coarse ore stockpile, which can then be reclaimed by feeders and a conveyor belt.
- Grinding ore in a SAG mill, ball mill, and pebble crusher before processing in a lead and zinc flotation cell circuit. The SAG mill will operate in closed circuit with a vibrating screen and a pebble crushing circuit. The ball mill will operate in closed circuit with hydro-cyclones, with a product size of $P_{80}$150um.
- Primary cyclone overflow will be approximately 40% solids.
- The lead flotation circuit will consist of rougher flotation, a regrinding stage and two stages of cleaning flotation circuits. The zinc flotation circuit will consist of rougher flotation, a regrinding stage and three stages of cleaning flotation circuits.
- Final concentrates will be thickened, filtered and stored for loading in trucks for shipment.
- Zinc flotation tailings will feed into the leaching circuits, consisting of:
    - Agitation leaching with cyanide followed by four-stage CCD thickening.
    - Recovery of silver from the pregnant leach solution in a Merrill-Crowe zinc precipitation/cementation plant.
    - Melting the zinc precipitate with fluxes to produce a silver doré bar, the final product of the direct leach processing facility.
    - Two stages of cyanide recovery with wash thickening, followed by slurry cyanide neutralization using sulphur dioxide and oxygen (with copper sulphate as a catalyst) prior to disposal in a slurry to the TSF.
    - Residual and precipitation water from the TSF will be returned for use in the process plant. Plant water stream types include: process water, freshwater, barren solution and potable water.
    - Storing, preparing, and distributing reagents used in the process. Reagents include: lime, copper sulphate ($CuSO_4$), sodium cyanide (NaCN), SIPX and A-3418 as collectors, methyl isobutyl carbinol (MIBC) and DF1012 as frother, and a polyacrylamide flocculant. Additionally zinc dust, anti-scalant, DE, sulphur, and several refinery fluxes, will be used within the leaching process.

#### 17.2.3.1 Primary Crusher, Overland Conveyors and Storage

The primary crushing, overland conveyance and storage processes for the sulphide ore are the same as for the oxide ore.

#### 17.2.3.2 Grinding

The primary and secondary grinding circuits for the sulphide ore are the same as for the oxide ore, with the exception that the trash screen overflow material from the cyclone cluster will flow by gravity to the lead rougher conditioner tank for lead rougher flotation. Grind size will be $P_{80}$150um.

#### 17.2.3.3 Lead Flotation

Flotation reagents will be added to the slurry in the lead rougher conditioner tank before flowing by gravity to the lead rougher flotation circuit. Rougher flotation will consist of six lead rougher flotation cells with a level controlling drop-box between each cell. Lead rougher concentrate will gravity flow to the regrind mill cyclone feed sump. Lead rougher tailings will gravity flow to the lead rougher tailings sump where the lead rougher tailings pumps (one operating, one standby) will pump the lead rougher tailings to the zinc rougher conditioner tank. The lead rougher concentrate will be sampled by the lead rougher concentrate sampler, while lead rougher tailings will be sampled by the lead rougher tailings sampler.

Slurry in the lead regrind mill cyclone feed sump will feed the lead regrind circuit. The lead regrind mill will operate in closed circuit with the lead regrind cyclone cluster. This cyclone cluster is fed by the lead regrind mill cyclone feed pumps (one operating, one standby) from the lead regrind mill cyclone feed sump. The lead regrind cyclone underflow will report to the lead regrind mill, while the lead regrind cyclone overflow will report to the lead cleaner conditioner tank. The lead regrind mill will grind to 30 µm final grind size. Alternately, the lead regrind mill cyclone feed pump can bypass the lead regrind cyclone and be pumped directly to the lead cleaner conditioner tank. Flotation reagents will be added to the lead cleaner conditioner tank before gravity flowing to the lead-first cleaner flotation cells. Lead-first cleaner flotation has four cells per level controlling drop-box. The tailings will gravity feed the lead-first cleaner flotation tailings sump. Concentrates will gravity feed the lead-first cleaner concentrate sump. Slurry in the lead-first cleaner flotation tailings sump will be pumped with lead-first cleaner flotation tailings pumps (one operating, one standby) to the zinc rougher conditioner tank for the zinc rougher flotation circuit. Slurry in the lead-first cleaner concentrate sump will be pumped to the lead-second cleaner concentrate flotation cells with the lead-first cleaner concentrate pumps (one operating, one standby). Lead-second cleaner flotation cells will have three cells per level controlling drop-box. The tailings will gravity flow to the lead first cleaner, to mix with the feed from the lead cleaner conditioner tank. Concentrates will gravity flow to the lead-second cleaner concentrate sump. A lead collector will be added to the feed box of the lead-second cleaner flotation cells. Lead concentrate in the lead-second cleaner concentrate sump will gravity flow to the lead concentrate thickener.

Samplers will be installed at the following locations:

- Cyclone overflow sampler, on the discharge of the trash screens, for process and metallurgical control;
- Lead rougher concentrate sampler, on the discharge of the rougher flotation cell before the lead regrind mill cyclone feed sump, for process control;
- Lead rougher tailings sampler, on the discharge of the rougher flotation cell before the zinc rougher conditioner tank, for process control;
- Lead second cleaner concentrate sampler, on the discharge pipe prior to the lead concentrate thickener, for process and metallurgical control;
- Lead first cleaner tailings sampler, on the tailings discharge pipe after the lead-first cleaner flotation tailings pumps, for process and metallurgical control; and
- Lead first cleaner concentrate sampler, on the discharge pipe after the lead-first cleaner concentrate pumps, for process control.

Lead flotation cell blowers (one operating, one standby) will be installed for the lead flotation cells.

Flotation reagents will be added at several points in the lead flotation circuits.

#### 17.2.3.4 Zinc Flotation

The zinc rougher conditioner tanks (two operating in series) will receive the lead rougher tailings and the lead first cleaner tailings where the zinc flotation reagents are introduced. The first conditioner tank gravity feeds the second conditioner tank which feeds the zinc rougher flotation cells. The zinc rougher flotation cells have one cell per level controlling drop-box. The tailings will gravity feed the zinc rougher tailings sump. Concentrate gravity feeds the zinc regrind mill cyclone feed sump.

The zinc rougher tailings sump will receive the zinc rougher tailings and the zinc first cleaner flotation tailings before being pumped to the tailings thickener feed box for final tailings dewatering. For process and metallurgical control, these final tailings will be sampled by the zinc final tailings sampler. Slurry in the zinc regrind mill cyclone feed sump will feed the zinc regrind circuit. The zinc regrind mill will operate in closed circuit with the zinc regrind cyclone cluster. These cyclones will be fed by the zinc regrind mill cyclone feed pumps (one operating, one standby) from the zinc regrind mill cyclone feed sump. The zinc regrind cyclone underflow will report to the zinc regrind mill, while the zinc regrind cyclone overflow will report to the zinc cleaner conditioner tank for the addition of zinc cleaner flotation reagents. The zinc regrind mill will grind to a 35 micron final grind size. Slurry in the zinc cleaner conditioner tank will gravity flow to the zinc-second cleaner tailings pump box, to mix with the zinc-second cleaner tailings, before being pumped by the zinc-second cleaner tailings pumps (one operating, one standby), to the zinc-first cleaner flotation cells. The zinc first cleaner flotation cell will have four cells per level controlling drop-box. The zinc-first cleaner tailings will gravity flow to the zinc-first cleaner flotation tailings pump box to be pumped with the zinc-first cleaner flotation tailings pumps (one operating, one standby) to the zinc rougher tailings sump. The zinc-first cleaner concentrate will gravity flow to the zinc-first cleaner concentrate pump box to be pumped with the zinc-first cleaner concentrate pumps (one operating, one standby) to the zinc-second cleaner flotation cells. The zinc-second cleaners will have four cells per level controlling drop-box and the zinc-second cleaner concentrate will gravity flow to the zinc-second cleaner concentrate pump box to be pumped to the zinc-third cleaner flotation cells. The zinc-third cleaners also have four cells per level controlling drop-box. The zinc-third cleaner tailings will gravity flow to the zinc-second cleaner flotation cells together with the zinc-first cleaner concentrate. The zinc-third cleaner concentrate will gravity flow to the zinc-third cleaner concentrate box.

Samplers will be installed at the following locations:

- Zinc rougher feed sampler, on the discharge of the zinc rougher conditioner tank, for process and metallurgical control;
- Zinc rougher tailings sampler, on the discharge of the zinc rougher flotation cell and before the zinc rougher tailings sump, for process control;
- Zinc rougher concentrate sampler, on the discharge pipe before the zinc regrind cyclone feed sump, for process control;
- Zinc final tailings sampler, on the discharge pipe after the zinc rougher tailings pumps, for process and metallurgical control;
- Zinc-first cleaner concentrate sampler, on the discharge pipe after the zinc-first cleaner concentrate pumps, for process control;
- Zinc-first cleaner tailings sampler, on the discharge pipe after the zinc-first cleaner flotation tailings pumps, for process and metallurgical control;and
- Zinc-third cleaner concentrate sampler, on the discharge pipe prior to the zinc concentrate thickener, for process and metallurgical control.

Zinc flotation cell blowers (one operating, one standby) will be installed for the zinc flotation cells.

Flotation reagents will be added at several points in the zinc flotation circuits.

#### 17.2.3.5 Concentrate Thickening and Filtration

##### 17.2.3.5.1 Lead Concentrate

The final lead concentrate will flow by gravity to the lead concentrate thickener. Thickener overflow will gravity flow to the lead concentrate thickener overflow sump and be pumped with the lead concentrate thickener overflow pumps (one operating, one standby) to the lead rougher conditioner tank. Thickener underflow will be pumped by the lead concentrate thickener underflow pump, a variable-speed, progressive cavity pump (one operating, one standby), to the agitated lead concentrate stock tank.

The lead concentrate will be pumped from the lead concentrate stock tank to the plate and frame lead concentrate filter by the lead concentrate filter feed pump (one operating, one standby).

Lead filter cake batches will discharge onto the lead dry concentrate feed conveyor to be sent to the lead concentrate storage area, where the batches will be weighed with the belt scale located under the conveyor.

Lead filtrate and filter wash water will be collected in the lead filtrate storage tank and will be returned to the lead concentrate thickener by the lead filtrate solution pumps (one operating, one standby).

Concentrate will be reclaimed from the storage area by front-end loader onto highway haulage trucks.

A lead concentrate filter blower will be installed for the lead concentrate filter.

An air compressor lead concentrate filter will be installed for the lead concentrate filter.

17.2.3.5.2 Zinc Concentrate

Zinc concentrate will follow the same process as the lead concentrate but in a separate, zinc only circuit.

17.2.3.6 Flotation Tailings Handling

The zinc rougher tailings sump will receive the zinc rougher tailings and the zinc-first cleaner flotation tailings before being pumped to either the leach circuits or to the tailings thickener feed box for flocculant mixing, and then gravity flow to the tailings thickener. Overflow from the tailings thickener will flow to the tailings thickener overflow sump and will be pumped with horizontal centrifugal tailings thickener overflow pumps (one operating, one standby) to the process water tank. Underflow from the tailings thickener will be pumped by three variable-speed, tailings thickener underflow pumps (three operating in series, three standbys) to the leach circuit.

17.2.3.7 Leach Circuit

The flotation tailings will report to the high rate pre-leach thickener. Flocculant and dilution water will be added to the thickener feed to aid in settling. The withdrawal rate of the settled solids will be controlled by variable-speed, pre-leach thickener underflow pumps (one operating, one standby) to maintain either thickener underflow density or thickener solids loading. The thickener underflow will be pumped to the leach tank feed box. The thickener overflow will flow via pre-leach thickener overflow pumps (one operating, one standby) to the barren solution distribution tank. Thickener underflow will be sampled by a leach feed sampler before entering the leach tank feed box. Here, slurry will be distributed to two parallel circuits of six leach tanks that operate in series and arranged such that the slurry can advance from leach tank to leach tank by gravity flow and finally be collected in the leach tank general discharge box. The leach tanks (16.7 m diameter, 16.7 m height) will provide 30 hours of total plug-flow retention time at 40% solids with 3,570 $m^3$ total working volume.

Cyanide solution can be added to the first and third tanks of each tank line. Lime will be piped to the first and second tank of each tank line. Leach tank blowers will add air as needed into both agitated leach circuits. Lime and cyanide will be added as needed to both agitated leach circuits.

17.2.3.8 CCD Thickeners

The leached slurry from the sulphide tailings will be processed in the CCD thickeners in the same manner as the oxide ore as described in Section 17.2.2.3.

17.2.3.9 Tailings Detoxification

Detoxification of the flotation tailings will be performed in the same manner as for the oxide tailings, as described in Section 17.2.2.4.

17.2.3.10 Merrill Crowe-Refinery

Silver will be recovered from the sulphide flotation tailings pregnant solution by the Merrill Crowe process, which is the same process described for the oxide ore in Section 17.2.2.5.

17.2.3.11 Sulphide Reagents

Reagents requiring handling, mixing, and distribution include:

- Cytec 3418A (collector)
- Sodium Isopropyl Xanthate (SIPX, collector)
- Lime
- Methyl Isobutyl Carbinol (MIBC, frother)
- DF-1012 (Frother)
- Copper Sulphate
- Flocculant
- Sodium Cyanide (dispersant)
- Anti-scalant

17.2.3.11.1 Sodium Isopropyl Xanthate (SIPX, collector)

SIPX will be added to the zinc rougher and cleaner conditioner tanks.

| | |
|---|---|
| Delivered form | dry powder in super sacks |
| Method of storage | super sacks on pallets and as water solution |
| Mix tank content | 5% |
| Day tank content | 5% |
| Usage rate, g/t mill feed | 30 |

17.2.3.11.2 Lime

Lime will be added in the SAG mill, ball mill, lead rougher tailings sump and lead regrind mill cyclone feed sump.

| | |
|---|---|
| Delivered form | bulk 35 tonne truck |
| Method of storage | in silo |
| Mix tank content | 15% |
| Day tank content | 15% |

| | |
|---|---|
| Usage rate, g/t in lead rougher | 670 |
| Usage rate, g/t in zinc rougher | 110 |
| Usage rate, g/t in cyanide leaching | 2,000 |
| Usage rate, g/t in detoxification | 1,000 |
| Total usage g/t | 3,780 |

17.2.3.11.3 Methyl Isobutyl Carbinol ("MIBC", frother)

MIBC will be added to the lead rougher conditioner tank, lead cleaner conditioner tank, zinc rougher conditioner tank, zinc rougher flotation cells and zinc cleaner conditioner tank.

| | |
|---|---|
| Delivered form | liquid in 160 kg drum |
| Method of storage | in tank |
| Mix tank content | undiluted |
| Day tank content | undiluted |
| Usage rate, g/t mill feed | 80 |

17.2.3.11.4 Dowfroth DF-1012 (Frother)

DF-1012 Frother will be added in the lead rougher flotation cells and zinc rougher conditioner tank.

| | |
|---|---|
| Delivered form | liquid in 220 L drums |
| Method of storage | drums on pallets and in tank |
| Mix tank content | undiluted |
| Day tank content | undiluted |
| Usage rate, g/t mill feed | 50 |

17.2.3.11.5 Copper Sulphate

Copper sulphate will be added to the zinc rougher conditioner tank and zinc regrind mill cyclone feed sump.

| | |
|---|---|
| Delivered form | dry powder in 25 kg bags |
| Method of storage | bags on pallets |
| Mix tank content | 20% |
| Day tank content | 20% |
| Usage rate, g/t in zinc flotation | 98.7 |
| Usage rate, g/t in cyanide detoxification | 85.5 |
| Total usage, g/t | 184.2 |

17.2.3.11.6 Sulphur

Sulphur will be added to the detoxification tanks in gas form.

| | |
|---|---|
| Delivered form | dry powder in 25 kg bags |
| Method of storage | bags on pallets |
| Mix tank content | 20% |

| | |
|---|---|
| Day tank content | 20% |
| Usage rate, g/t | 570 |

17.2.3.11.7 Flocculant

Flocculant will be added to the tailings thickener and concentrate thickener to enhance settling.

| | |
|---|---|
| Delivered form | dry powder in 25 kg bags |
| Method of storage | bags on pallets |
| Solution storage content | 1% (Mix Tank), 0.1% (Feed to Thickener) |
| Usage rate, g/t mill feed | 50 |
| Usage rate, g/t mill feed (pre-leach, 4 CCD thickeners) | 180 |
| Total usage, g/t | 230 |

17.2.3.11.8 Sodium Cyanide (dispersant)

Sodium cyanide will be added to the lead rougher conditioner tank, and leach tanks

| | |
|---|---|
| Delivered form | Flo-Bin solid |
| Method of storage | Flo-Bin storage |
| Mix tank content | 10% |
| Day tank content | 10% |
| Usage rate, g/t, in lead flotation | 30 |
| Usage rate g/t, in leaching circuits | 700 |
| Total usage, g/t | 730 |

17.2.3.11.9 Anti-scalant

Anti-scalant will be added into the suction of the process water pump to reduce scale formation.

| | |
|---|---|
| Delivered form | totes |
| Method of storage | totes on pallets |
| Solution storage content | neat |
| Usage rate, g/t mill feed | 10 |

## 17.3 PROCESS DESIGN CRITERIA

### 17.3.1 General

The design of the Oxide (or direct-leach) and Sulphide (or flotation-leach) facilities are based on the following criteria which have been provided, calculated, or recommended. Each line has a code letter which identifies the source of the criteria according to the following designation:

| Code letter | Source |
|---|---|
| A | Client documents or instructions |
| B | Recommended by M3 |
| C | Industry standards |
| D | Vendor data |
| E | Calculated from other data |
| F | Consultants |
| G | Reference handbooks |

### 17.3.2 Ore Characteristics

| Run-of-Mine Ore Characteristics | Oxide | Sulphide | Code Letter |
|---|---|---|---|
| Maximum mine-run ore size, m | 1.5 | 1.5 | A |
| Maximum ore size primary crusher feed, m | 900 | 900 | A |
| Ore specific gravity | 2.57 | 2.85 | A |
| Ore bulk density, primary crushed feed, t/m$^3$ | | | |
| Top | 2.0 | 2.0 | A |
| Swell factor for 6” | 1.75 | 1.75 | B |
| Ore abrasion index, Bond, (Ai), average | 0.40 | 0.40 | A |
| Ore work index, kWh/t | | | |
| Crushing work index, Bond, (CWi) | 9.9 | 16.7 | A |
| Rod mill work index, Bond, (RWi) | 16.8 | 16.7 | A |
| Ball mill work index, Bond, (BWi) | 18.4 | 18.0 | A |
| Regrind mill work index, Bond, (Wi) | | | G |
| Lead Rougher Concentrate | | TBD | G |
| Zinc Rougher Concentrate | | TBD | G |

JKTech SAG testing parameters for Oxide:

| A | b | A x b | Sg | ta |
|---|---|---|---|---|
| 64.75 | 0.84 | 54.65 | 2.22 | 0.56 |

SGS JKDW test 2011 for Oxide:

| DWi | DWi | Mia | Mih | Mic | A | b | A x b | Sg | ta |
|---|---|---|---|---|---|---|---|---|---|
| 3.56 | 22.5 | 15.20 | 9.90 | 5.15 | 61.8 | 0.99 | 59.90 | 2.07 | 0.75 |

JKTech SAG testing parameters for Sulphide:

| | A | b | A x b | Sg | ta |
|---|---|---|---|---|---|
| Basal Conglomerate | 64.4 | 1.13 | 72.77 | 2.77 | 0.68 |
| Sediment | 52.1 | 0.73 | 38.03 | 2.85 | 0.35 |
| C-Horizon Andesite | 57.6 | 0.6 | 34.56 | 2.54 | 0.35 |

SGS JKDW test 2011 for Sulphide:

| | Dwi | Dwi | Mia | Mih | Mic |
|---|---|---|---|---|---|
| Basal Conglomerate | 3.81 | 25 | 12.1 | 8.0 | 4.1 |
| Sediment | 7.47 | 73 | 20.1 | 15.2 | 7.9 |
| C-Horizon Andesite | 7.32 | 71 | 16.7 | 8.6 | 57.6 |

| | Oxide | Sulphide | Code Letter |
|---|---|---|---|
| Ore moisture content, % | | | |
| Design | 4 | 4 | A |
| Minimum | 1 | 1 | A |
| Maximum | 7 | 7 | A |

### 17.3.3 Production Design Rate

| | | | |
|---|---|---|---|
| Ore crushing and milling rate, average, t/a | 12,000 | 16,000 | B / A |

### 17.3.4 Metal Production Design Rate

Oxide Metal Production Design

| | Head Grade (g/t) | Recovery (%) | Production (kg/d) |
|---|---|---|---|
| Silver | 130.6 | 42.22 | 661.7 |

Sulphide Metal Production Design

| Basic Design | Cu | Zn | Pb | Ag |
|---|---|---|---|---|
| Mine Head Grades (%) | 0.043 | 0.763 | 0.359 | - |
| Mine Head Grades (g/t) | - | - | - | 84.86 |
| Lead Rougher Flotation Recovery (%) | 60.0 | 45.0 | 84.6 | 81.2 |
| Lead 1st Cleaner Flotation Recovery (%) | 72.0 | 10.0 | 91.0 | 88.1 |
| Lead 2nd Cleaner Flotation Recovery (%) | 85.0 | 28.0 | 93.0 | 91.1 |
| Zinc Rougher Flotation Recovery (%) | 60.0 | 95.5 | 25.0 | 20.0 |
| Zinc 1st Cleaner Flotation Recovery (%) | 75.0 | 94.0 | 40.0 | 62.0 |
| Zinc 2nd Cleaner Flotation Recovery (%) | 80.0 | 94.0 | 80.0 | 78.0 |
| Zinc 3rd Cleaner Flotation Recovery (%) | 85.0 | 96.5 | 90.0 | 80.0 |
| Overall Plant Recovery (%) | 67.3 | 82.7 | 78.80 | 72.91 |
| Production, tpd, average | 4.63 | 100.9 | 45.26 | 0.99 |

| Future Design | Cu | Zn | Pb | Ag |
|---|---|---|---|---|
| Mine Head Grades (%) | 0.062 | 1.65 | 0.56 | - |
| Mine Head Grades (g/t) | - | - | - | 92.36 |
| Lead Rougher Flotation Recovery (%) | 64.0 | 45.0 | 93.0 | 81.2 |
| Lead 1st Cleaner Flotation Recovery (%) | 72.0 | 22.0 | 93.0 | 91.0 |
| Lead 2nd Cleaner Flotation Recovery (%) | 85.0 | 46.0 | 96.0 | 96.0 |
| Zinc Rougher Flotation Recover, (%) | 60.0 | 95.0 | 25.0 | 18.5 |
| Zinc 1st Cleaner Flotation Recovery (%) | 75.0 | 95.0 | 40.0 | 62.0 |
| Zinc 2nd Cleaner Flotation Recovery (%) | 80.0 | 96.5 | 80.0 | 78.0 |
| Zinc 3rd Cleaner Flotation Recovery (%) | 85.0 | 98.0 | 90.0 | 80.0 |
| Overall Plant Recovery (%) | 69.8 | 90.0 | 90.2 | 78.1 |
| Production, tpd, average | 6.9 | 237.6 | 80.8 | 1.154 |

### 17.3.5 Operating Schedule

| Primary Crushing | Oxide | Sulphide | Code Letter |
|---|---|---|---|
| Days per year | 365 | 365 | A |
| Hours per day | 24 | 24 | A |
| Shifts per day | 2 | 2 | A |
| Hours per shift | 12 | 12 | A |
| Shifts per week | 14 | 14 | A |
| % Availability (excluding start-up) | 62.5 | 62.5 | A |
| Ore crushing rate, design, t/h | 1,067 | 1,067 | E |

Grinding, Leaching, CCD, Cyanide Destruction, Flotation, Concentrate Thickeners and Tailings Handling

| | Oxide | Sulphide | Code Letter |
|---|---|---|---|
| Days per year | 365 | 365 | A |
| Hours per day | 24 | 24 | A |
| Shifts per day | 2 | 2 | A |
| Hours per shift | 2 | 12 | A |
| Shifts per week | 14 | 14 | A |
| % Availability (excluding start-up) | 92 | 92 | A |
| Milling rate, design, t/h | 543 | 725 | E |

| Filtration | Oxide | Sulphide | Code Letter |
|---|---|---|---|
| Days per year | | 365 | A |
| Hours per day | | 24 | A |
| Shifts per day | | 2 | A |
| Hours per shift | | 12 | A |
| Shifts per week | | 14 | A |
| % Availability (excluding start-up) | | 83.33 | A |
| Filtration rate | | TBD | E |

### 17.3.6 Equipment Sizing

| Primary Crusher | Oxide and Sulphide | Code Letter |
|---|---|---|
| Number | 1 | A |
| Type | Gyratory | A |
| Size, mm x mm | 1,372 x 1,905 | A |
| Crusher feed, $F_{100}$, mm | 900 | A |
| Crusher product, $P_{80}$, mm | 150 | A |
| Flow sheet operating average, tpd | 16,000 | A |
| Power installed, kW | 450 | D |
| Power installed, hp | 600 | D |
| Operation, h/d | 15 | A |
| Maximum, t/h | 2,555 | D |
| Flowsheet operating average, t/h | 1,070 | D |

| SAG Mill | Oxide | Sulphide | Code Letter |
|---|---|---|---|
| Number | 1 | | B |
| Size, diameter x EGL, m | 9.75 x 4.25 | | D |
| Mode of operation | closed circuit with vibrating screen | | D |
| Circulating load, flow sheet design, % | 25 | | A |
| Critical speed, % | 75 | | B |
| Ball loading, % v/v | 2 | 10 | B |
| Mill feed slurry, % solids | 70 | 70 | B |
| Mill feed rate operating average, t/h | 543 | 725 | BA |
| Ball mill Wi, kWh/t @ 150 µm, Design value | | 18.0 | E |
| Ball mill Wi, kWh/t @ 106 µm, Design value | 18.4 | | E |
| Feed size, 80% passing, µm | 150,000 | 150,000 | B |
| Product size, 80% passing, µm | 1,200 | 1,200 | B |
| Power required, kW, calculated | 4,318 | 5,742 | B |
| Power installed, kW | 8,039 | 8,039 | B |
| Power installed, hp | 11,000 | 11,000 | B |
| Availability, % | 92 | 92 | A |

| Pebble Crusher | Oxide and Sulphide | Code Letter |
|---|---|---|
| Number | 1 | D |
| Type | Standard Cone | D |
| Size | HP 400 | |
| Setting, closed side, mm | 13 | D |
| Crusher feed, $F_{80}$, mm | 50 | B |
| Crusher product, $P_{80}$, mm. | 13 | B |
| Capacity, flowsheet design, t/h | 190 | A |
| Capacity, maximum operating, t/h @13 mm OSS | 230 | B |
| Power required, kW, calculated | 315 | D |
| Power installed, kW | 378 | D |

| Ball Mill | Oxide | Sulfide | Code Letter |
|---|---|---|---|
| Number | 1 | | B |
| Size, diameter x EGL, m | 6.70 x 10.05 | | D |
| Mode of operation | closed circuit with hydrocyclone | | D |
| Circulating load, flow sheet design, % | 300 | | A |
| Percent of critical speed | 72 | | B |
| Ball loading, % v/v | 36 | | B |
| Mill feed slurry, % solids | 70 | 65 | B |
| Mill feed rate operating average, t/h | 543 | 725 | A |
| Ball mill Wi, kWh/mt @ 150, µm Design value | | 18.0 | E |
| Ball mill Wi, kWh/mt @ 106µm, Design value | 18.4 | | E |
| Feed size, 80% passing, µm | 1,200 | 1,200 | B |
| Product size, 80% passing, µm | 106 | 150 | B |
| Power required, kW, calculated | 7,414 | 7,414 | B |
| Power installed, kW | 8,500 | 8,500 | B |
| Power installed, hp | 11,000 | 11,000 | B |

#### 17.3.6.1 Oxide Equipment Sizes

| Pre-Leach Thickener | Oxide | Code Letter |
|---|---|---|
| Number | 1 | A |
| Model | Bridge-supported | A |
| Type | High rate | A |
| Feed rate, tonnes per operating day | 12,000 | A |
| Feed density (% solids) | 15 | A |
| Underflow density (% solids) | 40 | A |
| Area factor, $m^3/m^2h$ | 3.8 | A |
| Size, diameter, m | 33 | E |

| CCD Thickener | Oxide | Code Letter |
|---|---|---|
| Number | 4 | A |
| Model | Bridge-supported | A |
| Type | High rate | A |
| Wash ratio | 2.4:1 | B |
| Feed rate, t/h | 12,000 | B |
| Feed density (% solids) | 35 | A |
| Underflow density (% solids) | 60 | A |
| Area factor, $m^3/m^2h$ | 2.5 | A |
| Size, diameter, m | 25 | E |

| Cyanide Recovery Thickener | Oxide | Code Letter |
|---|---|---|
| Number | 2 | A |
| Model | Bridge-supported | A |
| Type | High rate | A |
| Feed rate, t/h | 543 | B |
| Feed density (% solids) | 40 | A |
| Underflow density (% solids) | 50 | A |
| Area factor, $m^3/m^2h$ | 3.3 | A |
| Size, diameter, m | 23 | E |

| Tailings Thickener | Oxide | Code Letter |
|---|---|---|
| Number | 1 | A |
| Model | Bridge-supported | A |
| Type | High rate | A |
| Feed rate, t/h | 709.7 | B |
| Feed density (% solids) | 22 | A |
| Underflow density (% solids) | 55 | A |
| Area factor, $m^3/m^2h$ | 2.1 | A |
| Size, diameter, m | 39 | E |

| Leach Tanks | Oxide | Code Letter |
|---|---|---|
| Number | 12 | B |
| Type | open top w/agitator | B |
| Size, m | | |
| Diameter | 16.7 | E |
| Height | 16.7 | E |
| Freeboard | 0.4 | B |
| Mode of operation | 2 lines, 6 tanks in series | A |
| Residence time, hours, total | 41 | A |
| Residence time, hours, each | 6.8 | E |
| Operating Characteristics: | | |
| Tank feed rate, $m^3/h$ | 1,042 | E |
| Slurry, % solids w/w, design | 40 | E |

| Merrill-Crowe Plant | Oxide | Code Letter |
|---|---|---|
| Number | | |
| Clarifier filters (580-FL-001 to 003) | 3 (2 operating, 1 standby) | A |
| Precipitate filters (580-FL-004 to 006) | 3 (2 operating, 1 standby) | A |
| Pre-coat skids | TBD | B |
| Body feed skids | TBD | B |
| De-aeration tower (580-DE-001) | 1 | B |
| Zinc feeders | 2 | B |
| Mercury retorts | 2 | B |
| Induction furnace (580-FU-001) | 1 | B |
| Furnace exhaust system | 1 | B |

| Tailings Detoxification Tank | Oxide | Code Letter |
|---|---|---|
| Number | 2 | B |
| Type | open top w/agitator | B |
| Size, m | | |
| Diameter | 10.5 | E |
| Height | 10.5 | E |
| Freeboard | 0.4 | B |
| Mode of operation | series | B |
| Residence time, hours, total | 1.7 | A |
| Residence time, hours, each | 0.83 | E |
| Operating Characteristics: | | |
| Tank feed rate, $m^3/h$ | 1,029 | E |
| Slurry, % solids w/w, design | 40 | E |

17.3.6.2 Sulphide Equipment Sizes

| Lead Regrind Mill | Sulphide | Code Letter |
|---|---|---|
| Feed size, µm | 150 | B |
| Product size, µm | 30 | B |
| Circulating load, % | 300 | B |
| New feed rate, t/h | 36.3 | B |
| Overflow, % solids | 14 | B |
| Mill discharge, % solids | 60 | B |

| Zinc Regrind Mill | | |
|---|---|---|
| Feed size, µm | 150 | B |
| Product size, µm | 35 | B |
| Circulating load, % | 300 | B |
| New feed rate, t/h | 71.44 | B |
| Overflow, % solids | 14.3 | B |
| Mill discharge, % solids | 60 | B |

Lead Rougher Flotation Cell

| | 1st Phase | 2nd Phase | Code Letter |
|---|---|---|---|
| Number of cells | 6 | | B |
| Flotation cell type | tank | tank | A |
| Flotation slurry, pH | 7-9 | | A |
| Retention time lab test, minutes | 6 | | A |
| Retention time scale-up, lab to plant | 2 | 2.5 | A |
| Retention time design, min | 12 | 15 | A |
| Froth/aeration factor | 1.15 | 1.2 | B |
| Retention time, min | 13.8 | 18 | A |
| Number of rows | 1 | 1 | A |
| Air supply | Blowers | Blowers | A |
| Solids feed density, % | 30 | | B |

Lead First Cleaner Flotation Cell

| | 1st Phase | 2nd Phase | Code Letter |
|---|---|---|---|
| Number of cells | 8 | | B |
| Flotation cell type | conventional | | A |
| Flotation slurry, pH | 7-9 | | A |
| Retention time lab test, min | 5 | | A |
| Retention time scale-up, lab to plant | 2 | 2.5 | A |
| Retention time design, min | 10 | 12.5 | A |
| Froth/aeration factor | 1.25 | 1.4 | B |
| Retention time, minutes | 12.5 | 17.5 | A |
| Number of rows | 1 | 1 | A |
| Air supply | Blowers | Blowers | A |
| Solids feed density, % | 14 | 14 | B |

Lead Second Cleaner Flotation Cell

| | 1st Phase | 2nd Phase | Code Letter |
|---|---|---|---|
| Number of cells | 6 | | B |
| Flotation cell type | conventional | | A |
| Flotation slurry, pH | 7-9 | | A |
| Retention time lab test, min | 4 | | A |
| Retention time scale-up, lab to plant | 2 | 2.5 | A |
| Retention time design, min | 8 | 10 | A |
| Froth/aeration factor | 1.25 | 1.4 | B |
| Retention time, min | 10 | 14 | A |
| Number of rows | 1 | | A |
| Air supply | Blowers | Blowers | A |
| Solids feed density, % | 19.2 | 19.2 | B |

Zinc Rougher Flotation Cell

| | 1st Phase | 2nd Phase | Code Letter |
|---|---|---|---|
| Number of cells | 6 | | B |
| Flotation cell type | conventional | | A |
| Flotation slurry, pH | 7-9 | | A |
| Retention time lab test, min | 4 | | A |
| Retention time scale-up, lab to plant | 2 | 3 | A |
| Retention time design, min | 8 | 12 | A |
| Froth/aeration factor | 1.15 | 1.25 | B |
| Retention time, min | 9.2 | 15 | A |
| Number of rows | 1 | 1 | A |
| Air supply | Blowers | | A |
| Solids feed density, % | 25 | 25 | B |

Zinc First Cleaner Flotation Cell

| | 1st Phase | 2nd Phase | Code Letter |
|---|---|---|---|
| Number of cells | 8 | | B |
| Flotation cell type | conventional | | A |
| Flotation slurry, pH | 7-9 | | A |
| Retention time lab test, min | 5 | | A |
| Retention time scale-up, lab to plant | 2 | 3 | A |
| Retention time design, min | 10 | 15 | A |
| Froth/aeration factor | 1.25 | 1.5 | B |
| Retention time, min | 12.5 | 22.5 | A |
| Number of rows | 1 | 1 | A |
| Air supply | Blowers | | A |
| Solids feed density, % | 12.3 | 12.3 | B |

Zinc Second Cleaner Flotation Cell

| | 1st Phase | 2nd Phase | Code Letter |
|---|---|---|---|
| Number of cells | 8 | | B |
| Flotation cell type | conventional | | A |
| Flotation slurry, pH | 7-9 | | A |
| Retention time lab test, min | 4 | | A |
| Retention time scale-up, lab to plant | 2 | 3 | A |
| Retention time design, min | 8 | 12 | A |
| Froth/aeration factor | 1.25 | 1.5 | B |
| Retention time, minutes | 10 | 18 | A |
| Number of rows | 1 | 1 | A |
| Air supply | Blowers | | A |
| Solids feed density, % | 13.6 | 13.6 | B |

Zinc Third Cleaner Flotation Cell

| | 1st Phase | 2nd Phase | Code Letter |
|---|---|---|---|
| Number of cells | 8 | | B |
| Flotation cell type | conventional | | A |
| Flotation slurry, pH | 7-9 | | A |
| Retention time lab test, min | 4 | | A |
| Retention time scale-up, lab to plant | 2.25 | 3 | A |
| Retention time design, min | 9 | 12 | A |
| Froth/aeration factor | 1.25 | 1.5 | B |
| Retention time, min | 11 | 18 | A |
| Number of rows | 1 | 1 | A |
| Air supply | Blowers | | A |
| Solids feed density, % | 15 | 15 | B |

Lead Concentrate Thickener

| | 1st Phase | 2nd Phase | Code Letter |
|---|---|---|---|
| Number | 1 | | A |
| Model | Bridge-supported | | A |
| Type | High rate | | A |
| Feed rate, t/h | 4.51 | 6.18 | A |
| Area factor, $m^3/m^2h$ | 2.3 | 2.3 | A |
| Size, diameter, m | 5.0 | 6.0 | E |

Zinc Concentrate Thickener

| | 1st Phase | 2nd Phase | Code Letter |
|---|---|---|---|
| Number | 1 | | A |
| Model | Bridge-supported | | A |
| Type | High rate | | A |
| Feed rate, t/h | 10.42 | 20.34 | A |
| Area factor, $m^3/m^2h$ | 2.3 | 2.3 | A |
| Size, diameter, m | 6.0 | 9.0 | E |

Pulp Rheology

| | | |
|---|---|---|
| Non-Newtonian fluid | Plastic fluid | A |
| U' flow pulps, $N/m^2$ | <30 | A |

### 17.3.7 Reagents

Sodium Cyanide:

| | | |
|---|---|---|
| Consumption, kg/t mill feed | | |
| Flotation | 0.03 | A |
| Cyanide leaching | 1.00 | A |

Quicklime (Pebble Lime) (92% active):

| | | |
|---|---|---|
| Consumption rate, kg/t mill feed | | |
| Lead flotation | 0.67 | A |
| Zinc flotation | 0.11 | A |
| Cyanide leaching | 2.0 | A |
| Detoxification | 1.0 | A |

Frother (DF-1012):

| | | |
|---|---|---|
| Consumption, kg/t mill feed | | |
| Flotation | 0.10 | A |

Lead Promoter (A-3418):

| | | |
|---|---|---|
| Consumption, kg/t mill feed | | |
| Flotation | 0.03 | A |

Zinc Collector (SIPX):

| | | |
|---|---|---|
| Consumption, kg/t mill feed | | |
| Flotation | 0.03 | A |

Methyl Isobutyl Carbinol (MIBC):

| | | |
|---|---|---|
| Consumption, kg/t mill feed | | |
| Flotation | 0.08 | A |

Zinc Dust:

| | | |
|---|---|---|
| Consumption, g Zn/g Ag | | |
| Merrill-Crowe | 0.04 | A |

Diatomaceous Earth (DE):

| | | |
|---|---|---|
| Consumption, kg/t mill feed | | |
| Merrill-Crowe | 0.10 | A |

Copper Sulphate:

| | | |
|---|---|---|
| Consumption, g /t mill feed | | |
| Detoxification | 85.50 | E |
| Zinc flotation | 98.68 | A |

Sulphur (Sodium Metabisulfite):

| | | |
|---|---|---|
| Consumption, g $SO_2$/g $CN_{wad}$ | 2.0 | F |
| Detoxification | 2.0 | A |
| Consumption, $Na_2S_2O_5$ kg/t mill feed | 0.99 | E |

Sulphur (S):

| | | |
|---|---|---|
| Consumption, kg/t mill feed | 0.57 | B |

Anti-scalant:

| | | |
|---|---|---|
| Consumption, kg/t mill feed | | |
| Process Water | 0.01 | A |

Flocculant:

| | | |
|---|---|---|
| Consumption, kg/t mill feed | | |
| Pre-Leach Thickener | 0.03 | B |
| CCD Thickeners (4) | 0.12 | B |
| Cyanide Recovery Thickener | 0.03 | B |
| Zinc Concentrate Thickener | 0.01 | B |
| Lead Concentrate Thickener | 0.01 | B |
| Tailings Thickener | 0.03 | B |

## 17.4 PROJECTED REQUIREMENTS FOR WATER, PROCESS MATERIALS AND ENERGY

The projected requirements for oxide and sulphide reagent materials for the processing plant are detailed above in Sections 17.2.2.6 and 17.2.3.11 respectively.

### 17.4.1 Water System

The water supply for operations will primarily come from open pit dewatering, water wells, and run-off from precipitation events and reclaimed water contained in the TSF. Reclaim water from the TSF and water from the tailings thickeners will be stored in the process water tank for recycle to process operations. Water from the process water tank will be distributed by pumps to the concentrator usage points and gland water for pumps.

Water from the well field pumps will be pumped to the fresh/fire water tank. Water from the fresh/fire water tank will be used for reagent preparation, crushing area dust control and to feed the potable water system for general supply and eyewash/safety showers.

### 17.4.2 Water Requirements

A site-wide water balance analysis was performed by Tierra Group International (Tierra, 2012b) for the Pitarrilla Project to provide an estimate of freshwater make-up requirements for the mine operations and to estimate the expected fluctuations in the size of the reclaim water pond in the TSF. A description of the water requirements is included in Section 18.11.7.

The process plant receives water from various sources: moisture in run-of-mine ore, freshwater from groundwater pumping wells, reclaim water from the TSF, and any excess inflows from pit dewatering and ramp inflows that are not used directly for dust control. The main input parameter for the process plant is the tailings slurry that is 55% solids (by weight).

The TSF receives water contained in the slurry from the process plant, direct precipitation, and surface runoff. Free water within the TSF is lost to evaporation and seepage (minimal) or reclaimed to the process plant. Associated input parameters for the water balance are:

- The process plant requires a start-up volume of 606,000 $m^3$, equal to three months of pumping from the groundwater supply at 80 L/s. After the first year of operations, a water pool of approximately 600,000 $m^3$ must be maintained to provide the minimum 3 m water pool depth required for reclaim barge operation. Maintaining a minimum water pool limits the amount of available reclaim water during the dry season, because water storage must be carried over from the wet season to ensure that the pool lasts the duration of the dry season, and carryover storage is subject to significant evaporation losses within the TSF.

- Available reclaim water is restricted by the need to maintain the minimum water pool volume in the TSF. Monthly available reclaim is the water remaining in the TSF after evaporation. The actual reclaim is either the available reclaim or the process water demand, whichever is less.
- The TSF must maintain sufficient space above the operating pool to store the 50-year 24-hour flood event, with 2 m of residual freeboard.
- Seepage through the impoundment and dam is limited by the use of a geosynthetic liner; however, seepage losses may occur from potential liner defects. The unit seepage rate is estimated as 1,140 litres per hectare per day, and this rate is applied to the portion of liner covered by either tailings or water during any given time.

Due to evaporation losses and entrainment, coupled with low precipitation, the TSF water pool cannot provide sufficient reclaim water to fulfill process plant demand. On average, 53% of the required process water must be provided from freshwater make-up.

### 17.4.3 Processing Plant Energy Requirements

The total energy requirements for the processing plant will be in the range of 22.6 to 32.3 MW.

# 18 INFRASTRUCTURE

This section describes the infrastructure facilities that will support the Pitarrilla Project mine and process site facilities. The infrastructure facilities include the ancillary buildings, offices and support buildings, access roads into the plant site, source of electrical power and power distribution, fuel supply, storage and distribution, source of fresh water and water distribution, dewatering and drainage facilities, waste management, transportation and shipping, communications, mobile equipment, and the TSF.

## 18.1 SITE LAYOUT

To reduce the amount of earthworks required to construct the Pitarrilla Project, the majority of the process facilities are located in a compact arrangement near the mill buildings. The grinding, flotation and leaching-CCD circuit locations were selected on ground where the natural slope approximately matches the required slope for the foundation of the process buildings, thereby minimizing cut and fill, where bedrock is located close to the surface for improved foundation purposes, and in a location within close proximity to the open pit. Figure 18-1 and Figure 18-2 show the Pitarrilla Project site layout and process plant site plan, respectively.

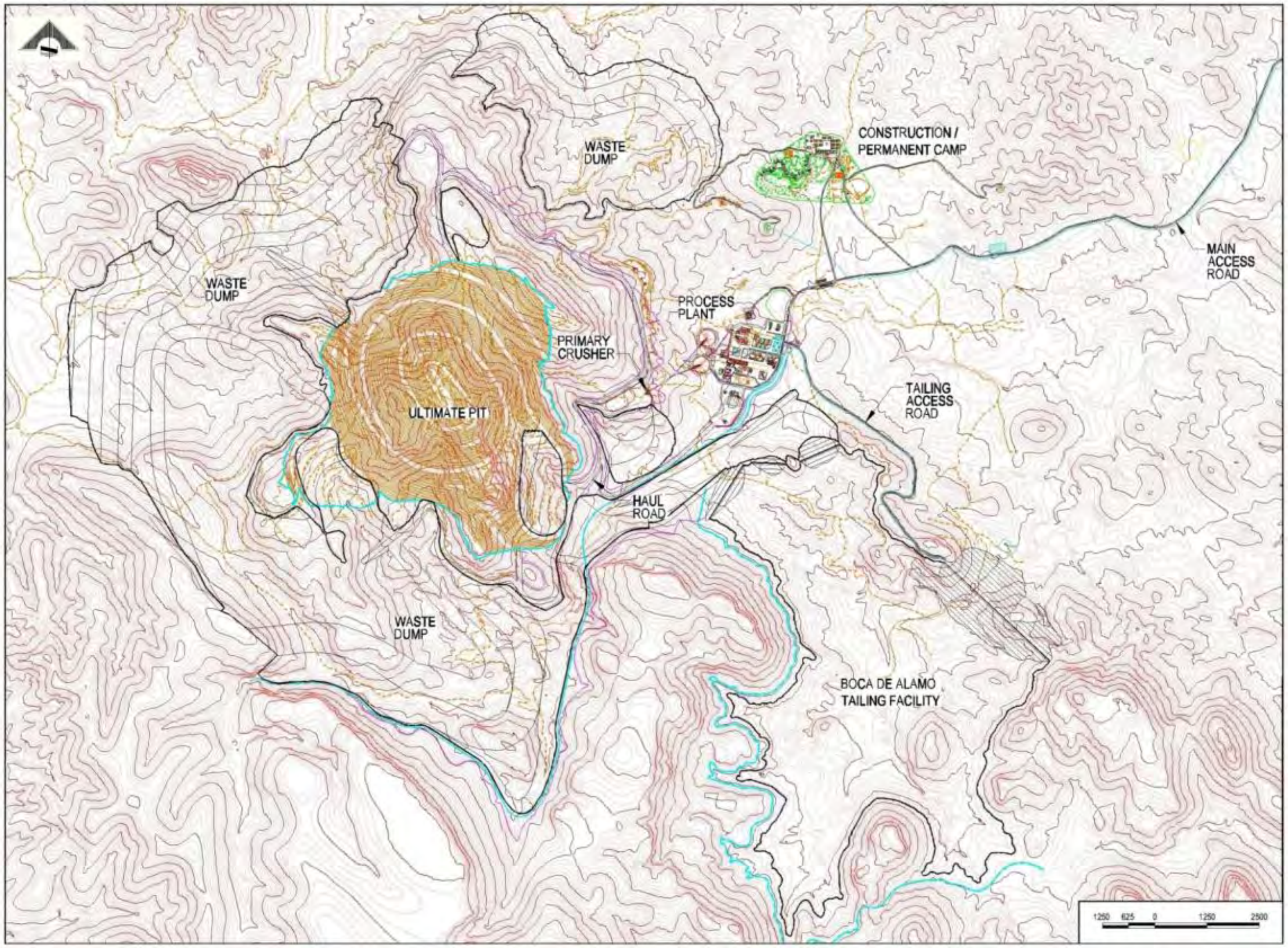



**Figure 18-1: Site Layout**

Source: M3, 2012

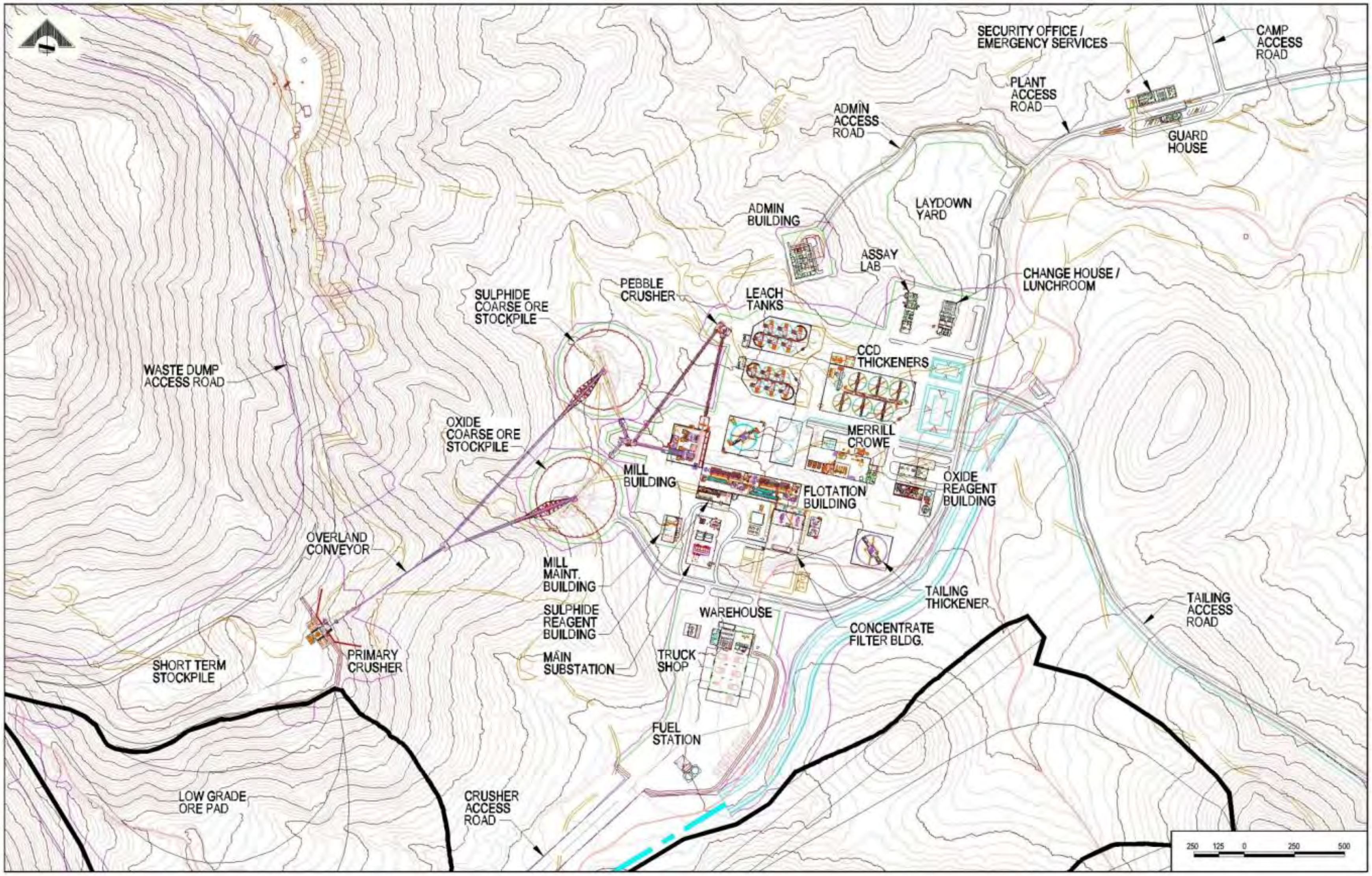



**Figure 18-2: Process Plant Area Site Plan**
Source: M3, 2012

## 18.2 ANCILLARY FACILITIES

The ancillary facilities necessary to support the Pitarrilla mine and process operations are outlined below:

- *Entry Gate/Guard House/Safety Area* – This area includes a small entry guard house with an adjacent medical clinic building, security office and emergency response building. The area also includes a truck scale and visitor temporary parking.
- *Administration Buildings* – Offices and conference rooms for management and staff.
- *Truck Shop/Warehouse Facility* – A six-bay mine maintenance shop, adjacent warehouse, small vehicle maintenance facility and mezzanine.
- *Truck Wash* – A one-bay wash facility with four water cannons and a water re-circulation and cleaning system.
- *Mill Maintenance Facility* – A maintenance building that will service processing equipment.
- *Laboratory* – Wet and metallurgical laboratories with offices and training rooms.
- *Change House/Lunch Room* – A building with showers, lockers and food service areas.
- *Fueling Station* – A station with diesel and gasoline storage and dispensing.

## 18.3 OPERATIONS CAMP

The operations camp includes facilities for approximately 320 beds and will be completed in time for start-up of the process facilities. Table 18-1 summarizes the facilities included in the operations camp.

**Table 18-1: Operations Camp Facilities**

| Description | Number |
|---|---|
| Supervisors' dormitories, 32 beds | 3 modules |
| Regular dormitories, 288 beds | 12 modules |
| Roofed deck | 7 units |
| Dining hall for 220 people with kitchen | 1 |
| Executive houses | 20 |
| Recreation fields | 7 |
| Access roads, parking lots, water and sewer lines | 1 |
| Waste water treatment plant | 1 |
| Gym and swimming pool | 1 |
| Administration offices, laundry facility, maintenance area | 1 |
| Convenience store | 1 |

## 18.4 ON-SITE AND OFF-SITE ROADWAYS

### 18.4.1 Main Access Road

The primary access road to the Pitarrilla Project site will be 47 km long and will be developed and/or upgraded as follows:

1. Approximately 36.7 km of an existing, unpaved, narrow public road north from Highway 45 will be upgraded. The road will be widened and straightened, drainage structures will be added, and accessibility (by re-routing the road in certain areas) will be improved.

2. A 150 m long concrete bridge will be constructed to cross the Nazas River.

3. From the bridge, a new 9.7 km unpaved access road will be constructed to the entrance guard station of the Pitarrilla Project site.

An existing access road, via Casas Blancas, is presently used for access to the project site; however, this road will not be used as the primary access road and hence minimal, if any, improvements will be performed on it.

#### 18.4.2 Primary On-site Roadways

Primary, on-site, unpaved roads will be 8 m wide and sloping 2% away from the crown to the road edges. The maximum grade will be 7%. Culverts will be placed as necessary for surface water run-off control. Water trucks will spray roads with water and possibly a chemical stabilizer for dust control.

#### 18.4.3 Secondary On-site Roadways

Secondary, on-site, unpaved roads will generally be 6 m wide with safety berms. Grades will vary according to use and terrain. These roads will provide access to less-frequented areas such as the TSF, explosives storage facility, power facilities, water pumping stations and pipelines, and waste rock storage areas.

### 18.5 POWER SUPPLY & DISTRIBUTION

#### 18.5.1 General

The Pitarrilla Project will utilize an electrical interconnection to the national transmission grid to supply power to the mine site. Silver Standard has requested that up to 40 MW of power be provided by Comisión Federal de Electricidad ("CFE"), Mexico's national transmission utility, along a 115 kV transmission line. CFE has stated that it could provide power to the project in two stages: an initial 17 MW from its existing Nuevo Ideal substation and the final 40 MW upgrade, once the build-out to its Canatlán substation is completed. CFE is also investigating an overall upgrade to 230 kV from its Canatlán substation to the Pitarrilla Project site; however, information on this upgrade was not complete by the date of this Technical Report.

#### 18.5.2 Main Substation

The main substation will be constructed in close proximity to the grinding area of the mill. The plant loads up to approximately 32.3 MVA will be shared by two, 25/33/42 MVA step-down transformers which will convert the incoming 115 kV to the utilization voltage of 13.8 kV. The transformers are sised so that one transformer can carry the total load in the event of a transformer failure. The substation step-down transformers will be protected by a 115 kV SF6 circuit breaker. Switchgear (15 kV), located in an elevated electrical house within the substation, will provide additional protection to the transformers and to the feeders used for distributing power to the various processing areas. A 1.5 MVA standby generator will be included as part of the substation to provide power to critical loads, such as thickeners and some pumps, as well as provide power globally for lights, etc. as needed. A capacitor bank/harmonic filter will be provided in the substation to correct for low power factors and mitigate disharmonic transmission.

#### 18.5.3 On-site Power Distribution

Power distribution from the main substation to other area substations will utilize underground and overhead 13.8 kV circuits. The largest power loads are in the grinding circuits. Both the SAG and ball mills will be each powered by two 4,100 kW (5,500 hp) motors. The SAG mill has variable speed induction motors and the ball mill has wound rotor motors. The mill area is fed from the substation with underground feeders at 13.8 kV. The flotation area, leaching and CCD areas are also fed by underground cables. All underground cables will be in concrete-encased duct bank and will utilize PVC conduits. Local area transformers will be provided to reduce voltages to the utilization voltages of 4,160 V and 480 V. A 13.8 kV power line will serve the primary crusher and mine areas. Another 13.8 kV line will serve the laboratory, administration building, and security and medical facilities, and will extend to the water well field for the booster pumps located approximately 10 km east of the site.

#### 18.5.4 Process Control System

A process control system will be installed to provide remote start, control, and monitoring of all process and water well systems. A power management system will also be provided. The master control room in the mill will house the main operator control stations. Provision will be made for an expert control system for the SAG mill.

### 18.6 WATER SUPPLY & DISTRIBUTION

#### 18.6.1 Delivery System

The freshwater make-up requirement for the Pitarrilla Project facilities is estimated to be approximately 115 to 150 L/s over the life of the mine. The well fields and water supply pipeline will be designed for this peak demand.

The proposed Pitarrilla Project well field is comprised of a minimum of three individual wells, each designed to pump 83 L/s of ground water to a booster station. With a water table level located several hundred metres below surface, the individual wells will be outfitted with submersible pumps. The booster station will consist of four pumps, each with a 127 L/s capacity. From the booster station, water will be pumped to a fresh/fire water storage tank via an 11 km long, 10-inch (254 mm) water line. The above ground water supply pipeline route from the well site will be primarily located along the main access road to the site.

### 18.6.2 Plant Water Distribution

The freshwater system will consist of a gravity distribution network from the fresh/fire water storage tank to the process facilities requiring freshwater. Freshwater will be pumped from the booster station to a common freshwater and fire water tank located near the process plant. The bottom capacity of the tank is to be used as the fire water reserve while fresh water usage for the plant will be taken from the capacity above the fire water reserve. Flow of fire water and freshwater is provided by both gravity and pumping, depending on the destination.

The fire water system will consist of a pressurised distribution network from the fresh/fire water storage tank to a system of hydrants around the ancillary and process facilities. Fire protection hose cabinets will be provided in process buildings and sprinkler systems in the administration building, change house, laboratory, warehouse, camp kitchen and laundry.

The potable water system will consist of a potable water treatment package, a potable water tank, and a distribution network capable of delivering potable water to all ancillary buildings, process facilities, restrooms, operations camp and safety showers.

The process water system will utilize the TSF for the majority of the process water usage. Supernatant water from the facility will be pumped via a reclaim water system to a process water tank located at the mill area. Water from the process water tank will be fed to the mill building.

## 18.7 WASTE MANAGEMENT

### 18.7.1 Landfill

Solid waste will be disposed in an onsite Class III industrial landfill. Presently, the landfill will be located east of the TSF, although the location may be changed.

### 18.7.2 Sewage

Sewage will be disposed of using standard septic tank and leach field systems. Septic tank systems will be installed for sanitary disposal of waste from the administration building, safety/security office, assay/metallurgical laboratory, mine maintenance facility, plant process facilities and mine dry. Septic tanks and leach fields are sised in accordance with building occupancy and type of use.

## 18.8 TRANSPORTATION AND SHIPPING

### 18.8.1 Lead and Zinc Concentrates

Lead and zinc concentrates will be transported by truck from the mine site for distribution to customer warehouses via highway transport. The planned production rate for lead concentrate averages 19,800 wmt/a and 50,500 wmt/a for zinc concentrate. However, production rates vary widely from years where no concentrate will be produced up to 80,500 wmt/a for lead concentrate and 289,500 wmt/a for zinc concentrate. The tractor trailers used to haul the product have a capacity of 24 wmt. Thus the traffic due to product shipment is estimated to be on average 270 trucks per month or nine trucks per day or at peak 1,285 trucks per month or 43 trucks per day.

#### 18.8.2 Silver Doré

Silver will be produced from the processing facility and transported via secure transport means to customer warehouse facilities. Doré production will vary from minimum years of 1,000,000 ozs to peak years of just over 8,000,000 ozs assuming 98% purity. This equates to weekly production varying from around 19,000 ozs to 150,000 ozs (or approximately 600 to 5,000 kg).

#### 18.8.3 Diesel Fuel

Diesel fuel is a major consumable for the mine equipment. Diesel fuel is available from suppliers in tank trucks with a capacity of approximately 10,000 litres per tank. At a consumption rate peaking at 42 million litres per year, the delivery rate will be about 350 truckloads per month (12 trucks per day). Diesel delivery can be scheduled seven days per week during the day between shift changes.

#### 18.8.4 Gasoline

Gasoline use is limited to the light vehicle fleet. Average consumption is estimated to be 20,000 litres per month.

#### 18.8.5 SAG and Ball Mill Grinding Balls

Grinding balls are a major consumable for the grinding area. Grinding balls are received in bulk by bottom dump or end dump trucks with a capacity of 24 tonnes. At an annual requirement of about 10,750 tonnes when processing sulphide ore, approximately eight to nine deliveries will be required each week. Approximately seven to eight truck loads are required per week when processing oxide ore.

#### 18.8.6 Regrind Mill Grinding Balls

Regrind mill grinding balls are also a major consumable for the grinding area when processing ore for flotation. Grinding balls are received in bulk by bottom dump or end dump trucks with a capacity of 24 tonnes. At an annual requirement of about 175 tonnes, approximately one delivery will be required each month.

#### 18.8.7 Miscellaneous Consumables

Miscellaneous consumables consist of reagents used in the process, wear parts used in the crushing and grinding process, and explosive powder and caps used by the mine. Reagents used in the flotation circuit are: lime, 3418A Promoter, Dowfroth 1012, sodium cyanide, copper sulphate, SIPX, MIBC, flocculant and anti-scalant. Reagents used in the leach circuit include lime, sodium cyanide, flocculant, copper sulphate, diatomaceous earth, zinc dust, refining fluxes and sulphur. Wear parts used in the crusher and grinding process include primary crusher liners, SAG and ball mill liners and screen panels, and regrind mill liners. All miscellaneous reagents and consumables are assumed to arrive at the plant site in one to four trucks per day (more when processing oxide ore) and unloaded into their respective storage vessels or warehousing areas.

## 18.9 COMMUNICATIONS

Communication and information systems will provide for high-quality voice and data transmission throughout the existing plant operation.

The telephone utility, Telmex, will supply telephone services for the Pitarrilla Project mine site. The project telephone system will be tied into the Telmex system via a fiber optic cable that will be hung on the 115 kV transmission line's overhead ground wire from the CFE Nuevo Ideal substation to the Pitarrilla Project mine site main substation. Cellular phones and radios (handheld/mobile/base stations), with dedicated frequencies and a repeater pair, will also be provided for remote operations throughout the mine site.

A new, high-capacity wide area network ("WAN"), with either a satellite internet connection or the above high-speed fiber optic connection, will be utilised for broadband data communications. A local area network ("LAN") at site will have sufficient capacity for use by operations, maintenance, engineering, and administration personnel. Computers connected to the LAN will be able to transfer information, use centralised server systems, and be linked for e-mail, voice and data transfer internally via the LAN, and externally via a WAN connection.

## 18.10 SITE MOBILE EQUIPMENT

Mobile equipment for the facilities will comprise mine production equipment, mine support equipment and process plant support equipment. The mine production and support equipment is discussed in detail in Section 16. The process plant support equipment and areas of use is itemised in Table 18-2.

**Table 18-2: Process Plant Support Equipment**

| Equipment | Class | Notes |
|---|---|---|
| Front-end Loader | 5.7 $m^3$ capacity | |
| Concentrate Loader | 5.7 $m^3$ capacity | |
| Excavator | 20 ton capacity | |
| Wheel Loader | 10.7 ton | Backhoe/loader configuration |
| Motor Grader | 14.4 ton | |
| Dozer (track) | 25.7 ton | For DRS and miscellaneous use |
| Water Truck | 10 ton off-road | |
| 4x4 Pickup Trucks (4) | | |
| All Terrain Forklift | 11.3 tonne | Telehandler |
| Indoor Forklifts (3) | 5 ton | Propane-powered |
| Man Lift | | |
| Personnel Busses (2) | 25 Passengers Each | |
| Haul Truck (articulated) | 30 ton | For DRS and miscellaneous use |
| Boom Trucks (2) | 24 ton | |
| Mobile Cranes (2) | 45 ton | |

DRS = Development Rock Stockpiles

## 18.11 TAILINGS STORAGE FACILITY

### 18.11.1 General

The TSF was designed by Tierra Group International of Denver, Colorado, U.S.A. The TSF is located in an area south of the process plant known as Boca de Alamo as shown on Figure 18-3. This site is the most efficient site near the process plant with a storage-to-dam fill ratio of 14.3 (by volume). The TSF is contained by natural ridges with additional containment provided by four dams: the Main Dam in the northeast and three Saddle Dams. Dams will be built predominantly with rockfill with zones of finer-grained material on the upstream portions. The TSF will be constructed in seven stages by downstream construction as shown on Figure 18-4. Only the Main Dam is needed for Stages 1 and 2 with construction of Saddle Dams 1 and 2 starting during Stage 3. Saddle Dam 3 is required during the Stage 5 expansion.

The TSF can be expanded to 159 Mt of capacity with the addition of two extra dam raises (Stages 6 and 7) to match the revised mine plan, which increased the amount of potential mineral resources to 157 Mt over an extended mine life of 30 years of tailings production. The Stages 6 and 7 expansions would utilize similar downstream construction methodology as Stages 2 to 5. Saddle Dams 1 to 3 would merge to form a single, continuous dam and an additional saddle dam (Saddle Dam 4) would be constructed on the ridgeline between the Main Dam and Saddle Dam 2 to provide containment for the additional storage capacity. An ultimate dam crest elevation of 1690 m would be reached for the 159 Mt capacity TSF.

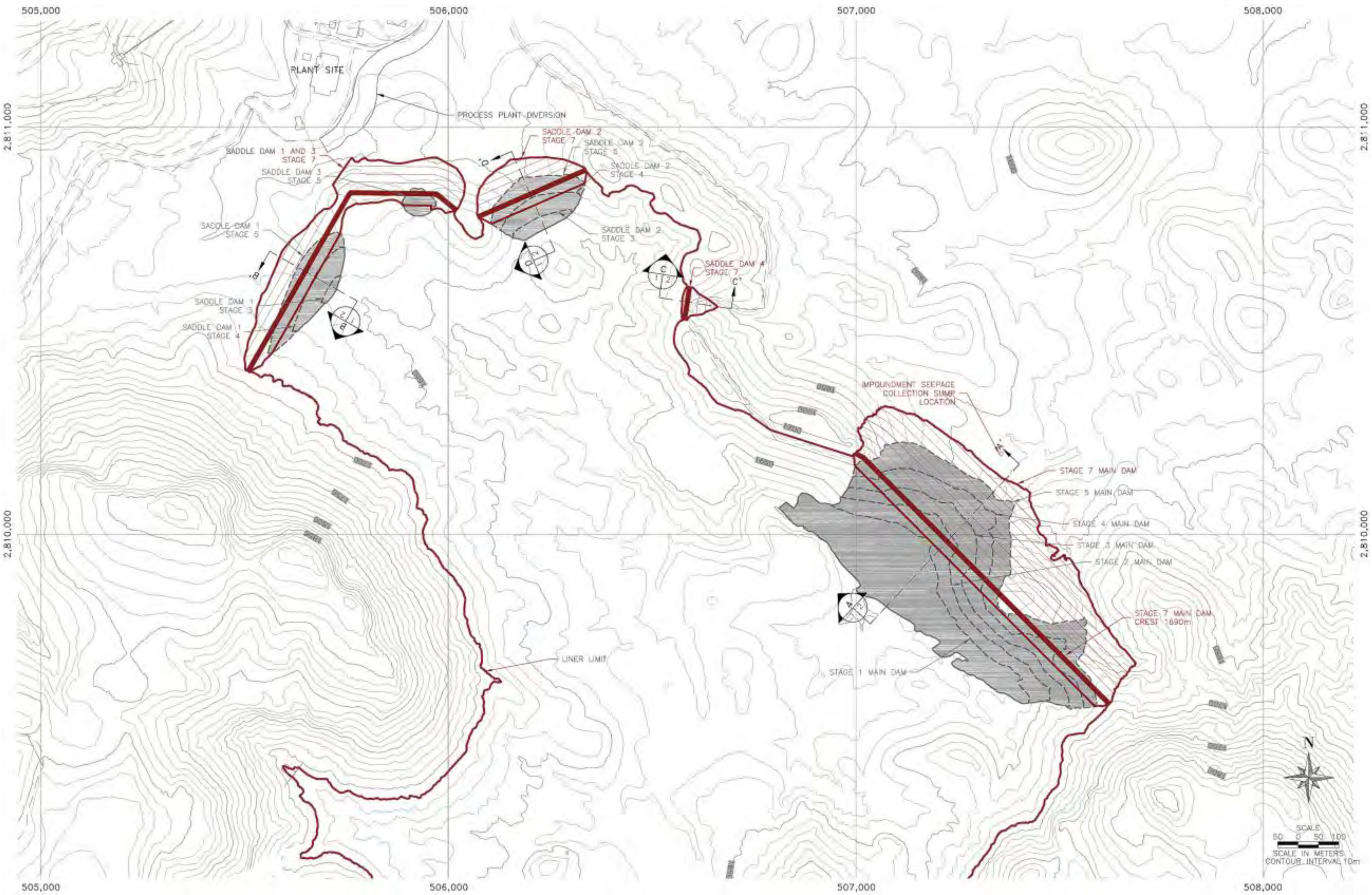



**Figure 18-3: TSF General Arrangement**
Source: Tierra Group International, 2012

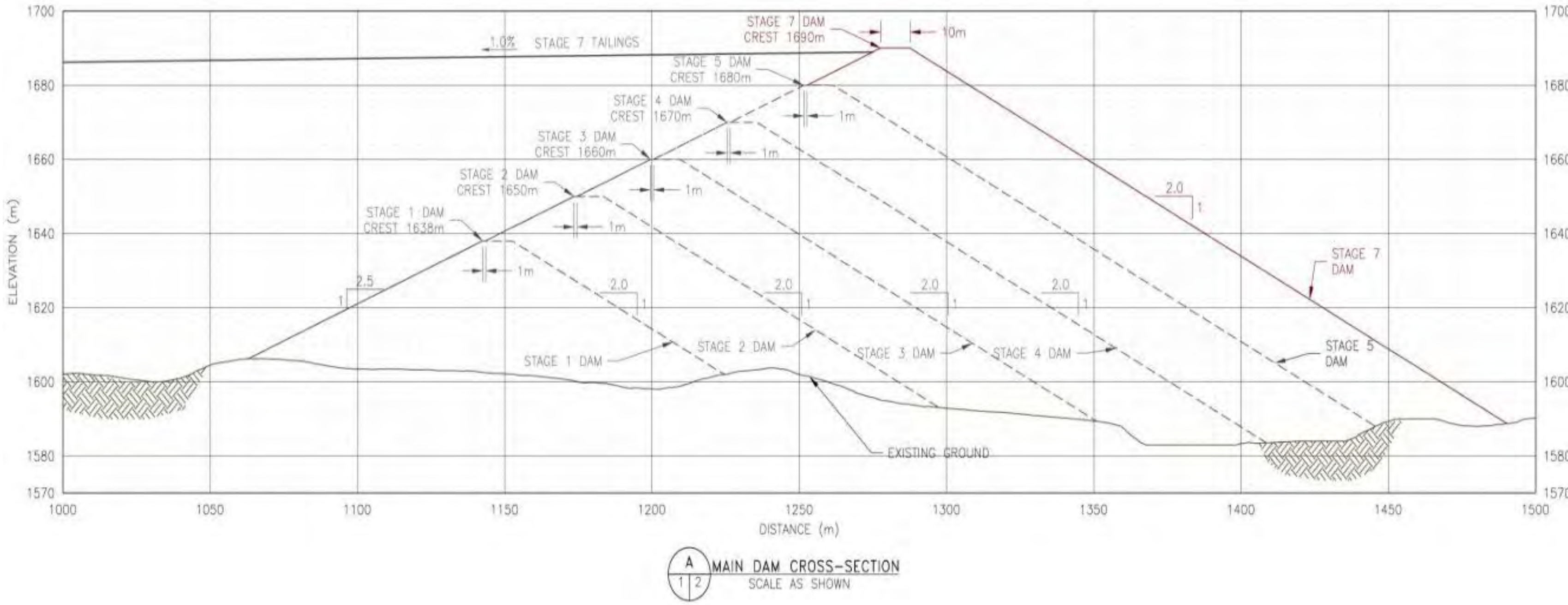



**Figure 18-4: TSF Main Dam Cross-Section (Stages 1 to 7)**
Source: Tierra Group International, 2012

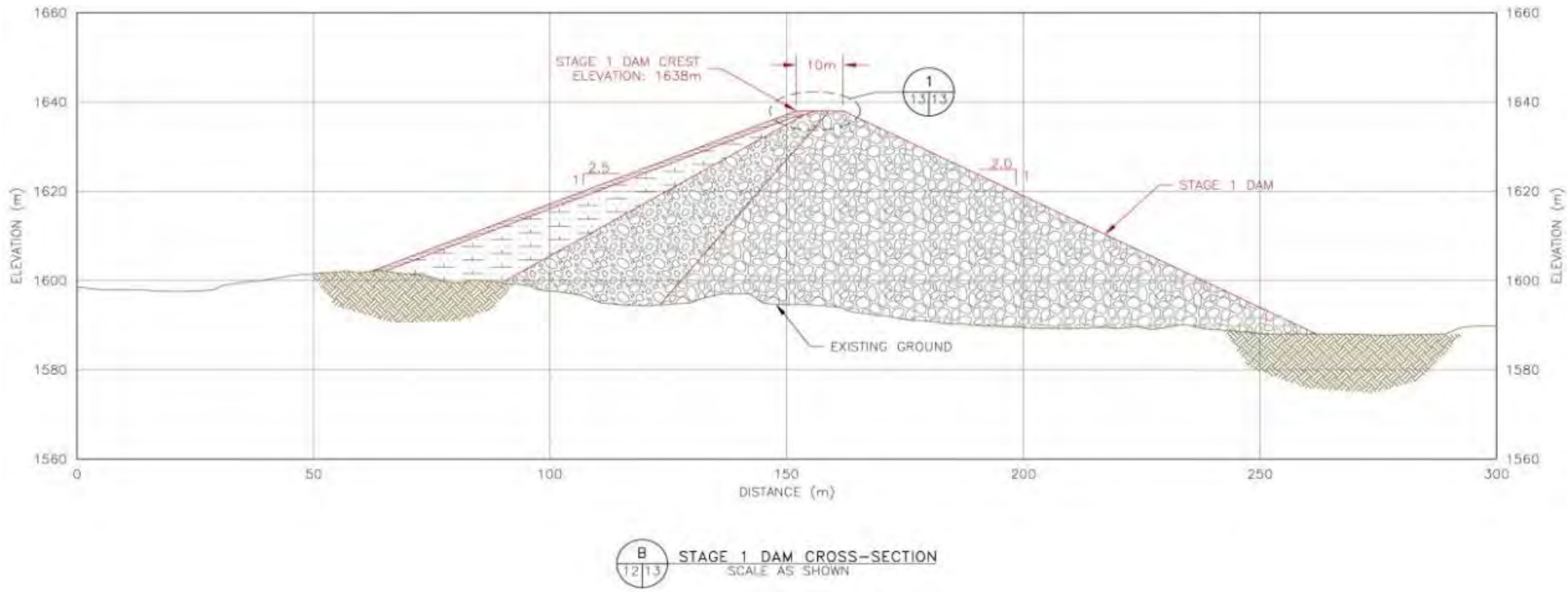



**Figure 18-5: TSF Main Dam Cross-Section (Stage 1 Only)**
Source: Tierra Group International, 2012

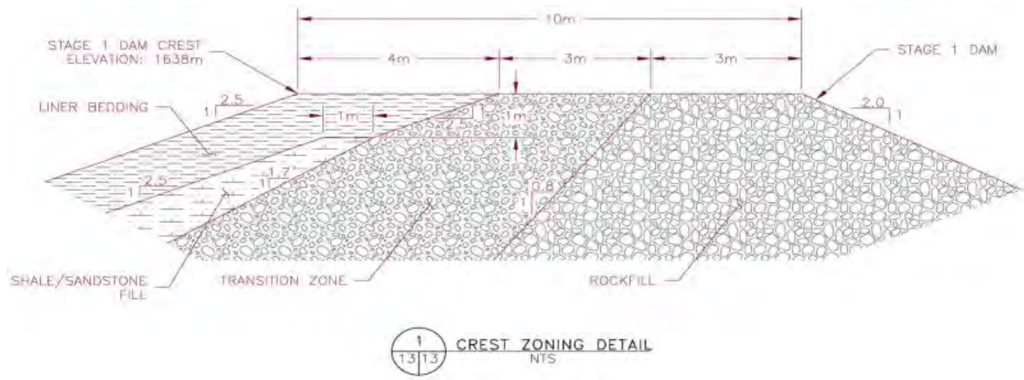



**Figure 18-6: TSF Main Dam Cross-Section Detail (Stage 1 Only)**

Source: Tierra Group International, 2012

**Table 18-3** summarizes the TSF crest elevations and construction costs for each stage.

**Table 18-3: TSF Embankment Stages**

| Stage | Required Crest Elevation (m) | Cumulative Storage Capacity (Mt) | Incremental Stage Raise Cost (US$) | Incremental Cost/Additional Storage (US$/tonne) |
|---|---|---|---|---|
| 1 | 1638 | 9 | $20,808,000 | $2.31 |
| 2 | 1650 | 22 | $7,177,000 | $0.55 |
| 3 | 1660 | 43 | $6,043,000 | $0.29 |
| 4 | 1670 | 71 | $6,614,000 | $0.24 |
| 5 | 1680 | 112 | $7,845,000 | $0.19 |
| 6 | 1686 | 140 | $6,902,000 | $0.25 |
| 7 | 1690 | 159 | $5,134,000 | $0.27 |
| **Total** | | **159** | **$60,523,000** | **$0.38** |

### 18.11.2 Dam Zones

The Main Dam and saddle dams will be built predominantly with rockfill borrowed from the plantsite excavation, a borrow source within the impoundment basin, and waste rock from the open pit mining operations. A 3 m wide zone (measured horizontally) of clayey sand fill will be constructed on the upstream face of the dams providing a bedding layer for the geosynthetic liner system as shown on Figure 18-3 and Figure 18-4. A minimum 3 m wide transition zone (measured horizontally) will be constructed between the liner bedding and rockfill materials to act as a filter preventing potential fine soil migration from the liner bedding to the rockfill. The combination of the geosynthetic liner system and free-draining rockfill simplifies dam construction obviating the need for internal dam drains. Saddle dams will be of similar geometry to the Main Dam.

Potential fine soil migration (piping) within the dam will be minimised by constructing a transition zone between the liner bedding and rockfill. The transition zone will consist of silty sands or gravels that are filter compatible between the liner bedding and rockfill. Borrow sources for this material include the plantsite area and portions of the fault blocks too coarse for use as liner bedding fill. Other sources of transition zone fill include colluvium found at the base of most hills in the TSF area.

The Stage 1 dam transition zone will include an additional zone comprised of inter-bedded shale and sandstone and/or colluvium as shown on Figure 18-5 and Figure 18-6. The Stage 1 dam has a larger transition zone than Stages 2 through 7 to accommodate the excess cut materials from the plantsite.

Rockfill makes up the largest portion of the dams. The Stage 1 dam will include materials from the plantsite grading activities and will be excavated from rockfill borrow sources within the impoundment where steep rock faces require re-grading prior to installing the geosynthetic liner.

Stage 2 rockfill will be borrowed from a mine waste stockpile while Stages 3 through 7 will use run-of-mine waste rock hauled directly to the dams by the mine haul truck fleet. Rockfill compaction will be accomplished by the haul truck fleet in addition to a large compactor along the edges.

### 18.11.3 Site Description

#### 18.11.3.1 TSF Site Geology

The TSF site and its contributing drainage basin consist primarily of extrusive Tertiary/Oligocene volcanic rock units. Older, underlying Cretaceous sedimentary units are exposed in the broad, relatively shallow valleys located north of the proposed dam. Surficial soils within the proposed facility are weathered-in-place bedrock and deposited alluvium/colluvium including stream deposits.

#### 18.11.3.2 Seismicity

Deterministic and probabilistic seismic hazard analyses were prepared for the Pitarrilla site during previous TSF design work (MWH, 2011b). The design basis earthquake from the deterministic analysis was selected for the Pitarrilla TSF design which corresponds to a moment magnitude (Mw) 6.2 earthquake event and site specific peak ground acceleration (“PGA”) of 0.23g.

### 18.11.4 Impoundment Containment

Given the relatively high hydraulic conductivity of the surficial soils (approximately $10^{-4}$ cm/s) and bedrock ($10^{-3}$ to $10^{-6}$ cm/s), a geosynthetic liner system was incorporated into the facility design to contain tailings solids and water. The lining system selected consists of a 60-mil linear low-density polyethylene (“LLDPE”) geomembrane liner underlain by fine-grained soil as liner bedding. The fine-grained soil underlying the geomembrane (liner bedding) will protect the liner from coarse particles that could puncture the liner and provides additional seepage protection. Liner bedding will have a minimum thickness of 0.3 m. Clayey sand covers most of the impoundment area with an average depth of 1.2 m.

The face of the dams will be lined with geosynthetic clay liner (“GCL”) to function as a secondary liner. The GCL provides an added seepage barrier to minimize flow through the dams and provides cushioning for the geomembrane.

Underdrains have been incorporated in the TSF design as a means to relieve elevated fluid pressures and to reduce the potential for build-up of groundwater levels beneath the TSF, although such a condition is unlikely. The underdrains mitigate impacts of TSF leakage by providing an ability to collect and concentrate leakage by capturing most of the seepage through the liner. An additional underdrain will be constructed conveying spring flow identified in the Main Dam footprint. These springs are not under the impoundment so flow emanating from the springs will be discharged to the existing drainage downstream of the Main Dam.

### 18.11.5 Slope Stability Analysis

Seepage and slope stability analyses were performed to determine factors of safety ("FOS") against dam slope failure for the various stages of the dam and its foundation. Iterative analyses were performed to assure that the FOS set forth in the design criteria were achieved utilizing the least amount of dam fill material possible, while developing practical, constructible dam geometry.

All stability analyses were conducted using a modeled phreatic surface. Boundary conditions used for the seepage analysis consisted of pressure head equal to the pond surface upstream of the dam and zero pressure head at the embankment toe. The geosynthetic liner was not included in the seepage or stability analyses. Preliminary seepage and stability modeling on select sections showed the dam geometry is not affected by including or not including the geomembrane.

Seepage and slope stability analyses were conducted for the Main Dam and Saddle Dams 1 and 2. Results of the slope stability analysis show that all cross-sections analyzed meet minimum FOS values for the loading conditions analyzed.

### 18.11.6 Hydraulic Structure Design

#### 18.11.6.1 Design Criteria

The minimum hydrologic design criteria for the TSF storm storage is the 50-year 24-hour storm event, with 2 metres of freeboard (SEMARNAT NOM-141, 2003). NOM-141 states that diversions must be designed for "normal and extreme runoff" without specifying what the "extreme runoff" event is. Since NOM-141 specifies the 50-year 24-hour event as the appropriate design storm, the 50-year 24-hour event was used for diversion design. No minimum freeboard requirement is specified by SEMARNAT for diversions.

#### 18.11.6.2 Design Storm and Runoff Modeling

A design storm is described in terms of the total rainfall depth, storm duration, and temporal distribution of rainfall over the storm duration. Storm water diversions, sediment traps and TSF operational freeboard were evaluated for the 50-year 24-hour design storm depth of 108 mm.

#### 18.11.6.3 Diversion Design

Surface water diversion around the TSF is necessary so *Aguas Nacionales* (National Waters as defined by CONAGUA are not captured. The TSF diversion allows surface water to be routed to adjacent drainages for use by downstream users. The TSF diversion was designed for the 50-year 24-hour storm event from the majority of the up-gradient watershed, and re-route it around the facility.

Fifty-year peak design flows predicted for the surface water diversion ranged from 44.1 $m^3/s$ to 51.6 $m^3/s$, increasing in the downstream direction as additional drainage area is accumulated. A diversion structure was designed at the diversion inlet to route flow into the diversion channel. Material cut during channel construction will be used to build the diversion structure. Fine-grained fill will be placed 1 m deep at the surface of the diversion structure to limit infiltration. Riprap will prevent erosion of the fine-grained fill.

The diversion channel was assumed to be excavated in bedrock, but the typical design nonetheless included 2:1 (H:V) side slopes within the channel, to allow for lining if required. Cut slopes above the channel level were assumed to be 1:1 for estimating purposes, but can be expected to vary from 0.5:1 to 2:1 during construction based on conditions encountered in the field. An access road is included in the project design to allow for inspection, repair, and maintenance.

18.11.6.4 Spillway

An emergency spillway suitable to pass the 50-year 24-hour event, with 1 m of residual freeboard to the dam crest (within the impoundment for wind/wave effects) was sised for the facility. The emergency spillway is unlikely to be used given the short time window (approximately one year) that it will be in service, but is nevertheless required to ensure that a large flood occurring during that period would not compromise the integrity of the dams. The emergency spillway is located in the southeastern portion of the impoundment near the TSF diversion outfall.

### 18.11.7 Water Balance Inputs

Principal inputs to the water balance for the TSF are driven by climatology, process characteristics, settling characteristics of the tailings, and the geometry of the TSF basin itself. Inputs are summarised below, organised according to their origin: climatology and hydrology, process plant, TSF, and mine pit / operations. With these inputs, a water balance was prepared by Tierra Group International for the TSF.

18.11.7.1 Climatology and Hydrology

Available climate data includes long-term regional data from CONAGUA and data collected from a PITMET. A summary of climatic conditions is shown in Table 18-4.

**Table 18-4: Climate Summary[1]**

| Parameter | Range | Annual Average |
|---|---|---|
| Temperature | 13.4 to 23.5 °C | 18.5 °C |
| Relative Humidity | 20 to 70% | - |
| Wind Speed | 3 to 5 m/s<br>(20 to 30 m/s maximum gusts) | 3.8 m/s |
| Solar Radiation | - | 227.6 Watts/$m^2$ |
| Barometric Pressure | 80.3 to 80.7 KPLa | 80.5 KPLa |
| Annual Precipitation | - | 407 mm |
| Annual Evaporation[2] | - | 2,130 mm |

[1] Data recorded at PITMET

[2] Evaporation correlated for Pitarrilla site with data from nearby CONAGUA meteorological station (El Palmito II)

18.11.7.2 Process Plant

The process plant receives water from various sources: moisture in run of mine ore, freshwater from groundwater pumping wells, reclaim water from the TSF, and any excess inflows from pit dewatering and ramp inflows that are not used directly for dust control. The main input parameter for the process plant is that the tailings slurry is 55% solids (by weight).

18.11.7.3 TSF

The TSF receives water contained in the slurry from the process plant, direct precipitation, and surface runoff. Free water within the TSF is lost to evaporation and seepage (minimal) or reclaimed to the process plant. A water balance overview, including these inputs and outputs for the TSF, is shown in Figure 18-7.

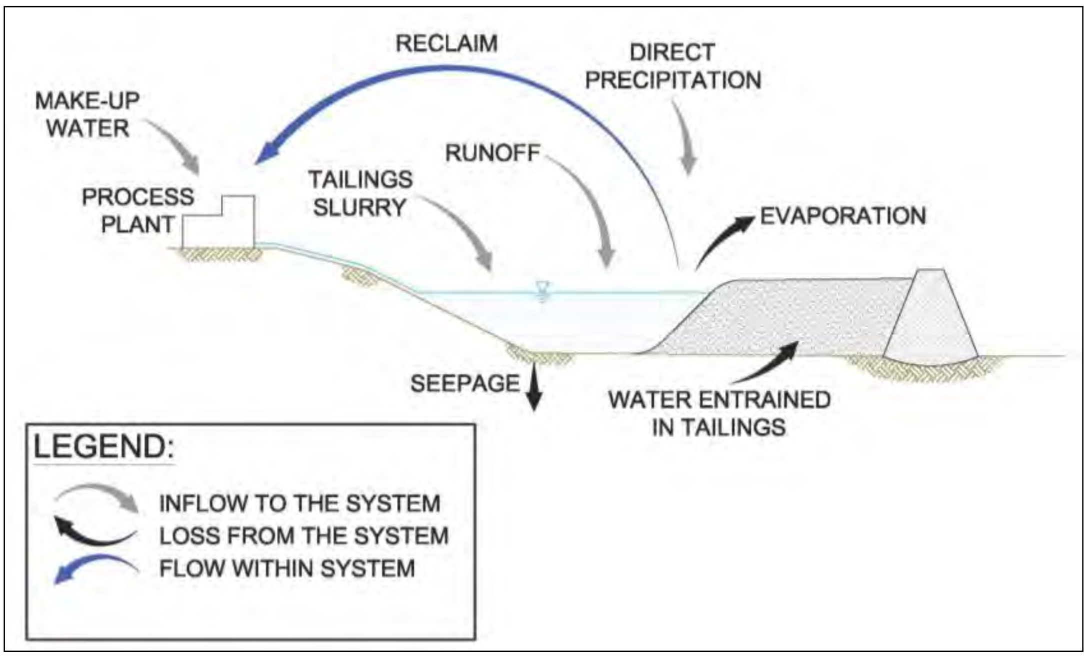



**Figure 18-7: Water Balance Overview**
Source: Tierra Group International, 2012

Associated input parameters for the TSF water balance are:

- After the first year of operations, a water pool of approximately 600,000 $m^3$ must be maintained, to provide the minimum 3 m water pool depth required for reclaim barge operation.
- Available reclaim water is the water remaining in the TSF after evaporation, and is restricted by the need to maintain the minimum water pool volume in the TSF.

- The TSF must maintain sufficient space above the operating pool to store the 50-year 24-hour flood event, with 2 m of residual freeboard.
- Seepage through the impoundment and dam is limited by the use of a geosynthetic liner; however, seepage losses may occur from potential liner defects. The unit seepage rate is estimated as 1,140 litres per hectare per day.

18.11.7.4 Mine Pit / Operations

The mine pit and other operations require water for dust control, which largely consumes the water recovered from pit dewatering activities. The mining operation produces the following inflows:

- Ore moisture is 4% by mass; and
- Pit dewatering provides a constant 10 L/s, starting in Mine Plan Year 12 (when the pit first intersects groundwater). Pit dewatering is largely consumed by dust control demands, except in the final years of operations when it may be introduced into the process plant.

18.11.7.5 Site-Wide Water Demands

Site-wide water demands accounted for in the water balance include:

- The camp consumes a constant 8.8 L/s for potable water and sanitation; and
- Dust control water demand for the pit, stockpiles, and waste rock dumps varies, ranging from 4.1 to 21.8 L/s, and averages 17.8 L/s throughout the mine life.

### 18.11.8 Reclaim / Freshwater Demand

Freshwater make-up is required seasonally for the majority of the TSF life under the base case climate conditions, and ranges from 0 L/s up to 133 L/s, averaging 67 L/s. Peak freshwater make-up requirements are 85 L/s during Stage 1, and 110 L/s during Stage 2. The corresponding peak total freshwater demand (process make-up plus the net demand from the rest of the site) is 115 L/s during Stage 1, 140 L/s during Stage 2, and 150 L/s for the life of the project. Later in the facility life, larger active beach and water pool areas, coupled with higher production rates, lead to increasing freshwater demand.

The TSF receives more upland runoff (due to increased liner area) as it is enlarged by successive dam raises, but suffers increased evaporation losses owing to the larger active beach and water pool area. The increase in evaporation losses exceeds the increase in precipitation runoff, so that more of the process water demand must be satisfied from freshwater during the dry season. Ore moisture is a minor component of the water balance, contributing 1.9 to 7.5 L/s according to fluctuations in the rate of ore delivery from the mine; while mine dewatering flows are largely consumed for dust control. The net effect is increasing freshwater demand throughout the facility life.

# 19 MARKETING AND CONTRACTS

## 19.1 MARKETING

Lead and zinc concentrate marketing for the Project is based on a study completed in July 2012 by BMMS, a concentrate marketing consultant based in Germany. The BMMS report considered Pitarrilla lead and zinc concentrate specifications as provided by Silver Standard and potential future market conditions as the basis for its analysis and conclusions. The BMMS report was reviewed in detail by the QP and it was concluded that the recommendations of the BMMS report support the conclusions of the Technical Report.

At the time of this Technical Report, no agreements to sell Pitarrilla lead and zinc concentrate have been entered into with smelters or traders.

### 19.1.1 Lead Concentrate

The Pitarrilla lead concentrate is estimated to have medium lead grade and high silver grade. The medium lead grade is similar to many other Mexican mines and may require higher grade lead concentrates for blending. The abundant availability of medium quality lead concentrate in Mexico advocates examining both domestic and offshore smelters to process the Pitarrilla lead concentrate. Domestic and offshore smelters may find the Pitarrilla lead concentrates attractive due to the high silver grade, relatively low copper grade and low level of impurities. The Pitarrilla lead concentrate may be processed domestically, as well as offshore by smelters in Asia (excluding China), Australia, Europe and North America. Currently, Chinese smelters have limited capacity to process lead concentrate with high quantities of silver as current London-Shanghai silver price differentials and Chinese Value-Added Tax policies applicable to silver metal, make Chinese smelter terms for payable silver economically prohibitive. International concentrate traders present another alternative for blending or processing the Pitarrilla lead concentrate.

Global lead smelting capacity is sufficient to process the expected quantity of Pitarrilla lead concentrate. To minimize transportation costs, both domestic and offshore lead smelters will be considered to process the Pitarrilla lead concentrates. However for the Feasibility analysis, costs for delivery to overseas smelters were utilised.

The following sales terms were estimated for the Pitarrilla lead concentrate and used in the Pitarrilla financial analysis.

<u>Payable Metal Factors:</u>

Lead: 95% of content (minimum deduction 3 units)
Silver: 95% of content (minimum deduction 50 g/dmt)
Gold: 95% of content (minimum deduction 1 g/dmt)

Treatment Charge:

$280.00 per dmt based on a lead price of $1,800/tonne
Escalator: +6% above or -4% below a lead price of $1,800/tonne

Refining Charges:

Silver: 5% of actual silver price
Gold: $6.00 to $8.00 per ounce payable gold.

### 19.1.2 Zinc Concentrate

The Pitarrilla zinc concentrate is estimated to be a lower-grade concentrate with relatively high silver content. The low zinc grade is similar to many other Mexican mines and zinc smelting capacity is expected to be available to process Pitarrilla zinc concentrate. The abundant availability of zinc concentrate in Mexico advocates examining both domestic and offshore smelters to process that from Pitarrilla. The iron content of Pitarrilla zinc concentrate may require added investigation to place them. The Pitarrilla zinc concentrate may be processed domestically, as well as offshore by smelters in Asia, Australia, Europe and North America. International concentrate traders present another alternative for blending or processing the Pitarrilla zinc concentrate.

Global zinc smelting capacity is sufficient to process the expected quantity of Pitarrilla zinc concentrate. To minimize transportation costs, domestic and offshore zinc smelters should be considered to process the Pitarrilla zinc concentrate.

The following sales terms were estimated for the Pitarrilla zinc concentrates and used in the Pitarrilla financial analysis.

Payable Metal Factors:

Zinc : 85% of content (minimum deduction 8 units)
Silver: Less 3 ounces per dmt and pay for 75% of content
Gold: Not payable

Treatment Charge:

$260.00 per dmt based on a zinc price of $2,000/tonne
Escalator: +6% above or -4% below $2,000/tonne

Refining Charges:

Silver: $0.50 to $1.00 per ounce payable silver
Gold: $6.00 to $8.00 per ounce payable gold.

#### 19.1.3 Transport and Shipping

For the Pitarrilla Feasibility Study (M3, 2012), sale of concentrate to overseas customers was assumed. A total cost of transport, handling, and all-in shipping costs of $132/dmt were applied to both the lead and zinc concentrates.

#### 19.1.4 Doré

Pitarrilla will produce silver doré throughout the Project life. Since the doré sales market is relatively broad and competitive, it is expected that the doré sales terms will be typical and consistent with standard industry practices and similar to contracts for the supply of doré elsewhere in the world.

Payable Metal Factors:

Ag: 99.0% of content
Au: 99.5% of content

Treatment Charge:

$0.40 per oz of doré

### 19.2 METAL PRICES

The projected metal prices of silver, lead and zinc as used for all economic analyses in this Technical Report are presented in Table 19-1.

**Table 19-1: Projected Metal Prices of Silver, Lead and Zinc**

| Year | Silver (US$ per ounce) | Lead (US$ per tonne) | Zinc (US$ per tonne) |
|---|---|---|---|
| -3 | 30.00 | 1,984 | 2,094 |
| -2 | 30.00 | 1,984 | 2,094 |
| -1 | 27.50 | 1,984 | 2,094 |
| 1 | 27.50 | 1,984 | 2,094 |
| 2 | 27.50 | 1,984 | 2,094 |
| 3 to end of life | 25.00 | 1,984 | 2,094 |

### 19.3 CONTRACTS

No material contracts for concentrates, doré, consumables, energy, equipment or labor have been executed as of the effective date of this Technical Report.

# 20 ENVIRONMENTAL AND REGULATORY AGENCY CONSIDERATIONS

## 20.1 INTRODUCTION

Environmental studies on the Pitarrilla Project were prepared by Environmental consultants SRK (2012) and Clifton (2012) as preparation for the production of an EIA. Their work, supported by existing studies referenced in the text, is presented in this section.

The Pitarrilla Project area has a low population density; primarily the land is used for the grazing of cattle.

Potential environmental impacts to surface soils, water, ecology and air quality will be mitigated as part of the mining operations, which have been developed to comply with the Mexican environmental regulations. The studies conducted at the site included characterization of the topography, geomorphology, geology, soils, water (surface water and groundwater), climate, air quality, flora, and fauna. The environmental setting and the proposed environmental monitoring plans are discussed below.

## 20.2 MEXICAN LEGAL FRAMEWORK

Environmental permitting of the mining industry in Mexico is mainly administered by the federal government body SEMARNAT, the federal regulatory agency that establishes the minimum standards for environmental compliance. Guidance for the federal environmental requirements is mainly derived from the LGEEPA. Article 28 of the LGEEPA specifies that SEMARNAT must issue prior approval to parties intending to develop a mine and mineral processing plant. An EIA (by Mexican regulations called a Manifestación de Impacto Ambiental, or "MIA") is the document that must be filed with SEMARNAT for its evaluation and, if applicable, further approval by SEMARNAT through the issuance of an Environmental Impact Authorization, whereby approval conditions are specified where works or activities have the potential to cause ecological imbalance or have adverse effects on the environment. The need for the mining industry to comply with Mexican environmental laws and regulations is supported by Article 27 section IV of the Ley Minera and Articles 23 and 57 of the Reglamento de la Ley Minera. Article 5 Section X of the LGEEPA authorizes SEMARNAT to provide the approvals for the works specified in Article 28. The LGEEPA also contains articles that speak directly to soil protection, water quality, flora and fauna, noise emissions, air quality, and hazardous waste management. The Ley de Aguas Nacionales provides authority to the CONAGUA, an agency within SEMARNAT, to issue water extraction concessions, and specifies certain requirements to be met by applicants.

Another important piece of environmental legislation is the LGDFS. Article 117 of the LGDFS indicates that authorizations must be granted by SEMARNAT for land use changes to industrial purposes. An application for change in forestry land use or Cambio de Uso de Suelo Forestal ("CUSF"), must be accompanied by a technical study that supports the ETJ. In cases requiring a CUSF, a MIA for the change of forestry land use is also required.

Mining projects also must include an AR and PPA.

The Ley General para la Prevención y Gestión Integral de los Residuos or LGPGIR also regulates the generation and handling of hazardous waste coming from the mining industry.

Guidance for the environmental legislation is provided in a series of NOM. These regulations provide specific procedures, limits and guidelines and carry the force of law.

## 20.3 ENVIRONMENTAL PERMITS, LICENSES AND AUTHORIZATIONS

There are three main SEMARNAT permits required prior to construction and these are: MIA, CUS with the accompanying ETJ and if applicable the corresponding forestry land use MIA and AR. A construction permit is required from the local municipality and an archaeological release letter is required from the INAH. An explosives permit is required from the SEDENA before construction begins. Water discharge and usage must be granted by CONAGUA. A project-specific environmental license (*Licencia Única Ambiental,* LAU), which states the operational conditions to be met, is issued by SEMARNAT when the agency has approved the project operations. The key permits and the stages at which they are required are summarised in Table 20-1.

**Table 20-1: Permitting Requirements**

| Permit | Mining Stage | Agency |
|---|---|---|
| Environmental Impact Assessment – MIA | Construction/Operation/Post-operation | SEMARNAT |
| Land Use Change – ETJ & Forestry Land Use MIA | Construction/Operation | SEMARNAT |
| Risk Analysis – AR | Construction/Operation | SEMARNAT |
| Construction Permit | Construction | Municipality |
| Explosive & Storage Permits | Construction/Operation | SEDENA |
| Archaeological Release | Construction | INAH |
| Water Use Concession | Construction/Operation | CONAGUA |
| Water Discharge Permit | Operation | CONAGUA |
| Project-specific License (LAU) | Operation | SEMARNAT |
| Accident Prevention Plan | Operation | SEMARNAT |

The project has acquired permits for mineral exploration and construction of initial project works, including water concessions, ramp, hazardous waste generator and the archaeological release. The permitted activities and the corresponding permit numbers are listed in Table 20-2.

**Table 20-2: Permitted Activities and the Permit Numbers**

| No. | Activity | Change of Land Use | EIA/Permit Number | Disturbance (hectares) |
|---|---|---|---|---|
| 1 | Exploration drilling, roads and drill pads, Monarch Resources de México, S.A. de C.V., 1996. | Not applicable | Notification of intent in accordance with NOM-120-ECOL-1997 | 5.60 |
| 2 | Exploration drilling, roads and drill pads, Silver Standard México, S.A. de C.V. 2003 to 2005. (Phase 1) | Not applicable | Notification of intent in accordance with NOM-120-ECOL-1997 | 11.638 |
| 3 | Opening of roads for mineral exploration and geotechnical studies. Silver Standard México, S.A. de C.V. from 2006 to 2010. (Phase 2) | SG/130.2.2/002440 | SG/130.2.1.1/002059 | 9.3252 |
| 4 | Modification of Silver Standard Mexico, S.A. de C.V. Phase 2 facilities: camps, office, workshop/warehouse, water tank, tailings pad, breaking of the tunnel ramp for exploration, reservoir for organic soil, access roads and drill pads. | Not applicable | SG/130.2.1.1/002400 | Not applicable |
| 5 | In 2006, Silver Standard Durango, S.A. de C.V. obtained from CONAGUA the permit to extract water from a well 50 metres deep, located at the camp for an annual extraction volume of 36,000 cubic metres. | Not applicable | Proof of registration folio: 03665. | Not applicable |
| 6 | Silver Standard Durango, S.A. de C.V. Preparation for beneficiation plant, large ramp and waste dump "B", camp, drill pads and access roads. | SG/130.2.2/000728/11 | SG/130.2.1.1/00625/11 | 95.3572 |
| 7 | Silver Standard Durango, S.A. de C.V. in October 2011 gave notice to CONAGUA that they were to create five (5) exploration wells for possible mine water sources. | Not applicable | BOO.E.23.1.1/.-1905 | Not applicable |
| 8 | On December 9, 2011, Silver Standard Durango, S.A. de C.V. was granted a registration number as a generator of hazardous wastes. | Not applicable | No. SSDMJ1001811 | Not applicable |
| 9 | February 2012, Silver Standard Durango, S.A. de C.V. obtained permission to drill 15 exploration holes over existing roads and pads, requiring no change in land use. | Not applicable | SG/130.2.1.1/000599/12 | Not applicable |
| 10 | March 13, 2012, CONAGUA issued a favorable technical opinion on the feasibility of extracting stone materials in four borrow sites. | Not applicable | BOO.E.23.1.1/0405 | Not applicable |
| 11 | March 2012 Pitarrilla Phase 4 geotechnical study for tailings dam and plant was authorised. | SG/130.2.2/001244/12 | SG/130.2.1.1/000958/12 | 1.9799 |

| No. | Activity | Change of Land Use | EIA/Permit Number | Disturbance (hectares) |
|---|---|---|---|---|
| 12 | April 2012, Silver Standard, obtained permission to drill 24 exploration holes, access roads, and drill pads in the mineralised area, Phase 5. | SG/130.2.2/001243/12 | SG/130.2.1.1/001077/12 | 0.4088 |
| 13 | April 2012, Silver Standard obtained permission for 4 exploration holes and 12 exploration trenches in the future waste area. | Not applicable | SG/130.2.1.1/000990/12 | Not applicable |
| 14 | April 2012, Silver Standard was authorised to conduct a seismic exploration program in the mineralised area of the Pitarrilla project. | Not applicable | SG/130.2.1.1/001098/12 | Not applicable |
| 15 | Agreement with INAH Durango for the archaeological survey, submitted April 15, 2012. | Not applicable | October 2011 | Not applicable |
| 16 | Silver Standard notified CONAGUA of the intent to install three (3) wells for possible sources of mine water. | Not applicable | BOO.E.23.1.1/0639 | Not applicable |

An environmental permit application was submitted on July 4, 2012 for the construction of various new facilities, including the principal access road, a permanent camp for operations personnel, a powerline, a metallurgical laboratory, a maintenance workshop, a landfill, and other minor works. An additional environmental permit application for the construction and operation of a bridge over the Nazas River, airport runway for private airplanes, and Telmex-Telcel communications tower was submitted on October 9, 2012. Review of the permits by SEMARNAT typically requires 60 to 120 days.

Environmental permitting documents for the open pit, crusher, processing plant, waste rock dumps and TSF will be ready for submittal to SEMARNAT once all of the surface land right acquisitions are completed.

## 20.4 ENVIRONMENTAL MONITORING SYSTEM

Various commitments have been made by SSR to prepare environmental management plans. These plans will be overseen by its environmental department to ensure that the plans are in full compliance with Mexican environmental regulations. The environmental monitoring system will have a number of individual management plans, including the following:

- Emergency response and spill contingency plan;
- Fuel storage and handling plan;
- Tailings containment area management plan;
- Waste management plan;
- Waste rock management plan;
- Wildlife protection plan; and
- Air quality management plan.

## 20.5 ENVIRONMENTAL MONITORING PROGRAM

Mexican environmental regulations require that monitoring programs be conducted and that the results be reported to SEMARNAT. To date, only the baseline studies have been undertaken, but a variety of monitoring programs are planned during construction activities and during operations. An environmental monitoring program was prepared as part of a pending environmental permit application. The environmental permit application will be submitted once all land acquisitions are completed. The objective of the monitoring program is to:

- document planned preventive, mitigation and compensation measures for potential impacts;
- to document their implementation; and
- and to review their efficiency in order to make improvements.

The monitoring programs will also indicate when required measures should be applied to site preparation, construction or operation.

The anticipated monitoring programs for pre-operations and their frequency are shown in Table 20-3.

**Table 20-3: Monitoring Plan**

| Action | Criteria / Considerations | Applicable Regulations | Monitoring Points | Frequency |
|---|---|---|---|---|
| Air quality monitoring | PM10, Total Suspended Particulates, perimeter points based on meteorological baseline data | NOM-043, | Perimeter | Semiannual inspections, Annual sampling |
| Noise monitoring | Decibels | NOM-081 | Perimeter | Annual |
| Surface water quality monitoring | Zero discharge from site | NOM-001 (based on use of the receiving body of water) and baseline results | Nazas River | Bi-annual inspection and sampling |
| Groundwater quality monitoring | Parameters to be determined based on results of baseline monitoring | NOM-127 and baseline results | Four monitor wells to be installed | Quarterly |
| Fauna registry | Based on species and numbers of fauna; protected status species to be removed if encountered | Compensation commitment | Operation areas for removal; registry in entire project area | Quarterly inspections during site preparation and bi-annual during construction, summary report |

| Action | Criteria / Considerations | Applicable Regulations | Monitoring Points | Frequency |
|---|---|---|---|---|
| Flora registry | Monitor survival rates, remove protected species when in areas of disturbance | Compensation commitment | Removal in operations areas, replanting protection in entire project areas | Quarterly inspections during preparation and bi-annual during construction with summary report, compensation biannual |
| Plant nursery | Document number and type of plants produced and planted | Restoration commitment | Replanting during reclamation | During reforestation activities |
| Soil | Collect and save organic material; remediate contaminated soils; install and maintain erosion controls | Compensation commitment | Organic soil stockpile; soil remediation area; erosion control areas | Annual inspection |
| Areas of disturbance | Registry for reclaimed areas (hectares) | Compensation/restoration commitment | As needed | Bi-annual inspection and report with documentation and recommendations |
| Socio-economics | Training programs, development of non-mining activities, social programs, reclamation of land, greenhouse production | Social commitment | Nearby communities | Annual survey and report |

## 20.6 ENVIRONMENTAL BASELINE DATA

A variety of studies have been completed in order to characterize the natural environment of the area. The project is outside of any protected areas designated by federal, state or municipal entities. The closest state designated protected area is the Fernandez Canyon, located about 113 km east of the project.

### 20.6.1 Climate

The area of Pitarrilla is in an area characterised as a steppe, which is grassland plains with trees located only at water sources. There are three different climate zones in the area, which are:

1) semi-arid with hot summers;
2) semi-arid with cool winters; and
3) dry with cool winters.

The majority of precipitation falls in the summer. The average annual temperature is 18.3 °C, with a maximum of 26.8 ºC and a minimum of 9.8 ºC. This information is derived from data obtained at public weather stations in the area. PITMET, which has a shorter monitoring history than the public stations, has provided data that gives an estimated average annual temperature of 17.9 ºC, with a maximum temperature of 22.8ºC and a minimum temperature of 12.4 ºC.

The total annual precipitation varies between 375 to 405 mm, based on public weather stations in the area. The local station has an annual average precipitation of 407 mm. The precipitation falls primarily from June to October and occurs as short, intense rainfall. Hail, snow and electrical storms also occur in the area. The area can be impacted by tropical storms or depressions, but would be on the edge of the hurricane trajectories. The local weather station has registered wind velocities averaging 3.9 m/s with gusts up to about 30 m/s. Winds are generally from the east and are strongest in the months of April and May.

### 20.6.2 Air Quality

Due to the semi-arid nature of the area, the project area is susceptible to dust generation, especially during the driest months of January to May. The dust sources in the area are primarily related to the livestock and vehicular traffic on unpaved roads. A perimeter air quality study conducted in 2010 for PM10 particulates at two sampling points indicates that the background PM10 concentrations were 31 and 34 micrograms per cubic metre (ug/m$^3$), which is well below the permissible standard of 120 ug/m$^3$ (Air and Safety Environmental Specialists, S.A. de C.V., 2010). The study also included sampling for total suspended particulates at four sampling points in and near the project site. The concentrations ranged from about 81 to 170 ug/m$^3$, all of which were below the permissible limit of 210 ug/m$^3$.

### 20.6.3 Noise

A noise study conducted in 2012 as part of the MIA baseline studies indicated that noise values ranged from 21 to 77 decibels (dB), with an average of 27.3 dB. The study included both fixed and mobile emission sources. A value of 65 dB at fixed source is considered to be high and could exceed a permissible limit, depending on the length of exposure. The principle noise sources were associated with gusts of wind and traffic, especially large trucks. The areas of the highest levels of noise were around the exploration drilling sites, near the town of San Francisco de Asís, due to development work and traffic, and at windy, high elevation points. None of the highest values were considered to be constant.

### 20.6.4 Surficial Geology and Soils

The surface geology in the project area includes sedimentary and volcanic rocks of the Casas Blancas, Cardenas, Pitarrilla and Peña Ranch formations. These are described in more detail in Section 7 of this document.

The soil types in the regional have been characterised as Lithosol, Phaeozem and Regosol, with Regosol being the predominant type at the site. This is according to the Mexican agency for statistics and geography (Instituto Nacional de Estadística y Geografía) (“INEGI”), which uses the Food and Agriculture Organization of the United Nations (“FAO”) classification system. The system is based on the soil-forming factors. Regosols are a very weakly developed mineral soil in unconsolidated materials, typical of arid and semi-arid areas and in mountain regions. According to the Mexican agency for biodiversity (Comisión Nacional para el Conocimiento y Uso de la Biodiversidad) (“CONABIO”) the soil types are predominantly Rendzinas with lesser occurrence of Regosol and Phaeozem.

A soil survey conducted in 2012 by Clifton Associates as part of the baseline study for the MIA identified a wide variety of soil types, although the principal types were Regosols, Leptosol and Phaeozem.

### 20.6.5 Flora

The project is located in the high plains (Altiplano) region of Mexico, a large area noted for its high altitude and low winter temperatures. The vegetation varieties in this arid and semi-arid region are quite extensive. The region is comparable to high desert and can include chaparral, mesquite-grassland and arid tropical scrub. There are a considerable number of endemic species within the Altiplano.

A vegetation baseline study was completed by Centro de Ecología Regional A.C. in June 2010. Three vegetation types were identified in the project area:

1) pine and oak forest,

2) “matorral xerofilo”, which includes high–desert chaparral, and

3) riparian forest, which is a forested area adjacent to a water source.

An inventory of the vegetation types in the area includes 29 plant families and 66 species. The most abundant families present are Asteraceae (commonly referred to as the aster, daisy or sunflower family) and Cactaceae (cactus family). Three species (Mamillaria marksiana, Pinus pinceana y Thelocactus heterochromus) are classified as at risk per the Mexican regulation NOM-059-SEMARNAT-2010, which lists native species and their risk status. An additional 14 species were considered of special interest due to potential for commercial or decorative use or due to the difficulty to propagate the species.

The relative importance of the species was calculated based on the species value. The species with the highest value were ocotillo, cat claw mimosa, acacia and mesquite. The calculated species diversity was 1.791, which is considered to be low. This low level is attributed to the degradation experienced by the area due to decades of use for agriculture and grazing.

### 20.6.6 Fauna

The project is located within the reptile-fauna regions of the Sierra Madre Occidental and the Chihuahuan Desert. Species characteristic of the region are badger, lynx, white-tail deer, wild turkey, mountain lion and black bear. Due to the presence of the Lazaro Cardenas dam and the Nazas River, there are migratory birds, such as the snow goose, bufflehead and common merganser that can frequent the area. Although some regional studies were done to identify mammals and reptiles in the region, there is little information regarding wildlife in the project area (Centro de Ecología Regional A.C., 2010).

A survey was conducted in February 2012 (Clifton Associates, 2012) as part of the baseline study for the environmental permit. During the field survey, a total of 83 species of four terrestrial vertebrate groups were logged. Birds composed the major part of the species noted, including 61 species. There were a total of 15 mammalian species. No amphibians were found, however, this was expected since the survey was done during the winter. There were seven reptilian species noted, including two species that have threatened status under Mexican regulation NOM-059 (Coluber flagellum and Thamnophis cyrtopsis). Of the 61 bird species identified, only one (Buteogallus anthracinus) had a special protection status. Of the 15 mammals identified, only one (Taxidea taxus) has threatened status.

Eighteen species of migratory birds were registered during the survey, which was taken at the end of the migratory season.

There are some species in the area that have commercial interest or are used for medicinal purposes. These species include one reptile, three birds and five mammals.

The habitats of the native species have been negatively impacted by the use of the area for agriculture and ranching. The wildlife is limited to the less-accessible areas such as ravines, canyons and high peaks. Within the project area there are possible wildlife habitats, especially the southern part of the site in the highest elevations and in areas of small ranches.

#### 20.6.7 Surface Hydrology

The Pitarrilla mine property is located in the Rio Nazas - Agua Nava hydrologic basin. The property is located across three groundwater basins, including Buenos Aires, La Victoria, and San Jose de Nazareno. The northeastern corner of the property extends into the La Zarca Revolución basin. The groundwater basins are contained in the Presa Lazaro Cardenas and Agustin Melgar hydrologic sub-basin (Figure 20-1).

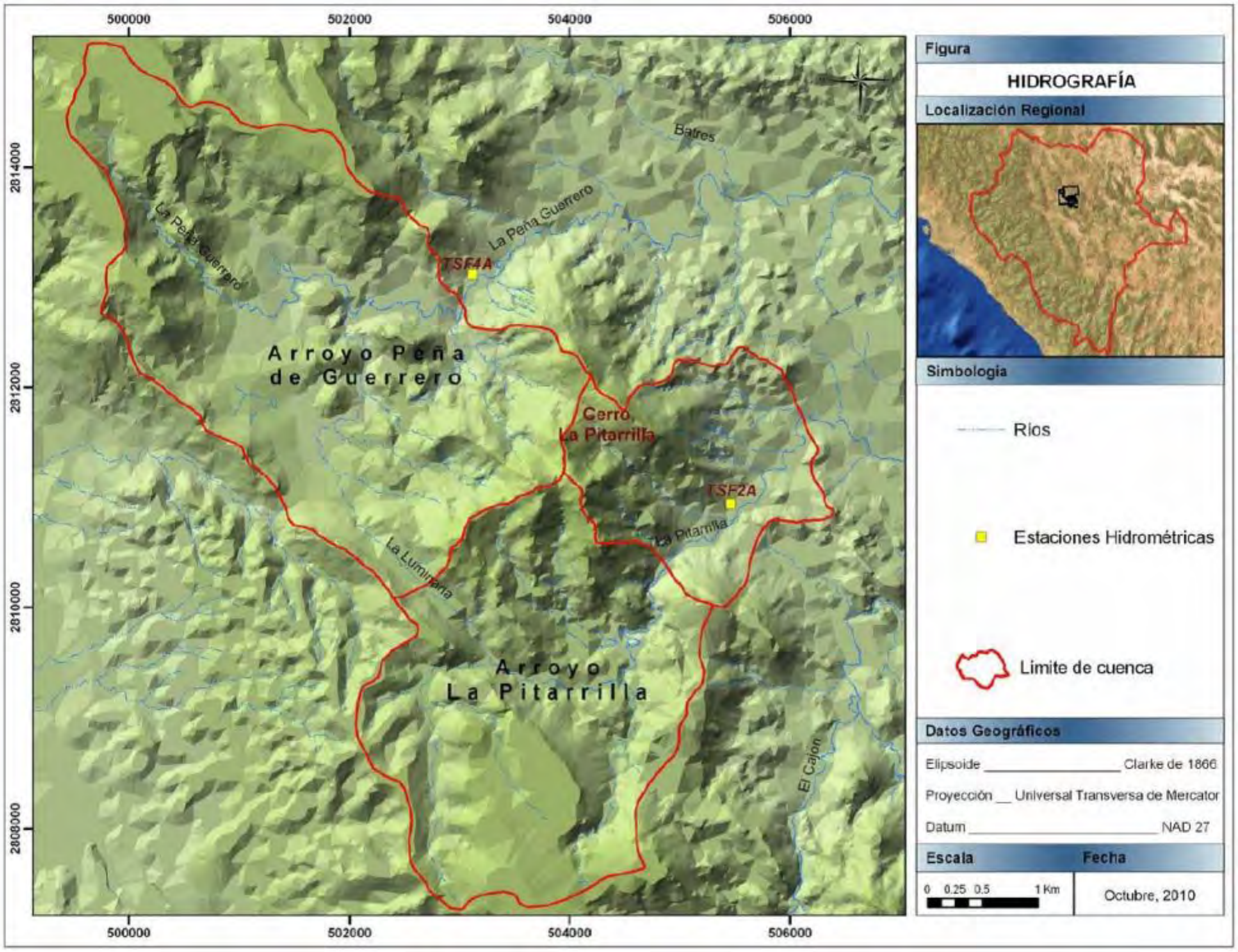



**Figure 20-1: Hydrographic and Regional Watersheds**
Source: IDEAS, 2010

The distinguishing hydrologic features of the area include the Lazaro Cardenas reservoir, which is located up-gradient and north of the mine property, and the Nazas River, which is the source of water for the Lazaro Cardenas reservoir. Controlled discharge from the reservoir is the principal source of water for the reach of the Nazas River that runs through the mine property.

The legal status of surface water in the Nazas - Agua Nava basin is complicated. On the one hand, a 2008 federal water availability study states that the Presa Lazaro Cardenas and Agustin Melgar sub-basins have 236.3 $Mm^3$ and 276.6 $Mm^3$ of available water, respectively, at the downstream edge of each basin. However, a 1932 “decreto de veda” bans the assignation of any additional surface water rights for the river and all hydrologic basins upstream of the reservoir. For SSR to obtain surface water from the Presa Lazaro Cardenas or Agustin Melgara basins, existing concessions would need to be purchased.

Baseline data was collected during a one-time surface water sampling program at five locations, including two locations at Presa Lazaro Cardenas (one in the lake and a second at the spillway), two samples from the Nazas River and a sample from Arroyo Petrorillos (Figure 20-2).

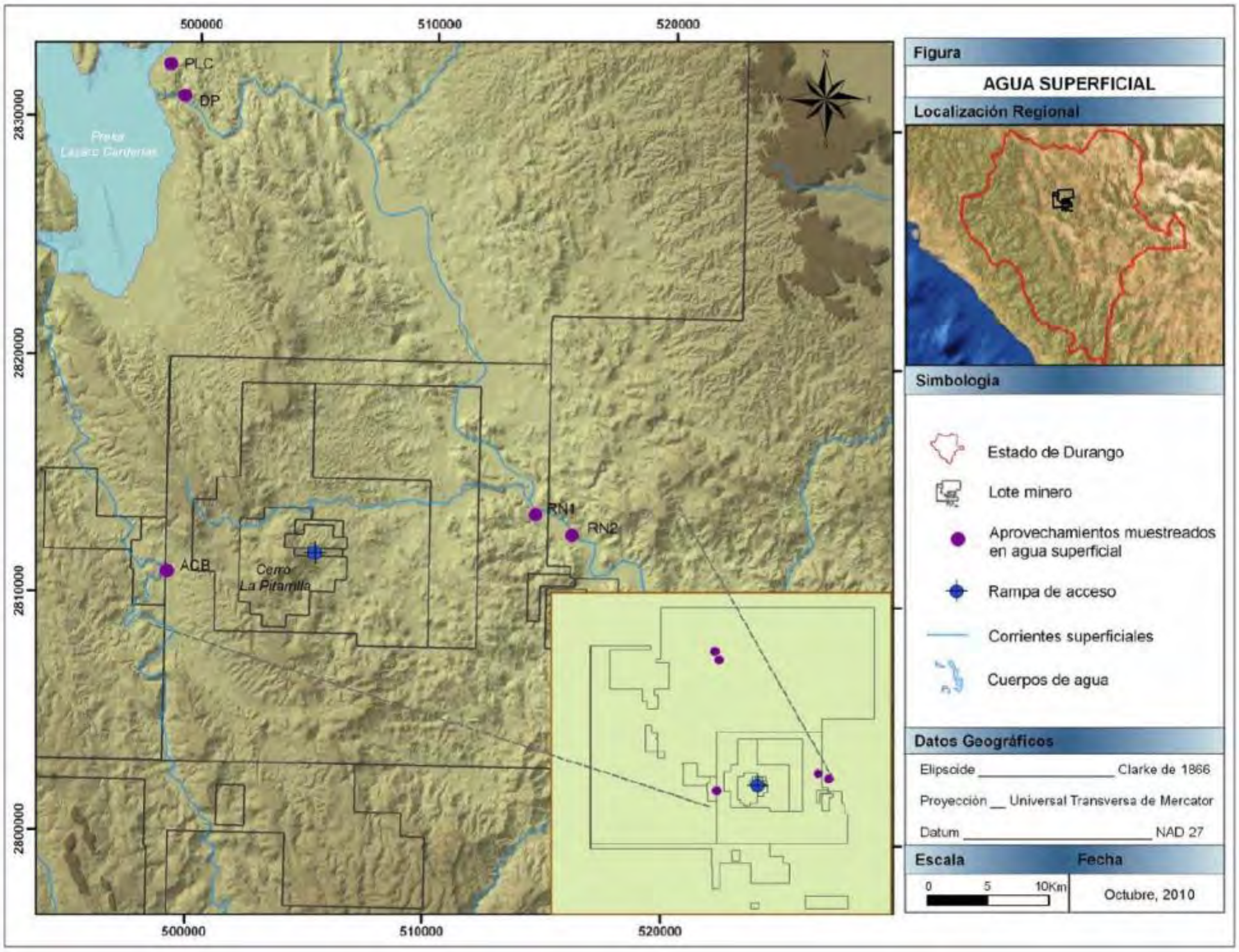



**Figure 20-2: Pitarrilla Monitoring Points**
Source: IDEAS, 2010

Samples were analyzed for major ions, nitrate, oil and grease, surfactants, biological oxygen demand, chemical oxygen demand, phosphorus, total coliform and fecal coliform. The results of the sample analyses show that except for fecal coliform the water quality was generally good and rated as potable at all locations (IDEAS, 2010). All of the samples were over the federal limit for fecal coliform, likely due to sewerage runoff from the local domestic sources and/or livestock.

### 20.6.8 Groundwater

Groundwater production potential was evaluated using a combination of geophysics (TEM and magnetic), review of local geology and structural and exploratory drilling. Based on the data developed by IDEAS and by Bufete Minero y Servicios de Ingeniería ("BMS"), groundwater flow appears to be contained partially in the primary porosity of alluvial sediments and the coarser units of the Mezcalera Formation (Peña Ranch Fm equivalent), and partially in the secondary porosity of an extensive fault and fracture system that permeates the mine property (IDEAS, 2010 and BMS, 2011). Regional groundwater elevations across the mine property range from 1730 masl (on the west side of the property) to about 1380 masl (on the eastern part of the property). This is based on data collected in July 2012 by SRK. Groundwater in the vicinity of the Nazas River is artesian.

The aquifer units include (by increasing depth) the recent alluvium, Upper Volcanic Group (ignimbrite, rhyolite, rhyolite porphyry and rhyolite tuff), the Lower Volcanic Group (andesite, dacite and conglomerate) and Mezcalera Formation (sandstone to siltstone). Hydraulic conductivity results from the IDEAS study ranged from 1.06x10-9 m/s (9.4x10-5 m/d) to 2.59x10-6 m/s (2.23x10-1 m/d). The transmissivity ranged between 5.32x10-7 $m^2$/s (0.04 $m^2$/d) and 1.04 x10-3 $m^2$/s (89.8 $m^2$/d). These values are reflective of the fractured aquifer formed primarily of low permeability rocks. The hydrogeologic work done by KPL in the area of the proposed open pit noted moderate to low rock mass permeability, with hydraulic conductivities ranging from 1x10-6 to 1x10-9 m/s, which is similar to the results obtained by the IDEAS study. Groundwater elevations ranged from 1663 to 1694 masl (KPL, 2012b).

The main fault orientations in the area of the project have been described by KPL (2012) as follows:

NW/SE Faults - These features typically dip between 30 and 60° to the northeast. The spacing between adjacent faults on a regional scale is probably in the order of 100 m to 200 m, with tighter spacing expected near the deposit.

Felsic Dykes - These dykes are parallel to the NW/SE faults and appear to have been inserted along these structures. While it is likely that some of the original faults have been re-healed/obliterated during the insertion, there is evidence of faulting on the boundaries of the dykes.

NE/SW Faults - These features dip between 60° and 70° to the northwest. The spacing between adjacent faults on a regional scale is thought to be in the order of 150 m to 600 m, with tighter spacing expected near the deposit.

ESE/WNW Faults - These features dip 70° to the North. Only two of these faults have been confidently identified within the vicinity of the deposit. The spacing between adjacent faults on a regional scale is thought to be in the order of 450 m.

Based on the IDEAS study, the aquifer is confined by a fine-grained unit that varies in thickness from 200 m to 500 m, overlying a confined aquifer ranging in thickness from 50 m to 400 m. Five exploratory boreholes were drilled by IDEAS in 2010 to depths from 231 m to 502 m and completed as piezometers. A sixth well was drilled and completed as a test well (WW-010-006). Another six exploratory boreholes were drilled in 2012 by BMS, of which two were completed as test wells. The wells were located where preliminary results showed minimal confining layer and either a major fault or convergence of structural features. Most of the exploratory wells produced less than 10 l/s, with the exceptions of wells that appear to intercept the Tata Lucas fault (in fractured andesite), where a total production rate of almost 90 l/s was obtained from two wells for a period of 36 hours.

Groundwater production potential appears to be best near the Nazas River, where three major faults converge. This tendency is displayed by the artesian conditions, field parameters and results of aquifer characterization tests. Airlift pumping and recovery tests in five wells and pumping tests in two additional wells indicate low conductivity in the vicinity of Cerro Pitarrilla (10-9 m/s) and moderate hydraulic conductivity along the Nazas River (2.6 x10-6 m/s). To date, the only area with groundwater production potential that might meet the mine requirement of 150 l/s is along the Tata Lucas fault. SSR plans to continue the groundwater production studies with the objective of identifying the water production potential in 2013. Due to the 1932 ban on water use from the Presa Lazaro Cardenas basin, the aquifer basins that could be developed for groundwater extraction include La Victoria and San Jose de Nazareno basins, although the "extractable volume" has not been calculated by CONAGUA.

The 2010 groundwater sampling program included general characterization of groundwater collected from four shallow hand-dug wells, eight springs, one well and mine inflow along the ramp. The results show three distinct groundwater regimes:

1) the area north of Cerro Pitarrilla (calcium-sulphate);

2) the area of springs west and south of Cerro Pitarrilla (bicarbonate-calcium); and

3) the deeper wells along the Nazas River (bicarbonate-sodium) (IDEAS, 2010).

Groundwater quality varies greatly. All the samples from the area north of Cerro Pitarrilla exceeded drinking water standards for total dissolved solids, sulfate, bicarbonate, calcium, and coliform. Samples from springs east and southwest of Cerro Pitarrilla, as well as the deep well, have good water quality. The differences in water quality and major ion geochemistry clearly show three different groundwater regimes.

As part of the baseline study for groundwater quality, three sampling events were conducted in 2012 to characterize pre-operations water quality based upon the Mexican regulation for potable water (NOM-127). Groundwater levels were also measured during the study measured and a potentiometric map was produced (SRK, 2012). Parameters that were detected above the permissible limits per NOM-127 were arsenic, fluoride, hardness, iron, sodium, sulphate, and total dissolved solids.

The depth to groundwater ranged from groundwater at surface (artesian) to about 70 m below ground surface. The groundwater elevation ranged from about 1380 masl to 1720 masl, based on July 2012 data (SRK, 2012). Groundwater flow directions vary at the site due to local pumping influences, but in general groundwater flow follows the topography and flow is towards drainages and controlled by the faults and fractures in the geologic setting The predominant groundwater flow is towards the Nazas River (to the east).

### 20.6.9 Archeological Resources

An archeological survey conducted by the *Universidad Nacional Autónoma de México* (UNAM) identified zones with rock paintings east and west of the project site. The Mexican authority responsible for archeology, INAH, has conducted a review of the site because of an agreement made between SSR and the agency (INAH, 2011). There was an area in the southern part of the site with rock paintings, however the site was not deemed to be significant and no restrictions for development were noted. A formal release letter from the institute is pending.

## 20.7 POTENTIAL IMPACTS AND MITIGATION MEASURES

### 20.7.1 Landscape

The landscape will be impacted by clearing and grubbing and the construction of the mining facilities. At closure there will be some facilities left in place which include the waste rock dumps, open pit and tailings impoundment. All impacts following closure will be controlled and mitigated through planned closure and reclamation activities.

### 20.7.2 Air Quality

Dust control measures will be implemented during construction and operations. During construction, dust will be controlled by watering the roads and areas under construction. During operations, water trucks will be used to control dust on the roads. Sprayers will be employed as necessary in the crushing areas. Emissions controls will be part of the laboratory design. At closure, the reclamation will be conducted to minimize dust generation. The tailings area will be covered with soil and reclaimed during closure.

### 20.7.3 Noise

Noise caused by operating machinery will be mitigated where possible and worker hearing protection will be required in areas that exceed safety standards. Machinery will be subject to routine maintenance to ensure optimal operating performance. Engineering controls to reduce noise will be considered as part of the noise reduction plan.

### 20.7.4 Water Resources

Prevention and mitigation measures being considered to protect surface and groundwater quality include surface erosion control around the facilities (channels that convey natural and impacted surface water separately), a low permeability liner for the TSF to control infiltration, and mitigation of any acidic or metal-contaminated water from waste rock.

Surface water monitoring will be conducted on a routine basis during construction, operations and post-operations to assess any impacts to surface water quality. Groundwater monitor wells will be installed up-gradient and down-gradient of the mining operations and sampled to evaluate impacts to the groundwater quality.

### 20.7.5 Fauna

Protected plant species will be relocated during construction activities. Native species will be used during reclamation activities to the extent practical.

### 20.7.6 Flora

Actions that are planned to mitigate vegetation impacts include compensation payments to the forest fund for land use rights, organic topsoil recovery during clearing and re-use of this material in the closure phase, and implementation of a flora species protection program during all stages of the project. Recovery of protected species and reclamation activities will include soil scarification to enhance plant growth and planting native species to restore the affected areas.

### 20.7.7 Waste Management

Contamination prevention and mitigation measures will be implemented to protect soil, surface water and groundwater quality. A hazardous and nonhazardous waste handling program that includes a spill prevention plan will be used where process chemicals are stored and used. Wastes generated during construction and operations will be managed according to the provisions of the General Law for Prevention and Integrated Waste Management. A landfill will be permitted and constructed to receive non-hazardous solid waste in compliance with NOM-083. Solid wastes will be minimised through re-use and recycling programs.

Hazardous wastes will be disposed of off-site in compliance with NOM-052, NOM-053 and NOM-054.

Oxide and sulphide tailings will be produced during the mining of the oxide and sulphide ores. A geochemistry study was conducted in accordance with NOM-141, which stipulates the design criteria for tailings impoundments design, operation and closure; and NOM-052, which classifies hazardous wastes. Both the oxide and the sulphide tailings samples were classified as potentially acid-generating. Tailings will be stored in an impoundment with a low-permeability liner in order to minimize infiltration of leachate to groundwater and to avoid environmental impacts (Tierra Group, 2012).

Waste rock was characterised in accordance with NOM-157 (SRK, 2012a). Total concentrations of arsenic, cadmium, antimony and lead were detected above the permissible limits in a composite sample prepared based upon the geologic formations that would potentially be placed in the waste rock dump. In addition, the acid generation potential result was 2.45, which is considered to be potentially acid-generating per NOM-157.

The results of 50 geochemistry tests for acid-generation potential of the waste rock were correlated to the sulphur and calcium concentrations in the assay database (KPL, 2012b). This correlation was used to classify the waste rock that will be deposited in the waste rock dumps during the mine life. The waste rock will be handled such that PAG rock will be chemically isolated and neutralised with NAG rock and limestone, maintaining a neutral pH in the waste rock dump. The PAG waste will be deposited in the waste dumps and encapsulated within the NAG waste rock at high elevations and well away from drainages to prevent interaction with runoff. The addition of calcium-rich rock fill will also be carried out during PAG waste deposition to provide adequate buffering capacity within the PAG waste dumps (KPL Piésold, 2012b).

Water management measures will be used to minimize contamination. These measures will include minimizing water infiltration and contact with oxygen, via the following components:

- NAG waste rock caps along the edges and tops of the waste dumps at the end of the mine life to minimize infiltration of runoff;

- Rock drains located in the valley bottoms to drain the waste dumps;
- Sedimentation basins, flow monitoring weirs and groundwater monitoring wells located in the valley bottoms and at locations downstream of the ultimate waste dump toes to capture sediment and to monitor surface and groundwater quality prior to reporting to the environment; and
- A diversion channel located at the south edge of the Southwest Waste Dump to divert flows from a drainage upstream of the waste dump footprint and to the environment.

A more detailed geochemistry study will be conducted and a waste management plan will be developed based on the results of the more detailed study.

Wastewater from buildings will be treated in a septic tank in compliance with NOM-006.

#### 20.7.8 Cumulative Environmental Effects

The proposed mining project is not considered to be a risk to the environment as appropriate environmental monitoring and control systems will be implemented to protect the flora and fauna species identified at the site. The identified impacts to soil, water and air will be minimised by using proper mitigation and control measures. The facilities that will remain after closure, such as the open pit and the waste rock dumps, will be managed to minimize environmental impacts.

### 20.8 COMMUNITY RELATIONS AND SOCIAL RESPONSIBILITIES

The project is located on the border between the two municipalities of El Oro and Indé, in the northern Mexican state of Durango. The nearest villages are Casas Blancas immediately to the west of the project in El Oro, and San Francisco de Asís, located approximately 12 km to the northeast in Indé. The estimated 2010 population of El Oro is just over 10,000 inhabitants. The municipality of Indé has a lower population, with a 2010 estimate of just under 3,800 residents.

In the municipality of El Oro there are 60 persons who speak an indigenous language. The community of Boquilla Colorada is composed of residents of Tarahumara ethnicity. In the municipality of Indé that are 51 persons who speak an indigenous language.

Locals are employed almost exclusively in cattle ranching, with some farming, mostly to produce livestock feed. A few individuals work at government posts (schools, clinics, municipality), infrastructure (telephone company, power company), or at the Lázaro Cárdenas dam. Many families have one or more members that have found work outside of Mexico, especially in the United States. Most villages report that there remain a number of young people in search of work. There is a general expectation that the project will increase employment opportunities and improve the standard of living.

A consultant carried out a number of socio-economic studies in 2010 for the proposed project area (Robert Boutilier & Associates, 2010). The results indicate that relations with local stakeholders are good, and there is strong support for the project (Robert Boutilier & Associates, 2010). A survey conducted with 62 interviews indicated a high level of satisfaction with SSR and a desire for the project to proceed in order to improve local employment opportunities, although in some areas there were concerns about the project. The concerns mentioned most frequently were the potential of water contamination and the influx of outsiders that bring crime (specifically drug addiction) and reduce safety. The closest community (Casas Blancas) gave the project the highest satisfaction ratings.

SSR has implemented a community relations program that includes environmental, medical, educational, infrastructure and social support. The company holds regular meetings with the community leaders to plan projects. Examples of the social projects completed in 2012 have included medical assistance from the project paramedics at Casas Blancas, environmental projects (a workday to clean areas at the Nazas River and planting trees at La Victoria), construction projects to improve infrastructure (media room at San Francisco de Asís high school, improvements to suspension bridge at the Nazas River, installation of livestock fences and improvement of a path, improvement to a water supply well), and a commitment to assist with water supply wells for livestock at San Rafael de Jicorica.

Based on second quarter 2012 statistics, 47 employees at Pitarrilla are from the local communities or nearby states. Of the 47 employees, 76% were from directly-impacted communities, 15% were from other areas in Durango State, 2% were from Chihuahua and 7% were from Sonora. In addition, subcontracted employees included 56 people from Casas Blancas, San Francisco de Asís, San Rafael de Jicorica, San Jose de Ramos and La Trinidad.

## 20.9 CLOSURE AND RECLAMATION

### 20.9.1 Introduction

In accordance with the mine plan, the mine will cease operation after production year 17. Plans to facilitate closure have been incorporated into the design of the facilities, such that the need for extensive re-sloping and re-handling of materials is minimised at closure. Progressive reclamation will be included in the mine planning and operations, which will minimize the effect of the mine on the environment during operations.

### 20.9.2 Regulatory Framework

Mine reclamation is addressed in Article 27 of the Mexican Constitution, which sets two broad standards for reclamation:

- Mexico retains ownership of the mineral rights at all times and concession holders only have rights to mined materials. As such, the Mexican government may establish the conditions of reclamation; and

- Mexico has an obligation to take mitigation measures to protect natural resources and restore the ecological balance.

Multiple Mexican regulations apply to closure conditions, including NOM-138-SEMARNAT/SS-2003, NOM-141-SEMARNAT-2003, NOM-147-SEMARNAT/SSA1-2004, and NOM-157-SEMARNAT-2009, as outlined as follows:

NOM-001-SEMARNAT-1996 sets the permissible limits for wastewater discharges.

NOM-083-SEMARNAT-2003 establishes the environmental protection requirements for solid waste and special management waste landfills, including the closure of the landfill.

NOM-138-SEMARNAT/SS-2003 establishes maximum permissible limits for hydrocarbons in soil. Should limits be exceeded, an environmental and human health risk assessment may be conducted to determine its characterization and remediation options.

NOM-141-SEMARNAT-2003 specifically includes post-closure requirements for the tailings impoundments. In general, the regulation requires that measures be taken to ensure that tailings impoundments do not release particulates to the atmosphere; that discharges from the tailings do not impact surface water or groundwater; and that the impoundments do not fail. If the tailings are potentially acid-generating, then the tailings must be covered or placed subaqueously to prevent the formation of acid drainage, or neutralised using other materials. Reclamation species should not promote acid formation in the subsurface, and the species should be native to the region. If mitigation of acid drainage is required, then measurements must be taken to prevent impacts to water, soil and sediments. The slopes must be stabilised.

NOM-147-SEMARNAT/SSA1-2004 establishes soil remediation levels for concentrations of arsenic, barium, beryllium, cadmium, chromium, hexavalent chromium, mercury, nickel, silver, lead, selenium, thallium and/or vanadium. The regulation includes specifications for site characterization (such as the number of samples), the conceptual site model, and an alternative method to determine remediation levels based on a risk assessment.

NOM-157-SEMARNAT-2009 establishes the requirements for mine waste management plans. Section 5.6 of the regulation describes the criteria for storage and final deposition of wastes. The criteria include identification of the environment that could be impacted by operations; the engineering and maintenance specifications to maintain physical stability; control measures to avoid wind and water erosion; and measures to prevent acid drainage, metal leaching and runoff. Post-closure criteria include monitoring of water bodies that could be impacted and reforestation using stockpiled soil and native species of the area.

As part of the permitting requirements, a detailed closure plan will be prepared and submitted to SEMARNAT prior to starting operations.

### 20.9.3 Objectives

The initial closure objectives include the following:

- Demonstrate compliance with relevant Mexican laws and regulations, SSR corporate standards and any relevant legislative requirements;
- Protect public and employee health, safety and welfare;
- Limit or mitigate residual adverse environmental effects of the project;
- Mitigate socio-economic impacts of the project following decommissioning and subsequent closure as far as reasonably possible;
- Help protect local community values;

- Provide a reasonable basis on which the financial consequences of closure can be estimated, recognised and managed, including consideration of any tax consequences; and
- Avoid or minimize costs and long-term liabilities to the company, government and public.

### 20.9.4 Future Land Use

It is anticipated that following closure, some of the reclaimed project area can be used for purposes other than mining. The designated uses may be one or a combination of the following:

- Natural habitat for wildlife and native plants, and -
- Land with potential for farming (either agriculture or livestock).

The open pit will not have a designated future use.

### 20.9.5 Closure Plan

The overall reclamation and closure will occur in four phases, with the first three phases carried out over a three-year period. The initial phase will involve the decommissioning of all facilities and infrastructure; the second phase will be the demolition and removal of mining and processing facilities; the third phase will be demolition of mine infrastructure at the completion of mining; and the fourth phase will be post-closure monitoring to confirm the success of closure measures.

It is assumed that the airstrip, roads, water distribution system and other components of the site infrastructure may remain at the end of the mine life, based on the final use of the property.

Descriptions of the general methods to be used for mine closure are described in the following paragraphs.

#### 20.9.5.1 Characterization

Prior to site closure, soils and water will be characterised to determine whether they have been impacted by mining operations. In addition, mine waste materials will be characterised to determine whether the potential for acid-rock drainage or metals leaching exists.

#### 20.9.5.2 Open Pit

Pit closure will include fencing around the pit to prevent access. The fence will be located approximately 3 m to 5 m from the pit edge, depending on edge stability, to ensure that it can be anchored securely. A safety berm will be constructed to prevent surface runoff into the pit walls. The area along the fence will be contoured and vegetated to minimize water flow and erosion into the pit from the edge. Access to the pit lake will be secured through a gate installed at the pit access ramp to ensure pit lake monitoring is possible if needed.

#### 20.9.5.3 Waste Rock Areas

- The waste dumps at closure will occupy approximately 576 ha. The roads to the waste dumps will not be reclaimed as they will be needed for long-term monitoring of the dumps. Descriptions of the slope preparation, conceptual cover design and cover placement are presented below.

- Slope Preparation. The plan is to re-grade the benches before reclamation activities to achieve more natural looking contours. The slopes will be re-graded to 3:1 (H:V) or flatter. This will provide stable slopes and slopes that can receive an engineered cover. The final waste dump bench crests will be rounded and the faces re-sloped.

- Cap. The goal of the closure design of waste dumps will be to reduce the discharge of acidified water by minimizing infiltration into the top of the facility as well as reducing infiltration through the slopes. There is expected to be approximately 2.7 Mt of NAG waste at the end of mine life to provide a cap on the waste dumps. This waste rock will be placed at the storm water diversion channel at the Southwest Waste Dump to minimize the infiltration of acidic water into the waste dump in this area. The remainder of the NAG waste rock will be placed above locations where PAG waste rock was placed during reclamation and closure phases 1 to 3.

#### 20.9.5.4 Tailings Storage Facility

The tailings impoundment at closure will occupy approximately 288ha (Figure 20-4). Descriptions of the dam slope preparation, conceptual cover design and surface water diversions are presented below.

- Dam slope preparation. The downstream slope of the tailings dam will be constructed at a 2:1 (H:V) slope that is conducive to reclamation. Since the final dam raise will be built using the downstream method of construction, this material will be the newest material placed on the dam and will be made up of rockfill. A soil cover will be placed on the downstream face of the dam as the rockfill will not be conducive to vegetation growth.

- Engineered cover. The goal of the closure design of the tailings impoundment is to minimize precipitation infiltration into the top of the tailings surface and reduce the likelihood of acid generation and metal leachate within the tailings. The top of the impoundment will be covered with an engineered cover comprised of layers of (in order of placement) lime slurry, soil and growth medium. The entire surface of the tailings impoundment will then be re-vegetated with native grasses and plants. A series of surface diversion channels will be excavated across the reclaimed surface in a dendritic pattern to collect and direct precipitation to the southeast corner of the impoundment where the Stage 5 spillway will be re-sised for closure conditions. The surface water diversion channel along the southern limit of the impoundment would also be re-sised for closure to continue to direct surface water runoff from upstream of the tailings impoundment away from entering the impoundment. The reclaim barge and associated pipework and the tailings distribution pipework will also be removed.

20.9.5.5 Buildings and Infrastructure

Decommissioning is that aspect of closure that addresses the chemicals and hazardous materials remaining on site upon cessation of production activities and prepares the site for its closure and post-closure period. Decommissioning will include activities such as:

- Equipment decontamination;
- Removal of hazardous materials;
- Disassembly and disposal of mine infrastructure;
- Re-profiling;
- Topsoil replacement; and
- Preparation of watercourses.

An engineering survey of buildings will be conducted by a competent person with a checklist to determine the condition of framing, floors and walls, and the possibility of unplanned collapse of any portion of the structure. The fire equipment will be left in place until other equipment is removed. The survey will be documented in writing.

20.9.5.6 Waste Management

No special handling is anticipated to be needed at the time for closure for the solid wastes. The organic wastes will have been composted, and this material will be available for reclamation efforts.

Wastes classified as hazardous per NOM-052-SEMARNAT-2005 will be managed in accordance with a waste management plan to prevent contamination to the environment and for worker safety.

### 20.9.6 Post-Closure Monitoring

The effectiveness of the closure activities will be verified during closure. Field activities related to the demolition and removal or remediation of facilities will be documented. The type and quantity of materials removed and type and quantity of materials remaining on site will be documented. Field activities will be monitored for quality control and quality assurance. Changes to design plans will be documented on final design drawings.

Post-closure monitoring activities will begin concurrently with closure and are estimated to last 20 years for the purpose of the closure costing. The length of post-closure monitoring may change if compliance with regulatory criteria is not achieved during that period. The amount of sampling and testing requirements may also change due to changes in regulatory requirements. The costs for repairing the mitigation structures were included in the mine closure budget. Repair work will be conducted on the structures until they are shown to be self-sustaining.

### 20.9.7 Closure Costs and Financial Assurance

The reclamation and closure cost was estimated by a third-party contractor. The closure process will include the decommissioning, demolition, rehabilitation (i.e., backfilling, covering, re-contouring, and revegetation), and post-closure monitoring of the mine site. Appropriate methods will be in place for decommissioning hazardous materials and equipment. Future regulations may either extend or reduce the time frame of this phase.

The cost of closing the various mine facilities is a function of the design and layout of the facilities and the closure design plan chosen for each facility. The costs included in the estimation are for the activities related to bulk earthworks, soil cover and revegetation, decommissioning, demolition, and waste removal and waste treatment. More specifically, the cost estimate includes the following activities: removal of electrical lines and cables; dismantling and disposal of structures, equipment, and buildings; demolition of concrete and headwalls; haulage and placement of topsoil; revegetation of topsoil; tailings impoundments and waste rock dumps slope re-contouring, haulage of borrow material, and installation of irrigation system.

Rehabilitation of the major facilities was estimated in the cost, and those facilities are the process plant, tailings impoundments, waste rock dumps, some infrastructure, contaminated areas, and utilities. The rehabilitation process that will be used during site closure will be to backfill foundations, cover with growth medium, re-contour for positive drainage, and revegetate with native species. The housing, roads and airstrip will remain after closure.

The estimated cost for the site closure is $75,796,000 with a contingency of +/- 35%. The contingency is factored in the closure cost and not the post-closure care and maintenance. The variation of 35% is considered to be appropriate for a conceptual level engineering project (US Forest Service, 2004). The post-closure monitoring is assumed to be 20 years but the actual time required for monitoring will be determined based on site-specific conditions.

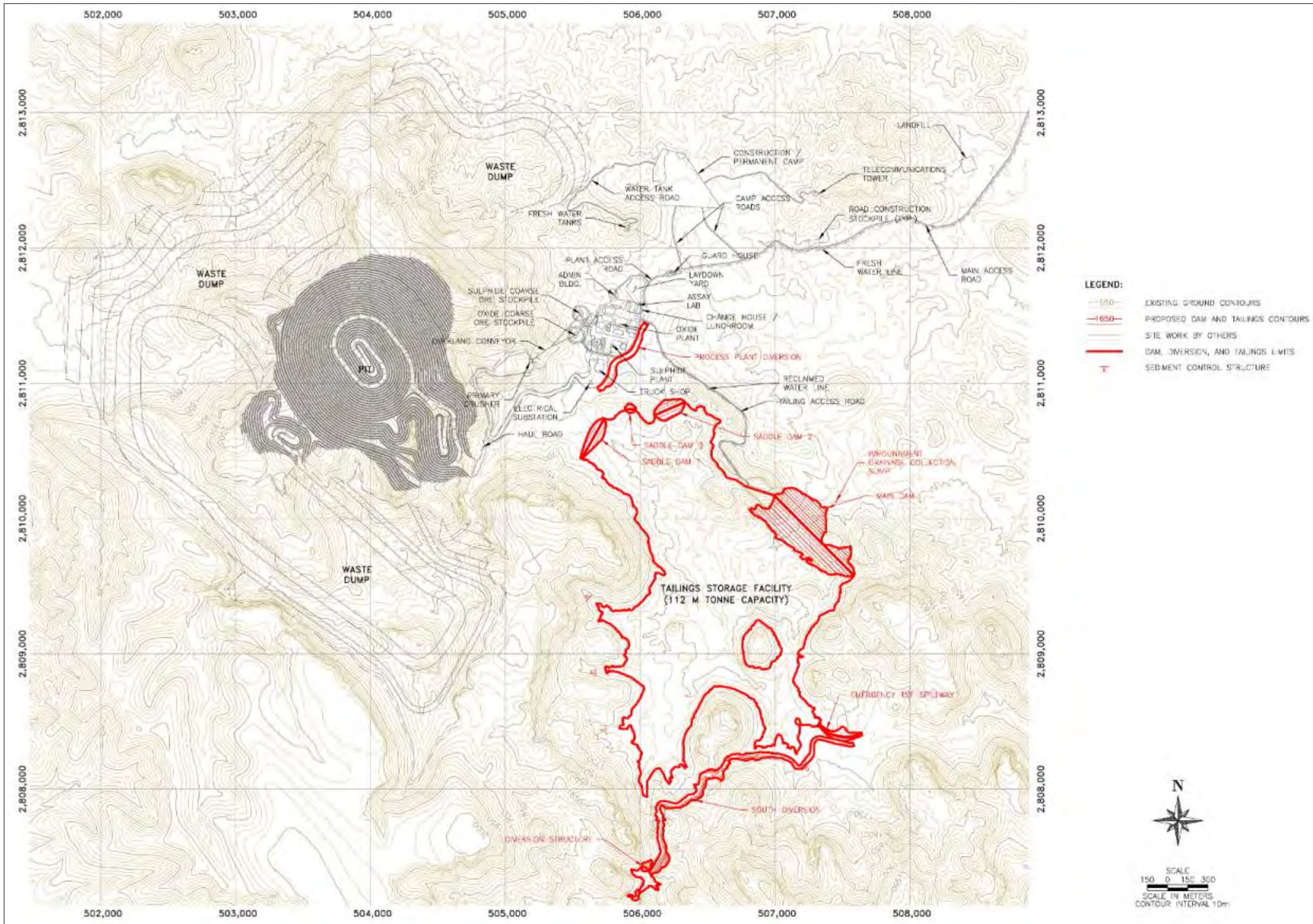



**Figure 20-3: Tailings Storage Facility Location and General Site Plan**
Source Tierra Group, 2012

# 21 CAPITAL AND OPERATING COSTS

Capital and operating cost estimates were developed to evaluate the economic feasibility of the Pitarrilla Project. All project costs incurred prior to the declaration of commercial production (but not including sunk costs prior to the construction decision) are considered initial capital costs and are summarised in Table 21-1, Table 21-3, Table 21-4 and Table 21-5. Declaration of commercial production is defined as at the end of the third consecutive month of 80% of nameplate or greater mill tonnage throughput. Capital costs that occur after the declaration of commercial production are considered sustaining capital and are summarised in Table 21-2 and Table 21-5. The operating and maintenance costs for the Pitarrilla Project operations are summarised by areas of the operation, and shown in Table 21-17.

## 21.1 CAPITAL COST

SSR compiled cost data from all contributors into a single master capital cost estimate which is summarised in Table 21-1. M3 Engineering developed a large part of the capital detail and their sub-summary is included in Table 21-3 and Table 21-4. The data for the estimate was supplied by the following organizations:

- **Independent Mining Consultants, Inc. (“IMC”) (Tucson, AZ)** – Sourced open pit mining equipment purchase costs.

- **Tierra Group International, Ltd. (“TGI”) (Lakewood, CO)** – Designed the TSF. TGI provided a detailed estimate indicating initial capital and sustaining capital costs for the TSF.

- **Knight-Piésold (“KPL”) (North Bay, Ontario)** – Estimated costs associated with the preparation of the waste dumps and water management.

- **SRK Consulting (“SRK”) (Tucson, AZ)** – Estimated closure costs for the mine, plant site, and TSF.

- **Silver Standard Resources (Vancouver, BC)** – Estimated the owner’s costs for the project, pre-production mine operating costs and required mining fleet quantity and estimated permanent camp costs. Silver Standard completed the final summary of the overall project financial tables.

- **M3 Engineering & Technology Corporation (“M3”) (Tucson, AZ)** – Estimated capital and operating costs for the processing facilities and project infrastructure. M3 developed initial project capital indirect costs and provided electrical power costs for all facilities.

- **Comisión Federal de Electricidad (“CFE”) (Gomez Palacio, Durango)** – Provided capital costs for construction of a 115 kV power line connecting the site to the national power grid including associated grid upgrades.

**Table 21-1: Summary of Anticipated Initial Capital Costs (including contingency)**

| Capital costs | Millions |
|---|---|
| Mine development and mobile equipment | $155.7 |
| Plant | $308.3 |
| Infrastructure | $51.4 |
| Direct costs | $515.4 |
| Indirect costs | $80.9 |
| Owners costs | $37.3 |
| Total indirect costs | $118.2 |
| Contingency | $81.5 |
| Construction capital | $715.1 |
| Pre-operating mine, plant and G&A | $156.5 |
| Pre-operating net revenue | ($130.9) |
| Total pre-operating capital | $740.6 |

***Notes**: (1) A contingency of 5% has been applied to mine equipment and to light vehicles. (2) A contingency of 15% has been applied to all other expenditures during the first two years of the pre-production period, with the exception of capitalised operating mining cost which has the 15% contingency applied only to the first year of pre-production. (3) The capital cost estimates are based on second quarter 2012 pricing and will be subject to inflation that may occur prior to the construction decision and during the construction period.*

Capital costs incurred after the start of commercial production are considered sustaining capital costs. The sustaining capital for the Pitarrilla Project is expected to be $404 million, including a $25 million contingency. Sustaining capital for the plant is included in plant operating costs, with the exception of a planned $45 million flotation plant upgrade in year 16.

**Table 21-2: Summary of Sustaining Capital Costs**

| Sustaining capital costs | Millions |
|---|---|
| Plant and infrastructure | $6.1 |
| Flotation plant upgrade | $44.5 |
| Mine equipment | $254.0 |
| Mine equipment - major components | $50.2 |
| Tailings impoundment sustaining capital | $37.3 |
| Vehicles/topsoil/water management | $11.8 |
| **Total sustaining capital** | **$403.9** |

***Notes**: (1) A contingency of 5% has been applied to mine equipment and to light vehicles. (2) Sustaining capital for the plant is included in operating costs, with the exception of a planned $45 million expansion of the flotation plant in year 16.*

The detailed initial capital cost estimate is summarised by area in Table 21-3 below. Table 21-4 summarizes the plant and infrastructure capital costs on a more detailed level. A description of each cost area is provided in subsequent paragraphs.

**Table 21-3: Initial Plant and Infrastructure Capital Cost Summary by Facility**

| Area | Description | Estimated Cost (Millions) |
|---|---|---|
| Process Plant | Includes costs associated with primary crushing, crushed ore conveyance and storage, grinding, flotation, concentrate filtration and storage, leaching, CCD thickening and cyanide recovery, Merrill-Crowe and refinery, and reagent storage/distribution. | $195.28 |
| Tailings and Reclaim | Includes costs associated with construction of the tailings starter dam, cyanide destruction tanks, tailings pumping system, and water reclaim system. | $37.08 |
| Infrastructure | Includes site wide preparation and water diversion, costs for power/electrical, fuel supply, and fresh water systems. | $51.98 |
| Roads and Site Set-up | Includes costs for off-site and on-site roads, bridge, general site preparation, site vehicles and maintenance (rolling stock) equipment. | $29.27 |
| Buildings and Camp | Ancillary buildings (guard house, administration, truck shop, laboratory, mill maintenance, change house, fuel stations, explosives storage) and the permanent camp facility. | $25.43 |
| Freight and Duties | Freight and IMMEX (Mexican import duties) | $22.64 |
| ***Total Direct Costs*** | | ***$361.66*** |
| Indirect Costs | Includes estimated bussing costs, construction camp costs, construction power, engineering, procurement, construction management, programming, vendor commissioning and start-up spares. | $81.91 |
| Contingency | Calculated based on estimate accuracy in each category of the estimate for plant construction. | $66.54 |
| ***Total Initial Capital Cost Excluding mining and owners cost*** | | ***$510.11*** |

### Table 21-4: Detailed Initial Plant and Infrastructure Capital Cost Excluding Mining and Owners Costs

| Area | Description | Labour Hours | Contract ($M) | Subcontract ($M) | Construction Equipment ($M) | Bulk Materials ($M) | Process Equipment ($M) | Sub-Total ($M) |
|---|---|---|---|---|---|---|---|---|
| 000 | Site General | 169,608 | 2.267 | - | 4.868 | 6.127 | 5.594 | 18.856 |
| 005 | Access Road | 203,912 | 1.845 | - | 2.684 | 4.110 | - | 8.640 |
| 010 | Bridge | 28,176 | 0.245 | - | 0.834 | 0.690 | - | 1.768 |
| 100 | Primary Crushing | 238,072 | 2.137 | 0.013 | 2.253 | 3.872 | 4.897 | 13.173 |
| 150 | Overland Conveyor | 121,936 | 1.114 | - | 0.849 | 2.395 | 2.903 | 7.261 |
| 200 | SAG Feed Conveyor | 179,846 | 1.539 | - | 1.138 | 3.444 | 3.708 | 9.829 |
| 300 | Grinding & Classification | 495,475 | 5.002 | 0.371 | 3.711 | 12.020 | 31.670 | 52.774 |
| 400 | Lead Flotation & Regrind | 242,340 | 2.353 | 0.008 | 1.883 | 7.536 | 9.737 | 21.519 |
| 410 | Zinc Flotation & Regrind | 140,965 | 1.431 | 0.006 | 0.186 | 2.024 | 9.742 | 13.389 |
| 500 | Concentrate Thickening, Filtration & Storage | 187,877 | 1.688 | 0.004 | 1.276 | 5.021 | 3.766 | 11.756 |
| 540 | Pre-Leach Thickener & Leach Tanks | 193,624 | 2.136 | 0.003 | 1.054 | 3.571 | 19.245 | 26.009 |
| 560 | CCD Thickeners | 180,816 | 1.665 | 0.153 | 0.701 | 4.046 | 8.147 | 14.712 |
| 580 | Merrill-Crowe & Refinery | 140,211 | 1.390 | 0.002 | 1.066 | 4.244 | 7.883 | 14.585 |
| 600 | Tailings Disposal & Detox | 151,551 | 1.578 | 0.006 | 0.634 | 6.015 | 7.415 | 15.648 |
| 610 | Tailings Impoundment | - | - | 20.808 | - | 0.624 | - | 21.433 |
| 620 | Site Wide Water Diversion | 136,227 | 1.233 | 0.038 | 4.798 | 0.946 | - | 7.014 |
| 650 | Fresh Water System | 81,640 | 0.889 | 0.008 | 0.314 | 2.333 | 3.459 | 7.002 |
| 660 | Reclaim Water System | 68,258 | 0.724 | - | 0.388 | 1.637 | 3.714 | 6.463 |
| 700 | Main Substation | 61,729 | 0.547 | 0.842 | 0.077 | 1.144 | 5.000 | 7.610 |
| 750 | Power Transmission Line | - | - | 23.414 | - | 0.468 | - | 23.882 |
| 800 | Sulphide Reagents | 45,455 | 0.428 | 0.004 | 0.261 | 1.049 | 0.897 | 2.638 |
| 810 | Oxide Reagents | 107,707 | 1.062 | 0.004 | 0.584 | 1.909 | 4.072 | 7.631 |
| 910 | Guard House, Safety & Security | 21,814 | 0.217 | 0.002 | 0.035 | 0.630 | 0.897 | 0.972 |
| 920 | Administration Building | 45,877 | 0.448 | 0.002 | 0.048 | 1.183 | 0.028 | 1.708 |
| 930 | Truck Shop, Truck Wash & Warehouse | 191,971 | 1.814 | 0.002 | 0.520 | 4.333 | 1.388 | 8.056 |
| 950 | Mill Maintenance | 29,249 | 0.274 | 0.003 | 0.062 | 0.673 | 0.041 | 1.053 |
| 960 | Laboratory | 38,925 | 0.381 | - | 0.045 | 2.420 | 0.024 | 2.869 |
| 970 | Change House & Lunchroom | 36,120 | 0.355 | - | 0.040 | 0.925 | 0.014 | 1.334 |
| 980 | Fuel Stations | 22,103 | 0.221 | - | 0.104 | 0.393 | 0.863 | 1.581 |
| 990 | Explosives Storage | 2,525 | 0.026 | - | 0.018 | 0.036 | 0.500 | 0.580 |
| 999 | Construction & Permanent Camps | 74,611 | 0.728 | - | 0.728 | 5.821 | - | 7.276 |
| | Freight and Duty | | | | | | | 22.644 |
| | ***Subtotal Direct Costs*** | ***3,638,619*** | ***35.734*** | ***45.690*** | ***31.158*** | ***91.640*** | ***134.797*** | ***361.663*** |
| | % of Subtotal Direct Costs | | 10.5% | 13.5% | 9.2% | 27.0% | 39.8% | 100% |
| | Temporary Facilities, Power & Indirect Field Costs | | | | | | | 4.521 |
| | Construction Camp & Busing Costs | | | | | | | 10.916 |
| | Engineering, Procurement and Construction Management | | | | | | | 61.881 |
| | Pre-Ops, First Fills & Commissioning | | | | | | | 0.890 |
| | Start up Spare Parts | | | | | | | 3.707 |
| | ***Subtotal Indirect Costs*** | | ***-*** | ***-*** | ***-*** | ***-*** | ***-*** | ***81.915*** |
| | Contingency - Direct & Indirect Cost | | | | | | | 66.537 |
| | Escalation (Excluded) | | | | | | | - |
| | ***Total Installed Cost (Estimated)*** | | ***35.734*** | ***45.690*** | ***31.158*** | ***91.640*** | ***134.797*** | ***510.115*** |

### 21.1.1 Capital Cost Summary – Mine and Owner Costs

Capital cost includes all initial capital expenditures for the design, procurement and construction of project facilities, including owner's costs and contingency as well as the sustaining capital required to operate the mine during production. The estimated total mine and owners cost for the initial capital is $367.6 M. A further $353.3 M is spent in the same categories in the production period as sustaining capital. A breakdown of the mine and owner's capital cost is presented in Table 21-5.

Initial capital cost refers to all the capital expenditures incurred during the Pitarrilla Project's construction period, currently estimated as from Year -3 to Year -1. Sustaining capital refers to all the capital expenditure incurred during the actual productive life of the mine. This covers the period Year 1 to end of the mine life.

All mining and owners cost estimates were developed from first principles and where equipment purchases were required, budget equipment supplier quotes were obtained. Details of the manner of each estimate are included in the following sub-sections.

**Table 21-5: Mine and Owner's Capital Costs Breakdown**

| Description | Contingency | Contingency Application | Pre-production Capital Cost ($M) | Sustaining Capital Cost ($M) |
|---|---|---|---|---|
| Owner's Construction/Commissioning Team | 15% | All years | 4.0 | 0.0 |
| Light Vehicles | 5% | All years | 0.8 | 0.0 |
| Plant First Consumables | 15% | All years | 1.3 | 0.0 |
| Capitalised Operating Mining Cost | 15% | All years | 6.9 | 0.0 |
| Capitalised Operating Plant Cost | 15% | All years | 1.4 | 0.0 |
| Capitalised Operating Administration Cost | 15% | All years | 29.5 | 0.0 |
| **Total Owner's Cost (with contingency)** | | | **44.0** | **0.0** |
| | | | | |
| **Mine Equipment** | **5%** | All years | **155.7** | **254.0** |
| | | | | |
| **Mine Equipment Major Components** | | **Not applied** | **0.0** | **50.2** |
| | | | | |
| **Tailings (TSF) Sustaining Capital** | | **Not applied** | **2.3** | **37.3** |
| | | | | |
| Light Vehicles | 5% | All years | 0.0 | 7.1 |
| Mine First Consumables | 15% | All years | 7.8 | 0.0 |
| Topsoil Stripping Sustaining Capital | 15% | Year -3, Year -2 only | 1.0 | 2.9 |
| Waste Dump Water Management | 15% | Year -3, Year -2 only | 0.4 | 1.7 |
| **Vehicles/Topsoil/Water Management** | | | **9.1** | **11.8** |
| | | | | |
| Capitalised Operating Mining Cost | | Not applied | 115.4 | 0.0 |
| Capitalised Operating Plant Cost | | Not applied | 33.8 | 0.0 |
| Capitalised Operating Administration Cost | | Not applied | 7.4 | 0.0 |
| **Capitalised Operating (excluding Owner's Cost Portion)** | | | **156.5** | **0.0** |
| | | | | |
| **Total Mine and Owner's Capital** | | | **367.6** | **353.3** |

**Note:** Any difference in summation is due to rounding.

A contingency factor of 5% was applied to the mine equipment and light vehicles for all periods. All other areas have a 15% contingency applicable to the first two years of the pre-production period and the owners cost having a general 15% contingency with the exception noted for light vehicle purchases. The owners cost without contingency amounts to $37.3 M.

Operating costs incurred during the pre-production years were transferred to pre-production capital costs. These include revenue, capitalised operating mining costs, capitalised operating plant costs, and capitalised operating administration costs.

A list of the required mine equipment and machinery at peak production is presented as follows:

- 10 x Production rock drills
- 3 x Pre-Split rock drills
- 4 x 21 $m^3$ Hydraulic Shovels
- 2 x 19 $m^3$ (Pit) Front End Loaders
- 1 x 12.3 $m^3$ (ROM) Front End Loader
- 27 x 150 tonne Haul Trucks
- 5 x 100 tonne Haul Trucks
- 1 x Scaler (small excavator)
- 2 x 100 tonne Water Trucks
- 9 x Dozers
- 3 x Graders
- Operated Ancillary Equipment (described in Table 21-9)
- Non-Operated Ancillary Equipment (described in Table 21-10)

A purchase schedule taking into account the equipment rebuilds and replacement is presented in Table 21-6 and Table 21-7.

## Table 21-6: Mine Equipment Purchase Schedule (Pre-Production Months)

| Description | Units | Year -3:7 | Year -3:8 | Year -3:9 | Year -3:10 | Year -3:11 | Year -3:12 | Year -2:1 | Year -2:2 | Year -2:3 | Year -2:4 | Year -2:5 | Year -2:6 | Year -2:7 | Year -2:8 | Year -2:9 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| No.Days | days | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 |
| Number of Units Purchased | pre-existing | | | | | | | | | | | | | | | |
| Production Drill rig | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 |
| Pre-split Drill Rig | 0 | 0 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 |
| Pit Shovel | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 |
| Pit FEL | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| 150t Haul Trucks | 0 | 0 | 0 | 0 | 0 | 1 | 1 | 0 | 1 | 1 | 1 | 0 | 0 | 2 | 2 | 1 |
| 100t Haul Trucks | 0 | 3 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| ROM FEL | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Scaler | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Water Truck | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 |
| Dozers | 1 | 1 | 1 | 1 | 0 | 0 | 1 | 1 | 1 | 0 | 0 | 1 | 0 | 0 | 0 | 0 |
| Graders | 0 | 1 | 1 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Operated Ancillary | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Non-Operated Ancillary | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |

| Description | Year -2:10 | Year -2:11 | Year -2:12 | Year -1:1 | Year -1:2 | Year -1:3 | Year -1:4 | Year -1:5 | Year -1:6 | Year -1:7 | Year -1:8 | Year -1:9 | Year -1:10 | Year -1:11 | Year -1:12 | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| No.Days | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | 30.4 | |
| Number of Units Purchased | | | | | | | | | | | | | | | | |
| Production Drill rig | 0 | 1 | 0 | 0 | 1 | 0 | 0 | 1 | 1 | 0 | 1 | 0 | 1 | 0 | 0 | 8 |
| Pre-split Drill Rig | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 3 |
| Pit Shovel | 1 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 4 |
| Pit FEL | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 2 |
| 150t Haul Trucks | 2 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 | 0 | 0 | 23 |
| 100t Haul Trucks | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 5 |
| ROM FEL | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 |
| Scaler | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 |
| Water Truck | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 2 |
| Dozers | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 8 |
| Graders | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 3 |
| Operated Ancillary | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 |
| Non-Operated Ancillary | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 |

## Table 21-7: Mine Equipment Purchase Schedule (Production Years)

| Description | Units | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 | Year 11 | Year 12 | Year 13 | Year 14 | Year 15 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| No.Days | days | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 |
| Number of Units Purchased | | | | | | | | | | | | | | | | |
| Production Drill rig | | 0 | 2 | 0 | 0 | 0 | 1 | 1 | 4 | 0 | 1 | 0 | 0 | 0 | 0 | 0 |
| Pre-split Drill Rig | | 0 | 0 | 0 | 0 | 0 | 2 | 1 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 |
| Pit Shovel | | 0 | 0 | 0 | 0 | 0 | 0 | 2 | 2 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Pit FEL | | 0 | 0 | 0 | 0 | 0 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 1 |
| 150t Haul Trucks | | 0 | 0 | 0 | 4 | 0 | 3 | 12 | 9 | 0 | 0 | 0 | 0 | 0 | 0 | 7 |
| 100t Haul Trucks | | 0 | 0 | 0 | 0 | 0 | 3 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 3 | 0 |
| ROM FEL | | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Scaler | | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 |
| Water Truck | | 0 | 0 | 0 | 0 | 0 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 1 |
| Dozers | | 0 | 0 | 0 | 0 | 1 | 4 | 4 | 0 | 0 | 0 | 0 | 0 | 0 | 2 | 2 |
| Graders | | 0 | 0 | 0 | 0 | 0 | 2 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 2 | 1 |
| Operated Ancillary | | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 |
| Non-Operated Ancillary | | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 |

| Description | Year 16 | Year 17 | Year 18 | Year 19 | Year 20 | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| No.Days | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | |
| Number of Units Purchased | | | | | | | | | | | | | | | |
| Production Drill rig | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 9 |
| Pre-split Drill Rig | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 4 |
| Pit Shovel | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 4 |
| Pit FEL | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 5 |
| 150t Haul Trucks | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 4 | 0 | 0 | 0 | 0 | 0 | 0 | 39 |
| 100t Haul Trucks | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 6 |
| ROM FEL | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 2 |
| Scaler | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 3 |
| Water Truck | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 5 |
| Dozers | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 14 |
| Graders | 0 | 0 | 0 | 0 | 0 | 0 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 7 |
| Operated Ancillary | 0 | 0 | 0 | 0 | 0 | 0 | 0.3 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 2 |
| Non-Operated Ancillary | 0 | 0 | 0 | 0 | 0 | 0 | 0.3 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 2 |

The mine equipment prices per unit applied in the capital cost estimation were based on purchase price quotations obtained from manufacturers effective October 1, 2012 (+/- 2%). This is summarised in Table 21-8.

**Table 21-8: Mine Equipment Unit Purchase Price**

| Mine Equipment | Type/Make | $K/unit |
|---|---|---|
| Production Drill rig | | 1,173 |
| Pre-split Drill Rig | | 731 |
| Pit Shovel | 21 $m^3$ Shovel | 7,400 |
| Pit FEL | 19 $m^3$ Bucket | 5,150 |
| 150t Haul Trucks | | 2,656 |
| 100t Haul Trucks | | 1,432 |
| ROM FEL | 12.3 $m^3$ Bucket | 1,503 |
| Scaler | Excavators, 2 $m^3$ and 4 $m^3$ | 1,326 |
| Water Truck | 100 tonne chassis | 1,521 |
| Dozers | 65 tonne Track Dozer | 1,292 |
| Graders | 275hp | 850 |
| Operated Ancillary | Minor equipment | 7,890 |
| Non-Operated Ancillary | Mine Communications, Pumps, Lights | 1,992 |
| Note: Purchase Price updated to 10/01/12 (+/-2%) | | |

A detailed list of the operated ancillary equipment including the number of units to be purchased and the estimated unit prices is presented in Table 21-9.

**Table 21-9: Mine Operated Ancillary Equipment**

| Operated Ancillary | No of units Purchased | K$/unit |
|---|---|---|
| Compactor 825H | 1 | 881 |
| Blasthole Stemmer (skid steer ) | 1 | 38 |
| Blasters Flatbed Truck (2 t) | 1 | 54 |
| ANFO/Slurry Truck (40,000 lb) | 2 | 630 |
| Fuel/Lube Truck 4,000 gal (15,000 L) | 2 | 414 |
| Road Bed Water Truck, 18,000 L | 1 | 390 |
| Flatbed Truck (8 - 10 ton) | 1 | 47 |
| Crane Truck (8 - 10 ton) | 1 | 66 |
| Tire Handler | 1 | 888 |
| Mechanics Truck | 4 | 290 |
| Welding Truck | 1 | 158 |
| Tractor & Lowboy (75 t) | 1 | 2,000 |
| Shop Forklift | 1 | 86 |
| RT Forklift | 1 | 35 |

Similarly, a detailed list of the non-operated ancillary equipment including the number of units to be purchased and the estimated unit prices is presented in Table 21-10.

**Table 21-10: Mine Non-Operated Ancillary Equipment**

| Not Operated Ancillary | No of units Purchased | K$/unit |
|---|---|---|
| light Plants | 12 | 14 |
| Mine Radios | 72 | 1 |
| Mine Communications Network | 1 | 368 |
| Water Pipe - Dewatering | 1 | 23 |
| Mine Pumps | 1 | 23 |
| Mine Dispatch System | 1 | 1,344 |

Equipment numbers reflect considerations made for rebuilds and replacements after their economic service lives have been attained. Equipment purchases were scheduled to ensure that the appropriate amount of equipment arrived on site to match with the work requirements in all periods.

Light vehicle quantities were estimated for the pre-production and operating periods inclusive of replacements on a 4 year period and equipment currently in operation.

Mine first consumables and spares include the initial diesel and ANFO charges. The estimate covers the first 25 months of the pre-production months. The estimated cost is approximately $7.8 M, including 15% contingency, as presented in Table 21-5.

Plant first consumables refers to SAG mill liners, hardware, screen panels, pebble crusher liners, SAG and ball mill initial ball charges, refinery supplies and safety equipment, operating tools and supplies and other consumable supplies. The estimated cost is approximately $1.3 M, including 15% contingency, as presented in Table 21-5.

SSR developed an owner's capital cost estimate for the Pitarrilla Project that includes owner-related costs such as land acquisition, training, owner's project management, and other costs not included in the EPCM scope. A detailed description of the owner's capital costs is outlined in Table 21-5 and the basis of these estimates are described in Section 21.1.3.7. $44.0 M is estimated as owner's cost inclusive of contingency.

Mine equipment major components' capital costs will be incurred for the purchase of these major components during the actual production years of the mine life. The annual estimate is based on 3.5% of the total operating mining cost incurred in that particular year in so far as there will be production three years after the major components are purchased. No equipment major components purchases are included in the last three years of the Project. Estimated total capital cost for equipment major components is approximately $50.2 M.

TSF sustaining capital refers to all the charges to be incurred during the four planned tailings dam raises, that is, Stages 2 through to 5 of the dam construction. The associated sustaining capital cost schedule is presented in Table 21-11; including the cost incurred to place mine waste from the open pit to the dam site, and all contractor related costs for the actual dam construction, dam monitoring instrumentation, impoundment and under-drain system. An estimated cost of $37.3 M will be incurred as sustaining capital. The expenditure to be incurred during the pre-production months is classified under the initial capital cost and it is estimated as $2.3 M (refer to Table 21-5).

SILVER STANDARD

# Table 21-11: Tailings Storage Facility Sustaining Capital Schedule

| Description | Units | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 | Year 11 | Year 12 | Year 13 | Year 14 | Year 15 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | days | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 |
| Tailings Dam Construction - Mine | K$ | 881 | 881 | 486 | 486 | 486 | 546 | 546 | 546 | 546 | 676 | 676 | 676 | 676 | 676 | 0 |
| Tailings Dam Construction - Contractor | K$ | 257 | 257 | 268 | 268 | 268 | 300 | 300 | 300 | 300 | 298 | 298 | 298 | 298 | 298 | 0 |
| Dam Monitoring Instrumentation - Contractor | K$ | 0 | 100 | 0 | 0 | 100 | 0 | 0 | 0 | 100 | 0 | 0 | 0 | 0 | 100 | 0 |
| Impoundment (basin) - Contractor | K$ | 1128 | 1128 | 1093 | 1093 | 1093 | 785 | 785 | 785 | 785 | 681 | 681 | 681 | 681 | 681 | 0 |
| Underdrain System - Contractor | K$ | 0 | 252 | 0 | 0 | 155 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Underdrain Outfall - Contractor | K$ | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Over 112Mt Capacity | K$ | | | | | | | | | | | | | | | 1518 |
| **Tailings Impoundment Sustaining Capital** | **K$** | **2267** | **2619** | **1847** | **1847** | **2102** | **1630** | **1630** | **1630** | **1730** | **1655** | **1655** | **1655** | **1655** | **1755** | **1518** |

| Description | Year 16 | Year 17 | Year 18 | Year 19 | Year 20 | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 | Year 30 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | |
| Tailings Dam Construction - Mine | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | **8782** |
| Tailings Dam Construction - Contractor | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | **4007** |
| Dam Monitoring Instrumentation - Contractor | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | **400** |
| Impoundment (basin) - Contractor | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | **12081** |
| Underdrain System - Contractor | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | **407** |
| Underdrain Outfall - Contractor | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | **0** |
| Over 112Mt Capacity | 1518 | 1518 | 1215 | 1139 | 1139 | 1139 | 1518 | 889 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | **11593** |
| **Tailings Impoundment Sustaining Capital** | **1518** | **1518** | **1215** | **1139** | **1139** | **1139** | **1518** | **889** | **0** | **0** | **0** | **0** | **0** | **0** | **0** | **37270** |

Topsoil stripping capital refers to the charges associated with stripping the waste dumps, the pit/mine areas, mine roads, stockpile areas and the plant site during the actual production years of the mine life. The stripping cost for the TSF is already included in the TSF construction costs and hence excluded from this section. Topsoil stripping costs are based on estimated quantities of anticipated disturbed areas as a result of the mining operations and mentioned above. These areas were provided by SSR and the equivalent unit cost per hectare was provided by TGI. Pre-production stripping cost is classified under initial capital expenditure and is estimated at approximately $1.0 M (refer to Table 21-5). Total sustaining capital for topsoil stripping is estimated as $2.9 M.

Waste dump water management capital cost estimate was broken down as initial capital cost and sustaining capital cost per waste dump phase for five total phases and then annualised.

Analysis completed by KPL for the waste dump design included, *inter alia*, development of a water management and sediment control program for the waste dumps. The 1 in 200 year storm was used to size rock drains in valley bottoms below the waste dumps and a diversion channel along the south side of the waste dumps. The 1 in 2 year storm event was used to design sedimentation basins and flow monitoring weirs downstream of the waste dump toes. Monitoring wells are also included in the design to measure the groundwater quality over the mine life and at closure.

The waste dump cost estimate includes material, quantities and installation measures to construct the waste dump components. A summary of the waste dump water management design is as follows:

- A drainage system in the valley bottoms at the base of the waste dumps to allow the waste dumps to drain and ensure a low phreatic surface within the waste, both during operations and post-closure;
- A diversion channel to route water away from the SW Waste Dump during large storms events;
- Weirs and sedimentation basins to contain sediment and to allow for monitoring of the surface water quality exiting the waste dumps; and
- Monitoring wells downstream of the weirs to monitor the groundwater quality downstream of the waste dumps.

The pre-production capital cost is estimated at approximately $0.4 M. Similarly, the sustaining capital portion is estimated at approximately $1.7 M.

Revenue within the pre-production period refers to the NSR of production during the pre-production period. This covers a period of ten months and is estimated at $130.9 M.

Capitalised operating mining cost refers to all of the total operating mining cost incurred during the pre-production months. This is estimated at $115.4 M excluding owner's cost portion. This includes the initial operating mining cost to establish the initial access to the pit, to construct the ROM Pad, and to commence pit mining from Breccia Ridge 1 East, Cordon Colorado and Breccia Ridge Phase 1.

Capitalised operating plant cost refers to all of the operating plant costs incurred during the pre-production months. The total capitalised operating plant cost excluding owner's cost portion is estimated at $33.8 M.

Capitalised operating administration cost refers to all of the pre-production operating costs relating to administration, community relations, health, safety and security, environmental and permits, and human resources. The total capitalised operating administration cost excluding owner's cost portion is estimated at $7.4 M.

### 21.1.2 Detailed Work Break Down Structure and Scope of Facilities

The capital cost estimate for the Pitarrilla Project was organised into the work break down structure areas, as described below in Table 21-12.

**Table 21-12: Detailed Work Breakdown Structure and Scope of Facilities**

| Area | Description |
|---|---|
| **000** | **Site General**<br>Systems or facilities that cross multiple areas of the plant. Included is the overall site plan grading, in plant roads, utility distribution, drainage trenches, plant site storm water diversions, communications, fencing, and construction support such as roads, laydown areas, work pads and ponds. |
| **010** | **Road Upgrade**<br>Includes upgrades to existing roads from Federal Highway 45 to the plant site, approximately 47km total. |
| **020** | **Bridge**<br>Includes the construction of a bridge near San Francisco de Asis to span the Nazas River. |
| **050** | **Mine**<br>Included are facilities such as mine equipment and the mine dispatch system. Mine maintenance facilities such as the truck shop and truck wash are considered part of the ancillary facilities in area 930. The powder magazines and ammonium nitrate silos are also in area 990. |
| **100** | **Primary Crushing and Storage**<br>This area consists of the primary crushing facilities and the ROM feed stockpile area. Reclaim from the ROM feed stockpile will be by front end loader to feed the primary crusher pocket. |
| **150** | **Overland Conveyor**<br>This area consists of the discharge conveyor from the primary crusher and includes the overland conveyor, two stacking conveyors, and two coarse ore stockpiles. The reclaim feeders and reclaim conveyors under the stockpile are in area 200. |

| Area | Description |
|---|---|
| **200** | **SAG Feed Conveyor**<br>This area consists of the two reclaim tunnels under each coarse ore stockpile and the conveyors to feed the SAG mill. Also included is the ball feed equipment to add grinding balls to the SAG feed conveyor. |
| **300** | **Grinding & Classification**<br>This area consists of the grinding circuit including the SAG mill, ball mill, recirculating conveyors, a pebble mill, and cyclone classification. |
| **400** | **Lead Flotation & Regrind**<br>This area consists of the lead flotation circuits, including rougher flotation cells, two stage cleaner circuits, and regrind circuit. |
| **410** | **Zinc Flotation & Regrind**<br>This area consists of the zinc flotation circuits including rougher flotation cells, three stage cleaner circuits, and regrind circuit. |
| **500** | **Concentrate Thickening and Filtration**<br>This area consists of the concentrate thickeners for lead and zinc as well as filtration facilities. Also included are the lead and zinc concentrate product storage and load out facilities. |
| **540** | **Pre-leach Thickener & Leach Tanks**<br>This area consists of the pre-leach thickener and the leach tank facility. |
| **560** | **CCD Thickeners**<br>This area consists of the counter-current decantation ("CCD") thickeners and the cyanide recovery thickeners. |
| **580** | **Merrill-Crowe & Refinery**<br>This area includes the Merrill-Crowe and the refinery. |
| **600** | **Tailings Disposal & Detoxification**<br>This area consists of the tailings dewatering thickener, the cyanide destruction tanks, the associated pumping systems, and the overland pipeline (tailings distribution pipeline) to the TSF. |
| **610** | **Tailings Impoundment**<br>This area consists of the TSF starter dam and geosynthetic liner for the tailings impoundment (initial stage). |
| **620** | **Site Wide Water Diversions**<br>This area consists of all drainage and water diversion channels around the site, including a surface water diversion structure and diversion channel for the TSF and a channel for the process plant. |
| **650** | **Fresh Water Systems**<br>This area consists of the fresh water system and includes groundwater wells, well development, overland pipeline, booster stations, fresh/fire water storage tank, water treatment, and distribution to the various areas on the site. |
| **660** | **Process Water System**<br>This area consists of the TSF reclaim water system, overland pipeline to the plant site, the process water storage tank, and the recycled process water system. |

| Area | Description |
|---|---|
| **700** | **Main Substation**<br>This area consists of the plant main substation, 115kV step down transformers to 13.8kV distribution switchgear, the power distribution in underground ducts or overhead lines to other transformers or switchgear in the various areas of the plant, and power lines to the fresh water pumps and fresh water booster stations. Distribution from the transformers to other switchgear, motor control centers to the various equipment and motors is included in the relevant areas. This area also includes the mine pit loop power line. |
| **750** | **Overhead Power Transmission Lines**<br>The new, high voltage 115kV power line from the utility company (CFE's) switchyard to the plant's main substation. |
| **800** | **Sulphide Reagents**<br>This area consists of receiving, storage, mixing, and metering of reagents for the flotation process. |
| **810** | **Oxide Reagents**<br>This area consists of receiving, storage, mixing, and metering of reagents for the leaching process, including the lime slaking package and the sulphur burner. |
| **910** | **Guard House, Safety & Security**<br>This area consists of the guard house, security building and safety building, including the ambulance garage and the truck scale. |
| **920** | **Administration Building**<br>This area consists of the administration building and the mine services facilities. |
| **930** | **Truck Shop, Truck Wash & Warehouse**<br>This area consists of the truck shop, truck wash, warehouse and light vehicle maintenance area. |
| **950** | **Mill Maintenance**<br>This area consists of the mill maintenance building. |
| **960** | **Laboratory**<br>This area consists of the assay laboratory. |
| **970** | **Change House & Lunchroom**<br>This area consists of the on-site change house and lunchroom facilities. |
| **980** | **Fuel Stations**<br>This area consists of the light and heavy vehicle fuel stations. |
| **990** | **Explosives Storage**<br>This area consists of the explosives storage, detonator storage, and ammonium nitrate prills storage silos. |
| **999** | **Construction & Permanent Camps**<br>This area consists of the owner's construction and permanent camps and all support facilities. |

The flotation circuit in the plant will be expanded in Year 16 to provide sufficient capacity required for the increasing lead and zinc metal grades, as predicted in the mining schedule. The expansion will include the following components:

- Lead 1st cleaner flotation cells
- Lead 2nd cleaner flotation cells
- Lead concentrate filtering
- Lead concentrate storage
- Zinc rougher concentrate regrinding mill, complete with cyclones
- Zinc 1st cleaner flotation cells
- Zinc 2nd cleaner flotation cells
- Zinc 3rd cleaner flotation cells
- Zinc concentrate filter
- Zinc concentrate storage

These will be housed in a new building, situated adjacent to the initial flotation and concentrate dewatering section of the plant.

A construction cost estimate was prepared by M3 for this flotation circuit expansion, and details are shown in Table 21-13. These costs will be incurred as sustaining capital costs.

**Table 21-13: Flotation Circuit Expansion Costs**

| Area | Description | Total ($M) |
|---|---|---|
| 405 | Lead Flotation Expansion | 8.6 |
| 415 | Zinc Flotation and Regrind Expansion | 9.6 |
| 505 | Concentrate Thickening and Filtration Expansion | 11.4 |
| | **Total Direct Level Costs** | **29.7** |
| | Indirect Costs (50%) | 14.8 |
| | **Total Cost** | **44.5** |

### 21.1.3 Basis of Capital Estimate

#### 21.1.3.1 General Assumptions and Clarifications

The capital cost estimate is based on an open pit mine operation treating both sulphide and oxide ores on a campaign basis. Sulphide mining, crushing, grinding, flotation, and tailings facilities are based on treatment and production of an average of 16,000 tonnes of ore per day. Oxide mining, crushing, grinding, leaching, CCD, cyanide recovery and destruction, and tailings are based on treatment and production of 12,000 tonnes of ore per day.

Mill grade ore will be transported to a primary gyratory crusher and crushed to a nominal size of 150 mm and stockpiled in one of two coarse ore stockpiles.

During sulphide ore processing, material will proceed from the stockpile for further treatment in the grinding and flotation facility to produce a lead concentrate and/or zinc concentrate product. The concentrate will be loaded into trucks for shipping to a local smelter or to a port in Mexico for oceanic shipping to an overseas smelter. Sulphide tailings material will be pumped to the oxide circuit for leaching and achieving further silver recovery.

During oxide ore processing, material will proceed from the stockpile for further treatment in the grinding, cyanide leaching, and CCD thickening, before proceeding to a Merrill-Crowe and refinery facility to produce doré. Cyanide will be recovered before tailings material is thickened and pumped to a conventional TSF, located to the southeast of the process plant.

The capital cost estimate is based on second quarter 2012 US funds and it is considered to be within a ±15% level of accuracy. Strictly speaking, actual project costs could be expected, therefore, to range from 15% above the estimated amount to 15% below the estimated amount. Practically speaking, however, projects are typically completed in the 0 to +15% range. The estimate accuracy is a separate issue from contingency. Specifically, contingency is intended to account for costs that are expected to be incurred, but which cannot be quantified with the level of information available. As such, the contingency represents a budget amount to be expended against unforeseen in-scope changes or developments and to allow for uncertainties in the current scope. Further, it should be noted that contingency does not include coverage for out-of-scope items or activities that may arise during project execution. Examples of such things that are not covered by contingency could be:

- delays due to weather;
- labor unrest/strikes;
- terrorism;
- force majeure;
- legislation changes;
- addition of unforeseen facilities, or
- other similar aspects that have not been included to some extent in the project estimate presented here.

Freight, permits and shipment insurance were included at 10% of the equipment cost for US-sourced equipment. Budget quotes for these items were obtained for foreign sourced equipment. Based on previous experience with Mexican projects, import duties are not anticipated for North American sourced equipment. There may be import duties on goods from Europe and Asia.

Costs have been included for plant acceptance and initiation of operations as per the following:

- Mechanical completion – by Contractor;
- Commissioning – by Contractor;
- Initial fills – by Owner but estimated by M3 Engineering;
- Start-up – by Owner; and
- Demonstration test – by Owner.

The following items have been excluded from the capital cost estimate, unless otherwise noted:

- Sunk costs prior to the Pitarrilla Feasibility Study (M3, 2012);
- Hedging, escalation or project financing costs;
- Costs and scope/facility adjustments to accommodate future expansion;
- All taxes such as Mexican sales tax;
- Special project incentives related to project schedule acceleration and productivity improvement; and
- Exchange rate fluctuations.

21.1.3.2 Basis of Estimate – Mine

SSR completed most of the mine planning activities for the Pitarrilla Feasibility Study (M3, 2012).

SSR also retained the services of IMC to verify the process undertaken by SSR through making independent process calculations of each major step of the estimation process. The estimates of mining-related capital costs were based on the assumption that the mine will be directly operated and managed by SSR.

Costs are presented in $USD based on Q2 2012 prices. No allowances were made for escalation of capital or operating costs beyond 2012.

Initial pit access and ROM pad development will be achieved through mining of borrow pits. Material from these borrow pits will be used to construct critical accesses to the pit. A set of minor designs completed to facilitate the pit development is described in detail in the mining methods section of this Technical Report. SSR estimated the requirements for pre-production equipment for mine development, as presented in Table 21-6 above.

The general scope limits of this estimate are the mine equipment, light vehicles, mine first consumables and spares, plant first consumables and spares, owners construction and commissioning team, mine equipment major components, TSF sustaining capital, topsoil stripping sustaining capital, waste dump water management, revenue within the pre-production period, and operating mining, plant and administration costs transferred to capital during the pre-production period. Initial plant and infrastructure, as well as flotation plant upgrade capital cost estimates, are discussed in other sections of this Technical Report.

Mine development, construction, and operations activities are based on a two-shifts-per-day, 9.5 operating hours-per-shift schedule.

21.1.3.3 Basis of Estimate – Process Plant, Tailings Storage Facility and Infrastructure

Documents available to the estimators included the following feasibility level items.

| | | |
|---|---|---|
| a) | Process Design Criteria | Yes |
| b) | Equipment List | Yes |
| c) | Equipment Data Sheets | Yes |
| d) | Equipment Specifications | Major equipment only |
| e) | Construction Specifications | No |
| f) | Flow sheets | Yes |
| g) | P&IDs | Yes |
| h) | General Arrangement Drawings | Yes |
| i) | Architectural Drawings | Yes |
| j) | Civil Drawings | Yes |
| k) | Concrete Drawings | No |
| l) | Structural Steel Drawings | No |
| m) | Mechanical Drawings | No |
| n) | Electrical Single Lines | Yes |
| o) | Electrical Physicals | No |
| p) | Instrumentation Schematics | No |
| q) | Instrument List | Yes |
| r) | Pipeline Schedule | No |
| s) | Valve List | No |
| t) | Cable and Conduit Schedule | No |
| u) | Equipment Quotations | Yes, for most process equipment |
| v) | Material Quantity Estimates | |
| | • Concrete | Yes |
| | • Steel | Yes |
| | • Civil Earthwork | Yes |
| | • Large Overland Pipe | Yes |

Labour rates are based on similar projects executed in Mexico in recent years. Craft labour has been estimated at the following blended labour crew rates, as shown in Table 21-14:

**Table 21-14: Blended Wage Rates**

| Labour Crew Rates | Base Rate $/Hour | Overtime | Indirect Costs Supervision | Other Indirect Costs | Total Rate $/Hour |
|---|---|---|---|---|---|
| Civil work | 9.05 | Included | Included | Included | 9.05 |
| Concrete work | 6.85 | Included | Included | Included | 6.85 |
| Concrete forming & architectural | 6.85 | Included | Included | Included | 6.85 |
| Reinforcing steel | 9.05 | Included | Included | Included | 9.05 |
| Structural steel | 9.75 | Included | Included | Included | 9.75 |
| Equipment installation | 11.96 | Included | Included | Included | 11.96 |
| Piping installation | 12.35 | Included | Included | Included | 12.35 |
| Electrical and instrumentation | 9.05 | Included | Included | Included | 9.05 |

Indirect field labour costs include field payroll burdens, overhead, field supervision, supervisory burdens and utilities, as well as trailers, vans and pickup trucks, cribbing, water trucks, safety insurance, and construction permits. Not included in the hourly rates are contractor costs for camp facilities and meals. The base rate includes benefits and the contractor's margin.

The estimate assumes that the project will carry a single Engineering, Procurement and Construction Management ("EPCM") contractor, responsible for coordination of all aspects of the project.

Method of Contracting

The contracting approach for the Pitarrilla Project can significantly impact the overall project cost and schedule. M3 has assumed the contracting methodology outlined below:

- EPCM approach with a single engineering, procurement and construction management contractor.;
- Encourage qualified local and national contractors to bid;
- Use two or more contractors rather than a single, large contractor for major trades; this encourages competition and reduces risk (i.e. non-performance); and
- Use low unit price contracts and lump sum contracts adequately supported by engineering, as opposed to cost plus or cost reimbursable contracts or EPC approach.

Construction subcontracts anticipated for the Pitarrilla Project include civil, concrete, structural steel, mechanical, piping, electrical and instrumentation. Construction of the Starter Dam for the TSF will also be performed by a contractor. Design, supply and erect contracts are anticipated for the access roads, the bridge over the Nazas River, the 115 kV transmission line, the existing roadway modifications, the pre-engineered buildings and the camp facilities.

Temporary construction sanitary facilities and construction camps will be provided by the general contractor and/or subcontractors. Any existing owner's facilities are for the owner's use and, in general, will not be available to construction personnel.

Temporary construction water will be provided by existing wells and a truck loading station currently on site. The general contractor and/or subcontractors will be responsible to haul temporary water from this station as necessary. Off-site contracts for the fresh water supply system can precede mobilization to site and may be available to provide water during construction.

The Owner will provide site security services, source of construction water, telephone lines, and power for the general contractor or subcontractors. The general contractor or subcontractors will provide their own hand-held radio communication system. The general contractor and subcontractors will be responsible for their own drinking water and portable toilets and all utility hookups (e.g., telephone and power into construction trailer) as well as delivery of construction water. Subcontractors will be responsible for their own temporary in-field construction power.

The Owner will not supply any construction equipment (such as forklifts and crane for unloading or water trucks for dust suppression) to the Pitarrilla Project.

It has been assumed that construction work areas will be accessible to the contractors 24 hours per day, seven days per week. Allowance is not included in the estimate for standby time for inefficiencies resulting from work stoppages or interferences initiated by operations or others.

Contractors can have their office trailer and laydown yard near the construction site. Construction personnel can park their construction vehicles near the construction site. Personal vehicles will be parked off-site and personnel bussed to the site.

The general contractor and each subcontractor will be responsible for the receiving of all materials and equipment in their scope. Any items that have been received prior to construction or inadvertently received by the Owner will be loaded and transported to the construction site by the appropriate contractor. In general, Owner personnel will not receive shipments during contractor's off-hours. Each contractor will be responsible for the security of received material, including warehouse facilities and fenced storage facilities.

21.1.3.4 Material Take-offs

Material unit prices for the Pitarrilla Project were estimated using costs gained through contacts with local regional suppliers, information from recently-constructed projects, and M3's in-house database of historical pricing.

Civil work quantities for general excavation, grading and backfill were taken off the site plot plans, general arrangement drawings and grading plans. TGI provided quantity estimates for the TSF construction, including a surface water diversion structure and diversion channels.

Concrete quantities were developed from general arrangement drawings and experience with similar projects. An allowance has been made for lean concrete.

Building architectural costs are based on new construction. Engineered buildings were based on material take-offs from general arrangement drawings. Pre-engineered buildings were estimated based on current in-house data. Internal architectural finishes were factored based on building area.

Structural steel quantities were developed from the general arrangement drawings and experience with similar installations. Quantities include allowances for miscellaneous steel, including base plates, bracing, bolts and gussets.

Take-offs were made for mechanical steel including plate work, abrasion resistant liners, ductwork, and, based on the general arrangement drawings, equipment lists and experience with similar installations.

Overland piping quantities were developed from the site plot plans and general arrangement drawings. Other general piping quantities were based on P&IDs, general arrangements, and experience with similar installations. Valves and miscellaneous fittings were factored based on experience with similar installations.

Electrical take-offs were based on the electrical single line diagrams, plot plans, and experience with similar installations. Bulk electrical materials were factored based on experience with similar installations.

Instrumentation take-offs were based on the P&ID's, instrument log, and experience with similar installations.

Construction equipment costs were estimated according to the tasks performed and the crew hours involved. Construction equipment is included as a direct cost.

21.1.3.5 Basis of Estimate – Indirect Costs

For the tabulation of this estimate, indirect field labour costs were included in the direct field labour costs. Based on experience with similar projects, an allowance of $1.8 M (0.5% of direct costs) was provided as an indirect field cost for a single construction mobilization effort.

EPCM services were estimated at 16.5% of the total direct field costs. The estimate was priced on the basis of a mid-sised engineering and construction management company performing the work.

A $2.50 per man-hour allowance was made for a contractor-supplied construction camp. This allowance includes money for shelter, food and housekeeping for construction workers during the project. An additional $0.50 per man-hour allowance was made for bussing of workers to the site during the construction project, for a total of $3.00 per man-hour.

An allowance of $0.9 M (0.25% of direct costs) was made for construction power and utilities.

An allowance of $0.7 M (0.2% of direct costs) was made for a construction trailer complex for the EPCM group.

An allowance of $1.5 M (1% of plant equipment) was made for supervision of specialty equipment. This includes the cost of vendor representatives on site for equipment installation.

An allowance of $0.4 M (0.3% of plant equipment) was made for pre-commissioning and $0.4 M (0.3% of plant equipment) for commissioning.

An allowance of $3.7 M (2.5% of plant equipment) was made for capital and commissioning spare parts. Operating spare parts are not included.

21.1.3.6 Basis of Estimate – Contingency

Estimate accuracy is a separate issue from contingency. Specifically, contingency is intended to account for costs that are expected to be incurred, but which cannot be quantified with the level of information available. As such, the contingency represents a budget amount to be expended against unforeseen in-scope changes or developments and to allow for uncertainties in the current scope. Further, it should be noted that contingency does not include coverage for out-of-scope items or activities that may arise during project execution.

The contingency was calculated by applying a percentage factor for each cost category. The factor was established giving consideration to the level of detail and accuracy of available information in each area of the estimate, as shown in Table 21-5.

#### 21.1.3.7 Basis of Estimate – Owner's Costs

As earlier mentioned, the owner's capital cost for the Pitarrilla Project was developed by SSR and includes owner-related costs such as land acquisition, training, insurance, owner's project management, construction and commissioning team, and other costs not included in the EPCM scope. Below is an explanation of the items included in the owner's cost estimate.

- Project Light Vehicles – All light vehicles purchased during the pre-production period.
- Plant First Consumables – All pre-production charges relating to SAG mill liners, hardware, screen panel, pebble crusher liners, SAG and ball mill initial ball charges, refinery supplies and safety equipment, operating tools and supplies and other consumable supplies; until start of the plant.
- Owner's Construction and Commissioning Team – All charges relating to the activities of owner's construction and commissioning team comprising manuals (mostly labor component), a project manager, a construction manager, an engineering manager, four construction supervisors, a project controls manager, a field coordinator, project drawing control and project administration.
- Capitalised Operating Mining Cost – All pre-production operating mining cost.
- Capitalised Operating Plant Cost – All pre-production operating plant cost.
- Capitalised Operating Administration Cost – All pre-production operating administration cost incurred until the start of the plant. The total estimated capitalised operating administration cost including owner's cost portion is $36.9 M. This cost includes some special assignment cost items required for the start-up of the operations. Some of the special assignment cost items are:
    - Education/Seminars and Scholarships
    - Consultants and External Services
    - Air Charter
    - Travel Cost (construction management)
    - Vehicle Expenses (construction management)
    - Software upgrades and support contracts
    - Safety Supplies
    - Recruitment and relocation cost

- Distributed power cost
- Environmental impact report preparation, monitoring and permitting;
- Land access and water rights;
- Legal and audit fees;
- Hiring and relocation allowances for project and operation teams;
- Builder's all-risk, building and third party liability insurance;
- Owner employee training;
- Employee transportation to/from site;
- Owner camp operating costs;
- Community and governmental relations;
- Property taxes;
- Independent consultants;
- Insurance (all types)

- Contingency on Owner's Costs – Provided as an allowance based on historical experience with similar projects. The applicable contingency percentages are as presented in Table 21-5.

### 21.1.4 Capital Cost Estimate Major Governing Assumptions

The following major assumptions have been used to guide the estimate assembly and provide a consistent basis for various aspects. In particular, all costs incurred prior to a construction decision by Silver Standard are considered sunk costs and not included in the capital cost estimate.

Many of the various assumptions used have been based on M3's experience with similar historical capital projects. This section documents those assumptions to clarify the basis of the associated costs/budgets. The area-by-area detailed assumptions are outlined as follows.

#### 21.1.4.1 Mill and Process Plant

A mobile crane will be used for primary crusher maintenance.

#### 21.1.4.2 Tailings Storage Facility

- The TSF embankment will use downstream construction methodology and will comprise three zones.

    - Zone 1 will consist of three metres (measured horizontally) of clayey sand fill, constructed on the upstream face of the dam to provide a bedding layer for the geosynthetic lining system.

    - Zone 2 will consist of a minimum of three metres (measured horizontally) of silty sands or gravels that are filter compatible between the liner bedding and rockfill.

  - Zone 3 will consist of rockfill borrowed from the plant site and tailings impoundment area for Stage 1 construction and waste rock from the open pit mining operations for Stages 2 through 5.

- Initial starter dam construction will be followed by the first dam raise (as a sustaining capital cost) approximately two years after mill start-up.

- Local borrow material, from areas within the tailings impoundment area and the plant site, will be used for starter dam construction.

- The impoundment and upstream face of the embankment will be covered with a geosynthetic liner as the bedrock in the TSF area generally has a high hydraulic conductivity which does not reduce significantly with depth; therefore, a low permeability, geosynthetic liner is required to minimize potential seepage.

- A diversion channel and diversion structure will be constructed for the TSF and the plant site during initial capital construction. Local borrow materials from the excavation of the TSF diversion channel will be used for the construction of the TSF diversion structure. General fill from the excavation of the diversion channels will be placed on the upstream side of the diversion structure to prevent water from pounding against the structure.

#### 21.1.4.3 Utilities

Per results of water exploration and associated analyses, a minimum of three groundwater wells are planned to be installed in the well field.

The cost for the main power transmission line and grid connection was supplied by CFE. This cost will be borne by the Pitarrilla Project. The estimate was reviewed by the project team and judged to be reasonable.

### 21.1.5 Closure Costs

#### 21.1.5.1 Summary

A closure cost estimate was developed, based on past experience for Mexican projects, and to be suitable for permit requirements to manage and minimize the long term impacts of the Pitarrilla Project. The closure and reclamation plans may be adjusted throughout the mine life.

The closure costs are estimated in second quarter 2012 dollars, and indirect costs are added, including supervision labour. No added contingency, allowances or offsets have been included in the closure cost estimate. Total estimated closure cost is $76 M and the cost breakdown is shown in Table 21-15.

The closure and reclamation effort would begin in Mine Plan Year 26, three years prior to plant closure, and continue for 20 years inclusive of the monitoring period. However, for financial estimation, costs after closure have been accumulated to the last period of plant processing.

Some closure and reclamation activities may be initiated earlier in the mine life as facilities are completed, such as reclamation of waste dumps once they've reached their full capacity. The assumption for the financial analysis is, however, that this does not begin until three years prior to plant closure. Other facilities, such as the TSF, cannot be reclaimed until all of the ore processing is completed.

#### 21.1.5.2 Basis of Estimate for the Mine and Plant Closure and Reclamation Costs

Below is a list of assumptions of the closure and reclamation cost estimate:

- The following facilities will remain intact for the local communities and will not require reclamation or demolition: public roads and bridges, including all access roads, the camp and associated infrastructure, and the transmission line.
- All buildings and structures not associated with the camp are to be dismantled and removed for salvage, with a net zero cost to SSR, which follows international standards for closure costing that do not allow credit for salvage of equipment and scrap metal.
- All concrete not covered by at least one metre of fill will be broken up and removed, and either disposed of in a landfill or crushed and used as fill on site.
- Buried services including piping may be left in place, provided at least one metre of fill covers it.
- All equipment and assets will be removed from the site.
- Slopes of re-vegetated areas (i.e. waste dumps, tailings dam) are based on 2-dimensional areas from design drawings.
- Closure of buildings assumes that a backfill will be placed over broken foundations.
- Soil contamination is assumed in industrial areas.
- Type of growth medium and re-vegetation will be based upon a future biological study.
- A source for growth media and backfill is assumed to be available on the mine property.
- Reclamation costs for the borrow site have not been included.
- Post-closure data review and additional reporting are not included.
- Personnel costs include severance pay.
- Groundwater sampling costs are based on sample collection and laboratory costs from similar projects.
- Assumes that no groundwater treatment will be needed in the future.
- Assumes that discharge from the mine will end once dewatering stops and the depth of the pit will require dewatering during a portion of the mine operation.
- Assumes adequate diversion channels and natural drainages exist on site to divert water away from the tailings impoundment and industrial areas. Therefore, no additional surface water controls were included in the closure cost.
- Assumed a contingency of 35% for estimated costs as based on guidelines from US Forest Service bond guidance (2004).
- Some tasks are based on man-hours. The task timing is not intended to be in a single time period. For example, a level of effort of "3 weeks" is not intended to be scheduled during a single 3-week period.

For the purposes of social responsibility, a document will be prepared regarding the closure commitments, and it is anticipated that this document will indicate the proposed conditions of the areas after closure.

In addition, Owner-owned equipment was also used to develop costs for reclamation. These costs are based on the estimated costs for operations work (i.e. dozers for waste dump slope shaping and tailings cover spreading).

The planned closure costs and schedule are summarised in Table 21-15 and Table 21-16.

**Table 21-15: Closure Cost Summary**

| **Item Description** | **SubTotal (M$)** |
|---|---|
| Tailings Impoundment | 18 |
| Waste Rock Dump Area | 14 |
| Open Pit Area | 1 |
| Sulphide Plant, Oxide Plant and Ancilliary Facilities | 4 |
| Crusher Area | 2 |
| Technical Studies & Engineering Designs | 6 |
| Closure Management | 9 |
| Subtotal of All Capital Cost Items * | 54 |
| Contingencies (35%) ** | 19 |
| Subtotal with Contingencies | 72 |
| Post-Closure Care & Maintenance | 3 |
| **Total Reclamation and Closure Cost (USD)** | **76** |

Notes:

* Costs assume no water treatment and no special handling for mine wastes (for example, the waste rock dumps).

** 35% contingency is based on international standards for engineering at conceptual level. Contingency based on guidelines from US Forest Service bond guidance (2004).

## Table 21-16: Conceptual Closure and Reclamation Schedule

| Activity | Closure activities during operations | Pre-Closure | | | Closure | | | Post-Closure |
|---|---|---|---|---|---|---|---|---|
| | | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 - Year 45 |
| **Closure Planning** | | | | | | | | |
| Notification to Agencies (LGPGIR, Artíclo 46 and Artíclo 68) | | | | | | | | |
| Risk Assessment | | | | | | | | |
| Environmental Aspects | | | | | | | | |
| Social Aspects | | | | | | | | |
| Schedule | | | | | | | | |
| Reclamation | | | | | | | | |
| **Social Responsibility** | | | | | | | | |
| Compromiso de Cierre | | | | | | | | |
| Severance Package Payout | | | | | | | | |
| **Characterization** | | | | | | | | |
| Tailings Impoundment | | | | | | | | |
| *Geochemistry Testing* | | | | | | | | |
| *Geotechnical Testing & Drilling* | | | | | | | | |
| Infrastructure | | | | | | | | |
| *Hydrocarbons Testing & Removal* | | | | | | | | |
| Groundwater | | | | | | | | |
| **Engineering Design & Planning** | | | | | | | | |
| Tailings Impoundment | | | | | | | | |
| Mine Pit | | | | | | | | |
| Infrastructure | | | | | | | | |
| Waste Dumps | | | | | | | | |
| **Decommissioning** | | | | | | | | |
| Facilities | | | | | | | | |
| *Tailings Impoundment* | | | | | | | | |
| *Mine Pit* | | | | | | | | |
| *Waste Dumps* | | | | | | | | |
| *Process Ponds* | | | | | | | | |
| Infrastructure | | | | | | | | |
| *Electric Power* | | | | | | | | |
| *Water Management* | | | | | | | | |
| *Buildings & Ancillary Facilities* | | | | | | | | |
| **Demolition** | | | | | | | | |
| Physical Structure & Management Infrastructure | | | | | | | | |
| *Salvage* | | | | | | | | |
| **Reclamation** | | | | | | | | |
| Facilities | | | | | | | | |
| *Tailings Impoundment* | | | | | | | | |
| *Mine Pit* | | | | | | | | |
| *Waste Dumps* | | | | | | | | |
| *Community* | | | | | | | | |
| Infrastructure | | | | | | | | |
| *Water Management* | | | | | | | | |
| *Buildings & Ancillary Facilities* | | | | | | | | |
| *Landfill* | | | | | | | | |
| **Post-Closure Monitoring** | | | | | | | | |
| Groundwater | | | | | | | | |
| *Monitoring Wells* | | | | | | | | |
| Physical Inspections/Monitoring | | | | | | | | |
| *Geotechnical* | | | | | | | | |
| *Erosion* | | | | | | | | |
| *Revegetation* | | | | | | | | |
| *Seepage Monitoring* | | | | | | | | |

## 21.2 OPERATING AND MAINTENANCE COSTS

This section addresses the following costs:

- Mine operating and maintenance costs;
- Process plant operating and maintenance costs; and
- General and administration costs.

The operating and maintenance costs for the Pitarrilla Project operations are summarised by area of the operation, and shown in Table 21-17. Cost centres include mine operations, process plant operations, laboratory, and the general and administration area. Operating costs were determined on an annual basis for both flotation and leaching operation. The products to be produced are a zinc concentrate, a lead concentrate and silver doré. 1,514,600 dry metric tonnes of zinc concentrate will be produced yielding 1,669 Mlbs of zinc and 29.4 Mozs of silver. The lead concentrate production will be 603,700 dry metric tonnes producing 582 Mlbs of lead and 185.4 Mozs of silver. The doré will contain 118.5 Mozs of silver. The life of mine unit cost per tonne of total ore milled is estimated to be $27.29 (including pre-production tonnes).

**Table 21-17: Life of Mine Operating Costs (Inclusive of Capitalised Operating)**

| | | | |
|---|---|---|---|
| Direct Leach Ore (kt) | 43,356 | | |
| Flotation/Leach Ore (kt) | 113,234 | | |
| Total Ore Mined (kt) | 156,590 | | |
| Waste Mined (kt) | 933,685 | | |
| Total Ore and Waste Mined (kt) | 1,090,275 | | |
| | **Life of Mine Cost (K$)** | **$/ore tonne mined** | **$/total material mined** |
| **Mine Operations** | | | |
| Drilling | 162,516 | 1.04 | 0.15 |
| Blasting | 269,534 | 1.72 | 0.25 |
| Loading | 188,143 | 1.20 | 0.17 |
| Haulage | 491,354 | 3.14 | 0.45 |
| Mine Support | 245,200 | 1.57 | 0.22 |
| General Mine | 214,755 | 1.37 | 0.20 |
| Total Mine Operations | **1,571,523** | **10.04** | **1.44** |
| **Mill Operations** | | | |
| Total Primary Crushing | 75,219 | 0.48 | |
| **Flotation/Leach Ore** | | **$/flotation/leach ore tonne milled** | |
| Grinding | 606,726 | 5.36 | |
| Flotation | 303,472 | 2.68 | |
| Concentrate Thickening, Filtration | 102,834 | 0.91 | |
| Tailings Leaching CCD | 479,853 | 4.24 | |
| Tailings Refinery | 67,076 | 0.59 | |
| Tailings | 32,285 | 0.29 | |
| Total Flotation/Leach Ore | 1,592,245 | 14.06 | |
| **Direct Leach Ore** | | **$/direct leach ore tonne milled** | |
| Grinding | 288,663 | 6.66 | |
| Leach/CCD | 210,678 | 4.86 | |
| Merrill Crowe/Refinery | 34,762 | 0.80 | |
| Tailings | 21,733 | 0.50 | |
| Total Direct Leach Ore | 555,835 | 12.82 | |
| Plant Administration* | 196,741 | 1.26 | |
| Total Mill* | **2,420,040** | **15.45** | |
| General Administration* | **282,108** | **1.80** | |
| **Grand Total*** | **4,273,671** | **27.29** | |

Note:* Unit cost expressed as dollar per combined ore tonne milled.
Any difference in summation is due to rounding.

Operating costs assume no contingencies.

### 21.2.1 Mine Operating Costs

This section presents the parameters, basis, and exclusions for the mine operating costs. Where necessary, the costs and performances were developed by SSR to determine probable cost of mine operations at the Pitarrilla Project. Productivities and performance were developed from first principles.

The Pitarrilla Project will use a standard open pit mining method, mining at 214 ktpd at the peak production period.

The pit is designed to be mined in five main phases: Cordon Colorado, Breccia Ridge Phase 1, Breccia Ridge Phase 2, Breccia Ridge Phase 3 and Breccia Ridge Phase 4. The ultimate pit configuration is designed to be approximately 600 m deep, 1.8km in length and 1.2 km in breadth. The individual pit phase depths are presented on Table 21-18. The life of mine stripping ratio is 6:1.

**Table 21-18: Designed Pit Phase Depths**

| Pit Phase | Depth (m) |
|---|---|
| Cordon Colorado | 174 |
| Breccia Ridge 1 | 270 |
| Breccia Ridge 2 | 293 |
| Breccia Ridge 3 | 491 |
| Breccia Ridge 4 | 600 |

The mine will undertake conventional drilling and blasting activities with pre-split to assure stable wall rock conditions.

Primary loading operations will be mostly performed with a maximum of four 21 $m^3$ hydraulic shovels and two 19 $m^3$ capacity bucket front-end loaders. A single 12.3 $m^3$ capacity bucket front-end loader will be deployed at the ROM Pad for crusher feeding and for occasional use with auxiliary 100 tonne trucks when they are not being loaded by excavator. There will be two smaller excavators to load the auxiliary trucks. The larger capacity loaders will sometimes serve for crusher re-handle during maintenance periods on the main re-handle loader and when the auxiliary fleet is in action.

Primary hauling operations of ore and waste will be performed with a maximum of 28 x 150 tonne haul trucks.

Equipment maintenance activities will be performed at an on-site workshop.

A list of the required mining fleet and machinery is presented in Section 16.

Operating costs were compared to independent zero based estimates completed by IMC and found within 2%. Bench marking to other operations in Mexico was completed by SSR for comparative purposes and found satisfactory.

Operating Costs for the Pitarrilla Feasibility Study (M3, 2012) have been estimated by element for each cost area (drilling, blasting, hauling etc.) as follows:

- Diesel consumption
- Lubricants (including hydraulic oil)
- Tires and tubes
- Explosives - ANFO
- Emulsion and high explosives
- Explosive accessories
- Bits, steel and other drill wear parts
- Blast-hole stemming
- Mobile open cut equipment wear parts
- Materials and repairs for mechanical maintenance
- Mine dewatering
- Mine operation charges to plant
- Laboratory charges
- Office supplies
- Supplies for engineering and geology not covered by office supplies
- Safety supplies
- Salaries
- Wages
- Consultants and external services
- Software upgrades and support contracts
- Education/Seminars/Scholarships
- Importation costs
- Freight charges
- Travel cost
- Vehicle expenses
- Distributed Human Resource charges (for all mining personnel)
- Miscellaneous maintenance cost
- Miscellaneous pit cost

For summary purposes, the above cost elements were grouped into seven major elements as labour, labour on costs, diesel, explosives/drill parts/stemming, direct maintenance charges (no labor), mine dewatering and other. A pie chart detailing the cost allocations per each major cost element is shown in Figure 21-1.

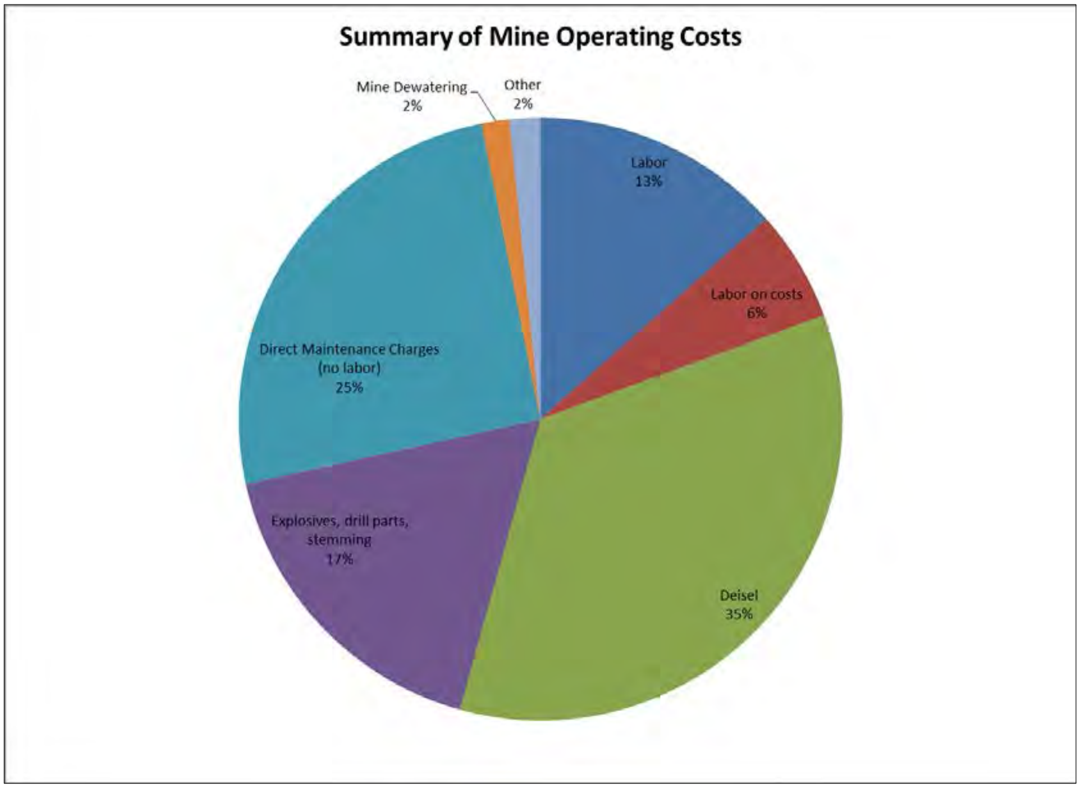



**Figure 21-1: Summary of Mine Operating Cost by Element**
Source: M3, 2012

Diesel consumption was estimated based on equipment operating hours, duty cycle and equipment numbers. Diesel forms approximately 35% of the total mine operating cost. Diesel consumption per operating equipment is summarised in Table 21-19.

Labour requirements for the mining operations were estimated based on equipment type, operating hours, roster arrangements, vacation support, and management requirements for each operational area and technical requirement. A summary of the annual labour requirement per operational area is presented in Table 21-20.

A maximum of 432 mine labour personnel are expected during peak production, with the number declining to 16 towards the end of the mine life. The life of mine average labour requirement is estimated at 220.

## Table 21-19: Diesel Consumption per Operating Equipment

| | Description | | Units | Year -3 | Year -2 | Year -1 | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 | Year 11 | Year 12 | Year 13 | Year 14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | Capital Period in Years | | | | | | | | | | | | | | | | |
| | No.Days | | days | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 |
| Deisel Consumption | | l/ophr | | | | | | | | | | | | | | | | | | |
| | Production Drill rig | 115 | klitres | 40 | 757 | 2790 | 3828 | 4785 | 4785 | 4785 | 4785 | 4785 | 4307 | 3828 | 3828 | 3350 | 3350 | 3350 | 2740 | 2239 |
| | Pre-split Drill Rig | 60 | klitres | 138 | 534 | 774 | 755 | 720 | 545 | 866 | 901 | 806 | 865 | 861 | 810 | 1094 | 1113 | 305 | 152 | 127 |
| | Pit Shovel | 290 | klitres | 0 | 852 | 4591 | 5301 | 5152 | 5242 | 5488 | 5689 | 5188 | 5389 | 5389 | 5786 | 5189 | 4790 | 3709 | 3075 | 2130 |
| | Pit FEL | 150 | klitres | 45 | 1294 | 1573 | 1574 | 1574 | 1574 | 1574 | 1574 | 1574 | 1574 | 1574 | 1574 | 1574 | 1574 | 1574 | 1574 | 1574 |
| | 150t Haul Trucks | 135 | klitres | 193 | 5355 | 14631 | 16638 | 16000 | 16401 | 19286 | 19564 | 20267 | 20201 | 19942 | 20300 | 20602 | 20593 | 17564 | 21338 | 19896 |
| | 100t Haul Trucks | 55 | klitres | 303 | 734 | 744 | 737 | 709 | 726 | 728 | 738 | 442 | 441 | 435 | 443 | 450 | 449 | 447 | 427 | 441 |
| | ROM FEL | 100 | klitres | 212 | 500 | 284 | 284 | 284 | 284 | 284 | 284 | 0 | 0 | 284 | 284 | 284 | 284 | 284 | 284 | 284 |
| | Scaler | 25 | klitres | 0 | 0 | 98 | 118 | 118 | 118 | 118 | 118 | 118 | 118 | 118 | 118 | 118 | 118 | 118 | 118 | 118 |
| | Water Truck | 55 | klitres | 96 | 432 | 527 | 522 | 502 | 514 | 515 | 523 | 522 | 520 | 514 | 523 | 531 | 530 | 528 | 504 | 520 |
| | Dozers | 65 | klitres | 251 | 1534 | 2716 | 2947 | 2899 | 2928 | 3008 | 3074 | 2642 | 2708 | 2976 | 3105 | 2911 | 2781 | 2429 | 2223 | 1916 |
| | Graders | 30 | klitres | 53 | 276 | 428 | 464 | 456 | 461 | 474 | 484 | 416 | 426 | 469 | 489 | 458 | 438 | 383 | 394 | 453 |
| | Operated Ancilary | 15 | klitres | 6 | 49 | 79 | 86 | 85 | 86 | 88 | 90 | 108 | 79 | 87 | 91 | 85 | 81 | 71 | 73 | 84 |
| | Ancilary | 55 | klitres | 0 | 148 | 261 | 284 | 279 | 282 | 289 | 296 | 254 | 261 | 286 | 299 | 280 | 268 | 234 | 214 | 184 |
| | Other | 10% % of total | klitres | 134 | 1246 | 2950 | 3354 | 3356 | 3395 | 3750 | 3812 | 3712 | 3689 | 3676 | 3765 | 3693 | 3637 | 3100 | 3312 | 2997 |
| | **Total Open Cut** | | **klitres** | **1470** | **13710** | **32446** | **36891** | **36920** | **37340** | **41254** | **41931** | **40836** | **40577** | **40438** | **41415** | **40618** | **40007** | **34096** | **36430** | **32964** |

| Description | Year 15 | Year 16 | Year 17 | Year 18 | Year 19 | Year 20 | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 | Year 30 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| No.Days | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 12044.4 |
| Production Drill rig | 2303 | 479 | 299 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 61412 |
| Pre-split Drill Rig | 158 | 182 | 21 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 11727 |
| Pit Shovel | 1067 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 74028 |
| Pit FEL | 1574 | 1137 | 722 | 389 | 365 | 365 | 365 | 487 | 487 | 487 | 487 | 462 | 365 | 365 | 324 | 0 | 33326 |
| 150t Haul Trucks | 16619 | 7479 | 5005 | 3693 | 3617 | 3617 | 3617 | 4013 | 4013 | 1577 | 1577 | 1498 | 1183 | 1183 | 1049 | 0 | 348513 |
| 100t Haul Trucks | 464 | 455 | 300 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 10614 |
| ROM FEL | 284 | 284 | 284 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 5261 |
| Scaler | 118 | 118 | 118 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 0 | 2200 |
| Water Truck | 548 | 537 | 177 | 73 | 73 | 73 | 73 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 9878 |
| Dozers | 1570 | 958 | 706 | 236 | 221 | 221 | 221 | 295 | 295 | 295 | 295 | 281 | 221 | 221 | 0 | 0 | 49087 |
| Graders | 556 | 339 | 250 | 28 | 26 | 26 | 26 | 70 | 139 | 139 | 139 | 133 | 105 | 105 | 45 | 0 | 9149 |
| Operated Ancilary | 103 | 63 | 46 | 16 | 15 | 15 | 15 | 10 | 19 | 19 | 19 | 18 | 15 | 15 | 6 | 0 | 1724 |
| Ancilary | 151 | 92 | 68 | 23 | 21 | 21 | 21 | 28 | 28 | 28 | 28 | 27 | 21 | 21 | 0 | 0 | 4700 |
| Other | 2551 | 1212 | 800 | 447 | 435 | 435 | 435 | 491 | 499 | 255 | 255 | 243 | 192 | 192 | 143 | 0 | 62162 |
| **Total Open Cut** | **28066** | **13336** | **8797** | **4913** | **4782** | **4782** | **4782** | **5403** | **5490** | **2810** | **2810** | **2670** | **2110** | **2110** | **1576** | **0** | **683781** |

## Table 21-20: Annual Labour Requirement

| | Description | Units | Year -3 | Year -2 | Year -1 | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 | Year 11 | Year 12 | Year 13 | Year 14 | Year 15 | Year 16 | Year 17 | Year 18 | Year 19 | Year 20 | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Capital Period in Years | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| | No.Days | days | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 |
| **Total Labor** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| | Drilling | ea | 7 | 19 | 33 | 32 | 37 | 35 | 38 | 39 | 38 | 36 | 33 | 33 | 33 | 33 | 24 | 19 | 14 | 15 | 4 | 2 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Blasting | ea | 3 | 7 | 11 | 11 | 11 | 11 | 12 | 12 | 11 | 12 | 11 | 12 | 11 | 11 | 9 | 7 | 6 | 4 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Mine Loading | ea | 1 | 12 | 18 | 18 | 17 | 17 | 18 | 18 | 17 | 18 | 18 | 19 | 17 | 16 | 14 | 13 | 11 | 9 | 5 | 3 | 2 | 1 | 1 | 1 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 |
| | Mine Haulage | ea | 15 | 54 | 84 | 83 | 80 | 82 | 95 | 97 | 97 | 96 | 95 | 97 | 98 | 98 | 84 | 81 | 73 | 60 | 29 | 19 | 12 | 11 | 11 | 11 | 13 | 13 | 7 | 7 | 7 | 5 | 5 | 5 |
| | Mine Support | ea | 16 | 42 | 55 | 54 | 53 | 54 | 55 | 56 | 49 | 49 | 54 | 57 | 54 | 52 | 47 | 44 | 43 | 43 | 30 | 21 | 5 | 4 | 4 | 4 | 5 | 7 | 7 | 7 | 7 | 5 | 5 | 1 |
| | Open Cut Mine Maintenance | ea | 28 | 92 | 134 | 130 | 130 | 130 | 143 | 145 | 139 | 138 | 139 | 142 | 139 | 138 | 119 | 111 | 101 | 88 | 48 | 30 | 12 | 12 | 12 | 12 | 13 | 14 | 11 | 11 | 11 | 8 | 8 | 5 |
| | Open Cut Mine General and Administration | ea | 18 | 38 | 57 | 54 | 58 | 57 | 63 | 65 | 61 | 59 | 56 | 56 | 58 | 61 | 46 | 43 | 37 | 42 | 17 | 14 | 6 | 6 | 6 | 6 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 4 |
| | **Total** | **ea** | **88** | **264** | **392** | **382** | **386** | **387** | **424** | **432** | **411** | **407** | **406** | **414** | **410** | **410** | **343** | **318** | **285** | **260** | **133** | **89** | **36** | **35** | **35** | **35** | **39** | **41** | **32** | **32** | **31** | **25** | **25** | **16** |

21.2.1.1 Drilling and Blasting

The current mine plan assumes that drilling and blasting will be performed on all material moved. Mining will occur in part on 15 m benches (100% waste zones) and 7.5 m benches (mixed ore and waste zones).

21.2.1.2 Drilling

It has been estimated that approximately 47% of the mined volume will be blasted on 15 m benches while the remaining 53% will be blasted on 7.5 m benches. This analysis is derived from a review of all ore benches for developing practical areas of double bench extraction. Further detailed extraction plans for dual and single benches are required for the detailed engineering process.

The life of mine drilling summary is presented in Table 21-21.

**Table 21-21: Production Drilling Summary by Material Type and Bench Height**

| Description | Units | Double Bench Waste | Mixed Waste | Ore | LOM Total |
|---|---|---|---|---|---|
| Bench Height | m | 15.00 | 7.50 | 7.50 | |
| Burden | m | 7.0 | 4.0 | 4.0 | |
| Spacing | m | 9.0 | 5.0 | 5.0 | |
| Sub drill | m | 1.75 | 1.20 | 1.20 | |
| Volume per Meter* | bcm/m | 53.60 | 16.38 | 16.38 | |
| Meters Drilled | km | 4084 | 11248 | 4024 | 19355 |
| Production Volume | kbcm | 218874 | 184230 | 65911 | 469015 |
| Production Volume | kt | 501420 | 432265 | 156590 | 1090275 |

Notes:
* Assume 95% pattern efficiency

The drill penetration rates selected in the Pitarrilla Feasibility Study (M3, 2012) as well as other key performance indicators and labor requirements, are discussed in Section 16.

The drilling and blasting operations would require 13 drill rigs during the peak of the mine life. This would comprise ten production rigs and three pre-split drill rigs. These estimates include a provision made for equipment rebuilds and replacements and assume 80% utilization of the production rigs and 75% mechanical availability.

Drilling operating unit cost per tonne mined is estimated at $0.15/tonne.

21.2.1.3 Blasting

A description of the blasting operations is presented in Section 16. Blasting costs were estimated using first principles and include, for major parameters, a powder factor of 0.17 kg/t for 7.5 m benches, and 0.13 kg/t for 15 m benches. ANFO was estimated for 95% of shots, with emulsion for overly wet areas estimated for the other 5%. Initiation will be through non-electric delay detonators and lead lines connected to detonator cord and surface line or row delays.

Stemming material will be used to fill between the explosive charge and the collar of each blast-hole to confine the explosive gases. This material will consist of screened mined waste to be loaded by a contractor onto 20 tonne trucks and dumped at specified areas near the shots. The cost of producing the stemming material is estimated at $30/tonne and was developed from first principles.

In addition, the mine plan estimates the following for explosives:

- 30 tonnes per day of Ammonium Nitrate Fuel Oil (“ANFO”), or Slurry + ANFO combined.
- The mine operation will load and shoot between 300 and 350 holes per day.
- The mine will receive 1 or 2 tanker loads of Ammonium Nitrate (“AN”) prills every day.

On the basis of the consumption estimates, a storage capacity of 240 tonnes on site has been estimated with matching ancillary magazines. A basic plan for the explosive storage facility was developed by M3 Engineering. The design consists of two structures for storing explosives and detonators separately and an ANFO storage tower.

Blasting operating unit cost per tonne mined is estimated at $0.25 /tonne.

Support equipment for the drilling and blasting operations will include:

- One blast-hole stemmer (skid steer);
- One blaster flatbed truck; and
- Two ANFO/Slurry trucks.

21.2.1.4 Loading Operations

Primary loading operations have been described in Section 16.10. The loading equipment was selected to match with the capacity of the haul trucks selected for efficient loading cycles.

A maximum of four shovels and two loaders were estimated based on cycle productivities and including their respective average mechanical availabilities of 85% and peak utilizations of 90%.

The loading operating unit cost per tonne mined is estimated at $0.17/tonne.

#### 21.2.1.5 Hauling Operations

Primary hauling operations of ore and waste have been described in Section 16.11.

A haul cost per mining source elevation was generated from the simulation results and used as guidance in the Pitarrilla Feasibility Study (M3, 2012). The final costs for haulage were generated from the productivities assigned to each ore and waste destination by phase within the final schedule spreadsheet.

The estimated number of hauling units in operation per year, including their respective mechanical availabilities and utilizations, is presented in Table 21-22.

The hauling operating unit cost per tonne mined is estimated at $0.45/tonne.

SILVER STANDARD

## Table 21-22: Actual Numbers of Hauling Units in Operation per Year

| | | | Capital Period in Years | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Equipment in Operation | | Units | Year -3 | Year -2 | Year -1 | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 | Year 11 | Year 12 | Year 13 |
| No.Days | | days | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 |
| | 150t Haul Trucks | | 2 | 14 | 23 | 23 | 23 | 23 | 27 | 27 | 28 | 28 | 28 | 28 | 28 | 28 | 24 | 24 |
| | 100t Haul Trucks | | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 |
| Mechanical Availability | | | | | | | | | | | | | | | | | | |
| | 150t Haul Trucks | | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% |
| | 100t Haul Trucks | | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% |
| Utilization | | | | | | | | | | | | | | | | | | |
| | 150t Haul Trucks | | 89% | 88% | 90% | 89% | 85% | 88% | 88% | 89% | 89% | 89% | 87% | 89% | 90% | 90% | 90% | 86% |
| | 100t Haul Trucks | | 89% | 88% | 90% | 89% | 85% | 88% | 88% | 89% | 89% | 89% | 87% | 89% | 90% | 90% | 90% | 86% |

| Equipment in Operation | | Year 14 | Year 15 | Year 16 | Year 17 | Year 18 | Year 19 | Year 20 | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| No.Days | | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 |
| | 150t Haul Trucks | 21 | 16 | 7 | 7 | 7 | 7 | 7 | 7 | 7 | 4 | 4 | 4 | 4 | 4 | 4 | 4 |
| | 100t Haul Trucks | 3 | 3 | 3 | 3 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Mechanical Availability | | | | | | | | | | | | | | | | | |
| | 150t Haul Trucks | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% |
| | 100t Haul Trucks | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% |
| Utilization | | | | | | | | | | | | | | | | | |
| | 150t Haul Trucks | 89% | 93% | 91% | 60% | 46% | 44% | 44% | 44% | 51% | 90% | 48% | 48% | 46% | 36% | 36% | 32% |
| | 100t Haul Trucks | 89% | 93% | 91% | 60% | 46% | 44% | 44% | 44% | 51% | 90% | 48% | 48% | 46% | 36% | 36% | 32% |

#### 21.2.1.6 Mine Support

Mine support functions are described in Section 16.12. The list of mine support equipment comprises pit wall scaling excavators, water trucks, dozers, graders and ancillary equipment (both operated (such as service trucks) and non-operated (such as lighting plants)). Operating costs based on hours of operation and duty cycle were estimated for all equipment. Numbers of equipment were based on empirical relationships to primary equipment such as loaders, lengths of active roads and other parameters. Road watering requirements were developed in detail from first principles.

Similar to the major mine equipment, the ancillary units estimated to be purchased include provision made for equipment rebuilds and replacements. Actual numbers of support equipment in operation per year, including their respective mechanical availabilities and utilizations, is presented in Table 21-23.

The operating unit cost per tonne mined for mine support is estimated at $0.22/tonne.

## Table 21-23: Actual Numbers of Mine Support Units in Operation per Year

| | | | Capital Period in Years | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | **Description** | **Units** | **Year -3** | **Year -2** | **Year -1** | **Year 1** | **Year 2** | **Year 3** | **Year 4** | **Year 5** | **Year 6** | **Year 7** | **Year 8** | **Year 9** | **Year 10** | **Year 11** | **Year 12** |
| | **No.Days** | **days** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** |
| **Equipment in Operation** | | | | | | | | | | | | | | | | | |
| | Scaler | | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| | Water Truck | | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 |
| | Dozers | | 5 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 |
| | Graders | | 2 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 |
| | Operated Ancilary | | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| | Ancilary | | 0 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 2 | 1 | 1 | 1 | 1 | 1 | 1 |
| | | | | | | 1.50 | 1.50 | 1.50 | 1.50 | 1.50 | 1.50 | 1.50 | 1.50 | 1.50 | 1.50 | 1.80 | 1.80 |
| **Mechanical Availability** | | | | | | | | | | | | | | | | | |
| | Scaler | | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% |
| | Water Truck | | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% |
| | Dozers | | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% |
| | Graders | | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% |
| | Operated Ancilary | | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% |
| | Ancilary | | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% |
| **Utilization** | | | | | | | | | | | | | | | | | |
| | Scaler | | 0% | 0% | 70% | 70% | 70% | 70% | 70% | 70% | 70% | 70% | 70% | 70% | 70% | 70% | 70% |
| | Water Truck | | 80% | 88% | 90% | 89% | 85% | 88% | 88% | 89% | 89% | 89% | 87% | 89% | 90% | 90% | 90% |
| | Dozers | | 32% | 51% | 79% | 85% | 84% | 85% | 87% | 89% | 77% | 79% | 86% | 90% | 84% | 81% | 70% |
| | Graders | | 32% | 51% | 79% | 85% | 84% | 85% | 87% | 89% | 77% | 79% | 86% | 90% | 84% | 81% | 70% |
| | Operated Ancilary | | 32% | 51% | 79% | 85% | 84% | 85% | 87% | 89% | 77% | 79% | 86% | 90% | 84% | 81% | 70% |
| | Ancilary | | 0% | 44% | 79% | 85% | 84% | 85% | 87% | 89% | 38% | 79% | 86% | 90% | 84% | 81% | 70% |
| | Total | | | | | 85% | 83% | 83% | 86% | 87% | 83% | 85% | 85% | 87% | 86% | 86% | 78% |

| | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | **Description** | **Units** | **Year 13** | **Year 14** | **Year 15** | **Year 16** | **Year 17** | **Year 18** | **Year 19** | **Year 20** | **Year 21** | **Year 22** | **Year 23** | **Year 24** | **Year 25** | **Year 26** | **Year 27** | **Year 28** | **Year 29** |
| | **No.Days** | **days** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** | **365** |
| **Equipment in Operation** | | | | | | | | | | | | | | | | | | | |
| | Scaler | | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| | Water Truck | | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Dozers | | 8 | 6 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 2 | 1 | 1 | 1 | 1 | 1 | 1 | 0 |
| | Graders | | 3 | 3 | 3 | 3 | 3 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| | Operated Ancilary | | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Ancilary | | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | | | 1.60 | 1.50 | 1.33 | 2.00 | 2.00 | 2.00 | 2.00 | 2.00 | 2.00 | 2.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 1.00 | 0.00 |
| **Mechanical Availability** | | | | | | | | | | | | | | | | | | | |
| | Scaler | | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% |
| | Water Truck | | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% | 77% |
| | Dozers | | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% | 85% |
| | Graders | | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% |
| | Operated Ancilary | | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% | 97% |
| | Ancilary | | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% | 87% |
| **Utilization** | | | | | | | | | | | | | | | | | | | |
| | Scaler | | 70% | 70% | 70% | 70% | 70% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% | 5% |
| | Water Truck | | 86% | 89% | 93% | 91% | 60% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% |
| | Dozers | | 73% | 83% | 83% | 63% | 46% | 15% | 14% | 14% | 14% | 39% | 77% | 77% | 77% | 73% | 58% | 58% | 25% |
| | Graders | | 73% | 83% | 83% | 63% | 46% | 15% | 14% | 14% | 14% | 39% | 77% | 77% | 77% | 73% | 58% | 58% | 25% |
| | Operated Ancilary | | 73% | 83% | 102% | 63% | 46% | 15% | 14% | 14% | 14% | 39% | 77% | 77% | 77% | 73% | 58% | 58% | 25% |
| | Ancilary | | 64% | 56% | 46% | 28% | 20% | 7% | 6% | 6% | 6% | 34% | 34% | 34% | 34% | 33% | 26% | 26% | 0% |
| | Total | | 74% | 79% | 88% | 74% | 50% | 27% | 26% | 26% | 26% | 44% | 71% | 51% | 51% | 49% | 39% | 39% | 27% |

21.2.1.7 Mine Maintenance

Mine maintenance function is described in Section 16.3. For the Pitarrilla Feasibility Study (M3, 2012), mine maintenance costs due to equipment support requirements are distributed on average over the life of mine as drilling (13%), loading (19%), hauling (35%) and mine support (33%).

21.2.1.8 Mine General and Administration

Mine general and administration functions are described in Section 16.14.

The operating unit cost per tonne mined for mine G&A is estimated at $0.20/tonne.

### 21.2.2 Process Plant Operating & Maintenance Costs

The process plants operating costs are summarised by areas of the plant and then by cost elements of labour, power, reagents, grinding media, wear items, maintenance parts and supplies and services. A summary of the sulphide and oxide plants operating costs is shown in Table 21-24 and Table 21-25. Additionally, a summary of the operating costs for leaching sulphide tailings material is shown in Table 21-26.

The sustaining capital for plant equipment replacement is included in this maintenance cost estimate.

### Table 21-24: Sulphide Plant Operating Cost – Life of Mine Operation

| | |
|---|---|
| **Sulphide Operations** | |
| **Cost Item** | LOM $/tonne |
| **Primary Crushing** | |
| Operating Labor and Fringes | 0.05 |
| Power | 0.10 |
| Liners | 0.02 |
| Maintenance | 0.12 |
| Supplies & Services | 0.03 |
| Subtotal Primary Crushing | 0.32 |
| **Grinding** | |
| Operating Labor and Fringes | 0.05 |
| Power | 2.07 |
| Grinding Media | 2.32 |
| Liners | 0.41 |
| Maintenance | 0.28 |
| Supplies and Services | 0.07 |
| Subtotal Grinding | 5.20 |
| **Flotation** | |
| Operating Labor and Fringes | 0.03 |
| Power | 0.58 |
| Reagents | 1.84 |
| Grinding Media | 0.05 |
| Grinding Media | 0.01 |
| Maintenance | 0.21 |
| Supplies and Services | 0.03 |
| Subtotal Flotation | 2.75 |
| **Concentrate Thickening,Filtration** | |
| Operating Labor and Fringes | 0.04 |
| Power | 0.13 |
| Reagents | 0.08 |
| Maintenance | 0.04 |
| Supplies and Services | 0.08 |
| Subtotal Concentrate Thickening, Filtration | 0.37 |
| **Tailings** | |
| Operating Labor and Fringes | 0.03 |
| Power | 0.13 |
| Reagents | 0.12 |
| Maintenance | 0.07 |
| Supplies and Services | 0.01 |
| Subtotal Tailings | 0.36 |
| **Ancillary** | |
| Operating Labor and Fringes | 0.12 |
| Power | 0.36 |
| Reagents | 0.03 |
| Maintenance | 0.07 |
| Water Charges | 0.16 |
| Supplies and Services | 0.08 |
| Subtotal Ancillary Services | 0.81 |
| **Total Process Plant** | **9.81** |

## Table 21-25: Oxide Plant Operating Cost – Life of Mine Operation

| **Oxide Operations** | |
|---|---|
| **Cost Item** | **LOM $/Tonne** |
| **Primary Crushing** | |
| Operating Labor and Fringes | 0.06 |
| Power | 0.11 |
| Liners | 0.02 |
| Maintenance | 0.13 |
| Supplies & Services | 0.04 |
| Subtotal Primary Crushing | 0.37 |
| | |
| **Grinding** | |
| Operating Labor and Fringes | 0.06 |
| Power | 2.74 |
| Grinding Media | 2.60 |
| Liners | 0.47 |
| Maintenance | 0.31 |
| Supplies and Services | 0.10 |
| Subtotal Grinding | 6.28 |
| | |
| **Leach/CCD** | |
| Operating Labor and Fringes | 0.04 |
| Power | 0.62 |
| Reagents | 3.92 |
| Maintenance | 0.49 |
| Supplies and Services | 0.05 |
| Subtotal Leach/CCD | 5.13 |
| | |
| **Merrill Crowe/Refinery** | |
| Operating Labor and Fringes | 0.05 |
| Power | 0.22 |
| Reagents | 0.32 |
| Maintenance | 0.14 |
| Supplies and Services | 0.07 |
| Subtotal Merrill Crowe | 0.79 |
| | |
| **Tailings** | |
| Operating Labor and Fringes | 0.04 |
| Power | 0.25 |
| Maintenance | 0.08 |
| Supplies and Services | 0.01 |
| Subtotal Tailings | 0.37 |
| | |
| **Ancillary** | |
| Operating Labor and Fringes | 0.15 |
| Power | 0.35 |
| Maintenance | 0.11 |
| Water Charges | 0.26 |
| Supplies and Services | 0.10 |
| Subtotal Ancillary Services | 0.97 |
| **Total Process Plant** | **13.90** |

**Table 21-26: Sulphide Tailings Leach Operating Cost – Life of Mine Operation**

| Sulphide Tailings Leach Operations | |
|---|---|
| **Cost Item** | **LOM $/Tonne** |
| **Leach/CCD** | |
| Operating Labor and Fringes | 0.04 |
| Power | 0.38 |
| Reagents | 3.19 |
| Maintenance | 0.29 |
| Supplies and Services | 0.04 |
| Subtotal Leach/CCD | 3.93 |
| **Merrill Crowe/Refinery** | |
| Operating Labor and Fringes | 0.05 |
| Power | 0.21 |
| Reagents | 0.22 |
| Maintenance | 0.05 |
| Supplies and Services | 0.05 |
| Subtotal Merrill Crowe | 0.58 |
| **Tailings** | |
| Operating Labor and Fringes | 0.00 |
| Power | 0.12 |
| Maintenance | 0.04 |
| Supplies and Services | 0.02 |
| Subtotal Tailings | 0.17 |
| **Ancillary** | |
| Operating Labor and Fringes | 0.00 |
| Power | 0.03 |
| Maintenance | 0.03 |
| Water Charges | 0.00 |
| Supplies and Services | 0.12 |
| Subtotal Ancillary Services | 0.17 |
| **Total Process Plant** | **4.85** |
| **G&A** | 0.05 |

Project operating costs were derived from an estimate completed by M3 that was based on a preliminary version of the mine plan. The SSR estimate was utilised for the Financial Model. Adjustments were made for increased ore grade (and expanded sulphide plant) in later years, camp costs and some mine operating cost charges to the plant. Table 21-27 compares the M3 and SSR operating cost estimates. M3 has reviewed the logic behind the SSR adjustments and agrees with the changes.

**Table 21-27: Life of Mine Operating Cost Summary Comparison**

| Description | Unit | M3's LOM Estimate | SSR's Adjusted LOM Estimate | Variance | Explanation of Variance | Added Cost for Camp and Mining Operations | Total SSR Estimate |
|---|---|---|---|---|---|---|---|
| Direct Leach (Oxide Process Plant) Operations | $/t | 13.90 | 13.98 | 0.08 | Negligible change | 0.52 | 14.50 |
| Flotation (Sulphide Process Plant Operations) | $/t | 9.81 | 10.40 | 0.59 | Increased operating costs due to high head grades in Years 9-21 | 0.52 | 10.91 |
| Sulphide Tailings Leach Operations | $/t | 4.85 | 4.83 | -0.02 | Negligible change | 0 | 4.83 |

21.2.2.1 Process Labour & Fringes

Process labour costs were derived from a staffing plan and based on prevailing annual labour rates in the area. Labour rates and fringe benefits for employees include all applicable social security benefits. A summary of the annual staffing requirement and gross annual labour cost is shown in Table 21-28.

**Table 21-28: Process Plant Annual Staffing Requirement and Labour Costs**

| Department | Number of Personnel | Total Annual Labour Cost (K$) |
|---|---|---|
| Mill Operations & Laboratory | 77 | 1,744 |
| Mill Maintenance | 49 | 905 |
| **Total** | **126** | **2,649** |

21.2.2.2 Power

Power consumption was based on the connected power load (kW) derived from the equipment list, discounted for operating time per day, and anticipated operating load level. The overall power rate is estimated at $0.1075 per kWh with a Life of Mine consumption for the sulphide operation of approximately 31.3 kWh per ore tonne and for the oxide operation is approximately 39.9 kWh per ore tonne. A summary of the power consumption and cost are shown in Table 21-29.

**Table 21-29: Summary of Process Area Electric Power Use – kWh/t**

| Sulphide Plant | kWhr/t |
|---|---|
| Primary Crushing | 0.9 |
| Grinding | 19.3 |
| Flotation | 5.4 |
| Concentrate Thickening, Filtration | 1.2 |
| Tailings | 1.2 |
| Ancillary | 3.3 |
| **Total Sulphide Plant** | **31.3** |

| Oxide Plant | kWhr/t |
|---|---|
| Primary Crushing | 1 |
| Grinding | 25.5 |
| Leach/CCD | 5.8 |
| Merrill Crowe/Refinery | 2 |
| Tailings | 2.3 |
| Ancillary | 3.3 |
| **Total Oxide Plant** | **39.9** |

21.2.2.3 Reagents

Consumption rates were determined from the metallurgical test data or industry practice. Budget quotations were received for reagents supplied from local sources, where available, with an allowance for freight to site. A summary of process reagent consumption and costs are included in Table 21-30 and Table 21-31 for sulphide and oxide, respectively.

**Table 21-30: Summary of Reagents – Sulphide Consumption Rates and Unit Prices**

| Reagent | Consumption kg/t ore | Unit Rate $/kg |
|---|---|---|
| Lime | 0.78 | 0.16 |
| 3418A Promoter | 0.03 | 10.74 |
| Dow Froth-1012 | 0.05 | 5.10 |
| SIPX | 0.03 | 2.63 |
| MIBC | 0.08 | 4.07 |
| Copper Sulphate | 0.10 | 4.18 |
| Sodium Cyanide | 0.03 | 2.29 |
| Flocculant - Conc. Thickening | 0.02 | 3.85 |
| Flocculant - Tailings | 0.03 | 3.85 |
| Antiscalant | 0.01 | 3.15 |

**Table 21-31: Summary of Reagents – Oxide Consumption Rates and Unit Prices**

| Reagent | Consumption kg/t ore | Unit Rate $/kg |
|---|---|---|
| Lime | 3.00 | 0.16 |
| Sodium Cyanide | 1.00 | 2.29 |
| Zinc Dust | 0.04 | 2.80 |
| Diatomaceous Earth | 0.10 | 1.10 |
| Copper Sulphate | 0.09 | 4.18 |
| Sulphur | 0.57 | 0.20 |
| Flux | 0.08 | 1.25 |
| Flocculant | 0.18 | 3.85 |

21.2.2.4 Maintenance Wear Parts and Consumables

Grinding media consumption and wear items (liners) were based on industry practice for the crusher and grinding operations. These consumption rates and unit prices are shown in Table 21-32.

**Table 21-32: Grinding Media and Wear Parts – Typical Year of Operation**

| Grinding Media & Wear Parts | Sulphide Plant | | | Oxide Plant | |
|---|---|---|---|---|---|
| | Consumption kg/t ore | Unit Rate $/kg | | Consumption kg/t ore | Unit Rate $/kg |
| Primary Crusher - Liners | 0.008 | 2.15 | | 0.011 | 2.15 |
| SAG Mill - Liners | 0.060 | 2.75 | | 0.050 | 2.75 |
| Ball Mill - Liners | 0.080 | 3.00 | | 0.110 | 3.00 |
| Regrind Mill - Liners | 0.002 | 2.75 | | 0.000 | 0.00 |
| SAG Mill - Balls | 0.760 | 1.35 | | 0.650 | 1.35 |
| Ball Mill - Balls | 1.060 | 1.22 | | 1.410 | 1.22 |
| Regrind - Balls | 0.030 | 1.69 | | 0.000 | 0.00 |

21.2.2.5 Maintenance Allowance

Allowances were made to cover the cost of maintenance of all items that were not specifically identified and to cover the cost of maintenance of the facilities. The allowance was calculated using the direct capital cost of equipment multiplied by 5% for each area. In addition, these costs were modified due to the tonnage being processed for each of the plants.

21.2.2.6 Process Supplies and Services

Allowances were provided in the process plant operating costs for outside consultants, outside contractors, vehicle maintenance, and miscellaneous supplies. The allowances were estimated using M3 Engineering's information from other operations and projects. The estimated unit cost per tonne is $0.30 for the sulphide operation and $0.36 for the oxide operation.

21.2.2.7 Plant General and Administration

The operating cost for the G&A areas were determined and summarised by cost element. The cost elements include labour, supplies, support infrastructure, services, and other expenses.

### 21.2.3 General and Administration

SSR have estimated the G&A costs including labour and fringe benefits for the administrative personnel, the human resources department and health, safety and environmental employees. Total staffing is estimated as 81 employees. Also included are operations camp costs, land payments, office supplies, communications, insurance, legal and auditing fees, consultants, employee cost and other expenses in the administrative area. The life of mine G&A costs are shown in Table 21-33.

The cost for camping and transport of workers in the mine and plant is back-charged to those respective areas (titled as Human Resources Charges). This is therefore presented as a negative cost to administration in the cost element as shown in Table 21-34.

## Table 21-33: General & Administration Cost – Life of Mine Cost

| | | | |
|---|---|---|---|
| **Total Admin Costs** | | | |
| | 901 Administration | $ | 184008 |
| | 908 Community Relations | $ | 12292 |
| | 914 Health, Safety and Security | $ | 22512 |
| | 922 Environmental & Permits | $ | 15091 |
| | 928 Human Resources | $ | 48204 |
| | Total | $ | **282108** |
| **Per t milled** | | | |
| | 901 Administration | $/t milled | 1.18 |
| | 908 Community Relations | $/t milled | 0.08 |
| | 914 Health, Safety and Security | $/t milled | 0.14 |
| | 922 Environmental & Permits | $/t milled | 0.10 |
| | 928 Human Resources | $/t milled | 0.31 |
| | Total | $/t milled | **1.80** |

## Table 21-34: General & Administration Cost – Life of Mine Costs by Element

| Costs by element | K$ | $/t milled |
|---|---|---|
| Wages | 28333 | 0.18 |
| Salaries | 65856 | 0.42 |
| Bus Transportation | 24585 | 0.16 |
| Workers Meals | 78672 | 0.50 |
| Recruitment and relocation cossts | 1713 | 0.01 |
| Education/Seminars/Scholarships | 9801 | 0.06 |
| Group Life Insurance | 24585 | 0.16 |
| Medical Expenses | 1574 | 0.01 |
| Employee Benefits | 4720 | 0.03 |
| Severance Expense | 3582 | 0.02 |
| Diesel | 5075 | 0.03 |
| Propane | 1967 | 0.01 |
| Operating supplies | 1151 | 0.01 |
| Safety supplies | 2638 | 0.02 |
| Office Supplies | 2302 | 0.01 |
| Distributed power costs | 5107 | 0.03 |
| Telephone and internet costs | 8976 | 0.06 |
| Costs associated to the external office | 575 | 0.00 |
| Consultants and external services | 13843 | 0.09 |
| Legal Fees | 8633 | 0.06 |
| Legal Expenses | 959 | 0.01 |
| Auditing and Tax Advice Services | 4891 | 0.03 |
| Air Charter | 1440 | 0.01 |
| Lease Cost Expense, Land rentals | 39292 | 0.25 |
| Technical service fees within country | 15346 | 0.10 |
| Techncial service fees from corporate head Office | 21120 | 0.13 |
| External road micelaneous charges | 2493 | 0.02 |
| Reclamation expense | 767 | 0.00 |
| Bank charges and fees | 192 | 0.00 |
| Public Relations | 3836 | 0.02 |
| Rent of houses | 1496 | 0.01 |
| Personnel events | 4656 | 0.03 |
| Travel Costs | 2391 | 0.02 |
| Vehicle Expenses | 2911 | 0.02 |
| Software upgrades and support contracts | 4824 | 0.03 |
| Hardware support contracts, parts and supplies | 1381 | 0.01 |
| Insurances, all types | 21110 | 0.13 |
| Human Resources Charges | -140685 | -0.90 |
| **Total** | **282108** | **1.80** |

# 22 ECONOMIC ANALYSIS

## 22.1 INTRODUCTION

The results of the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements in this section include, but are not limited to, statements with respect to the future price of silver, and base metals, the estimation of Mineral Reserves and Mineral Resources, the realization of Mineral Reserve estimates, the timing and amount of estimated future production, costs of production, capital expenditures, results of the permitting process, currency exchange rate fluctuations, requirements for additional capital, government regulation of mining operations and taxation, environmental risks, unanticipated reclamation expenses, title disputes or claims and limitations on insurance coverage.

Additional risk can come from actual results of changes in Project parameters as plans continue to be refined, possible variations in Mineral Reserves, grade or recovery rates, failure of plant, equipment or processes to operate as anticipated, accidents, labour disputes and other risks of the mining industry, and potential delays in obtaining additional governmental approvals.

The financial evaluation presents the determination of the key economic performance indicators for the Pitarrilla Project, including the Net Present Value ("NPV"), the payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return ("IRR") for the project. Annual cash flow projections are estimated over the life of the mine based on the estimates of capital expenditures, production costs and sales revenues. The sales revenue is based on the production of a zinc concentrate, a lead concentrate and silver doré. The lead and zinc concentrates contain a significant amount of silver. Silver accounts for 84% of the total net revenue. The plant recovers a total of 333,370 koz of silver, 582 Mlbs of lead, and 1,669 Mlbs of zinc over a 30 year process life (Table 22-3).

The estimates of initial and sustaining capital expenditures and site production costs have been developed specifically for this project and were presented in earlier sections of this report. Total initial capital totals $740.6 million and sustaining capital totals $403.9 million (see the report section Capital and Operating Costs for details).

The Pitarrilla Project has cash costs of $10.01/oz payable silver and production costs of $15.91/oz payable silver (Table 22-5). The NPV at assumed long term metal prices using a 5% discount rate is $737 million and the internal rate of return is 12.8% (Table 22-7). Payback of the initial capital occurs 7.4 years after commercial production commences (Table 22-7).

The economic analysis and supporting financial information, including capital and operating costs, have been developed in constant dollar terms.

The economic analysis uses the Probable Mineral Reserves as described in the Mineral Reserve Estimate of this report. Cash flow forecasts on an annual basis using the Mineral Reserves for the base case metal prices (Table 22-4) are included in Table 22-9.

## 22.2 MINE PRODUCTION STATISTICS

Mine production is reported as ore from the mining operation delivered to run-of-mine and long term stockpiles. The annual production figures were obtained from the mine plan as reported earlier in Section 16 - Mining Methods.

A simplified summary of mine production is shown in Table 22-1. The simplified table summarizes groups of years and presents the average values encountered per annum in those blocks. The blocks of time account for the construction period, the early mine production, late mine production and finally the stockpile re-handle period. The table only displays material mined, it does not display material re-handled from stockpile. The mined material is either placed on the stockpile or fed directly to the plant (note: this difference is not shown on Table 22-1, a more detailed yearly mine summary is supplied in Table 22-9).

### Table 22-1: Ore Mined Statistics

| Mining Statistics | | Construction (Yrs -3 to -1) Yearly Average | Years 1-9: Yearly Average | Years 10-18: Yearly Average | Years 19-30: Yearly Average | LOM Annual Yearly Average | TOTAL Including Construction (Yrs -3 to -1) | TOTAL Excluding Construction (Yrs -3 to -1) |
|---|---|---|---|---|---|---|---|---|
| **Direct Leach Ore (Oxide)** | | | | | | | | |
| Tonnes Mined | kt | 617 | 3,755 | 857 | 0 | 1,314 | 43,356 | 41,504 |
| Contained Metals Mined | | | | | | | | |
| Contained Silver | Koz | 1,995 | 11,244 | 2,258 | 0 | 3,864 | 127,510 | 121,525 |
| Contained Zinc | Klbs | 1,070 | 34,175 | 10,341 | 0 | 12,238 | 403,851 | 400,641 |
| Contained Lead | Klbs | 2,842 | 14,741 | 2,271 | 0 | 4,898 | 161,630 | 153,103 |
| Ore Head Grade | | | | | | | | |
| Silver Grade | g/t | 100.5 | 93.2 | 82.0 | 0.0 | 91.5 | 91.5 | 91.1 |
| Zinc Grade | % | 0.08% | 0.41% | 0.55% | 0.00% | 0.42% | 0.42% | 0.44% |
| Lead Grade | % | 0.21% | 0.18% | 0.12% | 0.00% | 0.17% | 0.17% | 0.17% |
| **Flotation / Leach Ore (Transitional /Sulphide)** | | | | | | | | |
| Tonnes Mined | Kt | 1,683 | 6,671 | 5,350 | 0 | 3,431 | 113,234 | 108,186 |
| Contained Metals Mined | | | | | | | | |
| Contained Silver | Koz | 6,101 | 19,970 | 17,014 | 0 | 10,641 | 351,157 | 332,855 |
| Contained Zinc | Klbs | 4,970 | 81,959 | 173,902 | 0 | 70,232 | 2,317,656 | 2,302,745 |
| Contained Lead | Klbs | 4,116 | 35,003 | 58,274 | 0 | 25,813 | 851,845 | 839,496 |
| Ore Head Grade | | | | | | | | |
| Silver Grade | g/t | 112.8 | 93.1 | 98.9 | 0.0 | 96.5 | 96.5 | 95.7 |
| Zinc Grade | % | 0.13% | 0.56% | 1.47% | 0.00% | 0.93% | 0.93% | 0.97% |
| Lead Grade | % | 0.11% | 0.24% | 0.49% | 0.00% | 0.34% | 0.34% | 0.35% |
| **Total Contained Metals Mined** | | | | | | | | |
| Tonnes Mined | Kt | 2,300 | 10,425 | 6,207 | 0 | 4,745 | 156,590 | 149,690 |
| Contained Metals Mined | | | | | | | | |
| Contained Silver | Koz | 8,096 | 31,214 | 19,272 | 0 | 14,505 | 478,667 | 454,380 |
| Contained Zinc | Klbs | 6,040 | 116,134 | 184,243 | 0.00% | 82,470 | 2,721,507 | 2,703,386 |
| Contained Lead | Klbs | 6,959 | 49,744 | 60,545 | 0.00% | 30,711 | 1,013,474 | 992,599 |
| Ore Head Grade | | | | | | | | |
| Silver Grade | g/t | 109.5 | 93.1 | 96.6 | 0.0 | 95.1 | 95.1 | 94.4 |
| Zinc Grade | % | 0.12% | 0.51% | 1.35% | 0.00% | 0.79% | 0.79% | 0.82% |
| Lead Grade | % | 0.14% | 0.22% | 0.44% | 0.00% | 0.29% | 0.29% | 0.30% |
| **Material Mined** | | | | | | | | |
| Ore Mined | Kt | 2,300 | 10,425 | 6,207 | 0 | 4,745 | 156,590 | 149,690 |
| Waste Mined | Kt | 30,860 | 64,206 | 29,250 | 0 | 28,293 | 933,685 | 841,105 |
| Total Material Mined | Kt | 33,160 | 74,631 | 35,457 | 0 | 33,039 | 1,090,275 | 990,795 |
| Strip Ratio | Ratio | 13.4 | 6.2 | 4.7 | 0.0 | 6.0 | 6.0 | 5.6 |

The ore quantities and ore grades delivered to the mineral processing plant are presented in Table 22-2 over various time periods as well as the life of mine. More detailed yearly mineral processing figures are supplied in Table 22-9.

**Table 22-2: Ore Milled Statistics**

| Processing Statistics | | Construction (Yrs -3 to -1) Yearly Average | Years 1-9: Yearly Average | Years 10-18: Yearly Average | Years 19-30: Yearly Average | LOM Annual Yearly Average | TOTAL Including Construction (Yrs -3 to -1) | TOTAL Excluding Construction (Yrs -3 to -1) |
|---|---|---|---|---|---|---|---|---|
| **Direct Leach Ore (Oxide)** | | | | | | | | |
| Tonnes Feed | Kt | 499 | 1,046 | 657 | 2,211 | 1,314 | 43,356 | 41,860 |
| Contained Metals Processed | | | | | | | | |
| Contained Silver | Koz | 1,774 | 4,818 | 2,170 | 4,941 | 3,864 | 127,510 | 122,188 |
| Contained Zinc | Klbs | 839 | 9,251 | 7,879 | 20,597 | 12,238 | 403,851 | 401,333 |
| Contained Lead | Klbs | 2,204 | 4,822 | 2,246 | 7,617 | 4,898 | 161,630 | 155,019 |
| Ore Head Grade | | | | | | | | |
| Silver Grade | g/t | 110.7 | 143.2 | 102.7 | 69.5 | 91.5 | 91.5 | 90.8 |
| Zinc Grade | % | 0.08% | 0.40% | 0.54% | 0.42% | 0.42% | 0.42% | 0.43% |
| Lead Grade | % | 0.20% | 0.21% | 0.16% | 0.16% | 0.17% | 0.17% | 0.17% |
| **Flotation / Leach Ore (Transitional /Sulphide)** | | | | | | | | |
| Tonnes Feed | Kt | 191 | 4,425 | 4,964 | 2,347 | 3,431 | 113,234 | 112,662 |
| Contained Metals Processed | | | | | | | | |
| Contained Silver | Koz | 1,221 | 16,321 | 16,900 | 4,042 | 10,641 | 351,157 | 347,495 |
| Contained Zinc | Klbs | 611 | 55,606 | 166,177 | 26,648 | 70,232 | 2,317,656 | 2,315,822 |
| Contained Lead | Klbs | 416 | 25,897 | 55,767 | 9,635 | 25,813 | 851,845 | 850,595 |
| Ore Head Grade | | | | | | | | |
| Silver Grade | g/t | 199.3 | 114.7 | 105.9 | 53.6 | 96.5 | 96.5 | 95.9 |
| Zinc Grade | % | 0.15% | 0.57% | 1.52% | 0.52% | 0.93% | 0.93% | 0.93% |
| Lead Grade | % | 0.10% | 0.27% | 0.51% | 0.19% | 0.34% | 0.34% | 0.34% |
| **Total Contained Metals Processed** | | | | | | | | |
| Tonnes Feed | Kt | 689 | 5,471 | 5,621 | 4,558 | 4,745 | 156,590 | 154,522 |
| Contained Metals Processed | | | | | | | | |
| Contained Silver | Koz | 2,995 | 21,139 | 19,070 | 8,983 | 14,505 | 478,667 | 469,683 |
| Contained Zinc | Klbs | 1,451 | 64,856 | 174,056 | 47,245 | 82,470 | 2,721,507 | 2,717,155 |
| Contained Lead | Klbs | 2,620 | 30,719 | 58,013 | 17,252 | 30,711 | 1,013,474 | 1,005,614 |
| Ore Head Grade | | | | | | | | |
| Silver Grade | g/t | 135.2 | 120.2 | 105.5 | 61.3 | 95.1 | 95.1 | 94.5 |
| Zinc Grade | % | 0.10% | 0.54% | 1.40% | 0.47% | 0.79% | 0.79% | 0.80% |
| Lead Grade | % | 0.17% | 0.25% | 0.47% | 0.17% | 0.29% | 0.29% | 0.30% |

## 22.3 PROCESS PLANT PRODUCTION STATISTICS

The design basis for the process plant is 16,000 tpd after mill availability (5,840 Mt per annum) for flotation/leach ore and 12,000 tpd (4,380 Mt per annum) for direct leach ore. The metal recoveries are projected to average 69.6% for silver, 61.3% for zinc and 57.4% for lead during the process plant life.

The metal production from the process plant is presented in Table 22-3 over various time periods including the mine life.

### Table 22-3: Metal Production from Processing Operations

| Processing Statistics Contained Metals - Process Plant Production | | Construction (Yrs -3 to -1) Yearly Average | Years 1-9: Yearly Average | Years 10-18: Yearly Average | Years 19-30: Yearly Average | LOM Annual Yearly Average | Total Including Construction (Yrs -3 to -1) | Total Excluding Construction (Yrs -3 to -1) |
|---|---|---|---|---|---|---|---|---|
| **Lead Concentrate** | | | | | | | | |
| Tonnage | Kt | 0 | 17 | 42 | 6 | 18 | 604 | 603 |
| Contained Metals | | | | | | | | |
| Silver | koz | 340 | 7,336 | 11,489 | 1,250 | 5,619 | 185,440 | 184,420 |
| Zinc | Kt | 0.0 | 1.3 | 5.4 | 0.5 | 2.0 | 66.3 | 66.3 |
| Lead | Kt | 0.1 | 6.8 | 19.7 | 2.0 | 7.9 | 262.1 | 261.9 |
| Concentrate Grade | | | | | | | | |
| Silver Grade | g/t | 56,136 | 13,572 | 8,511 | 6,312 | 9,555 | 9,555 | 9,511 |
| Zinc Grade | % | 4.34% | 7.67% | 12.80% | 8.61% | 10.99% | 10.99% | 11.00% |
| Lead Grade | % | 28.62% | 40.26% | 46.86% | 32.41% | 43.42% | 43.42% | 43.43% |
| **Zinc Concentrate** | | | | | | | | |
| Tonnage | Kt | 0.3 | 30.4 | 118.6 | 14.4 | 45.9 | 1,515 | 1,514 |
| Contained Metals | | | | | | | | |
| Silver | koz | 53 | 1,095 | 1,776 | 285 | 892 | 29,420 | 29,262 |
| Zinc | Kt | 0.1 | 12.5 | 56.7 | 5.6 | 20.9 | 690.6 | 690.4 |
| Lead | Kt | 0.0 | 0.1 | 0.1 | 0.0 | 0.1 | 1.8 | 1.8 |
| Concentrate Grade | | | | | | | | |
| Silver Grade | g/t | 5,958 | 1,120 | 466 | 617 | 604 | 604 | 601 |
| Zinc Grade | % | 28.11% | 41.05% | 47.83% | 39.09% | 45.60% | 45.60% | 45.61% |
| Lead Grade | % | 0.00% | 0.18% | 0.10% | 0.16% | 0.12% | 0.12% | 0.12% |
| **Doré** | | | | | | | | |
| Contained Metals | | | | | | | | |
| Silver | koz | 1,271 | 5,659 | 2,587 | 3,374 | 3,591 | 118,510 | 114,696 |
| **Total Processed Products** | | | | | | | | |
| Contained Metals | | | | | | | | |
| Silver | koz | 1,664 | 14,090 | 15,852 | 4,909 | 10,102 | 333,370 | 328,378 |
| Zinc | Kt | 0.1 | 13.8 | 62.1 | 6.1 | 22.9 | 757.0 | 756.7 |
| Lead | Kt | 0.1 | 6.8 | 19.8 | 2.0 | 8.0 | 263.9 | 263.8 |

## 22.4 SMELTER AND REFINERY RETURN FACTORS

The lead and zinc concentrates will be shipped from the site to a smelting company. Smelter treatment charges and refining charges will be negotiated as part of finalizing sales agreements.

The smelter charges used in the financial evaluation are presented in the Marketing and Contracts section of this report.

## 22.5 CAPITAL EXPENDITURES

### 22.5.1 Initial Capital Cost

The total initial capital for construction and pre-production mine development is described in detail in Capital and Operating Costs section of this report and totals $740.6 million, expended over a three-year period. The initial capital includes all pre-production capital expenditures for design, procurement and construction of project facilities, including owner's costs and contingency. Revenues realised during the construction period have been included as a reduction to capital costs. Declaration of commercial production is defined as at the end of the third consecutive month of 80% of nameplate or greater mill tonnage throughput. The financial model includes a $42.6 million transfer to inventory of capitalised operating expenditure at declaration of commercial production.

### 22.5.2 Sustaining Capital Costs

A schedule of capital cost expenditures during the production period was estimated and included in the financial analysis under the category of sustaining capital. These costs include the planned post start-up capital expenditures as described in Section 21. The total life of mine sustaining capital is estimated to be $403.9 million.

## 22.6 WORKING CAPITAL

All working capital is recaptured at the end of the mine life and the final value of these accounts is reduced to zero value. During the mine life, working capital is comprised of the following components:

### Current Assets

Trade Receivables:

- 90% of receipts of concentrate sales are assumed to be received at 30 days after revenue recognition, and the remaining 10% at 150 days. 100% of receipts on doré sales are assumed to be received 15 days after revenue recognition.

VAT Receivables:

- VAT Receivables are equal to payments on 30 days of VAT liabilities, calculated at the current rate of 16% of qualifying expenditures. Qualifying expenditures are assumed to be non-labour expenditures that are not imported.

Stockpile Inventories:

- Stockpiles are valued using an average cost method. Current stockpile quantities at each year-end are assumed to be the lesser of the actual stockpiles available and the estimated ore quantities processed in the following year. Stockpiled ore that is not anticipated to be processed within one year is classified as non-current.

Supplies Inventory:

- It was assumed that after the initial warehouse supply build during construction that this stock would remain constant throughout the production life.

Finished Goods Inventory:

- Concentrate inventory at year-end equals thirty days of annual concentrate production. Dore inventory at year-end equals seven days of annual production.

Income Tax Installments Prepaid:

- 100% of the year's estimated tax liability is prepaid by installment at each year-end. The year's tax liability is assumed to equal the prior year's income tax expense.

**Current Liabilities**

Trade and Other Accounts Payable:

- Trade payables are equal to 30 days of annual non-labour expenditures.
- Bonus Payable is equal to the estimated bonus expense for the current year.

The year-to-year working capital impacts are shown in the cash flow model in Table 22-9.

## 22.7 IVA TAXES

The VAT tax (in Spanish, the "Impuesto al Valor Agregado" ("IVA")) is assumed to be paid on domestic non-labour purchases. VAT payments are assumed to be recoverable within 30 days (note that the recoverable amount is reflected as a receivable in the working capital calculation).

## 22.8 REVENUE

Annual revenue is determined by applying estimated metal prices to the estimated annual payable metal for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The revenue is the net value of payable metals sold after treatment, refining and selling charges.

To determine the metal price assumptions underlying the revenue calculations, a review of consensus forecast information was undertaken relying principally on a survey completed by a leading Canadian financial firm, dated November 2012, which compiled commodity price forecasts from approximately thirty geographically diverse bank and financial institutions. The consensus forecast, provides annual prices for the period from 2012 through 2016 as well as long-term. Consistent with the financial modeling approach, the consensus forecasts and metal price assumptions utilised are in constant dollars.

Table 22-4 provides a tabular summary of the base case and three other metal price scenarios that were utilised in the economic analysis. All four metal price scenarios are based on the consensus forecasts. The downside and upside metal price assumptions were determined by taking the lower and upper ranges of the consensus forecasts excluding certain outliers. The spot price case is equal to the closing market price for each metal as of November 23, 2012.

**Table 22-4: Metal Price Scenarios for Sensitivity Evaluation**

| | | Downside Case | Base Case* | Upside Case | Spot Price Case |
|---|---|---|---|---|---|
| Silver Price | $/oz | $22.50 | $25.00 | $30.00 | $34.13 |
| Lead Price | $/lb | $0.80 | $0.90 | $1.10 | $0.99 |
| Zinc Price | $/lb | $0.85 | $0.95 | $1.10 | $0.87 |

* The base case silver price is assumed to $30 per oz in Yr -3 to Yr -2, $27.50 per oz in Yr-1 to Yr 2 and $25 per oz thereafter.

Sensitivity analyses were calculated using the four cases for metal prices outlined in Table 22-4. The project cash flow is based on the base case and annual summaries of the cash flow are included in Table 22-9.

## 22.9 EXCHANGE RATE

Currency used in this report is in United States dollars. The Mexican peso (“MXN”) to United States dollar (“USD”) forecast exchange rate was determined based on publicly published forecasts by leading US and Canadian financial institutions. For time periods beyond the forecasts available, a purchase power parity assumption was utilised whereby the MXN devalued in nominal terms at the inflation rate differential between a Mexican long term inflation assumption of 3.8% and an US long term inflation assumption of 1.5%. In real terms, utilizing purchase power parity the MXN is constant. Long term inflation assumptions were determined based on publicly published forecasts by leading US and Canadian financial institutions.

Within the financial model, exchange rate was assumed to equal 12.86 $MXN per $USD for Yr-3, 12.58 $MXN per $USD for Yr-2 and 12.5 $MXN per $USD thereafter.

## 22.10 TOTAL CASH COSTS AND TOTAL PRODUCTION COSTS

The total cash cost over the life of the mine is estimated to be $10.01 per payable ounce of silver and total production cost per payable ounce of silver is estimated to $15.91 (including the construction period). Table 22-5 summarizes the estimated cash and total production cost per payable ounce of silver over selected time periods as well as the life of mine.

**Table 22-5: Total Cash Costs and Total Production Costs (per Silver Oz Payable)**

| Operating Cost Statistics* | | Years 1-9: Yearly Average | Years 10-18: Yearly Average | Years 19-30: Yearly Average | LOM Production |
|---|---|---|---|---|---|
| Direct Mining Cost | $/oz Payable | $3.91 | $2.77 | $1.24 | $2.93 |
| Direct Process Plant | $/oz Payable | $6.29 | $6.32 | $14.51 | $7.82 |
| G&A | $/oz Payable | $0.67 | $0.60 | $1.60 | $0.81 |
| **Cash Operating Costs** | **$/oz Payable** | **$10.87** | **$9.69** | **$17.35** | **$11.56** |
| Shipping and Selling | $/oz Payable | $0.51 | $1.60 | $0.63 | $1.00 |
| Treatment and Refining | $/oz Payable | $1.91 | $4.26 | $1.89 | $2.91 |
| By-Product Credits | $/oz Payable | -$2.51 | -$9.32 | -$2.75 | -$5.46 |
| **Cash Offsite Costs** | **$/oz Payable** | **-$0.09** | **-$3.45** | **-$0.23** | **-$1.55** |
| **Total Cash Costs** | **$/oz Payable** | **$10.78** | **$6.24** | **$17.12** | **$10.01** |
| Non-Cash Costs | $/oz Payable | $4.81 | $8.37 | $2.50 | $5.90 |
| **Total Production Costs** | **$/oz Payable** | **$15.59** | **$14.61** | **$19.62** | **$15.91** |

* ***Notes:*** *(1) Non-cash costs include a period of depreciation and amortization of physical plant and equipment, asset retirement obligation assets, and capitalised mine development and pre-operating costs. The depreciation is compliant with IFRIC 20 relating to deferred stripping. (2) Life of project averages calculated over the full 32 year life of the Project.*

### 22.10.1 Reclamation & Closure

A calculation was completed to estimate the reclamation and closure costs for this project (see Environmental and Regulatory Agency Considerations for further details). The reclamation and closure cost of $75.8 million is included in production years 26-29 (note: ongoing expenditures after production year 29 have been accumulated within production year 29). The cash flow impacts of the reclamation and closure costs are shown in the cash flow section of Table 22-9.

For the purposes of accounting, the reclamation and closure expenditures have been discounted to production year 1 based upon a discount rate of 5%. The difference between this present and future value accretes from production years 1 to 25.

### 22.10.2 Salvage Value

A zero allowance for salvage value was included in the cash flow model shown in Table 22-9.

## 22.11 DEFERRED STRIPPING ASSET

Stripping of overburden is considered to provide future benefits to the mining project by providing improved access to ore. The cost is capitalised as a stripping activity asset and amortised using the units of production method over the ore body to which improved access is provided. This is consistent with IFRIC 20 that will be applicable if the Pitarrilla Project proceeds.

## 22.12 TAXATION

The Mexican income tax liability for the Pitarrilla Project has been calculated based upon the current income tax laws enacted in Mexico.

#### 22.12.1 Corporate Income Tax

The Pitarrilla Project is subject to a corporate income tax rate of 29% in 2012 and 28% in all subsequent years. Total current tax liability over the life of the mine, as shown in the cash flow model in Table 22-9 equals $871.3 million. Current taxes are forecast to be payable between years 5 and 21 of operations.

#### 22.12.2 Alternative Minimum Tax

Alternative Minimum Tax (in Spanish: "IETU") is also calculated for all periods in the mine life. Alternative Minimum Tax credits equal to 85.5 million MXN are assumed to be available as at the start of the construction period. In this financial assessment, no Alternative Minimum Tax liability is incurred over the life of the mine.

#### 22.12.3 Depreciation for Tax

Capital expenditures are depreciated for tax in accordance with the following table. For the purposes of this financial assessment, it has been assumed that accelerated depreciation is taken wherever possible, with the result that depreciation is taken using the "Elective Rate" as shown in Table 22-6.

**Table 22-6: Tax Asset Classes and Depreciation Methodology**

| | Assumed Depreciation Method/Life | |
|---|---|---|
| **Asset Class** | **Annual Rate** | **Elective Rate** |
| Vehicles | 25% | 100% |
| Computer equipment | 25% | 94% |
| Communication Equipment | 10% | n.a. |
| Furniture and fixtures | 10% | n.a. |
| Mine Plant Equipment | 5% | 87% |
| Mining equipment | 10% | 87% |
| Buildings & Facilities | 5% | 74% |
| Mineral properties | 5% | n.a. |
| Asset Retirement Obligation | 5% | n.a. |
| Mine development/Capitalised Pre-Operating | 5% | 100% |
| Soft costs incurred during construction | 100% | n.a. |

#### 22.12.4 Tax Loss Carry Forward Balances

Tax loss carry forwards for Corporate Income Tax purposes equal to 1,270 million MXN are available (with varying expiry periods) as at the start of the construction period. Tax losses can be carried forward for a period of ten years.

#### 22.12.5 Inflation/Deflation of Asset Balance

As the financial model is computed in real dollars, zero inflation or deflation is applied to calculations of depreciable asset balances and loss carry forwards (note that in accordance with Mexican income tax laws, depreciable asset balances and loss carry forwards will increase each year based upon inflation and therefore retain their full value in real terms).

### 22.13 EXCLUDED COSTS

Costs in the economic analysis exclude sunk costs (e.g. drilling costs and corporate overheads) prior to project construction commencement.

### 22.14 PROJECT FINANCING

It is assumed the Pitarrilla Project will be 100% equity financed.

### 22.15 ROYALTIES

No royalties are applicable to the Pitarrilla Project and none were applied.

### 22.16 BONUS/PROFIT SHARING

Profit sharing is not applied to the Pitarrilla Project. Equivalent of one month (1/12$^{th}$) of annual labour expenditures was applied as an average annual profit sharing bonus.

### 22.17 NET PRESENT VALUE, INTERNAL RATE OF RETURN, PAYBACK

The economic analyses for the Pitarrilla Project are summarised below in Table 22-7 for each of the metal price cases.

The Pitarrilla Project has cash costs of $10.01/oz payable silver and production costs of $15.91/oz payable silver (Table 22-5). The NPV at assumed long term metal prices using a 5% discount rate is $737 million and the internal rate of return is 12.8% (Table 22-7). Payback of the initial capital occurs 7.4 years after commercial production commences (Table 22-7).

**Table 22-7: Financial Analysis Results**

| Performance Metrics | | Downside Case | Base Case | Upside Case | Spot Price Case |
|---|---|---|---|---|---|
| Silver Price | $/oz | $22.50 | **$25.00** | $30.00 | $34.13 |
| Average Silver Price | $/oz | $22.50 | **$25.53** | $30.00 | $34.13 |
| Lead Price | $/lb | $0.80 | **$0.90** | $1.10 | $0.99 |
| Zinc Price | $/lb | $0.85 | **$0.95** | $1.10 | $0.87 |
| Diesel Price | $/litre | $0.70 | **$0.80** | $0.95 | $0.85 |
| $MXN per $USD | | 12.50 | **12.50** | 12.50 | 12.96 |
| Pre-tax NPV 5% | $M | $680 | **$1,176** | $1,972 | $2,552 |
| **After-tax NPV 5%** | **$M** | **$368** | **$737** | **$1,316** | **$1,741** |
| Pre-tax IRR | % | 11.5% | **15.8%** | 20.8% | 25.3% |
| **After-tax IRR** | **%** | **9.1%** | **12.8%** | **17.2%** | **21.2%** |
| Undiscounted Cash Flow | $M | $1,328 | **$2,015** | $3,187 | $3,948 |
| Payback After COD | Years | 10.4 | **7.4** | 4.8 | 3.8 |
| Cash Cost/Payable oz of Silver | $/oz Payable | $10.23 | **$10.01** | $9.65 | $10.47 |
| Production Cost/Payable oz of Silver | $/oz Payable | $16.10 | **$15.91** | $15.54 | $16.28 |

***Notes**: (1) Base case silver price is $27.50 per ounce in the final pre-production year and the first two years of production, and $25 per ounce thereafter, (2) The MXN:USD exchange rate is assumed to equal $12.86 in the second half of 2013 and the first half of 2014, $12.58 in the second half of 2014 and the first half of 2015 and $12.50 thereafter, (3) COD means Commercial Operation Date.*

## 22.18 SENSITIVITY ANALYSIS

The results of the sensitivity analysis on project after tax basis NPV are shown in Table 22-8 and Figure 22-1, Figure 22-2, Figure 22-3, and Figure 22-4. Table 22.8 is ordered such that the most sensitive items are at the top of the table grading to the least at the bottom of the table.

### Table 22-8: After Tax NPV Sensitivity Analysis

| | | | Metric Value | | | | NPV 5% (after tax) | |
|---|---|---|---|---|---|---|---|---|
| **Base Case Sensitivity** | | **-10% Change** | **Base Case** | **+10% Change** | | **-10% Change** | **Base Case** | **+10% Change** |
| All Metal Prices | $/oz & $/lb | Silver: $22.50<br>Lead: $0.81<br>Zinc: $0.86 | **Silver: $25**<br>**Lead: $0.90**<br>**Zinc: $0.95** | Silver: $27.50<br>Lead: $0.99<br>Zinc: $1.05 | $M | $381 | **$737** | $1,091 |
| All Mined Grades | Factor | 0.9 | **1.0** | 1.1 | $M | $412 | **$737** | $1,062 |
| Silver Price* | $/oz | $22.50 | **$25.00** | $27.50 | $M | $432 | **$737** | $1,042 |
| Operating Costs | $/tonne milled | $20.51 | **$22.99** | $25.47 | $M | $900 | **$737** | $574 |
| Dore and Concentrate Charges and Shipping | %net/gross revenue | 81.6% | **83.3%** | 84.4% | $M | $802 | **$737** | $674 |
| Initial Capital Costs | $M | $667 | **$741** | $815 | $M | $795 | **$737** | $680 |
| Tax Rate | % | 25.2% | **28.0%** | 30.8% | $M | $781 | **$737** | $694 |
| Diesel Price***** | $/litre | $0.72 | **$0.80** | $0.88 | $M | $766 | **$737** | $709 |
| Labour cost**** | $M | $430 | **$477** | $525 | $M | $759 | **$737** | $716 |
| Sustaining Capital Costs*** | $M | $363 | **$404** | $444 | $M | $755 | **$737** | $720 |
| MXN/USD Rate** | | 11.25 | **12.5** | 13.75 | $M | $712 | **$737** | $752 |

* Base case silver price is $30 per oz (Yr-3 to Yr -2), $27.50 (Yr-1 to Yr 2) and $25 thereafter. The cases assume the stated price in all years (note: that these prices impact the capital cost since revenues during the construction period are netted off against the capital cost).

** Exchange rate 12.86 $MXN per $USD (Yr-3), 12.58 $MXN per $USD (Yr-2) and 12.5 $MXN per $USD thereafter.

*** Excludes ARO.

**** Includes operating labour and labour during the capital period.

***** Includes diesel in the operating and capital periods.

| **Base Case Discount Rate Sensitivity** | | **0% Discount Rate** | **5% Discount Rate** | **7% Discount Rate** | **8% Discount Rate** | **10% Discount Rate** |
|---|---|---|---|---|---|---|
| Base Case NPV (after-tax) | $M | $2,015 | $737 | $460 | $350 | $175 |

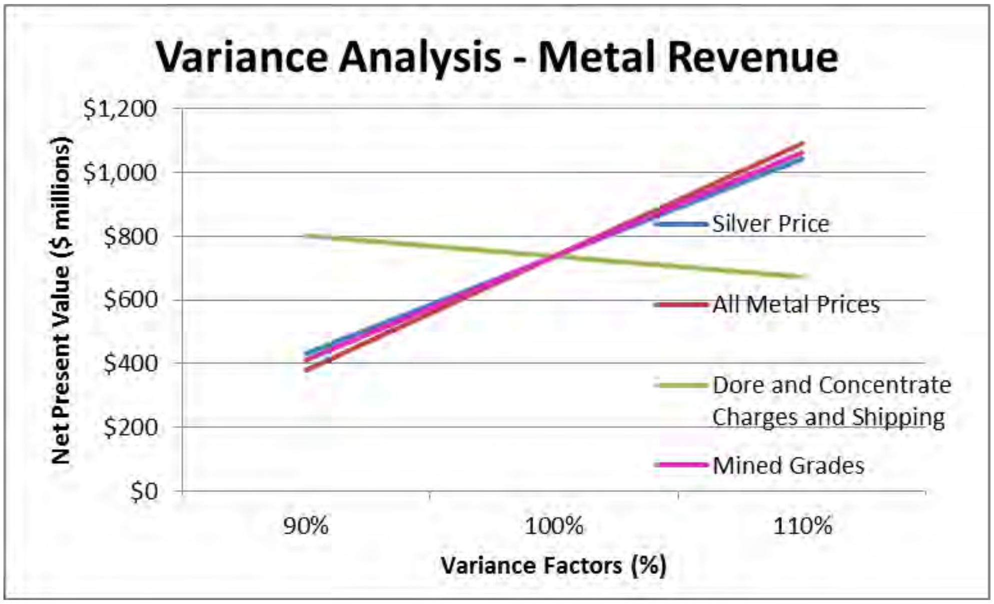



**Figure 22-1: Sensitivity Analysis on After Tax NPV at 5%: Metal Value**

Source: M3, 2012

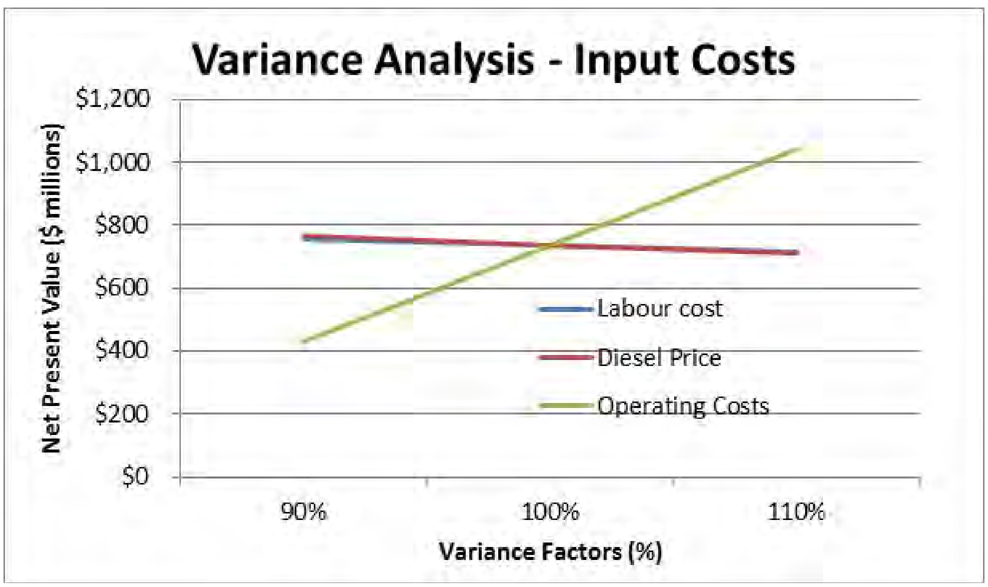



**Figure 22-2: Sensitivity Analysis on After Tax NPV at 5%: Input Costs**

Source: M3, 2012

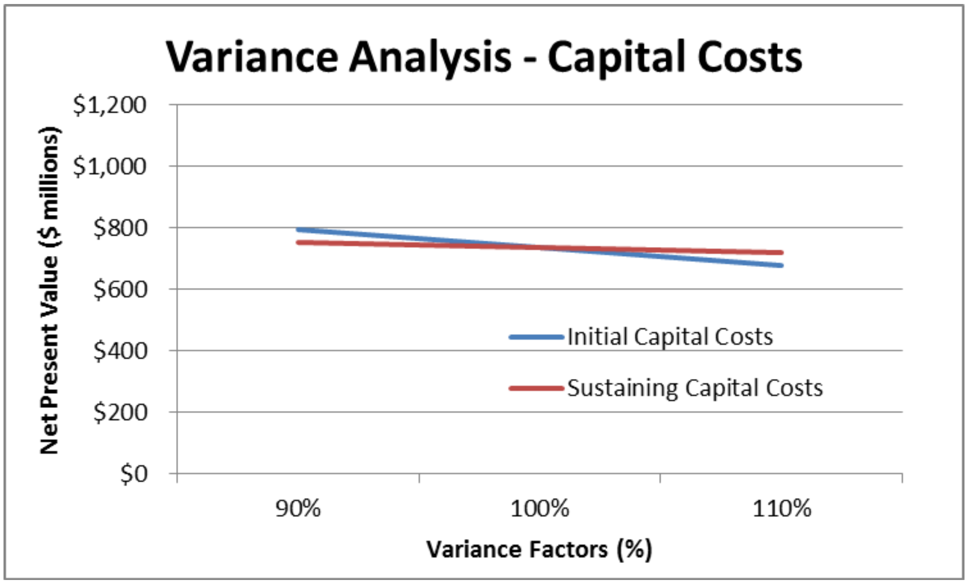



**Figure 22-3: Sensitivity Analysis on After Tax NPV 5%: Capital Costs**
Source: M3,2012

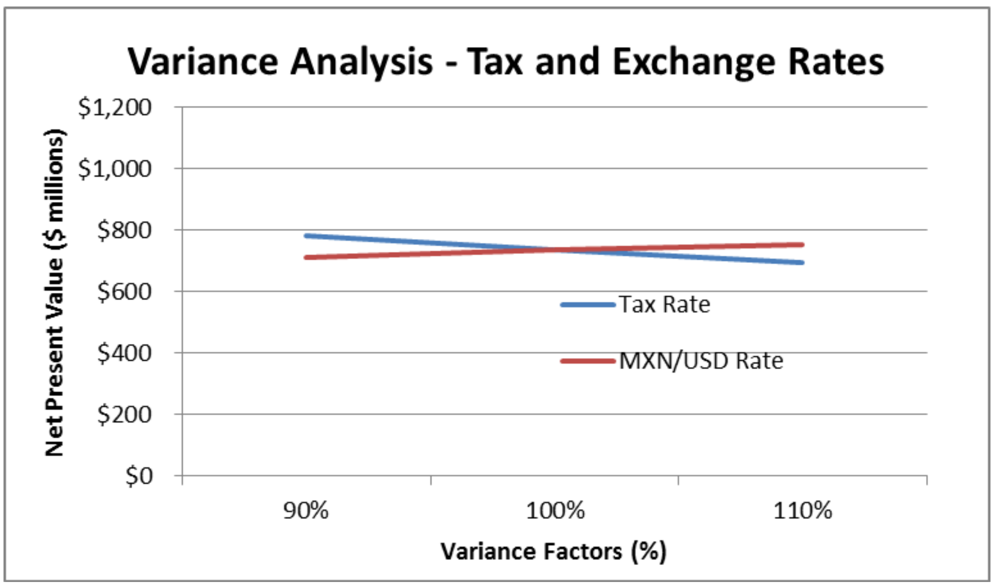



**Figure 22-4: Sensitivity Analysis on After Tax NPV at 5%: Tax and Exchange Rates**
Source: M3, 2012

Table 22-9 provides yearly summary of the mining activities, production schedule and cash flows. Table 22-10 provides additional yeary summary information on process plant feed grades and recovery.

## Table 22-9: Financial Model

| Base Case | Units | Total | Year -3 | Year -2 | Year -1 | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 | Year 11 | Year 12 | Year 13 | Year 14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Mining Operations** | | | | | | | | | | | | | | | | | | | |
| **Ore** | | | | | | | | | | | | | | | | | | | |
| Beginning Inventory | kt | 156,590 | 156,590 | 156,590 | 154,411 | 149,690 | 142,660 | 131,861 | 122,656 | 110,274 | 95,978 | 84,049 | 73,003 | 64,064 | 55,862 | 45,685 | 34,001 | 27,885 | 23,930 |
| Mined | kt | 156,590 | - | 2,179 | 4,721 | 7,029 | 10,799 | 9,205 | 12,382 | 14,296 | 11,929 | 11,046 | 8,938 | 8,203 | 10,176 | 11,684 | 6,117 | 3,954 | 3,665 |
| Ending Inventory | kt | - | 156,590 | 154,411 | 149,690 | 142,660 | 131,861 | 122,656 | 110,274 | 95,978 | 84,049 | 73,003 | 64,064 | 55,862 | 45,685 | 34,001 | 27,885 | 23,930 | 20,265 |
| Silver Grade | g/t | 95.1 | - | 113.1 | 107.8 | 91.2 | 89.9 | 89.1 | 98.3 | 99.5 | 93.7 | 93.6 | 91.2 | 85.3 | 89.4 | 88.6 | 84.2 | 102.6 | 86.7 |
| Lead Grade | % | 0.29% | 0.00% | 0.15% | 0.13% | 0.27% | 0.32% | 0.25% | 0.18% | 0.17% | 0.15% | 0.18% | 0.21% | 0.28% | 0.17% | 0.30% | 0.50% | 0.47% | 0.37% |
| Zinc Grade | % | 0.79% | 0.00% | 0.10% | 0.13% | 0.31% | 0.56% | 0.61% | 0.53% | 0.47% | 0.47% | 0.45% | 0.49% | 0.66% | 0.62% | 0.88% | 0.99% | 2.02% | 1.63% |
| Copper Grade | % | 0.03% | 0.00% | 0.01% | 0.01% | 0.02% | 0.02% | 0.02% | 0.01% | 0.01% | 0.01% | 0.01% | 0.01% | 0.04% | 0.04% | 0.05% | 0.03% | 0.07% | 0.04% |
| Contained Silver | kozs | 478,667 | - | 7,924 | 16,363 | 20,600 | 31,228 | 26,373 | 39,139 | 45,720 | 35,932 | 33,243 | 26,195 | 22,499 | 29,234 | 33,280 | 16,566 | 13,043 | 10,216 |
| Contained Lead | klbs | 1,013,474 | - | 7,186 | 13,689 | 41,583 | 76,583 | 51,326 | 49,210 | 53,493 | 40,033 | 44,259 | 41,004 | 50,205 | 38,884 | 76,216 | 67,797 | 41,218 | 30,122 |
| Contained Zinc | klbs | 2,721,507 | - | 4,866 | 13,255 | 47,374 | 134,411 | 123,804 | 143,389 | 148,202 | 122,374 | 110,148 | 95,582 | 119,918 | 139,405 | 226,275 | 132,843 | 176,331 | 131,973 |
| Contained Copper | klbs | 99,891 | - | 316 | 982 | 2,716 | 5,496 | 3,635 | 2,257 | 2,931 | 2,460 | 1,907 | 1,688 | 6,423 | 8,290 | 12,842 | 3,598 | 5,895 | 3,233 |
| **Waste** | | | | | | | | | | | | | | | | | | | |
| Beginning Inventory | kt | - | - | 3,542 | 31,242 | 92,581 | 158,502 | 219,934 | 282,634 | 346,172 | 410,004 | 470,701 | 534,480 | 600,367 | 670,434 | 732,892 | 789,464 | 838,462 | 881,101 |
| Mined | kt | 933,685 | 3,542 | 27,700 | 61,338 | 65,921 | 61,432 | 62,700 | 63,538 | 63,832 | 60,698 | 63,779 | 65,887 | 70,067 | 62,459 | 56,571 | 48,998 | 42,640 | 32,835 |
| Ending Inventory | kt | 933,685 | 3,542 | 31,242 | 92,581 | 158,502 | 219,934 | 282,634 | 346,172 | 410,004 | 470,701 | 534,480 | 600,367 | 670,434 | 732,892 | 789,464 | 838,462 | 881,101 | 913,936 |
| | | | | | | | | | | | | | | | | | | | |
| **Total Material Mined** | kt | **1,090,275** | **3,542** | **29,879** | **66,059** | **72,951** | **72,231** | **71,905** | **75,920** | **78,128** | **72,627** | **74,825** | **74,825** | **78,270** | **72,635** | **68,255** | **55,115** | **46,594** | **36,500** |
| **Strip Ratio** | Ratio | **5.96** | **-** | **12.71** | **12.99** | **9.38** | **5.69** | **6.81** | **5.13** | **4.47** | **5.09** | **5.77** | **7.37** | **8.54** | **6.14** | **4.84** | **8.01** | **10.78** | **8.96** |
| | | | | | | | | | | | | | | | | | | | |
| **Process Plant Operations** | | | | | | | | | | | | | | | | | | | |
| **Total Ore Processed** | kt | **156,590** | **-** | **-** | **2,067** | **5,074** | **5,256** | **5,694** | **5,621** | **5,548** | **5,475** | **5,475** | **5,548** | **5,548** | **5,548** | **5,548** | **5,694** | **5,767** | **5,840** |
| Silver Grade | g/t | 95.1 | - | - | 135.2 | 126.1 | 123.8 | 102.9 | 116.4 | 136.9 | 127.0 | 129.2 | 113.7 | 107.1 | 115.2 | 128.8 | 95.0 | 93.5 | 75.9 |
| Lead Grade | % | 0.29% | 0.00% | 0.00% | 0.17% | 0.22% | 0.37% | 0.33% | 0.21% | 0.21% | 0.17% | 0.21% | 0.24% | 0.33% | 0.22% | 0.38% | 0.51% | 0.43% | 0.31% |
| Zinc Grade | % | 0.79% | 0.00% | 0.00% | 0.10% | 0.27% | 0.57% | 0.69% | 0.56% | 0.52% | 0.48% | 0.47% | 0.51% | 0.75% | 0.68% | 1.08% | 1.03% | 1.66% | 1.23% |
| Copper Grade | % | 0.03% | 0.00% | 0.00% | 0.01% | 0.01% | 0.02% | 0.02% | 0.01% | 0.01% | 0.01% | 0.01% | 0.01% | 0.04% | 0.05% | 0.05% | 0.04% | 0.06% | 0.03% |
| Contained Silver | kozs | 478,667 | - | - | 8,984 | 20,563 | 20,915 | 18,834 | 21,040 | 24,419 | 22,361 | 22,744 | 20,279 | 19,098 | 20,550 | 22,969 | 17,400 | 17,334 | 14,243 |
| Contained Lead | klbs | 1,013,474 | - | - | 7,860 | 24,194 | 42,681 | 41,550 | 26,416 | 26,286 | 20,543 | 25,361 | 29,281 | 40,158 | 26,831 | 46,153 | 63,419 | 54,703 | 39,948 |
| Contained Zinc | klbs | 2,721,507 | - | - | 4,352 | 30,002 | 66,376 | 86,607 | 69,164 | 63,666 | 57,729 | 57,079 | 61,920 | 91,165 | 82,827 | 132,300 | 129,139 | 211,023 | 158,848 |
| Contained Copper | klbs | 99,891 | - | - | 455 | 1,451 | 2,837 | 2,495 | 1,306 | 1,347 | 1,236 | 1,081 | 1,358 | 5,195 | 6,335 | 6,628 | 4,553 | 8,048 | 4,404 |
| **Beginning of Period Stockpile Inventory** | kt | | | **-** | **2,179** | **4,832** | **6,788** | **12,331** | **15,842** | **22,604** | **31,351** | **37,806** | **43,377** | **46,767** | **49,422** | **54,050** | **60,186** | **60,609** | **58,796** |
| Silver Grade | g/t | | | - | 113 | 98 | 70 | 65 | 65 | 71 | 72 | 71 | 69 | 68 | 67 | 66 | 65 | 64 | 63 |
| Lead Grade | % | | | - | 0.15% | 0.12% | 0.20% | 0.24% | 0.21% | 0.19% | 0.18% | 0.17% | 0.17% | 0.17% | 0.17% | 0.16% | 0.17% | 0.17% | 0.17% |
| Zinc Grade | % | | | - | 0.10% | 0.13% | 0.21% | 0.36% | 0.39% | 0.42% | 0.43% | 0.43% | 0.43% | 0.43% | 0.44% | 0.45% | 0.47% | 0.47% | 0.46% |
| Copper Grade | % | | | - | 0.01% | 0.01% | 0.01% | 0.02% | 0.02% | 0.01% | 0.01% | 0.01% | 0.01% | 0.01% | 0.01% | 0.01% | 0.02% | 0.01% | 0.01% |
| **Ore from Stockpile to Plant** | kt | **67,145** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **1,813** | **2,175** |
| Silver Grade | g/t | 66 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 74 | 58 |
| Lead Grade | % | 0.21% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.34% | 0.20% |
| Zinc Grade | % | 0.61% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.87% | 0.56% |
| Copper Grade | % | 0.02% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.05% | 0.02% |
| **Ore Direct from Mine to Plant** | kt | **89,445** | **-** | **-** | **2,067** | **5,074** | **5,256** | **5,694** | **5,621** | **5,548** | **5,475** | **5,475** | **5,548** | **5,548** | **5,548** | **5,548** | **5,694** | **3,954** | **3,665** |
| Silver Grade | g/t | 117.3 | - | - | 135.2 | 126.1 | 123.8 | 102.9 | 116.4 | 136.9 | 127.0 | 129.2 | 113.7 | 107.1 | 115.2 | 128.8 | 90.5 | 102.6 | 86.7 |
| Lead Grade | % | 0.36% | 0.00% | 0.00% | 0.17% | 0.22% | 0.37% | 0.33% | 0.21% | 0.21% | 0.17% | 0.21% | 0.24% | 0.33% | 0.22% | 0.38% | 0.51% | 0.47% | 0.37% |
| Zinc Grade | % | 0.92% | 0.00% | 0.00% | 0.10% | 0.27% | 0.57% | 0.69% | 0.56% | 0.52% | 0.48% | 0.47% | 0.51% | 0.75% | 0.68% | 1.08% | 1.03% | 2.02% | 1.63% |
| Copper Grade | % | 0.04% | 0.00% | 0.00% | 0.01% | 0.01% | 0.02% | 0.02% | 0.01% | 0.01% | 0.01% | 0.01% | 0.01% | 0.04% | 0.05% | 0.05% | 0.03% | 0.07% | 0.04% |
| **Ore Differential Mine to Stockpile** | kt | **67,145** | **-** | **2,179** | **2,654** | **1,956** | **5,543** | **3,511** | **6,761** | **8,748** | **6,454** | **5,571** | **3,390** | **2,655** | **4,628** | **6,136** | **423** | **-** | **(0)** |
| Silver Grade | g/t | 66 | - | 113 | 86 | 1 | 58 | 67 | 83 | 76 | 65 | 59 | 54 | 40 | 58 | 52 | - | - | - |
| Lead Grade | % | 0.21% | 0.00% | 0.15% | 0.10% | 0.40% | 0.28% | 0.13% | 0.15% | 0.14% | 0.14% | 0.15% | 0.16% | 0.17% | 0.12% | 0.22% | 0.47% | 0.00% | 0.54% |
| Zinc Grade | % | 0.61% | 0.00% | 0.10% | 0.15% | 0.40% | 0.56% | 0.48% | 0.50% | 0.44% | 0.45% | 0.43% | 0.45% | 0.49% | 0.55% | 0.69% | 0.40% | 0.00% | 0.00% |
| Copper Grade | % | 0.02% | 0.00% | 0.01% | 0.01% | 0.03% | 0.02% | 0.01% | 0.01% | 0.01% | 0.01% | 0.01% | 0.00% | 0.02% | 0.02% | 0.05% | 0.00% | 0.00% | 0.00% |
| | | | | | | | | | | | | | | | | | | | |
| **Lead Concentrate** | | | | | | | | | | | | | | | | | | | |
| **Lead Concentrate** | kt | **603.7** | **-** | **-** | **0.6** | **6.7** | **12.5** | **22.8** | **13.4** | **17.4** | **13.5** | **16.4** | **20.1** | **28.6** | **19.9** | **34.0** | **47.0** | **41.5** | **30.1** |
| Silver Grade | g/t | 9,554.8 | - | - | 112,135.5 | 23,528.4 | 11,125.2 | 10,400.4 | 14,057.8 | 15,795.6 | 15,757.4 | 15,377.5 | 13,698.1 | 11,106.5 | 14,995.0 | 12,269.3 | 7,018.6 | 8,144.8 | 8,378.4 |
| Lead Grade | % | 43.42% | 0.00% | 0.00% | 28.62% | 37.56% | 44.87% | 43.05% | 40.96% | 41.02% | 35.45% | 36.90% | 37.92% | 41.72% | 35.68% | 44.00% | 47.40% | 44.61% | 40.25% |
| Zinc Grade | % | 10.99% | 0.00% | 0.00% | 4.34% | 4.98% | 8.44% | 8.78% | 9.57% | 7.50% | 8.54% | 6.32% | 5.90% | 7.88% | 8.18% | 10.55% | 7.89% | 16.39% | 16.06% |
| Ag Recovery | % | 52.81% | 0.00% | 0.00% | 27.86% | 35.29% | 36.45% | 44.66% | 34.07% | 44.94% | 41.88% | 47.31% | 53.24% | 64.84% | 58.71% | 69.86% | 66.86% | 65.64% | 56.90% |
| Lead Recovery | % | 67.83% | 0.00% | 0.00% | 28.55% | 42.04% | 39.67% | 56.03% | 51.38% | 65.14% | 58.50% | 62.12% | 64.31% | 71.29% | 64.15% | 75.93% | 78.85% | 75.74% | 66.83% |
| Zinc Recovery | % | 6.31% | 0.00% | 0.00% | 2.91% | 3.94% | 4.86% | 5.46% | 4.60% | 5.14% | 4.93% | 4.83% | 4.83% | 6.00% | 5.19% | 6.64% | 6.58% | 7.21% | 6.71% |
| Recovered Silver | kozs | 185,440 | - | - | 1,020 | 5,048 | 4,462 | 7,620 | 6,074 | 8,828 | 6,824 | 8,125 | 8,836 | 10,207 | 9,604 | 13,399 | 10,595 | 10,872 | 8,104 |
| Recovered Lead | kt | 262.1 | - | - | 0.2 | 2.5 | 5.6 | 9.8 | 5.5 | 7.1 | 4.8 | 6.1 | 7.6 | 11.9 | 7.1 | 14.9 | 22.3 | 18.5 | 12.1 |
| Recovered Zinc | kt | 66.3 | - | - | 0.0 | 0.3 | 1.1 | 2.0 | 1.3 | 1.3 | 1.2 | 1.0 | 1.2 | 2.3 | 1.6 | 3.6 | 3.7 | 6.8 | 4.8 |
| **Zinc Concentrate** | | | | | | | | | | | | | | | | | | | |
| **Zinc Concentrate** | kt | **1,514.6** | **-** | **-** | **0.8** | **8.6** | **20.3** | **41.1** | **29.1** | **31.7** | **29.5** | **27.6** | **31.4** | **54.2** | **43.2** | **80.9** | **90.0** | **147.9** | **107.5** |
| Silver Grade | g/t | 604.1 | - | - | 5,958.1 | 2,660.3 | 1,027.3 | 857.0 | 1,056.0 | 1,240.7 | 1,121.6 | 1,367.0 | 1,282.7 | 855.0 | 1,049.6 | 719.3 | 601.1 | 367.4 | 388.3 |
| Lead Grade | % | 0.12% | 0.00% | 0.00% | 0.29% | 0.28% | 0.25% | 0.18% | 0.16% | 0.17% | 0.15% | 0.19% | 0.20% | 0.16% | 0.14% | 0.12% | 0.15% | 0.08% | 0.09% |
| Zinc Grade | % | 45.60% | 0.00% | 0.00% | 28.11% | 34.35% | 42.70% | 43.13% | 41.82% | 41.25% | 40.30% | 38.20% | 38.12% | 42.96% | 40.69% | 46.17% | 45.48% | 48.17% | 46.79% |
| Ag Recovery | % | 8.38% | 0.00% | 0.00% | 4.32% | 5.14% | 5.47% | 6.64% | 5.55% | 6.43% | 6.52% | 7.07% | 7.81% | 9.47% | 8.91% | 9.75% | 10.97% | 10.55% | 9.43% |
| Lead Recovery | % | 0.48% | 0.00% | 0.00% | 0.42% | 0.40% | 0.35% | 0.43% | 0.43% | 0.49% | 0.54% | 0.54% | 0.53% | 0.52% | 0.56% | 0.50% | 0.47% | 0.50% | 0.52% |
| Zinc Recovery | % | 65.70% | 0.00% | 0.00% | 27.63% | 35.09% | 40.01% | 48.42% | 43.59% | 51.48% | 50.85% | 49.10% | 48.83% | 62.01% | 56.03% | 69.17% | 72.68% | 75.47% | 69.83% |
| Recovered Silver | kozs | 29,420 | - | - | 158 | 736 | 670 | 1,134 | 989 | 1,263 | 1,063 | 1,215 | 1,296 | 1,491 | 1,458 | 1,871 | 1,739 | 1,747 | 1,343 |
| Recovered Lead | kt | 1.8 | - | - | 0.0 | 0.0 | 0.1 | 0.1 | 0.0 | 0.1 | 0.0 | 0.1 | 0.1 | 0.1 | 0.1 | 0.1 | 0.1 | 0.1 | 0.1 |
| Recovered Zinc | kt | 690.6 | - | - | 0.2 | 3.0 | 8.7 | 17.7 | 12.2 | 13.1 | 11.9 | 10.6 | 12.0 | 23.3 | 17.6 | 37.3 | 40.9 | 71.2 | 50.3 |
| **Dore - Sulphide Leach** | | | | | | | | | | | | | | | | | | | |
| Sulphide Tailings Tonnes Feed | kt | 111,115 | - | - | 570 | 3,307 | 3,471 | 5,192 | 4,921 | 4,623 | 4,337 | 4,336 | 4,621 | 4,589 | 4,609 | 4,557 | 5,119 | 5,359 | 5,702 |
| Sulphide Tailings Silver Feed Grade | g/t | 38.2 | - | - | 135.5 | 80.2 | 63.7 | 49.8 | 68.0 | 64.3 | 60.3 | 56.2 | 43.5 | 27.4 | 35.7 | 26.7 | 21.3 | 22.9 | 26.2 |
| Silver Recovery of Sulphide Tailings | % | 35.00% | 0.00% | 0.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% |
| Recovered Silver (kozs) | kozs | 47,704 | - | - | 869 | 2,983 | 2,488 | 2,909 | 3,768 | 3,343 | 2,943 | 2,742 | 2,263 | 1,415 | 1,854 | 1,369 | 1,229 | 1,380 | 1,679 |
| **Dore - Oxide Leach** | | | | | | | | | | | | | | | | | | | |
| Tonnes Feed | kt | 43,356 | - | - | 1,496 | 1,752 | 1,752 | 438 | 657 | 876 | 1,095 | 1,095 | 876 | 876 | 876 | 876 | 438 | 219 | - |
| Silver Feed Grade | g/t | 91.5 | - | - | 110.7 | 111.1 | 154.0 | 125.8 | 152.0 | 169.5 | 172.3 | 158.3 | 130.7 | 119.2 | 148.8 | 134.5 | 110.3 | 109.6 | - |
| Ag Recovery | % | 53.75% | 0.00% | 0.00% | 55.17% | 62.76% | 61.99% | 69.46% | 53.36% | 59.86% | 54.55% | 59.68% | 60.15% | 63.13% | 61.52% | 65.90% | 65.80% | 70.23% | - |
| Recovered Silver | kozs | 70,806 | - | - | 2,945 | 3,927 | 5,377 | 1,230 | 1,713 | 2,858 | 3,309 | 3,325 | 2,215 | 2,119 | 2,578 | 2,497 | 1,022 | 542 | 0.00% |
| **Dore - Total** | | | | | | | | | | | | | | | | | | | |
| Recovered Silver | kozs | 118,510 | - | - | 3,814 | 6,910 | 7,865 | 4,139 | 5,482 | 6,202 | 6,252 | 6,067 | 4,478 | 3,534 | 4,432 | 3,866 | 2,251 | 1,922 | - |
| Dore - Oxide Leach | | | - | - | 1,496 | 1,752 | 1,752 | 438 | 657 | 876 | 1,095 | 1,095 | 876 | 876 | 876 | 876 | 438 | 219 | 1,679 |

# Table 22-9: Financial Model (continued)

| Base Case | Units | Total | Year 15 | Year 16 | Year 17 | Year 18 | Year 19 | Year 20 | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 | Year 30 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Mining Operations** | | | | | | | | | | | | | | | | | | | |
| **Ore** | | | | | | | | | | | | | | | | | | | |
| Beginning Inventory | kt | 156,590 | 20,265 | 7,889 | 2,398 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 156,590 |
| Mined | kt | 156,590 | 12,376 | 5,491 | 2,398 | - | - | - | - | - | - | - | - | - | - | - | - | - | 156,590 |
| Ending Inventory | kt | - | 7,889 | 2,398 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | - |
| Silver Grade | g/t | 95.1 | 106.0 | 119.6 | 101.4 | - | - | - | - | - | - | - | - | - | - | - | - | - | 95.1 |
| Lead Grade | % | 0.29% | 0.64% | 0.65% | 0.71% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.29% |
| Zinc Grade | % | 0.79% | 2.32% | 1.28% | 1.19% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.79% |
| Copper Grade | % | 0.03% | 0.09% | 0.07% | 0.06% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.03% |
| Contained Silver | kozs | 478,667 | 42,175 | 21,120 | 7,817 | - | - | - | - | - | - | - | - | - | - | - | - | - | 478,667 |
| Contained Lead | klbs | 1,013,474 | 173,979 | 79,178 | 37,509 | - | - | - | - | - | - | - | - | - | - | - | - | - | 1,013,474 |
| Contained Zinc | klbs | 2,721,507 | 633,223 | 155,094 | 63,040 | - | - | - | - | - | - | - | - | - | - | - | - | - | 2,721,507 |
| Contained Copper | klbs | 99,891 | 23,994 | 8,036 | 3,193 | - | - | - | - | - | - | - | - | - | - | - | - | - | 99,891 |
| **Waste** | | | | | | | | | | | | | | | | | | | |
| Beginning Inventory | kt | - | 913,936 | 928,935 | 932,570 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | - |
| Mined | kt | 933,685 | 14,999 | 3,634 | 1,116 | - | - | - | - | - | - | - | - | - | - | - | - | - | 933,685 |
| Ending Inventory | kt | 933,685 | 928,935 | 932,570 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 | 933,685 |
| **Total Material Mined** | kt | **1,090,275** | **27,375** | **9,125** | **3,514** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **1,090,275** |
| **Strip Ratio** | Ratio | **5.96** | **1.21** | **0.66** | **0.47** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **5.96** |
| **Process Plant Operations** | | | | | | | | | | | | | | | | | | | |
| **Total Ore Processed** | kt | **156,590** | **5,840** | **5,840** | **5,840** | **4,672** | **4,380** | **4,380** | **4,380** | **5,840** | **5,840** | **5,840** | **5,840** | **5,549** | **4,380** | **4,380** | **3,886** | **-** | **156,590** |
| Silver Grade | g/t | 95.1 | 136.7 | 152.3 | 72.9 | 74.3 | 79.6 | 79.6 | 75.6 | 53.6 | 53.6 | 53.6 | 53.6 | 54.6 | 60.9 | 60.9 | 60.9 | - | 95.1 |
| Lead Grade | % | 0.29% | 0.78% | 0.78% | 0.54% | 0.20% | 0.17% | 0.17% | 0.16% | 0.19% | 0.19% | 0.19% | 0.19% | 0.18% | 0.15% | 0.15% | 0.15% | 0.00% | 0.29% |
| Zinc Grade | % | 0.79% | 2.80% | 2.04% | 1.32% | 0.56% | 0.45% | 0.45% | 0.44% | 0.52% | 0.52% | 0.52% | 0.52% | 0.50% | 0.40% | 0.40% | 0.40% | 0.00% | 0.79% |
| Copper Grade | % | 0.03% | 0.11% | 0.09% | 0.05% | 0.02% | 0.01% | 0.01% | 0.01% | 0.02% | 0.02% | 0.02% | 0.02% | 0.02% | 0.01% | 0.01% | 0.01% | 0.00% | 0.03% |
| Contained Silver | kozs | 478,667 | 25,670 | 28,604 | 13,697 | 11,162 | 11,206 | 11,206 | 10,642 | 10,058 | 10,058 | 10,058 | 10,058 | 9,734 | 8,583 | 8,583 | 7,614 | - | 478,667 |
| Contained Lead | klbs | 1,013,474 | 100,544 | 100,190 | 69,689 | 20,641 | 16,097 | 16,097 | 15,696 | 23,975 | 23,975 | 23,975 | 23,975 | 22,143 | 14,235 | 14,235 | 12,628 | - | 1,013,474 |
| Contained Zinc | klbs | 2,721,507 | 360,782 | 263,132 | 170,487 | 57,965 | 43,592 | 43,592 | 42,483 | 66,307 | 66,307 | 66,307 | 66,307 | 61,082 | 38,436 | 38,436 | 34,097 | - | 2,721,507 |
| Contained Copper | klbs | 99,891 | 14,048 | 11,765 | 6,923 | 1,598 | 1,017 | 1,017 | 1,001 | 2,262 | 2,262 | 2,262 | 2,262 | 2,021 | 944 | 944 | 838 | - | 99,891 |
| **Beginning of Period Stockpile Inventory** | kt | | **56,622** | **63,157** | **62,808** | **59,367** | **54,695** | **50,315** | **45,935** | **41,555** | **35,715** | **29,875** | **24,035** | **18,195** | **12,646** | **8,266** | **3,886** | **-** | |
| Silver Grade | g/t | | 64 | 65 | 62 | 62 | 61 | 60 | 58 | 56 | 56 | 57 | 58 | 59 | 61 | 61 | 61 | - | |
| Lead Grade | % | | 0.17% | 0.20% | 0.19% | 0.17% | 0.17% | 0.17% | 0.17% | 0.17% | 0.17% | 0.17% | 0.16% | 0.16% | 0.15% | 0.15% | 0.15% | 0.00% | |
| Zinc Grade | % | | 0.45% | 0.60% | 0.53% | 0.48% | 0.47% | 0.47% | 0.47% | 0.48% | 0.47% | 0.46% | 0.45% | 0.43% | 0.40% | 0.40% | 0.40% | 0.00% | |
| Copper Grade | % | | 0.01% | 0.02% | 0.02% | 0.01% | 0.01% | 0.01% | 0.01% | 0.02% | 0.01% | 0.01% | 0.01% | 0.01% | 0.01% | 0.01% | 0.01% | 0.00% | |
| **Ore from Stockpile to Plant** | kt | **67,145** | **-** | **349** | **3,442** | **4,672** | **4,380** | **4,380** | **4,380** | **5,840** | **5,840** | **5,840** | **5,840** | **5,549** | **4,380** | **4,380** | **3,886** | **-** | |
| Silver Grade | g/t | 66 | - | 666 | 53 | 74 | 80 | 80 | 76 | 54 | 54 | 54 | 54 | 55 | 61 | 61 | 61 | - | |
| Lead Grade | % | 0.21% | 0.00% | 2.73% | 0.42% | 0.20% | 0.17% | 0.17% | 0.16% | 0.19% | 0.19% | 0.19% | 0.19% | 0.18% | 0.15% | 0.15% | 0.15% | 0.00% | |
| Zinc Grade | % | 0.61% | 0.00% | 14.02% | 1.42% | 0.56% | 0.45% | 0.45% | 0.44% | 0.52% | 0.52% | 0.52% | 0.52% | 0.50% | 0.40% | 0.40% | 0.40% | 0.00% | |
| Copper Grade | % | 0.02% | 0.00% | 0.48% | 0.05% | 0.02% | 0.01% | 0.01% | 0.01% | 0.02% | 0.02% | 0.02% | 0.02% | 0.02% | 0.01% | 0.01% | 0.01% | 0.00% | |
| **Ore Direct from Mine to Plant** | kt | **89,445** | **5,840** | **5,491** | **2,398** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | |
| Silver Grade | g/t | 117.3 | 136.7 | 119.6 | 101.4 | - | - | - | - | - | - | - | - | - | - | - | - | - | |
| Lead Grade | % | 0.36% | 0.78% | 0.65% | 0.71% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | |
| Zinc Grade | % | 0.92% | 2.80% | 1.28% | 1.19% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | |
| Copper Grade | % | 0.04% | 0.11% | 0.07% | 0.06% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | |
| **Base Case** | **Units** | Total | Year 15 | Year 16 | Year 17 | Year 18 | Year 19 | Year 20 | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 | Year 30 | |
| **Ore Differential Mine to Stockpile** | kt | **67,145** | **6,536** | **(0)** | **0** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | |
| Silver Grade | g/t | 66 | 79 | - | 37 | - | - | - | - | - | - | - | - | - | - | - | - | - | |
| Lead Grade | % | 0.21% | 0.51% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | |
| Zinc Grade | % | 0.61% | 1.89% | 0.00% | 0.41% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | |
| Copper Grade | % | 0.02% | 0.07% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | |
| **Lead Concentrate** | | | | | | | | | | | | | | | | | | | |
| **Lead Concentrate** | kt | **603.7** | **73.2** | **73.2** | **53.4** | **5.6** | **-** | **-** | **-** | **15.3** | **15.3** | **15.3** | **15.3** | **12.6** | **-** | **-** | **-** | **-** | **603.7** |
| Silver Grade | g/t | 9,554.8 | 8,123.4 | 9,199.5 | 5,373.3 | 4,575.6 | - | - | - | 6,312.2 | 6,312.2 | 6,312.2 | 6,312.2 | 6,312.2 | - | - | - | - | 9,554.8 |
| Lead Grade | % | 43.42% | 51.35% | 51.31% | 46.34% | 39.53% | 0.00% | 0.00% | 0.00% | 32.41% | 32.41% | 32.41% | 32.41% | 32.41% | 0.00% | 0.00% | 0.00% | 0.00% | 43.42% |
| Zinc Grade | % | 10.99% | 17.39% | 12.32% | 10.19% | 10.78% | 0.00% | 0.00% | 0.00% | 8.61% | 8.61% | 8.61% | 8.61% | 8.61% | 0.00% | 0.00% | 0.00% | 0.00% | 10.99% |
| Ag Recovery | % | 52.81% | 74.50% | 75.68% | 67.39% | 42.57% | 0.00% | 0.00% | 0.00% | 30.91% | 30.91% | 30.91% | 30.91% | 30.91% | 0.00% | 0.00% | 0.00% | 0.00% | 52.81% |
| Lead Recovery | % | 67.83% | 82.45% | 82.64% | 78.32% | 64.45% | 0.00% | 0.00% | 0.00% | 45.66% | 45.66% | 45.66% | 45.66% | 45.66% | 0.00% | 0.00% | 0.00% | 0.00% | 67.83% |
| Zinc Recovery | % | 6.31% | 7.78% | 7.56% | 7.04% | 6.08% | 0.00% | 0.00% | 0.00% | 4.39% | 4.39% | 4.39% | 4.39% | 4.39% | 0.00% | 0.00% | 0.00% | 0.00% | 6.31% |
| Recovered Silver | kozs | 185,440 | 19,126 | 21,647 | 9,230 | 824 | - | - | - | 3,109 | 3,109 | 3,109 | 3,109 | 2,558 | - | - | - | - | 185,440 |
| Recovered Lead | kt | 262.1 | 37.6 | 37.6 | 24.8 | 2.2 | - | - | - | 5.0 | 5.0 | 5.0 | 5.0 | 4.1 | - | - | - | - | 262.1 |
| Recovered Zinc | kt | 66.3 | 12.7 | 9.0 | 5.4 | 0.6 | - | - | - | 1.3 | 1.3 | 1.3 | 1.3 | 1.1 | - | - | - | - | 66.3 |
| **Zinc Concentrate** | | | | | | | | | | | | | | | | | | | |
| **Zinc Concentrate** | kt | **1,514.6** | **263.2** | **192.8** | **127.5** | **14.8** | **-** | **-** | **-** | **35.7** | **35.7** | **35.7** | **35.7** | **29.4** | **-** | **-** | **-** | **-** | **1,514.6** |
| Silver Grade | g/t | 604.1 | 334.7 | 510.2 | 403.0 | 386.7 | - | - | - | 616.7 | 616.7 | 616.7 | 616.7 | 616.7 | - | - | - | - | 604.1 |
| Lead Grade | % | 0.12% | 0.08% | 0.10% | 0.12% | 0.12% | 0.00% | 0.00% | 0.00% | 0.16% | 0.16% | 0.16% | 0.16% | 0.16% | 0.00% | 0.00% | 0.00% | 0.00% | 0.12% |
| Zinc Grade | % | 45.60% | 50.21% | 49.20% | 46.83% | 44.49% | 0.00% | 0.00% | 0.00% | 39.09% | 39.09% | 39.09% | 39.09% | 39.09% | 0.00% | 0.00% | 0.00% | 0.00% | 45.60% |
| Ag Recovery | % | 8.38% | 11.03% | 11.06% | 12.06% | 9.52% | 0.00% | 0.00% | 0.00% | 7.05% | 7.05% | 7.05% | 7.05% | 7.05% | 0.00% | 0.00% | 0.00% | 0.00% | 8.38% |
| Lead Recovery | % | 0.48% | 0.44% | 0.44% | 0.49% | 0.53% | 0.00% | 0.00% | 0.00% | 0.51% | 0.51% | 0.51% | 0.51% | 0.51% | 0.00% | 0.00% | 0.00% | 0.00% | 0.48% |
| Zinc Recovery | % | 65.70% | 80.74% | 79.48% | 77.22% | 66.45% | 0.00% | 0.00% | 0.00% | 46.46% | 46.46% | 46.46% | 46.46% | 46.46% | 0.00% | 0.00% | 0.00% | 0.00% | 65.70% |
| Recovered Silver | kozs | 29,420 | 2,832 | 3,163 | 1,652 | 184 | - | - | - | 709 | 709 | 709 | 709 | 583 | - | - | - | - | 29,420 |
| Recovered Lead | kt | 1.8 | 0.2 | 0.2 | 0.2 | 0.0 | - | - | - | 0.1 | 0.1 | 0.1 | 0.1 | 0.0 | - | - | - | - | 1.8 |
| Recovered Zinc | kt | 690.6 | 132.1 | 94.9 | 59.7 | 6.6 | - | - | - | 14.0 | 14.0 | 14.0 | 14.0 | 11.5 | - | - | - | - | 690.6 |
| **Dore - Sulphide Leach** | | | | | | | | | | | | | | | | | | | |
| Sulphide Tailings Tonnes Feed | kt | 111,115 | 5,504 | 5,574 | 5,659 | 1,148 | - | - | - | 5,789 | 5,789 | 5,789 | 5,789 | 4,762 | - | - | - | - | 111,115 |
| Sulphide Tailings Silver Feed Grade | g/t | 38.2 | 21.0 | 21.2 | 15.5 | 25.1 | - | - | - | 33.5 | 33.5 | 33.5 | 33.5 | 33.5 | - | - | - | - | 38.2 |
| Silver Recovery of Sulphide Tailings | % | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% | 35.00% |
| Recovered Silver (kozs) | kozs | 47,704 | 1,300 | 1,328 | 985 | 324 | - | - | - | 2,184 | 2,184 | 2,184 | 2,184 | 1,797 | - | - | - | - | 47,704 |
| **Dore - Oxide Leach** | | | | | | | | | | | | | | | | | | | |
| Tonnes Feed | kt | 43,356 | - | - | - | 3,504 | 4,380 | 4,380 | 4,380 | - | - | - | - | 745 | 4,380 | 4,380 | 3,886 | - | 43,356 |
| Silver Feed Grade | g/t | 91.5 | - | - | - | 81.9 | 79.6 | 79.6 | 75.6 | - | - | - | - | 60.9 | 60.9 | 60.9 | 60.9 | - | 91.5 |
| Ag Recovery | % | 53.75% | 0.00% | 0.00% | 0.00% | 56.28% | 55.06% | 55.06% | 53.40% | 0.00% | 0.00% | 0.00% | 0.00% | 45.48% | 45.48% | 45.48% | 45.48% | 0.00% | 53.75% |
| Recovered Silver | kozs | 70,806 | - | - | - | 5,193 | 6,170 | 6,170 | 5,683 | - | - | - | - | 664 | 3,903 | 3,903 | 3,463 | - | 70,806 |
| **Dore - Total** | | | | | | | | | | | | | | | | | | | |
| Recovered Silver | kozs | 118,510 | 1,300 | 1,328 | 985 | 5,517 | 6,170 | 6,170 | 5,683 | 2,184 | 2,184 | 2,184 | 2,184 | 2,460 | 3,903 | 3,903 | 3,463 | - | 118,510 |

# Table 22-9: Financial Model (continued)

| Base Case | Units | Total | Year -3 | Year -2 | Year -1 | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 | Year 11 | Year 12 | Year 13 | Year 14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Total Metal Recovery** | | | | | | | | | | | | | | | | | | | |
| **Metals Recovered** | | | | | | | | | | | | | | | | | | | |
| Silver | kozs | 333,370 | - | - | 4,993 | 12,694 | 12,997 | 12,892 | 12,544 | 16,293 | 14,139 | 15,406 | 14,609 | 15,232 | 15,494 | 19,136 | 14,585 | 14,541 | 11,125 |
| Lead | klbs | 581,898 | - | - | 362 | 5,578 | 12,452 | 21,792 | 12,236 | 15,840 | 10,624 | 13,485 | 16,909 | 26,483 | 15,808 | 33,166 | 49,357 | 41,100 | 26,905 |
| Zinc | klbs | 1,668,870 | - | - | 566 | 7,247 | 21,415 | 43,530 | 29,684 | 31,675 | 28,734 | 25,572 | 29,010 | 56,326 | 42,340 | 90,241 | 98,398 | 172,070 | 121,575 |
| **Recoveries** | | | | | | | | | | | | | | | | | | | |
| Recovered Silver | % | 69.65% | 0.00% | 0.00% | 55.57% | 61.73% | 62.14% | 68.45% | 59.62% | 66.72% | 63.23% | 67.74% | 72.04% | 79.76% | 75.40% | 83.31% | 83.82% | 83.89% | 78.11% |
| Recovered Lead | % | 57.42% | 0.00% | 0.00% | 4.60% | 23.06% | 29.17% | 52.45% | 46.32% | 60.26% | 51.72% | 53.17% | 57.75% | 65.95% | 58.92% | 71.86% | 77.83% | 75.13% | 67.35% |
| Recovered Zinc | % | 61.32% | 0.00% | 0.00% | 13.01% | 24.16% | 32.26% | 50.26% | 42.92% | 49.75% | 49.77% | 44.80% | 46.85% | 61.78% | 51.12% | 68.21% | 76.19% | 81.54% | 76.54% |
| **Payable Metals** | | | | | | | | | | | | | | | | | | | |
| **Lead Concentrate** | | | | | | | | | | | | | | | | | | | |
| Payable Silver | kozs | 176,168 | - | - | 969 | 4,795 | 4,239 | 7,239 | 5,770 | 8,387 | 6,483 | 7,718 | 8,394 | 9,697 | 9,124 | 12,729 | 10,065 | 10,329 | 7,699 |
| Payable Lead | kt | 244 | - | - | 0 | 2 | 5 | 9 | 5 | 7 | 4 | 6 | 7 | 11 | 7 | 14 | 21 | 17 | 11 |
| **Zinc Concentrate** | | | | | | | | | | | | | | | | | | | |
| Payable Silver | kozs | 18,657 | - | - | 117 | 533 | 457 | 758 | 676 | 876 | 731 | 849 | 901 | 996 | 996 | 1,221 | 1,102 | 977 | 765 |
| Payable Zinc | kt | 569 | - | - | 0 | 2 | 7 | 14 | 10 | 11 | 10 | 8 | 9 | 19 | 14 | 31 | 34 | 59 | 42 |
| **Dore** | | | | | | | | | | | | | | | | | | | |
| Payable Silver | kozs | 117,325 | - | - | 3,776 | 6,841 | 7,786 | 4,097 | 5,427 | 6,140 | 6,189 | 6,006 | 4,433 | 3,499 | 4,388 | 3,827 | 2,229 | 1,903 | 1,662 |
| **Total** | | | | | | | | | | | | | | | | | | | |
| Payable Silver | kozs | 312,150 | - | - | 4,862 | 12,169 | 12,482 | 12,094 | 11,873 | 15,403 | 13,403 | 14,574 | 13,728 | 14,192 | 14,508 | 17,777 | 13,396 | 13,209 | 10,126 |
| Payable Zinc | kt | 569 | - | - | 0 | 2 | 7 | 14 | 10 | 11 | 10 | 8 | 9 | 19 | 14 | 31 | 34 | 59 | 42 |
| Payable Lead | kt | 244 | - | - | 0 | 2 | 5 | 9 | 5 | 7 | 4 | 6 | 7 | 11 | 7 | 14 | 21 | 17 | 11 |
| **Income Statement** | | | | | | | | | | | | | | | | | | | |
| Silver | $/oz | | $ 30.00 | $ 30.00 | $ 27.50 | $ 27.50 | $ 27.50 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 |
| Zinc | $/lb | | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 |
| Lead | $/lb | | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 |
| **Net Revenues** | | | | | | | | | | | | | | | | | | | |
| Silver in Concentrates | (US$000) | $ 4,221,837 | $ - | $ - | $ - | $ 121,808 | $ 114,561 | $ 169,044 | $ 145,719 | $ 201,627 | $ 164,206 | $ 188,919 | $ 206,470 | $ 233,169 | $ 226,782 | $ 299,848 | $ 245,881 | $ 237,752 | $ 184,258 |
| Concentrates - Lead | (US$000) | $ 416,301 | $ - | $ - | $ - | $ 3,789 | $ 8,677 | $ 15,291 | $ 9,556 | $ 11,492 | $ 8,041 | $ 9,704 | $ 12,220 | $ 18,779 | $ 12,189 | $ 23,241 | $ 34,725 | $ 29,352 | $ 19,684 |
| Concentrates - Zinc | (US$000) | $ 1,021,019 | $ - | $ - | $ - | $ 3,936 | $ 12,213 | $ 25,378 | $ 18,993 | $ 19,590 | $ 17,901 | $ 15,807 | $ 17,549 | $ 33,566 | $ 27,131 | $ 54,347 | $ 60,546 | $ 101,049 | $ 76,611 |
| Dore - Silver | (US$000) | $ 2,867,570 | $ - | $ - | $ - | $ 185,015 | $ 213,623 | $ 104,197 | $ 135,032 | $ 153,150 | $ 154,704 | $ 150,241 | $ 111,581 | $ 87,916 | $ 109,270 | $ 95,947 | $ 56,487 | $ 47,724 | $ 41,663 |
| **Total Revenues** | (US$000) | **$ 8,526,726** | **$ -** | **$ -** | **$ -** | **$ 314,549** | **$ 349,073** | **$ 313,910** | **$ 309,300** | **$ 385,859** | **$ 344,852** | **$ 364,672** | **$ 347,820** | **$ 373,429** | **$ 375,371** | **$ 473,382** | **$ 397,638** | **$ 415,877** | **$ 322,216** |
| Cost of Sales | | | | | | | | | | | | | | | | | | | |
| Cash Cost of Inventory Sold | (US$000) | $ 3,594,944 | $ - | $ - | $ - | $ 103,374 | $ 105,846 | $ 114,298 | $ 122,403 | $ 124,633 | $ 125,014 | $ 123,305 | $ 123,818 | $ 125,419 | $ 126,072 | $ 126,645 | $ 130,381 | $ 135,282 | $ 136,973 |
| Shipping and selling cost | (US$000) | $ 306,951 | $ - | $ - | $ - | $ 2,215 | $ 4,750 | $ 9,270 | $ 6,172 | $ 7,113 | $ 6,228 | $ 6,391 | $ 7,464 | $ 12,008 | $ 9,151 | $ 16,655 | $ 19,856 | $ 27,467 | $ 19,955 |
| **Total Cash Cost of Sales** | (US$000) | **$ 3,901,895** | **$ -** | **$ -** | **$ -** | **$ 105,589** | **$ 110,596** | **$ 123,567** | **$ 128,575** | **$ 131,746** | **$ 131,242** | **$ 129,696** | **$ 131,282** | **$ 137,427** | **$ 135,223** | **$ 143,300** | **$ 150,237** | **$ 162,749** | **$ 156,928** |
| Operating Income Before Depreciation | (US$000) | $ 4,624,831 | $ - | $ - | $ - | $ 208,960 | $ 238,477 | $ 190,343 | $ 180,726 | $ 254,113 | $ 213,610 | $ 234,976 | $ 216,538 | $ 236,002 | $ 240,149 | $ 330,083 | $ 247,401 | $ 253,128 | $ 165,288 |
| Depreciation, Depletion and Amortization | (US$000) | $ 1,814,093 | $ - | $ - | $ - | $ 25,173 | $ 31,416 | $ 34,208 | $ 35,915 | $ 41,791 | $ 40,526 | $ 43,200 | $ 44,047 | $ 47,923 | $ 50,804 | $ 60,249 | $ 62,500 | $ 67,427 | $ 63,644 |
| **Earnings (Loss) Before Interest and Taxes** | (US$000) | **$ 2,810,739** | **$ -** | **$ -** | **$ -** | **$ 183,787** | **$ 207,061** | **$ 156,135** | **$ 144,811** | **$ 212,321** | **$ 173,085** | **$ 191,776** | **$ 172,492** | **$ 188,079** | **$ 189,344** | **$ 269,834** | **$ 184,901** | **$ 185,701** | **$ 101,644** |
| Interest Charges | (US$000) | $ 55,562 | $ - | $ - | $ - | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 |
| **Earnings (Loss) Before Taxes** | (US$000) | **$ 2,755,177** | **$ -** | **$ -** | **$ -** | **$ 181,564** | **$ 204,839** | **$ 153,912** | **$ 142,588** | **$ 210,099** | **$ 170,862** | **$ 189,553** | **$ 170,269** | **$ 185,856** | **$ 187,122** | **$ 267,611** | **$ 182,679** | **$ 183,479** | **$ 99,421** |
| Income Taxes | | | | | | | | | | | | | | | | | | | |
| Current Taxes | (US$000) | $ 871,309 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ 30,981 | $ 44,128 | $ 44,290 | $ 42,763 | $ 58,887 | $ 59,396 | $ 85,028 | $ 50,891 | $ 62,868 | $ 31,428 |
| Deferred Taxes | (US$000) | $ (16,943) | $ - | $ - | $ - | $ 62,464 | $ 4,735 | $ 5,126 | $ 3,384 | $ 677 | $ 2,397 | $ 135 | $ 1,005 | $ 922 | $ (1,465) | $ (6,618) | $ 699 | $ 541 | $ 262 |
| **Total Income Tax Expense** | (US$000) | **$ 854,366** | **$ -** | **$ -** | **$ -** | **$ 62,464** | **$ 4,735** | **$ 5,126** | **$ 3,384** | **$ 31,658** | **$ 46,525** | **$ 44,425** | **$ 43,767** | **$ 59,809** | **$ 57,931** | **$ 78,409** | **$ 51,591** | **$ 63,409** | **$ 31,690** |
| **Net Income After Taxes** | (US$000) | $ 1,900,811 | $ - | $ - | $ - | $ 119,100 | $ 200,103 | $ 148,786 | $ 139,204 | $ 178,441 | $ 124,337 | $ 145,129 | $ 126,502 | $ 126,047 | $ 129,190 | $ 189,202 | $ 131,088 | $ 120,070 | $ 67,732 |
| **Cash Flow** | | | | | | | | | | | | | | | | | | | |
| **Cash Flows from (or used in) Operating Activities** | | | | | | | | | | | | | | | | | | | |
| Operating Income Before Depreciation | (US$000) | 4,624,831 | - | - | - | 208,960 | 238,477 | 190,343 | 180,726 | 254,113 | 213,610 | 234,976 | 216,538 | 236,002 | 240,149 | 330,083 | 247,401 | 253,128 | 165,288 |
| Net Change in Non-cash working capital items | (US$000) | (587,572) | - | - | (48,639) | (93,567) | (50,064) | (52,640) | 1,549 | 14,635 | 19,206 | (11,194) | (37,169) | (37,434) | 11,716 | 6,239 | (21,536) | (47,546) | 164 |
| Asset Retirement Accretion/(Expenditure | (US$000) | (75,796) | | | | | | | | | | | | | | | | | |
| Subtotal: Cash from (used in) operating activities | (US$000) | $ 3,961,463 | $ - | $ - | $ (48,639) | $ 115,392 | $ 188,413 | $ 137,704 | $ 182,274 | $ 268,748 | $ 232,816 | $ 223,782 | $ 179,370 | $ 198,568 | $ 251,865 | $ 336,322 | $ 225,866 | $ 205,582 | $ 165,452 |
| **Cash Flows from (or used in) Investing Activities** | | | | | | | | | | | | | | | | | | | |
| **Initial Capital** | | | | | | | | | | | | | | | | | | | |
| Mining Equipment | (US$000) | (163,602) | (41,812) | (74,886) | (46,904) | | | | | | | | | | | | | | |
| Plant & Equipment | (US$000) | (509,170) | (150,686) | (239,529) | (118,954) | | | | | | | | | | | | | | |
| Mine development/Capitalized Pre-Operating | (US$000) | (20,659) | (21,981) | (49,839) | 51,160 | | | | | | | | | | | | | | |
| Other | (US$000) | (4,608) | (1,627) | (1,682) | (1,300) | | | | | | | | | | | | | | |
| **Subtotal - Initial Capital** | (US$000) | **$ (698,039)** | **$ (216,106)** | **$ (365,935)** | **$ (115,997)** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| **Sustaining Capital** | | | | | | | | | | | | | | | | | | | |
| Mining Equipment | (US$000) | (304,190) | | | | (3,118) | (5,656) | (3,157) | (14,651) | (4,952) | (45,034) | (67,716) | (50,465) | (3,396) | (5,337) | (3,310) | (2,732) | (2,591) | (30,070) |
| Plant & Equipment | (US$000) | (50,644) | | | | (6,121) | - | - | - | - | - | - | - | - | - | - | (44,523) | - | - |
| Mine development/Capitalized Pre-Operating | (US$000) | (41,970) | | | | (2,370) | (3,145) | (2,374) | (1,871) | (2,126) | (2,708) | (2,708) | (2,022) | (2,122) | (1,928) | (1,928) | (1,659) | (1,659) | (1,755) |
| Other | (US$000) | (7,054) | | | | (424) | (347) | (420) | (27) | (424) | (346) | (420) | (27) | (424) | (347) | (420) | (27) | (424) | (347) |
| Subtotal - Sustaining Capital | (US$000) | **$ (403,858)** | **$ -** | **$ -** | **$ -** | **(12,034)** | **(9,148)** | **(5,951)** | **(16,550)** | **(7,502)** | **(48,089)** | **(70,844)** | **(52,514)** | **(5,943)** | **(7,611)** | **(5,658)** | **(48,941)** | **(4,674)** | **(32,172)** |
| Net changes in construction-related payables | (US$000) | $ 26,953 | $ 15,965 | $ 10,988 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| (Increase)/Decrease in long term stockpiles | (US$000) | $ - | $ - | $ - | $ - | $ (9,683) | $ (26,427) | $ (25,914) | $ (70,210) | $ (96,819) | $ (76,662) | $ (56,559) | $ (31,753) | $ (40,263) | $ (75,102) | $ (86,867) | $ (19,579) | $ (3,597) | $ (18,577) |
| **Total Capital Expenditures** | (US$000) | **$ (1,074,944)** | **$ (200,141)** | **$ (354,948)** | **$ (115,997)** | **$ (21,717)** | **$ (35,576)** | **$ (31,864)** | **$ (86,760)** | **$ (104,322)** | **$ (124,751)** | **$ (127,402)** | **$ (84,267)** | **$ (46,205)** | **$ (82,714)** | **$ (92,525)** | **$ (68,520)** | **$ (8,271)** | **$ (50,749)** |
| Cash Flow before Taxes | (US$000) | $ 2,886,519 | $ (200,141) | $ (354,948) | $ (164,636) | $ 93,675 | $ 152,837 | $ 105,839 | $ 95,515 | $ 164,426 | $ 108,066 | $ 96,380 | $ 95,103 | $ 152,362 | $ 169,151 | $ 243,797 | $ 157,345 | $ 197,311 | $ 114,703 |
| Cumulative Cash Flow before Taxes | (US$000) | | $ (200,141) | $ (354,948) | $ (164,636) | $ 93,675 | $ 152,837 | $ 105,839 | $ 95,515 | $ 164,426 | $ 108,066 | $ 96,380 | $ 95,103 | $ 152,362 | $ 169,151 | $ 243,797 | $ 157,345 | $ 197,311 | $ 114,703 |
| Taxes | | | | | | | | | | | | | | | | | | | |
| Cash Income Taxes | (US$000) | $ 871,309 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ 30,981 | $ 44,128 | $ 44,290 | $ 42,763 | $ 58,887 | $ 59,396 | $ 85,028 | $ 50,891 | $ 62,868 | $ 31,428 |
| **Cash Flow after Taxes** | (US$000) | **$ 2,015,209** | **$ (200,141)** | **$ (354,948)** | **$ (164,636)** | **$ 93,675** | **$ 152,837** | **$ 105,839** | **$ 95,515** | **$ 133,445** | **$ 63,937** | **$ 52,090** | **$ 52,340** | **$ 93,475** | **$ 109,755** | **$ 158,769** | **$ 106,454** | **$ 134,443** | **$ 83,275** |
| Cumulative Cash Flow after Taxes | (US$000) | | $ (200,141) | $ (555,089) | $ (719,725) | $ (626,050) | $ (473,212) | $ (367,373) | $ (271,858) | $ (138,413) | $ (74,476) | $ (22,386) | $ 29,954 | $ 123,429 | $ 233,184 | $ 391,954 | $ 498,408 | $ 632,851 | $ 716,127 |

## Table 22-9: Financial Model (continued)

| Base Case | Units | Total | Year 15 | Year 16 | Year 17 | Year 18 | Year 19 | Year 20 | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 | Year 30 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Total Metal Recovery** | | | | | | | | | | | | | | | | | | | |
| **Metals Recovered** | | | | | | | | | | | | | | | | | | | |
| Silver | kozs | 333,370 | 23,257 | 26,138 | 11,867 | 6,525 | 6,170 | 6,170 | 5,683 | 6,002 | 6,002 | 6,002 | 6,002 | 5,601 | 3,903 | 3,903 | 3,463 | - | 333,370 |
| Lead | klbs | 581,898 | 83,339 | 83,234 | 54,925 | 4,920 | - | - | - | 11,069 | 11,069 | 11,069 | 11,069 | 9,106 | - | - | - | - | 581,898 |
| Zinc | klbs | 1,668,870 | 319,378 | 229,012 | 143,642 | 15,868 | - | - | - | 33,713 | 33,713 | 33,713 | 33,713 | 27,734 | - | - | - | - | 1,668,870 |
| **Recoveries** | | | | | | | | | | | | | | | | | | | |
| Recovered Silver | % | 69.65% | 90.60% | 91.38% | 86.64% | 58.46% | 55.06% | 55.06% | 53.40% | 59.67% | 59.67% | 59.67% | 59.67% | 57.55% | 45.48% | 45.48% | 45.48% | 0.00% | 69.65% |
| Recovered Lead | % | 57.42% | 82.89% | 83.08% | 78.81% | 23.84% | 0.00% | 0.00% | 0.00% | 46.17% | 46.17% | 46.17% | 46.17% | 41.12% | 0.00% | 0.00% | 0.00% | 0.00% | 57.42% |
| Recovered Zinc | % | 61.32% | 88.52% | 87.03% | 84.25% | 27.37% | 0.00% | 0.00% | 0.00% | 50.84% | 50.84% | 50.84% | 50.84% | 45.41% | 0.00% | 0.00% | 0.00% | 0.00% | 61.32% |
| **Payable Metals** | | | | | | | | | | | | | | | | | | | |
| **Lead Concentrate** | | | | | | | | | | | | | | | | | | | |
| Payable Silver | kozs | 176,168 | 18,169 | 20,565 | 8,768 | 783 | - | - | - | 2,954 | 2,954 | 2,954 | 2,954 | 2,430 | - | - | - | - | 176,168 |
| Payable Lead | kt | 244 | 35 | 35 | 23 | 2 | - | - | - | 5 | 5 | 5 | 5 | 4 | - | - | - | - | 244 |
| **Zinc Concentrate** | | | | | | | | | | | | | | | | | | | |
| Payable Silver | kozs | 18,657 | 1,532 | 1,938 | 952 | 105 | - | - | - | 451 | 451 | 451 | 451 | 371 | - | - | - | - | 18,657 |
| Payable Zinc | kt | 569 | 111 | 79 | 50 | 5 | - | - | - | 11 | 11 | 11 | 11 | 9 | - | - | - | - | 569 |
| **Dore** | | | | | | | | | | | | | | | | | | | |
| Payable Silver | kozs | 117,325 | 1,287 | 1,315 | 975 | 5,462 | 6,108 | 6,108 | 5,626 | 2,162 | 2,162 | 2,162 | 2,162 | 2,436 | 3,864 | 3,864 | 3,428 | - | 117,325 |
| **Total** | | | | | | | | | | | | | | | | | | | |
| Payable Silver | kozs | 312,150 | 20,987 | 23,818 | 10,696 | 6,350 | 6,108 | 6,108 | 5,626 | 5,567 | 5,567 | 5,567 | 5,567 | 5,237 | 3,864 | 3,864 | 3,428 | - | 312,150 |
| Payable Zinc | kt | 569 | 111 | 79 | 50 | 5 | - | - | - | 11 | 11 | 11 | 11 | 9 | - | - | - | - | 569 |
| Payable Lead | kt | 244 | 35 | 35 | 23 | 2 | - | - | - | 5 | 5 | 5 | 5 | 4 | - | - | - | - | 244 |
| **Income Statement** | | | | | | | | | | | | | | | | | | | |
| Silver | $/oz | | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | $ 25.00 | |
| Zinc | $/lb | | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | $ 0.95 | |
| Lead | $/lb | | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | $ 0.90 | |
| **Net Revenues** | | | | | | | | | | | | | | | | | | | |
| Silver in Concentrates | (US$000) | $ 4,221,837 | $ 393,866 | $ 483,660 | $ 228,512 | $ 34,781 | $ 337 | $ - | $ - | $ 63,917 | $ 70,913 | $ 70,913 | $ 70,913 | $ 59,381 | $ 4,600 | $ - | $ - | $ - | 4,221,837 |
| Concentrates - Lead | (US$000) | $ 416,301 | $ 55,632 | $ 60,949 | $ 40,678 | $ 6,468 | $ 62 | $ - | $ - | $ 6,712 | $ 7,447 | $ 7,447 | $ 7,447 | $ 6,236 | $ 483 | $ - | $ - | $ - | 416,301 |
| Concentrates - Zinc | (US$000) | $ 1,021,019 | $ 185,175 | $ 149,241 | $ 92,368 | $ 16,312 | $ 172 | $ - | $ - | $ 17,476 | $ 19,389 | $ 19,389 | $ 19,389 | $ 16,236 | $ 1,258 | $ - | $ - | $ - | 1,021,019 |
| Dore - Silver | (US$000) | $ 2,867,570 | $ 32,344 | $ 32,853 | $ 24,546 | $ 134,403 | $ 152,400 | $ 150,191 | $ 138,560 | $ 55,715 | $ 54,055 | $ 54,055 | $ 54,055 | $ 60,761 | $ 95,923 | $ 95,015 | $ 84,499 | $ 1,644 | 2,867,570 |
| **Total Revenues** | (US$000) | **$ 8,526,726** | **$ 667,018** | **$ 726,703** | **$ 386,104** | **$ 191,965** | **$ 152,970** | **$ 150,191** | **$ 138,560** | **$ 143,820** | **$ 151,804** | **$ 151,804** | **$ 151,804** | **$ 142,614** | **$ 102,264** | **$ 95,015** | **$ 84,499** | **$ 1,644** | **8,526,726** |
| Cost of Sales | | | | | | | | | | | | | | | | | | | |
| Cash Cost of Inventory Sold | (US$000) | $ 3,594,944 | $ 147,026 | $ 144,930 | $ 140,398 | $ 114,424 | $ 101,109 | $ 99,500 | $ 98,375 | $ 124,566 | $ 130,863 | $ 130,876 | $ 131,904 | $ 132,354 | $ 109,687 | $ 110,890 | $ 151,656 | $ 2,924 | 3,594,944 |
| Shipping and selling cost | (US$000) | $ 306,951 | $ 48,777 | $ 38,570 | $ 26,237 | $ 2,961 | $ - | $ - | $ - | $ 7,405 | $ 7,405 | $ 7,405 | $ 7,405 | $ 6,092 | $ - | $ - | $ - | $ - | 306,951 |
| **Total Cash Cost of Sales** | (US$000) | **$ 3,901,895** | **$ 195,803** | **$ 183,500** | **$ 166,635** | **$ 117,385** | **$ 101,109** | **$ 99,500** | **$ 98,375** | **$ 131,971** | **$ 138,268** | **$ 138,281** | **$ 139,309** | **$ 138,445** | **$ 109,687** | **$ 110,890** | **$ 151,656** | **$ 2,924** | **3,901,895** |
| Operating Income Before Depreciation | (US$000) | $ 4,624,831 | $ 471,215 | $ 543,203 | $ 219,469 | $ 74,580 | $ 51,860 | $ 50,691 | $ 40,185 | $ 11,849 | $ 13,536 | $ 13,522 | $ 12,495 | $ 4,168 | $ (7,423) | $ (15,875) | $ (67,157) | $ (1,280) | 4,624,831 |
| Depreciation, Depletion and Amortization | (US$000) | $ 1,814,093 | $ 103,812 | $ 113,295 | $ 92,991 | $ 68,083 | $ 59,457 | $ 58,396 | $ 55,836 | $ 69,691 | $ 73,211 | $ 73,131 | $ 74,633 | $ 78,514 | $ 63,549 | $ 67,400 | $ 111,123 | $ 2,147 | 1,814,093 |
| **Earnings (Loss) Before Interest and Taxes** | (US$000) | **$ 2,810,739** | **$ 367,403** | **$ 429,908** | **$ 126,478** | **$ 6,497** | **$ (7,596)** | **$ (7,705)** | **$ (15,650)** | **$ (57,843)** | **$ (59,675)** | **$ (59,609)** | **$ (62,138)** | **$ (74,346)** | **$ (70,972)** | **$ (83,275)** | **$ (178,280)** | **$ (3,428)** | **2,810,739** |
| Interest Charges | (US$000) | $ 55,562 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ 2,222 | $ - | $ - | $ - | $ - | $ - | 55,562 |
| **Earnings (Loss) Before Taxes** | (US$000) | **$ 2,755,177** | **$ 365,180** | **$ 427,686** | **$ 124,256** | **$ 4,274** | **$ (9,819)** | **$ (9,927)** | **$ (17,873)** | **$ (60,065)** | **$ (61,898)** | **$ (61,832)** | **$ (64,361)** | **$ (74,346)** | **$ (70,972)** | **$ (83,275)** | **$ (178,280)** | **$ (3,428)** | **2,755,177** |
| Income Taxes | | | | | | | | | | | | | | | | | | | |
| Current Taxes | (US$000) | $ 871,309 | $ 117,472 | $ 145,389 | $ 55,387 | $ 15,850 | $ 9,987 | $ 9,857 | $ 6,709 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | 871,309 |
| Deferred Taxes | (US$000) | $ (16,943) | $ (42,803) | $ (26,185) | $ (10,098) | $ 217 | $ 457 | $ 469 | $ 508 | $ (2,239) | $ (6,808) | $ (1,342) | $ (1,461) | $ (1,146) | $ (36) | $ (814) | $ (633) | $ 708 | (16,943) |
| **Total Income Tax Expense** | (US$000) | **$ 854,366** | **$ 74,668** | **$ 119,204** | **$ 45,289** | **$ 16,067** | **$ 10,444** | **$ 10,326** | **$ 7,216** | **-$ 2,239** | **-$ 6,808** | **-$ 1,342** | **-$ 1,461** | **-$ 1,146** | **-$ 36** | **-$ 814** | **-$ 633** | **$ 708** | **$ 854,366** |
| **Net Income After Taxes** | (US$000) | $ 1,900,811 | $ 290,512 | $ 308,482 | $ 78,967 | -$ 11,793 | -$ 20,263 | -$ 20,253 | -$ 25,089 | -$ 57,826 | -$ 55,090 | -$ 60,490 | -$ 62,899 | -$ 73,199 | -$ 70,936 | -$ 82,461 | -$ 177,646 | -$ 4,135 | $ 1,900,811 |
| **Cash Flow** | | | | | | | | | | | | | | | | | | | |
| **Cash Flows from (or used in) Operating Activities** | | | | | | | | | | | | | | | | | | | |
| Operating Income Before Depreciation | (US$000) | 4,624,831 | 471,215 | 543,203 | 219,469 | 74,580 | 51,860 | 50,691 | 40,185 | 11,849 | 13,536 | 13,522 | 12,495 | 4,168 | (7,423) | (15,875) | (67,157) | (1,280) | 4,624,831 |
| Net Change in Non-cash working capital items | (US$000) | (587,572) | 165,870 | 49,095 | 58,468 | (10,402) | (47,983) | (55,180) | (105,211) | (82,377) | (70,250) | (69,217) | (55,655) | (16,699) | (45,232) | (40,480) | 82,111 | 1,847 | (587,572) |
| Asset Retirement Accretion/(Expenditure | (US$000) | (75,796) | | | | | | | | | | | | (11,369) | (11,369) | (11,369) | (41,688) | - | (75,796) |
| Subtotal: Cash from (used in) operating activities | (US$000) | $ 3,961,463 | $ 637,084 | $ 592,299 | $ 277,937 | $ 64,178 | $ 3,877 | $ (4,489) | $ (65,026) | $ (70,528) | $ (56,714) | $ (55,695) | $ (43,160) | $ (23,901) | $ (64,024) | $ (67,724) | $ (26,733) | $ 567 | $ 3,961,463 |
| **Cash Flows from (or used in) Investing Activities** | | | | | | | | | | | | | | | | | | | |
| **Initial Capital** | | | | | | | | | | | | | | | | | | | |
| Mining Equipment | (US$000) | (163,602) | | | | | | | | | | | | | | | | | (163,602) |
| Plant & Equipment | (US$000) | (509,170) | | | | | | | | | | | | | | | | | (509,170) |
| Mine development/Capitalized Pre-Operating | (US$000) | (20,659) | | | | | | | | | | | | | | | | | (20,659) |
| Other | (US$000) | (4,608) | | | | | | | | | | | | | | | | | (4,608) |
| **Subtotal - Initial Capital** | (US$000) | **$ (698,039)** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **(698,039)** |
| **Sustaining Capital** | | | | | | | | | | | | | | | | | | | |
| Mining Equipment | (US$000) | (304,190) | (32,195) | (2,538) | (602) | (272) | (263) | (263) | (263) | (5,179) | (19,826) | (204) | (204) | (195) | - | - | - | - | (304,190) |
| Plant & Equipment | (US$000) | (50,644) | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | (50,644) |
| Mine development/Capitalized Pre-Operating | (US$000) | (41,970) | (1,518) | (1,518) | (1,518) | (1,215) | (1,139) | (1,139) | (1,139) | (1,518) | (889) | - | - | - | - | - | - | - | (41,970) |
| Other | (US$000) | (7,054) | (420) | (27) | (424) | (346) | (420) | 0 | (315) | (67) | (113) | (0) | (315) | (67) | (113) | - | - | - | (7,054) |
| Subtotal - Sustaining Capital | (US$000) | **$ (403,858)** | **(34,134)** | **(4,083)** | **(2,545)** | **(1,833)** | **(1,822)** | **(1,402)** | **(1,717)** | **(6,764)** | **(20,828)** | **(204)** | **(519)** | **(262)** | **(113)** | **-** | **-** | **-** | **(403,858)** |
| Net changes in construction-related payables | (US$000) | $ 26,953 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ 26,953 |
| (Increase)/Decrease in long term stockpiles | (US$000) | $ - | $ (237,848) | $ (51,839) | $ 1,097 | $ 70,860 | $ 74,494 | $ 74,494 | $ 123,597 | $ 95,357 | $ 95,357 | $ 95,357 | $ 84,792 | $ 52,699 | $ 84,569 | $ 75,024 | $ - | $ - | $ - |
| **Total Capital Expenditures** | (US$000) | **$ (1,074,944)** | **$ (271,981)** | **$ (55,922)** | **$ (1,448)** | **$ 69,027** | **$ 72,672** | **$ 73,092** | **$ 121,880** | **$ 88,593** | **$ 74,529** | **$ 95,153** | **$ 84,273** | **$ 52,438** | **$ 84,456** | **$ 75,024** | **$ -** | **$ -** | **$ (1,074,944)** |
| Cash Flow before Taxes | (US$000) | $ 2,886,519 | $ 365,103 | $ 536,376 | $ 276,490 | $ 133,205 | $ 76,549 | $ 68,604 | $ 56,854 | $ 18,064 | $ 17,815 | $ 39,458 | $ 41,113 | $ 28,537 | $ 20,432 | $ 7,299 | $ (26,733) | $ 567 | $ 2,886,519 |
| Cumulative Cash Flow before Taxes | (US$000) | | $ 365,103 | $ 536,376 | $ 276,490 | $ 133,205 | $ 76,549 | $ 68,604 | $ 56,854 | $ 18,064 | $ 17,815 | $ 39,458 | $ 41,113 | $ 28,537 | $ 20,432 | $ 7,299 | $ (26,733) | $ 567 | $ 2,886,519 |
| Taxes | | | | | | | | | | | | | | | | | | | |
| Cash Income Taxes | (US$000) | $ 871,309 | $ 117,472 | $ 145,389 | $ 55,387 | $ 15,850 | $ 9,987 | $ 9,857 | $ 6,709 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ 871,309 |
| **Cash Flow after Taxes** | (US$000) | **$ 2,015,209** | **$ 247,631** | **$ 390,988** | **$ 221,103** | **$ 117,355** | **$ 66,562** | **$ 58,747** | **$ 50,146** | **$ 18,064** | **$ 17,815** | **$ 39,458** | **$ 41,113** | **$ 28,537** | **$ 20,432** | **$ 7,299** | **$ (26,733)** | **$ 567** | **$ 2,015,209** |
| Cumulative Cash Flow after Taxes | (US$000) | | $ 963,758 | $ 1,354,746 | $ 1,575,849 | $ 1,693,203 | $ 1,759,766 | $ 1,818,513 | $ 1,868,658 | $ 1,886,723 | $ 1,904,537 | $ 1,943,995 | $ 1,985,108 | $ 2,013,645 | $ 2,034,076 | $ 2,041,376 | $ 2,014,642 | $ 2,015,209 | $ 2,015,209 |

# Table 22-10: Annual Process Plant Grades and Recoveries

| | | Description | Units | TOTAL | Year -3 | Year -2 | Year -1 | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 | Year 11 | Year 12 | Year 13 | Year 14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Process Plant Summary | | | | | | | | | | | | | | | | | | | | | |
| | Direct Leach | Tonnage | Kt | 43,356 | 0 | 0 | 1,496 | 1,752 | 1,752 | 438 | 657 | 876 | 1,095 | 1,095 | 876 | 876 | 876 | 876 | 438 | 219 | 0 |
| | | Silver Grade | g/t | 91.5 | 0.0 | 0.0 | 110.7 | 111.1 | 154.0 | 125.8 | 152.0 | 169.5 | 172.3 | 158.3 | 130.7 | 119.2 | 148.8 | 134.5 | 110.3 | 109.6 | 0.0 |
| | | Lead Grade | % | 0.17% | 0.00% | 0.00% | 0.20% | 0.29% | 0.30% | 0.31% | 0.19% | 0.11% | 0.11% | 0.16% | 0.17% | 0.17% | 0.12% | 0.14% | 0.12% | 0.16% | 0.00% |
| | | Zinc Grade | % | 0.42% | 0.00% | 0.00% | 0.08% | 0.30% | 0.48% | 0.60% | 0.52% | 0.40% | 0.26% | 0.40% | 0.41% | 0.43% | 0.71% | 0.69% | 0.52% | 0.60% | 0.00% |
| | | Silver Recovery | % | 55.53% | 0.00% | 0.00% | 55.34% | 62.76% | 61.99% | 69.46% | 53.36% | 59.86% | 54.55% | 59.68% | 60.15% | 63.13% | 61.52% | 65.90% | 65.80% | 70.23% | 0.00% |
| | | Lead Recovery | % | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| | | Zinc Recovery | % | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| | Flotation/Leach | Tonnage | Kt | 113,234 | 0 | 0 | 572 | 3,322 | 3,504 | 5,256 | 4,964 | 4,672 | 4,380 | 4,380 | 4,672 | 4,672 | 4,672 | 4,672 | 5,256 | 5,548 | 5,840 |
| | | Silver Grade | g/t | 96.5 | 0.0 | 0.0 | 199.3 | 134.0 | 108.7 | 101.0 | 111.7 | 130.8 | 115.7 | 121.9 | 110.5 | 104.8 | 108.9 | 127.7 | 93.8 | 92.9 | 75.9 |
| | | Lead Grade | % | 0.34% | 0.00% | 0.00% | 0.10% | 0.18% | 0.40% | 0.33% | 0.22% | 0.23% | 0.19% | 0.22% | 0.25% | 0.36% | 0.24% | 0.42% | 0.54% | 0.44% | 0.31% |
| | | Zinc Grade | % | 0.93% | 0.00% | 0.00% | 0.15% | 0.25% | 0.62% | 0.70% | 0.56% | 0.54% | 0.53% | 0.49% | 0.52% | 0.80% | 0.67% | 1.16% | 1.07% | 1.70% | 1.23% |
| | | Silver Recovery | % | 74.77% | 0.00% | 0.00% | 55.91% | 61.28% | 62.25% | 68.34% | 60.75% | 68.39% | 66.46% | 70.35% | 74.68% | 83.30% | 78.95% | 86.75% | 85.59% | 84.52% | 78.11% |
| | | Lead Recovery | % | 68.37% | 0.00% | 0.00% | 28.90% | 42.71% | 40.04% | 56.52% | 51.72% | 65.56% | 59.35% | 62.72% | 65.02% | 71.82% | 64.51% | 76.34% | 79.32% | 76.21% | 67.35% |
| | | Zinc Recovery | % | 71.94% | 0.00% | 0.00% | 31.72% | 39.23% | 44.79% | 53.87% | 48.17% | 56.62% | 55.84% | 53.92% | 53.71% | 67.98% | 61.25% | 75.85% | 79.28% | 82.68% | 76.54% |
| | Total | Tonnage | Kt | 156,590 | 0 | 0 | 2,067 | 5,074 | 5,256 | 5,694 | 5,621 | 5,548 | 5,475 | 5,475 | 5,548 | 5,548 | 5,548 | 5,548 | 5,694 | 5,767 | 5,840 |
| | | Silver Grade | g/t | 95.1 | 0.0 | 0.0 | 135.2 | 126.1 | 123.8 | 102.9 | 116.4 | 136.9 | 127.0 | 129.2 | 113.7 | 107.1 | 115.2 | 128.8 | 95.0 | 93.5 | 75.9 |
| | | Lead Grade | % | 0.29% | 0 | 0 | 0.17% | 0.22% | 0.37% | 0.33% | 0.21% | 0.21% | 0.17% | 0.21% | 0.24% | 0.33% | 0.22% | 0.38% | 0.51% | 0.43% | 0.31% |
| | | Zinc Grade | % | 0.79% | 0 | 0 | 0.10% | 0.27% | 0.57% | 0.69% | 0.56% | 0.52% | 0.48% | 0.47% | 0.51% | 0.75% | 0.68% | 1.08% | 1.03% | 1.66% | 1.23% |
| | | Silver Recovery | % | 69.65% | 0.00% | 0.00% | 55.57% | 61.73% | 62.14% | 68.45% | 59.62% | 66.72% | 63.23% | 67.74% | 72.04% | 79.76% | 75.40% | 83.31% | 83.82% | 83.89% | 78.11% |
| | | Lead Recovery | % | 57.42% | 0.00% | 0.00% | 4.60% | 23.06% | 29.17% | 52.45% | 46.32% | 60.26% | 51.72% | 53.17% | 57.75% | 65.95% | 58.92% | 71.86% | 77.83% | 75.13% | 67.35% |
| | | Zinc Recovery | % | 61.32% | 0.00% | 0.00% | 13.01% | 24.16% | 32.26% | 50.26% | 42.92% | 49.75% | 49.77% | 44.80% | 46.85% | 61.78% | 51.12% | 68.21% | 76.19% | 81.54% | 76.54% |

| | | Description | Units | TOTAL | Year 15 | Year 16 | Year 17 | Year 18 | Year 19 | Year 20 | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 | Year 30 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| MillFeed Summary | | | | | | | | | | | | | | | | | | | | |
| | Direct Leach | Tonnage | Kt | 43,356 | 0 | 0 | 0 | 3,504 | 4,380 | 4,380 | 4,380 | 0 | 0 | 0 | 0 | 745 | 4,380 | 4,380 | 3,886 | 0 |
| | | Silver Grade | g/t | 91.5 | 0.0 | 0.0 | 0.0 | 81.9 | 79.6 | 79.6 | 75.6 | 0.0 | 0.0 | 0.0 | 0.0 | 60.9 | 60.9 | 60.9 | 60.9 | 0.0 |
| | | Lead Grade | % | 0.17% | 0.00% | 0.00% | 0.00% | 0.17% | 0.17% | 0.17% | 0.16% | 0.00% | 0.00% | 0.00% | 0.00% | 0.15% | 0.15% | 0.15% | 0.15% | 0.00% |
| | | Zinc Grade | % | 0.42% | 0.00% | 0.00% | 0.00% | 0.47% | 0.45% | 0.45% | 0.44% | 0.00% | 0.00% | 0.00% | 0.00% | 0.40% | 0.40% | 0.40% | 0.40% | 0.00% |
| | | Silver Recovery | % | 55.53% | 0.00% | 0.00% | 0.00% | 56.28% | 55.06% | 55.06% | 53.40% | 0.00% | 0.00% | 0.00% | 0.00% | 45.48% | 45.48% | 45.48% | 45.48% | 0.00% |
| | | Lead Recovery | % | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| | | Zinc Recovery | % | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% |
| | Flotation/Leach | Tonnage | Kt | 113,234 | 5,840 | 5,840 | 5,840 | 1,168 | 0 | 0 | 0 | 5,840 | 5,840 | 5,840 | 5,840 | 4,804 | 0 | 0 | 0 | 0 |
| | | Silver Grade | g/t | 96.5 | 136.7 | 152.3 | 72.9 | 51.5 | 0.0 | 0.0 | 0.0 | 53.6 | 53.6 | 53.6 | 53.6 | 53.6 | 0.0 | 0.0 | 0.0 | 0.0 |
| | | Lead Grade | % | 0.34% | 0.78% | 0.78% | 0.54% | 0.29% | 0.00% | 0.00% | 0.00% | 0.19% | 0.19% | 0.19% | 0.19% | 0.19% | 0.00% | 0.00% | 0.00% | 0.00% |
| | | Zinc Grade | % | 0.93% | 2.80% | 2.04% | 1.32% | 0.85% | 0.00% | 0.00% | 0.00% | 0.52% | 0.52% | 0.52% | 0.52% | 0.52% | 0.00% | 0.00% | 0.00% | 0.00% |
| | | Silver Recovery | % | 74.77% | 90.60% | 91.38% | 86.64% | 68.86% | 0.00% | 0.00% | 0.00% | 59.67% | 59.67% | 59.67% | 59.67% | 59.67% | 0.00% | 0.00% | 0.00% | 0.00% |
| | | Lead Recovery | % | 68.37% | 82.89% | 83.08% | 78.81% | 65.18% | 0.00% | 0.00% | 0.00% | 46.17% | 46.17% | 46.17% | 46.17% | 46.26% | 0.00% | 0.00% | 0.00% | 0.00% |
| | | Zinc Recovery | % | 71.94% | 88.52% | 87.03% | 84.25% | 72.81% | 0.00% | 0.00% | 0.00% | 50.84% | 50.84% | 50.84% | 50.84% | 50.87% | 0.00% | 0.00% | 0.00% | 0.00% |
| | Total | Tonnage | Kt | 156,590 | 5,840 | 5,840 | 5,840 | 4,672 | 4,380 | 4,380 | 4,380 | 5,840 | 5,840 | 5,840 | 5,840 | 5,549 | 4,380 | 4,380 | 3,886 | 0 |
| | | Silver Grade | g/t | 95.1 | 136.7 | 152.3 | 72.9 | 74.3 | 79.6 | 79.6 | 75.6 | 53.6 | 53.6 | 53.6 | 53.6 | 54.6 | 60.9 | 60.9 | 60.9 | 0.0 |
| | | Lead Grade | % | 0.29% | 0.78% | 0.78% | 0.54% | 0.20% | 0.17% | 0.17% | 0.16% | 0.19% | 0.19% | 0.19% | 0.19% | 0.18% | 0.15% | 0.15% | 0.15% | 0.00% |
| | | Zinc Grade | % | 0.79% | 2.80% | 2.04% | 1.32% | 0.56% | 0.45% | 0.45% | 0.44% | 0.52% | 0.52% | 0.52% | 0.52% | 0.50% | 0.40% | 0.40% | 0.40% | 0.00% |
| | | Silver Recovery | % | 69.65% | 90.60% | 91.38% | 86.64% | 58.46% | 55.06% | 55.06% | 53.40% | 59.67% | 59.67% | 59.67% | 59.67% | 57.55% | 45.48% | 45.48% | 45.48% | 0.00% |
| | | Lead Recovery | % | 57.42% | 82.89% | 83.08% | 78.81% | 23.84% | 0.00% | 0.00% | 0.00% | 46.17% | 46.17% | 46.17% | 46.17% | 41.12% | 0.00% | 0.00% | 0.00% | 0.00% |
| | | Zinc Recovery | % | 61.32% | 88.52% | 87.03% | 84.25% | 27.37% | 0.00% | 0.00% | 0.00% | 50.84% | 50.84% | 50.84% | 50.84% | 45.41% | 0.00% | 0.00% | 0.00% | 0.00% |

# 23 ADJACENT PROPERTIES

Silver Standard is unaware of any adjacent properties to the Property that have publically disclosed information.

# 24 OTHER RELEVANT DATA AND INFORMATION

Silver Standard considers that the current Technical Report includes all relevant data and information such that the report is not misleading.

# 25 INTERPRETATION AND CONCLUSIONS

An updated Mineral Resources estimate has been prepared for the Pitarrilla deposit following the completion of Silver Standard's recent 2012 drilling program. This update includes all new drilling and assay information, available as of December 4, 2012. The Mineral Resources estimate is based on drill sample data of acceptable quality from a series of drilling programs conducted between 2003 and 2012. A combination of non-linear (Local Uniform Conditioning or LUC) and linear (Ordinary Kriging) estimation techniques were used to model the polymetallic mineralisation hosted in the deposit.

This Technical Report represents the most accurate interpretation of the Mineral Reserve available at the effective date of this report. The Mineral Resources and Mineral Reserve estimates generated from this new information are presented in Table 14-9, Table 14-10, and Table 15-6. The conversion of Mineral Resources to Mineral Reserve was made using industry-recognised methods of determining operational costs, capital costs, and plant performance. Thus, it is considered to be representative of actual and future operational conditions. This report has been prepared with the latest information regarding environmental and closure cost requirements and has indicated that future work is in progress.

Silver Standard is not aware of any significant risk or uncertainty that may materially affect the reliability or confidence in the Mineral Resources/Mineral Reserve estimates or projected economic outcomes.

Silver Standard is in the process of obtaining the remaining surface rights and rights-of-way required for the Project, and expects to complete this in 2013. All required surface rights are necessary prior to submitting the construction permit application.

The recommended development plan for Pitarrilla includes a large open cut mine with a flexible flotation and leach plant that would be capable of efficiently processing a significant portion of the resource. The mine would operate for 20 years, moving an average of 175,000 tonnes per day and mining a total of 157 million tonnes of ore over its life. The process plant would operate for 30 years including commissioning and start-up and would have primary crushing, SAG and Ball milling circuits with the capacity to process 12,000 tonnes per day of direct leach ore and 16,000 tonnes per day of flotation/leach ores. The direct leach ore would be treated in an agitated leach circuit, CCD thickening and Merrill Crowe refinery to produce silver doré. The flotation/leach ores would be processed in a two stage lead and zinc flotation circuit, with the tailings treated by agitated leaching for incremental recovery of silver. This circuit would produce lead and zinc concentrates along with incremental doré from the tailings leach. In the final 12 years of the operation, the plant would continue to process ore from stockpiles.

This mine would be one of the largest silver mines in Mexico, with production of 333 million ounces of silver, 582 million pounds of lead and 1,669 million pounds of zinc over a 32 year project life. Production of silver in the first 18 years will range from 5 to 26 million ounces per year, averaging 15 million ounces of silver per year.

Total initial capital totals $740.6 million over a three year period and sustaining capital totals $403.9 million. The project generates a cumulative, net income after-tax of US$2.015 billion over its 32-year life including three years of pre-production, generating an NPV(5% discount) of $737 million and an after-tax rate of return of 12.8%. The project has a 7.4 year payback.

Mineral Resources that exist below the Pitarrilla Feasibility open pit have previously been considered in underground mining scenarios, and may be evaluated at a future date.

The proposed project includes a budget for closure, which is sufficient to reclaim the mine and plant facilities in an environmentally sound and sustainable manner with suggested future land uses for agricultural and livestock grazing, similar to current land uses. The project would provide temporary employment for 1,500 to 2,000 construction workers and would generate up to 639 permanent jobs.

# 26 RECOMMENDATIONS

All work has been completed to support the Pitarrilla Feasibility Study (M3, 2012).

It is recommended that the EIA document be completed and submitted to SEMARNAT for environmental review. Formal application for a construction permit from SEMARNAT cannot be made until all land impacted by the project is under SSR control. The environmental review of the EIA document will expedite future consideration of the construction permit, when submitted, and is expected to reduce permitting risk. No significant costs remain on the application process.

Whilst the EIA process is underway, SSR will continue its community relations work, continue to investigate project financing alternatives, and advance a number of programs to reduce risk and advance critical infrastructure. These programs are described as follows:

**Process and Metallurgical Optimization**

- Complete a plant throughput variability study, using the tested grinding hardness by rock type.
- Complete a pilot plant campaign on the Transitional ore portion of the flotation/leach process type, using the combined flotation and leaching flow sheet.

**Geomechanical and Geotechnical Optimization**

The feasibility geomechanical report that supported the overall Pitarrilla Feasibility Study (M3, 2012) recommended some further studies be completed for detailed mine design, which included:

- Collect further data on the Smectite-Chlorite alteration and create a 3D model to confidently define the extent of rock that may be affected by strong alteration. Use this data to further refine the detailed mine design.
- Define the aspects of dynamic hydrogeological modelling with collection of specific pump water tests to confidently define the aspects of short term and local water ingress effects.
- Complete advanced numerical modelling of potential toppling failure modes to further assure confidence of slope angles in the pit sectors where this rock failure mode is indicated.
- Collect further data on later life dump footings for the NE waste dump.
- Continue investigation of ARD parameters and alternate mitigation strategies.

**Geological and Mining Optimization**

- Continue to define additional Measured Resources principally through data evaluation.
- Continue to evaluate risks and opportunities offered through mining equipment selection.

- Further examine mining schedules with finer detail focusing on the stockpile strategy.

**Infrastructure**

- Evaluate the 230 kV power line option once it has been provided by CFE.
- Continue to develop identified sources of water to meet the demand of the process plant.

The recommendations cover the period up to receipt of the EIA with a construction permit and concurrent completion of the risk reduction and opportunity programs. The cost of these optimization programs is estimated to be approximately $1 million in total.

Following a positive construction decision, the project would advance to detailed engineering and full project implementation as defined in the Pitarrilla Feasibility Study (M3, 2012), inclusive of refinements due to the latter risk reduction and opportunity programs. The recommendations in this Technical Report do not go beyond the construction decision.

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