Report to:

Reliant Ventures S.A.C.,
Silver Standard Resources Inc.,
as the Operator of the San Luis Project

Technical Report for the
SAN LUIS PROJECT FEASIBILITY STUDY

Ancash Department, Peru

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IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Reliant Ventures S.A.C. and Silver Standard Resources Inc. by Mine and Quarry Engineering Services, Inc. (“MQes”), R&R Engineering, Milne & Associates, Resource Modeling Inc, Resource Evaluation Inc., and Montgomery Watson Harzag Americas Inc. (the Authors). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Authors’ services, based upon: (i) information available at the time of preparation, (ii) data supplied by outside sources, and (iii) the assumptions conditions, and qualifications set for the in this report. This report is intended to be used Reliant Ventures S.A.C. and Silver Standard Resources Inc. subject to the terms and conditions of its contracts with the Authors. These contracts permit Reliant Ventures S.A.C. and Silver Standard Resources Inc. to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities laws, any other use of this report by any third party is at that party’s sole risk.

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3.1 INTRODUCTION AND BACKGROUND

The San Luis project is a greenfield exploration and development project owned by Reliant Ventures S.A.C., a joint venture formed between Silver Standard Resources Inc. ("SSR") and Esperanza Silver Corporation ("Esperanza"). SSR is the operating partner. A NI 43-101 compliant resource estimate in January, 2009 indicated the project contains approximately 484,000 tonnes (measured and indicated) of material with average gold and silver grades of 22.4g/t and 578.1g/t respectively.

This technical report summarizes the report entitled, *Feasibility Study – San Luis Project, Anchash Department, Peru,* which was commissioned to investigate the exploitation and processing of the Ayelén Vein system.

The subject of this feasibility study is the exploitation and processing of the Ayelén Vein system.

3.1.1 Project Location

The San Luis project is located in the Cordillera Negra of north-central Peru, 513 kilometers north-northwest of Lima and 113 kilometers east of the city of Casma. The project consists of forty mineral concessions covering an area of 33,438 hectares. Elevations range from 3,600masl at the San Luis project campsite to 4,850 masl at the summit of the nearby Cerro Huilcahuaín. The project location is shown in Figure 3-1.

Figure 3-1: Project Location

3.1.2 Project History

There are no reported historical mineral resources or documented production from the property prior to the discovery of the San Luis vein system in 2005 by the joint venture.

In 2006 and 2007 a systematic trenching and drilling program was carried out on the westernmost vein, known as the Ayelén Vein. During this time approximately 19,200 meters of drilling was completed in 108 diamond drill holes. Thirty four trenches were also completed.

In January 2009 Resource Modeling Incorporated ("RMI") and Resource Evaluation Incorporated ("REI") prepared an updated mineral resource estimate of the San Luis project, the results of which were publicly disclosed in the NI 43-101 Technical Report by Lechner and Earnest (2009).

There are a number of other precious and base metal occurrences in the region that are owned by other operators including Barrick Gold Corporation's Pierina gold mine which is situated approximately 25km east-southeast of the San Luis property.

3.1.3 Project Overview

The San Luis system is a volcanic hosted, low sulphidation, epithermal quartz, precious metals deposit. Gold occurs as electrum and silver is present as acanthite, electrum, and other silver sulphosalts.

The Ayelén vein is the better of the known vein structures. Trenching and diamond drilling have traced this structure along a strike of 340° to 345° for a length of over 720m, with down dip extensions of more than 325m. Surface mapping shows that the vein structure dips -75° to -85° west-southwesterly, but drilling results show that the controlling fault structure(s) dip vertically to -80° west-southwesterly. True thicknesses of individual vein segments vary from tens of centimeters to more than 10m, with an average width of 1.5m to 3.0m.

The feasibility study that is the focus of this Technical Report describes a plan to mine the Ayelén vein system at San Luis using overhand cut-and-fill mining methods at a rate of 417 tonnes per day (350 days per year). The mining voids will be backfilled with unconsolidated rock fill.

Ore will be stockpiled on surface before being crushed, ground and processed by a combination of gravity and cyanide leach operations at a rate of 400 tonnes per day (365 days per year).

Tailings will be pumped to a 667,000m^3 capacity Tailings Storage Facility (TSF) for impoundment. The TSF will include a rock-filled embankment, a double synthetic geomembrane liner system and a leak detection system between the membranes. An engineered underdrain will convey groundwater beneath the embankment and the geomembrane liner system.

3.1.4 Project Stakeholders

In addition to the joint venture owners of the project, the following list of project stakeholders have been identified:

- District Municipalities of Shupluy and Cochabamba.
- Governments of Shupluy and Cochabamba.
- Educational Institutions in the area of influence.
- Health care centres and stations in the area of influence.
- National Police Stations in the area of influence.
- Community of Ecash.
- Community of Cochabamba.
- Residents of Pueblo Viejo, Tambra, Miramar and Cochabamba.
- Religious Institutions in the area of influence.
- Community-based Organizations (Mothers' Club, Vaso de Leche Committee, APAFA).
- Ministry of Energy and Mines (MINEM).

Communications with these groups is ongoing through SSR's community relations group.

3.1.5 Project Access

The San Luis project site is accessible through a network of public roadways in the area, some of which currently support mining traffic. The main access to the project site is planned to be along public gravel roadways from the cities of Huaraz and Casma to the project's west gate. Although this access is a public road, part of it passes through property and controlled checkpoints at Barrick's Pierina mine.

A secondary access/egress point is also planned from the project site's east gate, however the public road in that area is relatively steep and large vehicles are not able to navigate its tight turns.

3.2 TECHNICAL SUMMARY

3.2.1 Geology and Mineral Resources

3.2.1.1 Regional and Local Geology and Mineralization

The San Luis property is situated within the Cordillera Negra terrain of the Peruvian Andes, which is characterized by sedimentary rocks (limestones, mudstones, shales, sandstones), quartzites, extrusive volcanic rocks (tuffs, ignimbrites, coarse pyroclastics, agglomerates, and lavas) and intrusive rocks (granodiorites, tonalites, andesites, dacites, and rhyolites). Recent surficial deposits in the region are mostly fluvial – glacial in character. Regional structure is related to two uplifts of the Andean belt that resulted in folding and faulting of basement rocks.

Local stratigraphy in the San Luis project area is dominated by flat-lying fine to coarse-grained andesites overlain by beds of interlayered andesite and volcanic breccia that strike northwesterly and dip from sub-horizontal to 30° southwest, which are the main hosts for the northwest to north-northwest-trending mineralized (low sulphidation) epithermal veins. Post-mineral faulting resulted in well developed vein breccia textures and tensional openings that allowed subsequent emplacement of rhyodacite dikes and later brecciation/displacement of both the dikes and veins. Vein gangue mineralization consists of quartz, chalcedony, calcite and minor adularia. Gold occurs as electrum and silver is present as acanthite, other silver sulphosalts, and electrum. Other sulphides include trace amounts of pyrite, chalcopyrite, galena and sphalerite. The veins display common epithermal textures. Ore "shoots" typically occur in dilational zones that are the result of a variety of local stresses, and often these stresses are repeated along the length of a vein structure, resulting in multiple ore-shoots. The two veins containing the mineralization that provides the mineral reserve are the Ayelén vein and the Inéz vein. These provide the basis for the feasibility study disclosed by this Technical Report.

3.2.1.2 Drill Hole and Trench Data

The assay data on which the mineral resource estimate for this technical report are based were generated from 136 diamond drill holes that produced predominantly HQ–diameter (63.5 mm) core. Most of the diamond core holes were drilled at steep angles to the east to provide acute intersection angles with the steep westerly-dipping veins. Core recovery in these holes overall was excellent, and core handling and sample chain of custody procedures were appropriate. In addition to the diamond core holes, a series of chip-channel samples were collected from surface trenches that were excavated on approximately 25-meter-spaced intervals along and perpendicular to the strike length of the outcropping Ayelén and Inéz veins.

Sample preparation and analysis was done by SGS del Perú S.A.C. in Lima, Peru (2006), and ALS Chemex Peru S.A. in Lima (2007). Both laboratories are accredited according to the ISO/IEC 172025 standard, which is specifically designed for Mineral Analysis Testing Laboratories. The sample preparation and analytical procedures used by these laboratories were appropriate for the type of epithermal gold-silver mineralization present in the San Luis deposit, and the sampling/assaying procedures were backed by acceptable quality assurance/quality control (QA/QC) protocols.

3.2.1.3 Composite Samples for Metallurgical Testing

The metallurgical composite samples taken to date from drill core adequately represent (both spatially and from a production time standpoint) the material scheduled for mining from the Ayelén deposit during the term of the Life of Mine Plan. The composites are valid for the metallurgical testwork planned by Mine and Quarry Engineering Services Inc. (MQes) in San Mateo, California.

3.2.1.4 Mineral Resource Estimation

Three-dimensional wireframes were constructed for both veins using lithology and precious metal grades. To the extent possible, the wireframes were also based on a geologic interpretation that was completed by SSR. A three pass inverse-distance cubed ($1/d^3$) interpolation plan was used to estimate block model gold and silver grades using one-meter-long composite data. Prior to compositing the sample data, raw gold and silver assays were capped based on a review of cumulative probability plots. Different capping limits were used for the Ayelén and Inéz veins due to differences in the distribution of metal for those veins.

A dynamic anisotropy search strategy was used that matched blocks and composites based on their distance from either the hanging wall or footwall contact of the main Ayelén vein. This approach resulted in a model that displays detailed high-grade mineralized shoots and intervening low-grade and/or waste selvages. A nominal number of composites were used to estimate block grades in order to minimize grade smearing. Bulk density values of 2.61g/cm^3 for vein material and 2.65g/cm^3 for all other lithologies were used for tabulating mineral resource tonnages.

The estimated blocks were classified into Measured, Indicated, and Inferred resource categories based on the distance of each block from data. Measured resources were only assigned to Ayelén blocks located within 15 meters of surface trench samples. Table 3-1 summarizes Mineral Resources for the San Luis project using a gold equivalent (AuEQV) cutoff grade of 6.0g/t. The resource estimate was developed based on gold and silver metal prices of US$600 and US$9.25 per troy ounce respectively

Table 3-1: Mineral Resource Summary at 6.0g/t AuEq Cutoff

Mineral Resource Category	Tonnes	Gold Grade g/t Au	Silver Grade g/t Ag	Contained Gold Ounces	Contained Silver Ounces
Measured	55,000	34.3	757.6	61,000	1,345,100
Indicated	429,000	20.8	555.0	287,000	7,658,200
Measured & Indicated	484,000	22.4	578.1	348,000	9,003,300
Inferred	20,000	5.6	270.1	3,600	174,900

3.2.2 Mineral Reserves

Resource and reserve grades were derived from a geologic block model provided by Resource Evaluation Inc. (REI) and Resource Modeling Inc. (RMI).

In the time between completion of the resource and reserve estimates, metal prices and operating costs changed due to market fluctuations. Along with metal prices of US$800/oz Au and US$12.50/oz Ag; metallurgical recoveries of 94.0% Au and 90.0% Ag and an operating cost of US$160.83/tonne of ore processed were used to arrive at a breakeven cutoff grade of 6.9g/t AuEq.

The stope shapes were originally designed to a breakeven cutoff grade of 6.0g/t AuEq based on preliminary estimates of operating costs. The final economic parameters result in a 6.9g/t AuEq breakeven cutoff grade. The difference in the reserve between the two cutoff grades is economically insignificant and accounts for 4.5% of the total reserve, well within the accuracy of a feasibility study. This is primarily due to the nature of the vein system which provides a distinct boundary between mineralized and nonmineralized material. Any below cutoff material in the mine plan is treated as planned dilution in that normal operating costs and block model grades are applied.

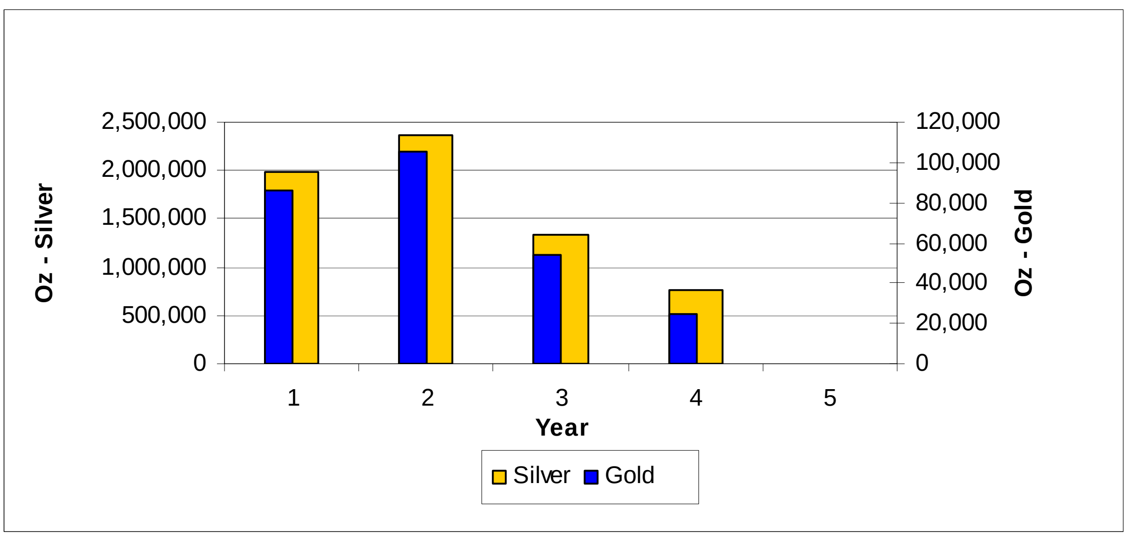
The fully diluted mineral reserve for the San Luis project has been calculated to be 503,313 tonnes of Proven and Probable material at average gold and silver grades of 17.95g/t and 446.14g/t respectively. The mineral reserve has been determined in accordance with CIM guidelines and is summarized in Table 3-2. Only measured and indicated resources as defined in National Instrument 43-101 were used to calculate the proven and probable mineral reserve.

Table 3-2: Mineral Reserve Summary

Material	Tonnes	Gold Grade g/t Au	Silver Grade g/t Ag	Contained Gold Ounces	Contained Silver Ounces
Proven	56,000	28.3	604.5	51,000	1,088,000
Probable	447,000	16.7	426.2	240,000	6,125,000
Total	503,000	18.0	446.1	291,000	7,213,000

After metallurgical processing recoveries, this reserve will yield an average of 78,000 ounces of gold and 1.86 million ounces of silver annually over its 3.5 year mine life. The life of mine metal production is shown in Figure 3-2.

Figure 3-2: Forecast Annual Metal Production



3.2.3 Mining Method and Mine Plan

Overhand cut-and-fill mining with unconsolidated rock fill has been chosen as the mining method due to the ore body width and dip as well as familiarity with this method in Peru.

A 10% factor for external dilution has been added to stope tonnages. Such dilution material is conservatively assumed to have no gold or silver content/value. Internal dilution is accounted for in the stope design at modeled grades. Mining recovery is considered to be 95%.

A mine production rate of 417 tonnes per day was chosen based on the ore body width, mining method, and appropriate mining productivities.

The development openings and support requirements are based on the preliminary mine geotechnical assessment of the Ayelén vein system made by Buenaventura Ingenieros S.A. ("BISA").

Ore will be drilled using pneumatically powered jacklegs and stoper drills before being blasted using ammonium nitrate and fuel oil (ANFO) explosives. Broken ore will be mucked to, and dropped down, a two compartment ore pass with captive electrically powered load-haul-dump (LHD) units. The ore will be retained in the ore pass by pneumatically operated chutes until being loaded into diesel powered haul trucks and hauled to the run of mine (ROM) ore stockpile at the surface.

Unconsolidated rock fill will provide support and working platforms for progressive mining cuts. The backfill material will primarily be quarried and screened from nearby colluvial deposits. The rock fill will be trucked to the mine and dumped into fill raises before being placed by captive electric LHDs. The fill floor will provide a base for work to begin on subsequent cuts.

Mining will occur on levels spaced 50 meters apart vertically. Five of the levels (4550, 4500, 4450, 4400, and 4350), will be accessed from surface portals while the lowest level, the 4300, will be accessed via a ramp from the 4350 level. The levels are named by nominal elevation of the first or lowest cut on the level. A 1.5m sill pillar will be left between each level to avoid breaking into the next level up.

All ventilation is designed to be mechanical, through primary surface fans, drawing clean air through the access drifts and exhausting through interior raises to the surface. Ventilation is designed to maintain airflow velocities in the main accesses to less than 350 meters per minute to avoid raising dust from the floors. The minimum ventilation requirements are estimated to be 4,476m^3/min. By utilizing the preproduction development fans for the main ventilation circuit, 5,947m^3/min of fresh air can be delivered to the mine.

Mine water drainage, estimated at 4.0l/s will gravity feed to the 4400 level through the raises and ditches and each level. Drainage water will be pumped from the 4300 level to the 4350 level prior to treatment and discharge from the mine workings.

Electricity will be distributed to the mine from the main substation at 4.16kV. Transformers located in key positions will lower the voltage to 480V for consumption in the mine by items such as fans, pumps, lighting and electric LHDs.

Two electrically powered 1,350cfm air compressors will provide compressed air for drilling. The high altitude of the project has been considered in sizing the air compressors, the power supply and all electric motors.

The mine will require 6 months of preproduction development prior to start of milling. An 8 month operational learning curve has been built in to the schedule to allow for training of the workforce and typical project startup issues/timing.

3.2.4 Process Metallurgy

Four metallurgical testwork programs have been completed investigating the metallurgical response of mineralized samples from the San Luis project. The first program was performed by Process Research Associates Ltd. ("PRA") of Richmond, British Colombia, Canada. The second program was performed at C.H. Plenge & CIA S.A. ("Plenge") in Lima, Peru. The third and fourth programs were performed by G&T Metallurgical Services Ltd. ("G&T") of Kamloops, British Colombia, Canada. In conjunction with the second G&T program (fourth program overall), a study including flocculant screening, sedimentation, rheology and filtration was undertaken by Pocock Industrial, Inc. of Salt Lake City, U.S.A. These programs investigated the recovery of gold and silver using gravity concentration, cyanide leaching and flotation, as well as defining mill design parameters. Petrographic analyses of core samples have been performed by Dr. Craig Leitch and BISA.

Comminution testwork indicates the Bond ball mill work index ranges between 9.5 and 16.5kWh/tonne for the samples tested. Crusher work indices range from 6 to 17.4kWh/tonne and abrasion indices range from as low as 0.026 to about 0.6.

Metallurgical samples identified as Ayelén 1 to 5 show low levels of sulphur, with most of the sulphur in sulphide form. Mercury, antimony and arsenic levels are also low. The samples are essentially comprised of pyrite and non-sulphide gangue. The non-sulphide gangue accounts for approximately 99.5% of the sample weight with about 0.5% pyrite making up the balance of the samples. At a primary grind sizing of about P_{80} = 50μm, the two-dimensional pyrite liberation ranges from about 36% to 53%. The unliberated pyrite is in binary form with non-sulphide gangue. On average, the binary particles were approximately 15% to 25% by weight pyrite. The liberation data suggests that it should be possible to produce a bulk sulphide concentrate with high pyrite recovery from these samples.

Testwork involving gravity concentration followed by cyanidation of composite samples showed total gold and silver recoveries improve at finer grinds down to P_{80}'s of approximately 74 microns. Grinds finer than this produced only marginal increases in gold and silver recoveries. Gold recoveries to the gravity concentrates improved at grinds finer than 80% passing 74 microns but gains in the gravity recoveries were balanced by lower extractions in related leach tests, resulting in little total recovery change. Silver recoveries to the gravity concentrate were low, 5% or less, at all grinds. Approximately 90% of the feed silver could be extracted during cyanide leaching.

Twenty four hour leach times produced lower and more variable recoveries than 72 hour leach times, especially for silver. Twenty four hour leach times extracted 65% to 77% of total gold and 57% to 92% of total silver, while 72 hour leach times extracted 71% to 77% of total gold and 76% to 94% of total silver. Five thousand ppm sodium cyanide is needed for high recoveries at 24 hour leach times, while 2,000ppm sodium cyanide produced high recoveries at 72 hour leach times. In both cases 0.5kg/t of $PbNO_3$ was also used.

Very high NaCN concentrations of 10,000ppm produced total gold and silver recoveries less than 0.5% higher than those produced by similar tests using 5,000ppm NaCN. These differences are within the testwork margin of uncertainty.

Tests with lead nitrate additions of 1.0kg/t and 0.5kg/t did not produce significant differences in metal recoveries.

A high grade rougher flotation concentrate containing 1,200 and 23,000g/t gold and silver respectively was produced from the Ayelén 1 sample. ADIS scans carried out on the bulk rougher concentrate and tailing revealed that the gold and silver occur mainly as liberated grains of gold and native silver in the rougher concentrate. The gold occurrences in the rougher tailing were also mainly liberated gold grains, but with an average diameter of 6µm. The silver occurrences in the rougher tailing were mainly as fine grained binary particles with goethite.

Sedimentation, rheology and filtration testing by Pocock showed that:

- Certain anionic and nonionic flocculants are effective settling aids with San Luis ore. A non-ionic, medium to high molecular weight flocculant was chosen for settling and filtration tests.
- Conventional thickener unit areas ranged from 0.186 – 0.299m^2/MTPD.
- High-rate thickener feed loading rates ranged from 2.6 – 4.2m^3/m^2hr. A rate of 2.6m^3/m^2hr was used by MQes as a design basis.
- Thickener underflow pulp densities of 65% or higher could be achieved.
- The composite representing the first three quarters of operation (Composite A) settled somewhat better than the intended medium grade composite (Composite C).
- Rheology studies showed acceptable slurry behaviour at pulp densities below 70% solids. Slurry pulp densities of 70% or more exhibited high viscosities and should be avoided.
- Filter cake moistures of approximately 20% could be achieved with vacuum filtration. Moistures of 15% could be achieved with pressure filtration.

Testwork to date indicates that grinding material to a P_{80} of approximately 74µm followed by gravity concentration and then cyanide leaching of the gravity tails provides the preferred alternative for recovery of gold and silver. Gravity pretreatment is beneficial as a low cost unit operation to remove coarse gold from the feed and thus improve leach kinetics. Cyanide leaching of gravity tails improves only slightly at grinds finer than 80% passing 74 microns. Tests using leaching times of 72 hours combined with P_{80} grinds of 74 microns typically produce gold recoveries of approximately 94% and silver recoveries of approximately 90%.

The process flow sheet selected for the San Luis project includes two-stage crushing and screening, fine ore storage, grinding and classification, gravity concentration, cyanide leaching, thickening and Counter Current Decantation (CCD) washing of leach discharge slurries. Leached tailings are discharged to a tailings storage facility near the process plant from which cyanide solution will be recycled to the process circuit. Pregnant leach solution will be deaerated prior to the injection of zinc dust to precipitate gold and silver. Precipitates will be collected in a filter press, then dried and smelted to produce doré bars for final refining offsite. The crushing and screening circuits as well as the smelting section will operate on day shift only. All other process facilities will operate 24 hours a day.

A geological model for the San Luis project has not been developed and consequently classification of mineralized material by rock type, geological domain, lithology etc. is not available. Indications from geological personnel are that mineralization type does not vary significantly within the resource. Testing of composites intended to represent high, average and low head grades showed little difference in total recoveries between the composite types. A composite representative of the first 3 quarters of the mine plan was also tested, with recoveries similar to those of the other three composites. Metallurgical testing has been carried out using industry-accepted procedures by reputable testing facilities. Although considerable effort was made to ensure the representivity of the samples with respect to the entire deposit, nevertheless it is possible that certain mineral assemblages were not identified or tested adequately.

3.2.5 Process Facilities

Gold and silver will be extracted from San Luis ore by a combination of gravity and cyanide leach processes. The plant will operate at a nominal capacity of 400 tonnes per day.

The process flow sheet includes two-stage crushing and screening, fine ore storage, grinding and classification, gravity concentration, cyanide leaching, thickening and CCD washing of leach discharge slurry. Leached tailings will be discharged to a tailings storage facility near the process plant with cyanide solution being recycled to the process circuit. Pregnant leach solution will be clarified and deaerated prior to the injection of zinc dust to precipitate gold and silver. Precipitates will be collected in a filter press, then dried and smelted to produce doré bars for final refining offsite. The crushing and screening circuits as well as smelting section will operate on day shift only. All other process facilities will operate 24 hours a day.

The processing criteria consider:

- The mining department will schedule ore deliveries and blend ore types to prevent plant throughput impacts from items such as over size rock at the primary crusher, plugging due to high moisture and excessive feed grade variations.
- A primary jaw crusher will reduce the run-of-mine (ROM) feed size to a P_{80} of 55mm.
- Secondary crushing will reduce the size of material to a nominal P_{80} of 10mm ($^3/_8$").
- Fine crushed material will be screened and fed to a fine ore bin. Material will discharge from the fine ore bin onto a belt feeder and then to a conveyor where lime is added before feeding into the ball mill. Cyanide solution will be added to the ball mill to control slurry density.
- Material will discharge from the ball mill and be cycloned. Cyclone underflow is fed to a rougher gravity concentrator. Gravity tails recycles back to the ball mill. Cyclone overflow is fed to the leach circuit.
- Gravity concentrate will be upgraded via cleaner tables, dried and fed to the smelting furnace. Cleaner table tails will be fed to the leach circuit.
- Gold and silver in the cyclone overflow will be leached using sodium cyanide solution. Leach discharge solution will be washed in a CCD circuit to produce a pregnant leach solution. The washed solids material will discharge to a Tailings Storage Facility (TSF).

- Gold and silver will be recovered from the pregnant leach solution using zinc dust in a Merrill-Crowe plant.
- Zinc precipitate will be dried, mixed with fluxes and smelted to produce silver-gold doré bars, the final product from the processing facility.

The processing circuit will operate as a closed circuit, with barren solution recycled. Plant streams will include pregnant solution, barren solution, process water and fresh water.

Reagents used in the process include sodium cyanide, lead nitrate, pebble lime, diatomaceous earth, zinc dust, flocculant and antiscalant. Storage, preparation and distribution of these reagents is included as part of the process plant.

Process control and monitoring is considered to be accomplished using a system of PLC's (Programmable Logic Controllers) interfaced with workstations.

3.2.6 Tailings Storage Facility

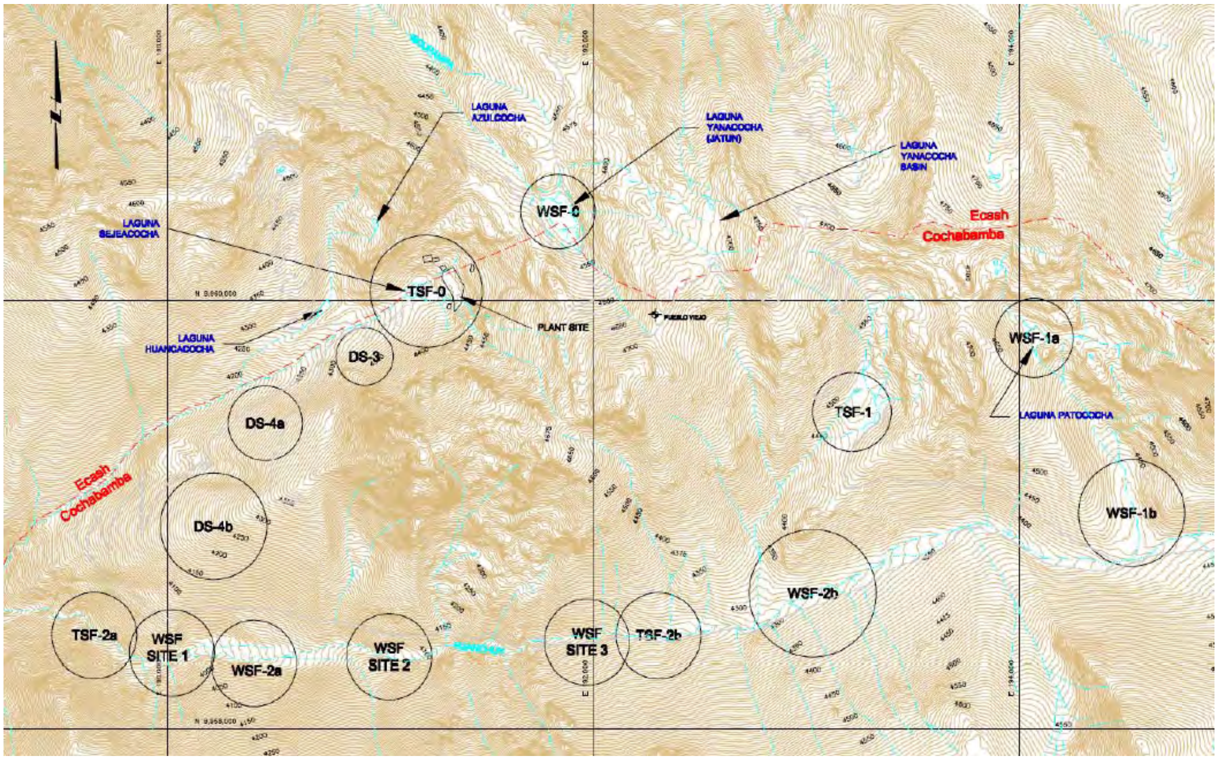
The design objective for the TSF is to store the tailings from the San Luis project mining and mineral processing operations in an environmentally responsible manner, both during operations and after mine closure. Based on this objective and applicable standards, the design criteria for the TSF included design for the Maximum Credible Earthquake (MCE) and Probable Maximum Precipitation (PMP) events, as well as international guidelines for environmental practice.

Based on the 2007 Canadian Dam Safety Guidelines criteria, the San Luis TSF dam was placed in the "High" to "Very High" Dam Classification based on the potential impact a failure of the dam could have on the population and the environment. The project design criteria were selected based on this classification.

A number of alternatives studies were performed prior to selecting the TSF, including a tailing deposition method alternatives study, a site alternatives study, and an embankment alternatives study. Based on the evaluation of technical, environmental, and social factors a TSF site adjacent to the process plant, TSF-0 site, was selected as the most suitable site for the development of the TSF. The TSF-1 site is considered the next best alternative but economically less attractive (see Figure 3-3).

Tailings storage is planned for the current mine plan of 503,313 tonnes. The density of the tailing was assumed to vary from 1t/m^3 at the start of the operations to 1.25t/m^3 at the completion of deposition. Assuming an average impounded tailing density of 1.25t/m^3, to accommodate the planned total production of approximately 500,000 tonnes, a storage capacity of 400,000m^3 would be required. Further, the embankment must have enough freeboard to contain the Probable Maximum Flood (PMF), which was conservatively estimated to be 211,000m^3 (excluding possible reduction of catchment volumes due to use of the diversion channels). This corresponds to a total capacity of 611,000m^3 and an ultimate embankment elevation of 4354m.

Figure 3-3: TSF and WSF Alternative Sites



The TSF is designed as a zoned rockfill embankment to be constructed using borrow materials from within the TSF impoundment and adjacent areas. The impoundment will be constructed using a double liner system with a leak detection system. A blanket drain will be constructed under the embankment to control groundwater flow from below the TSF. Diversion channels will be constructed around the perimeter of the TSF to control surface water. Due to the short mine life, the entire TSF embankment will be built during the capital construction phase. The impoundment will also be used to store start-up water prior to the start of mine operations.

The TSF was designed with a zoned embankment with a 3H:1V upstream slope and 2H:1V downstream slope. The upstream slope was selected based on liner installation requirements and the downstream slope was selected based on the results of the stability evaluations. The embankment crest width is 8m, the embankment crest elevation is 4354m and the maximum embankment height at the centerline is 23m.

For water management within the TSF, a floating barge and pump system is planned to recycle a certain quantity of water for re-use in the process plant. Tailings will be discharged by spigots from the embankment side of the TSF, to provide as long a tailings beach slope as possible, sloping downward from the west area of the TSF to the east corner of the TSF. The floating barge and pump system will be located in the area of ponded water in the east corner of the TSF (adjacent to the process plant).

Due to the anticipated cyanide concentrations in the process water, the area of ponded water in the TSF will be covered with appropriate netting or similar materials to prevent access of water fowl or other animals. Further, the impoundment area will also be enclosed by a fence to limit access.

The conceptual closure plans for the TSF consist of creation of a revegetated draining surface on the top of the tailings impoundment. A layered cover system will be placed to minimize infiltration into the TSF. Pipelines, pumps, lighting, the diversion channels and other facilities are planned to be removed. A post-closure spillway will be constructed to convey runoff from the top surface of the TSF and discharge into Quebrada Sejachocha, at the downstream toe of the embankment. The spillway will require erosion projection that will include riprap lining and grouted riprap in steeper areas.

3.2.7 Waste Rock Management

Although preliminary geochemical test results show that the mined materials are not expected to be acid generating, plans for management of waste rock and ore when brought to the surface include the following:

1. Use of waste rock for construction material on access roads and in the TSF dam.
2. Use of waste rock as underground mine backfill, if physically suitable.
3. Controlling runoff from pads, where ore and waste rock are stockpiled.

3.2.8 Power Supply

The project is estimated to require approximately 3.4MW of electrical power. Discussions with the regional power utility, Distriluz, indicated that the current grid capacity is fully utilized and surplus capacity is not available for the San Luis project. Further, although Distriluz is investigating potential capacity expansion projects, Distriluz concluded that they (or their local subsidiary, Hidrandina) could not assure provision of suitable, reliable power for the project. Accordingly, on-site diesel generation of power has been planned and allowed for in the project facilities.

The site power plant will consist of three 2,250kW diesel generators. Two generators will run at any given time with a third in either maintenance or standby mode. The project's high elevation has been considered in sizing the generators and other electrical components.

Electricity will be generated at 4.16kV and distributed at 13.8kV, in accordance with Peruvian and NEC standards. It will then be stepped down to the required voltage for each load being served, typically 480V or 220V.

3.2.9 Water Supply Structure (WSS)

3.2.9.1 Design Objective

The design objective of the WSS is to provide fresh water for the San Luis Project mining and mineral processing operations, while meeting downstream water release requirements. A number of alternatives were considered for the WSS. The Water Storage Facility-1 (WSF-1) site was selected as the preferred location after consideration of several locations and methods of water supply. Both a diversion structure and a water storage dam options were considered at this site.

Construction of a diversion structure was selected as a more practical and economic option considering the relatively small amount of required fresh water supply.

The proposed diversion structure on Quebrada Huanchuy will consist of a reinforced concrete weir wall, with upstream and downstream apron slabs, and reinforced concrete side walls. The diversion structure would be a low height concrete structure to provide a small pool of water for withdrawals via gravity flow to the pump forebay. The maximum height of the overflow weir wall would be 2m.

3.2.9.2 Other Infrastructure

Additional infrastructure items considered in this study include:

- On site access road construction and improvements to/from the mine, process plant, TSF, camp, security and connections with off-site roadways.
- Raw, fire, and potable water supply and distribution.
- Electrical power generation supply and distribution.
- Buildings and offices for administration, general site services, metallurgical laboratory, maintenance shop, warehouse, security and other structures.
- Ancillary facilities such as fuel storage, fire protection systems, and communications.
- Waste handling and sewage treatment systems.

3.2.10 Project Execution Schedule

The project execution plan for the San Luis project is summarized in Figure 3-4 on the following page. Key activities include the following durations:

Complete detailed engineering...10 months

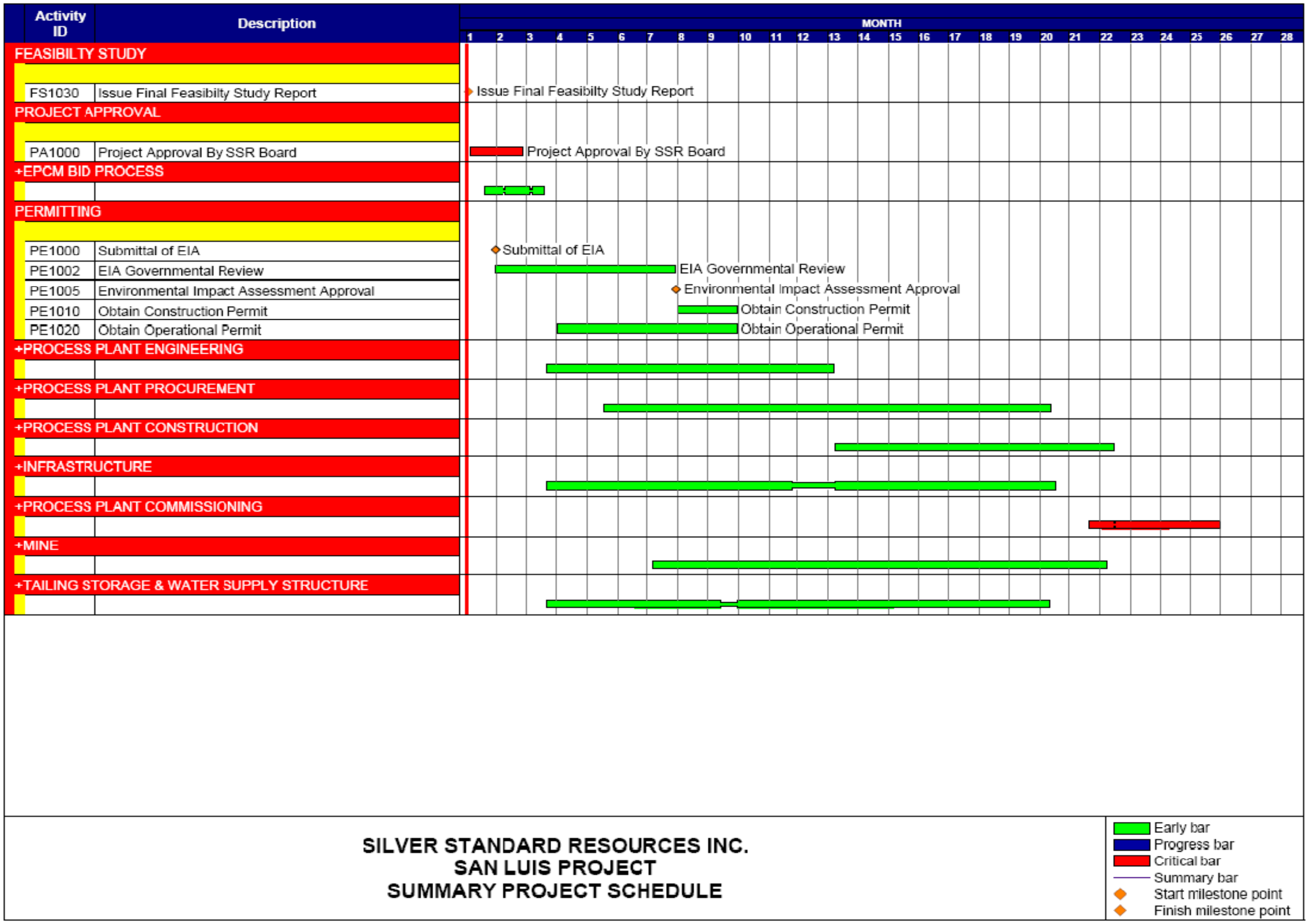
Mine development..6 months

Project construction..10 months

Plant commissioning ....3 months

The consolidated project schedule reflects a duration of 20 months from project approval to introduction of ore. Following that, commissioning and startup activities are planned to require an additional 3 months which will end with mechanical completion and handover. Subsequently, the owner will assume operation of the plant. With this in mind, the overall consolidated schedule is expected to require 25 months from the end of this feasibility study through to mechanical completion. The schedule is presented assuming that the project will advance seamlessly without delays, between the various phases.

Figure 3-4: Summary Project Execution Schedule



3.3 ENVIRONMENTAL AND PERMITTING

There are numerous Peruvian regulations applicable to the environmental, socioeconomic, archeological and cultural impacts of mining. These national and sector regulations focus on the protection, preservation and sustainable management of cultural features, natural resources, flora and fauna, as well as the application of air quality, water, and noise standards. In accordance with these requirements, an Environmental Impact Assessment (EIA) is being prepared by the joint venture.

At this writing, baseline sampling and monitoring is ongoing to support EIA completion. The parameters targeted in the baseline sampling program include:

- Meteorological data such as wind speed, precipitation, temperature, and humidity.
- Air quality data for PM_{10} particulates, SO_2, NO_2, CO, H_2S, and O_3.
- Ambient noise levels.
- Soil taxonomy.
- Surface water quality including physical, chemical, and biological properties.
- Flora and fauna surveys.
- Archeological assessments.
- Socioeconomic surveys.

Data from the ongoing baseline monitoring has been used with a variety of public and private climatic data to prepare the engineering design criteria for the project. This data will also be used to:

- Complete detail engineering for the project.
- To support completion of the environmental impact assessment.
- To support permit applications.

Cultural and socioeconomic data has been collected during the EIA and has been used to guide the Community and Stakeholder Relations Plans.

Environmental monitoring and analysis will continue through construction, operation and closure of the project.

The Conceptual Closure Plan for the project represents a preliminary plan describing the removal of the San Luis facilities and reclamation of the project site. Upon completion of the normal mine operations, the main closure activities will involve:

- Flushing and cleaning of processing vessels and pipe lines.
- Removal of fixed and mobile mining equipment, process equipment, electrical installations, steel structures, buildings and infrastructure facilities.

- Demolition and disposal on site of concrete structures.
- Reclamation and re-vegetation of affected project areas, including conditioning of slopes to prevent erosion.
- Sealing of mine portal and ventilation openings.
- Transfer of suitable mutually agreed infrastructure to local communities for their use and benefit.
- Residual process water evaporation from the TSF, followed by capping and re-vegetation of the TSF.

3.4 OCCUPATIONAL SAFETY AND HEALTH

In order to maintain a safe work environment, SSR will establish a site safety plan guided by its corporate safety philosophies. SSR will provide job task training to employees. Employees will then be required to demonstrate competency in the task before being assigned to the job.

Safety rules will be established by the Company and distributed through handbooks, bulletins and regularly scheduled safety meetings so that employees are aware of the rules and procedures. Adherence to company safety rules will be a condition of employment.

Emergency response teams comprised of project employees will be trained and equipped to handle a variety of emergencies. Some of these include underground mine rescue, chemical spill response, fire fighting and emergency medical response teams. Response teams will practice regularly and maintain a high state of readiness.

A clinic will be provided to provide basic first aid for employees. Employees needing further medical attention will be stabilized and then transported to an appropriate care facility. All persons attending the site will be subject to a medical assessment.

3.5 CAPITAL AND OPERATING COST ESTIMATES

The initial capital cost of the process plant and ancillary facilities to treat 400tpd of ore is estimated at US$90.4 million and is summarized in Table 3-3. All costs presented here are shown in first quarter 2010 US Funds, except for the mine components which are in third quarter 2009 US Funds.

Table 3-3: San Luis Project Capital Cost Estimate[1]

Area	Amount US$ (000's)	% of Total
Mine	8,123	9%
Process Plant & Infrastructure	50,984	56%
TSF and WSS	8,062	9%
Owner's Cost	11,972	13%
Contingency[2]	11,284	13%
Total	**90,426**	**100%**

Note: 1. Component contributions (including contingencies) for the mine, process & infrastructure, TSF & WSS, owner's costs were provided by Milne & Associates (with BISA support), MQes, MWH, and SSR respectively. 2. The overall contingency presented here is a combination of those provided by each contributor.

The operating cost for the San Luis project is estimated at US$160.83/t ore processed. The costs broken down into Mining, Processing, General and Administration are summarized in Table 3-4.

Table 3-4: Estimated Operating Cost Summary

Area	Average Annual Cost US$	Unit Cost (US$/t Ore)
Mine	8,613,713	59.00
Process	8,222,281	56.32
General and Administration	6,644,496	45.51
Total	**23,480,490**	**160.83**

3.6 MARKETING AND DOWNSTREAM COSTS

The San Luis project will produce doré bars containing gold, silver, and trace amounts of other non-marketable elements. The expected annual production of doré is approximately 211,000kg. The doré will be transported from site to a refinery located in Lima, Peru by a fully insured armored car service experienced in such work.

The refining process will be completed within 21 working days after receipt of doré. The refiner will charge "refining costs" to cover melting, assaying, refining and the production of bullion. The final charges will be negotiated between SSR and the refinery but they have been quoted at US$1.00/oz Au and US$0.50/oz Ag. Generally 99.8% to 99.95% of the gold contained in the bullion will be returned to SSR and the sale price will be fixed on the day of the refinery outrun with a settlement of 30 business days.

3.7 RISKS AND OPPORTUNITIES

This feasibility study is based on the best available data at the time of writing. However, certain reasonable assumptions and interpretations have been made to develop project concepts, designs and estimates With this in mind, certain risks and opportunities have been identified which may impact project capital cost, operating cost, metal recoveries, productivities and project economics. The following sections describe the main risks and opportunities in further detail.

3.7.1 General

3.7.1.1 Risks

Labor Availability - The availability of technical labor is a risk that can impact throughput and operating costs of the project. The mining and processing methods chosen for the project are conventional and widely used in Peru. An 8 month startup training curve has been built into the production schedule to mitigate this risk.

Political Risk - Political uncertainty associated with project stakeholders is a potential, but unlikely risk. While Silver Standard is working closely with all stakeholders, opposition to the project by one or several parties could delay construction and ultimately metal production from the project. An ongoing dialogue with all stakeholders and a Community Relations Plan will mitigate this risk.

Metal Price - Metal prices used in developing the resource estimate, reserve estimate and base case cash flow analysis are different. The effects of these price variations are uncertain.

The base case economic evaluation of $800 per ounce of gold is approximately 37% below the April 2010 spot price. The base silver price for the economic evaluation is $12.50, which is approximately 32% below the April 2010 spot price. This conservative approach minimizes the risk of changes in the metal price. A $100 change in the price per ounce of gold affects the IRR by approximately 7%.

3.7.1.2 Opportunities

Used Equipment - All new equipment has been assumed for the purposes of this study. Capital costs may be reduced if appropriately sized used equipment is available during project design. This opportunity will be explored during the procurement phase of the project.

Metal Sales - Forward metal sales could assure a higher return on investment, but this is not Silver Standard's practice. The metal prices chosen for this project are reasonable for an economic evaluation and are well below current spot prices.

Power Generation – The high cost of diesel generated power either through leasing or purchase indicates that further investigation into the local electrical utility, Hidrandina's capacities and infrastructure is warranted. Further use of water supply from Quebrada Huanchuy may offer the possibility of hydro-electric power generation as an alternate power source.

Toll Treatment – An alternative to onsite treatment of ore is to toll treat via a third party. This eliminates capital costs associated with the processing plant, tailings storage facility and associated failities. Identification of suitable third party operations may offer some benefits for processing of San Luis ore.

3.7.2 Geological

3.7.2.1 Risks

Geological Interpretation – The geologic continuity of the mineralization in the Ayelén and Inéz veins has been well documented on the surface, in the trenches and in vein outcrops. However, assumptions of the continuity of the mineralization at depth relies heavily on the drill hole intercepts (and the adequacy of the spacing of those intercepts), which in effect are "points in space". Until development of the mine opens up underground exposures (in raises and sublevels) of the Ayelén vein, a low to moderate risk exists that current assumptions of the continuity of the mineralization (which is based on hole-to-hole interpretations) are overly optimistic;

Mineral Resource Model – The validity of any mineral resource estimate is contingent on correct assumptions of data compositing, high grade outlier restrictions (capping or other limitation), estimation parameters (search distances and directions, composite selection) and resource classification criteria. For deposits like San Luis, which contain very high grade gold and silver, a risk always exists prior to development and production that individual high grade assays have exerted too great an influence in estimation of block grades in the feasibility study resource model. For the current San Luis mineral resource model, this risk is judged to be low to moderate;

3.7.2.2 Opportunities

Exploration Potential - A moderate opportunity exists for the discovery of additional mineable mineralization at depth in the Ayelén vein. Epithermal deposits hosted by volcanic rocks occasionally are found to be "telescoped", where a repeat of the precious metal (gold-silver) horizon occurs at depth. Because the deeper drill hole intercepts in the Ayelén vein did not intersect significant base metal (copper, lead, or zinc) mineralization, it does not appear that the bottom of the precious metal horizon in the Ayelén vein has been identified.

3.7.3 Mining

3.7.3.1 Risks

Stope Design – The layout and design of stopes is based on the grades of individual blocks in the resource model. If the estimation parameters have created an artificial "smoothing" of the block grades in the model (i.e., if the actual grades present are much more erratic than modeled), actual grade continuities may be worse than anticipated, internal dilution within the designed stopes could be higher than planned, and average production grades may fall short of plans and forecasts. For the San Luis model, this risk is considered low to moderate.

Mine Dewatering – Assumptions based on the geotechnical properties of drill core have been made with regards to the amount of water that will be encountered by mining excavations. The impact on operating cost of increased water flows should be minor given that the dewatering system is gravity fed from all but the two lowest levels.

Geotechnical Conditions – Geotechnical assessment of diamond drill core has been thorough, but there are no underground openings to check the validity of the interpretations. If ground conditions are worse than expected, capital costs and operating costs could increase due to extra ground support requirements. Less favourable geotechnical conditions in the ore and at the ore-waste contacts could increase dilution and decrease mining recoveries. The relatively narrow width of the mine openings at San Luis will mitigate some of this risk.

3.7.3.2 Opportunities

Surface Mining - A significant technical opportunity exists for mining the upper portions of the Ayelén and Inéz veins by open pit methods. Open pit mining would allow for the extraction and processing of lower grade and/or less continuous mineralization, and also local moderate to high grade portions of the deposit which are not of sufficient volume to support underground development. Social and environmental issues associated with open pit mining should be addressed before pursing this opportunity.

Pillar Recovery – The recovery of sill pillars has not been included in this mineral reserve estimate. There is an opportunity to extract some of these pillars at a reduced recovery late in the mine life.

Contract Development Mining – A tradeoff study comparing the cost of owner's crew preproduction development as presented in this study is economically favourable to contractor development costs. This opportunity has the potential of improving the project economics by delaying certain capital equipment purchases and higher preproduction productivities.

Contract Mining – As with the use of a contractor for preproduction development, there may be an opportunity to improve project economics by hiring a contractor to mine the ore as well as the waste. This would remove the need to purchase the majority of the mining equipment fleet, but increase operating costs. The operating cost increase could be at least partially offset by higher productivities.

Ventilation System – The main ventilation system has been designed to utilize the five auxiliary fans required for development as the main fans during operation. An operating vs. capital cost tradeoff should be completed to see if electrical cost savings can be made by purchasing one or two large main fans for the permanent ventilation system instead of using the five smaller fans.

Operating Learning Curve – An eight month production learning curve has been included in the schedule. If the operation can achieve full production sooner, then the throughput and cash flow will be improved.

3.7.4 Processing

3.7.4.1 Risks

Ore Variability – Metal recoveries may be negatively impacted if ore variability is greater than represented in the preliminary sampling programs. Process testwork to date has not indicated significant recovery differences between low, medium and high grade samples but orebody variation is insufficiently well defined to preclude mineralogical or other orebody influences.

Sample Oxidation – Sample composites used for metallurgical testing were not from freshly drilled core. Oxidation of the samples may have altered their metallurgical response.

3.7.4.2 Opportunities

Reagent Optimization - Lead nitrate dosages of 0.5kg/t produced recoveries as high as those produced by dosages of 1.0kg/t. Dosages less than 0.5kg/t should be investigated to determine the optimum amount of lead nitrate needed to assist in leaching.

Metal Recovery Improvements - Although tested recoveries of gold and silver are very high, due to the extremely high San Luis feed grades, final tailings still carry significant amounts of gold and silver, on the orders of 0.5 – 1.0g/t Au and 20 – 50g/t Ag. This is worth between $20/t - $40/t at metal prices of US$700/oz Au and US$11/oz Ag. The physical and chemical nature of these unrecovered values should be investigated by particle size analysis and a mineralogical study. Depending on the results of these programs, additional processing methods may be indicated for testing and possible inclusion in the process to enhance total recovery.

3.7.5 Tailings Storage Facility

The feasibility level TSF design is based on geologic mapping, test pit excavation and geotechnical and hydrogeologic drilling, review of rock core from the mineral exploration drilling program, laboratory testing and geophysics. The collected information was used to evaluate the engineering properties of foundation and subsurface conditions; selection of borrow materials; development of physical properties for rock and borrow materials; evaluate depth to groundwater; and characterize thickness of overburden materials. The risk and opportunity factors for the TSF are outlined below.

3.7.5.1 Risks

No significant risks have been identified related to the construction of the TSF at the proposed TSF-0 location.

3.7.5.2 Opportunities

Construction Material Preparation – Several components of the TSF dam require particle size limitations or specifications. For the estimated capital costs, the production of these materials is based on processing materials to produce the anticipated particle size for use in the dam. Assumptions were made for the required material processing based on limited laboratory test data. There is a potential that the borrow materials may require less processing, which may reduce the costs. This item is considered to present a minor opportunity.

Tailings Density – The results of the slurry consolidation testing indicated achievement of high tailing density over a short period of time. Due to concerns for the effect of flocculants on the results, a more conservative density was used in the design. Achievement of a higher tailing density would increase the capacity of the TSF beyond that shown in this report. The opportunity associated with this item is considered to be moderate.

3.7.6 Water Supply

3.7.6.1 Risks

Water Supply - Assumptions regarding fresh water supply for the project and the availability of water for supply to the project have been made. Based on natural variation and the short history of precipitation and streamflow records, there is an uncertainty for the reliability of Quebrada Huanchuy as a firm water supply source. This risk is considered to be moderate.

Water Permit - There is an uncertainty in obtaining the permits for diversion of water from the Quebrada Huanchuy for project use. Inability to secure sufficient fresh water supply from this source will require further investigation of other water supply alternatives. The risk associated with this is moderate.

3.7.7 Environmental Permitting

3.7.7.1 Risks

Environmental Permitting – The environmental permitting process is subject to ministerial, local community and public reviews. Potential delays could arise from a need to mitigate or address any concerns. Since such reviews involve an element of uncertainty, the project schedule has been developed based on recent experience with similar Peruvian projects. Delays to the permitting process may directly or indirectly result in increased capital costs and production delays.

Land Use Agreements – Land use agreements are required in the area of the project site to ensure surface access to the site. At this writing, such agreements are being discussed with representatives of several communities interfacing with the project site. Failure to reach a land use agreement with local communities could lead to project delays and possibly increased project costs.

3.7.8 Estimated and Actual Costs

3.7.8.1 Risks

Capital, Operating and Closure Costs - Capital and operating costs are subject to variations that may be due to a variety of influences, some of which are described within the body of this report. The contingencies developed and included in the cost estimate are intended to cover variations for in-scope items that have been identified. With this in mind, the capital cost estimate presented here is expected to be within the target accuracy of 15%.

Although the estimates presented here are made on a best efforts basis, cost increases could, however, be possible from unforeseen items that arise during the project which may be outside the current scope of the facilities and services envisioned in this report. Such increases could arise from factors beyond project control such as inflation, labor or political unrest, changes in legislation, strikes, force majeure, delays, weather conditions and others.

3.8 FINANCIAL ANALYSIS AND SENSITIVITIES

A base case cash flow model for the project has been developed using metal prices of US$800/oz gold and US$12.50/oz silver.

The metallurgical and refinery recoveries applied to the production plan are shown in Table 3-5.

Table 3-5: Metal Recoveries

	Recovery Factor	
Metal	**Plant**	**Refinery**
Gold	94.00%	99.00%
Silver	90.00%	99.50%

Additional off site costs included in the cash flow model include US$25,000 per year for transportation, security and refining at US$0.50/oz silver received, and US$1.00/oz of payable gold.

Depreciation on capital equipment has been taken at 20% per year, the maximum allowable rate under Peruvian law.

As required by Peruvian regulations, royalties are calculated at the rate of 1% of the first US$60 million of gross revenue and 2% of any gross revenue over US$60 million.

Depletion is not an allowable deduction under Peruvian tax laws and has not been included.

Given the short mine life and the feasibility study assumption that only new equipment will be purchased, salvage values of 25% on mining equipment and 12% of processing equipment have been assumed at the end of the mine life.

The Peruvian IGV tax is paid on various materials and services, but is refundable as a tax credit and has not been considered in the analysis.

An income tax rate of 30% has been included in the cash flow analysis.

100% equity financing has been assumed.

The base case cash flow analysis yielded the following economic indicators presented in Table 3-6.

An analysis of sensitivity to metal prices, capital costs, operating costs and metal production has been completed and is discussed in detail in Section 17 of this report.

Table 3-6: Base Case Economic Results

Economic Indicator	Units	Value
IRR		26.5%
NPV @ 5% discount rate	US$ x 1 million	39.2
Payback Period	Years	1.2
Cash Operating Cost (LOM)	US$ x 1 million	84.6
Total Operating Cost	**US$ per tonne ore**	**160.83**
Cost of Production	US$ per oz gold recovered	313.16

3.9 CONCLUSIONS

Conclusions and recommendations from this study are presented below.

3.9.1 Geology

The geology of the San Luis deposit (specifically the Ayelén and Inéz veins and the associated host rocks) is sufficiently understood to support the geologic interpretations that form the basis for the current mineral resource estimate.

The Mineral Resource estimate that forms the basis of the Mineral Reserves for this Feasibility Study is based on valid assay, specific gravity, lithologic, and alteration, and structural data. These data are supported by a sufficient QA/QC program, such that the resulting current Mineral Resource is a reasonable estimate of the in situ undiluted gold and silver mineralization present in the San Luis deposit, as it is defined by the existing drill holes.

The nine composite samples collected for the metallurgical testing completed as of the date of this Feasibility Report adequately represent (both spatially and from a production time standpoint) the material scheduled for mining from the Ayelén deposit during the term of the life of mine plan. Furthermore, these composites are valid for the metallurgical testwork performed by Mine and Quarry Engineering Services Inc. (MQes) and Process Research Associates Ltd. (PRA) that provided gold and silver recoveries that were used in tabulation of the Feasibility Study mineral reserves.

A significant portion of the Inéz vein currently is classified as Inferred mineral resources. SSR should assess the value of completing additional drilling on the Inéz vein in order to upgrade these Inferred resources to Measured or Indicated status. Based on drilling completed to date, the gold and silver grades within the Inéz vein are significantly lower and less consistent than those found in the Ayelén vein.

3.9.2 Mining

The mining method and extraction rate selected for the exploitation of the Ayelén vein is appropriate for the deposit’s width, dip and length. It is a proven mining method commonly used in Peru.

The productivities used to calculate the mine production rate are reasonable for narrow vein mining in Peru.

3.9.3 Processing

The process flowsheet to treat San Luis material has been developed based on results of metallurgical testwork to date. Additional metallurgical testwork will better define the metallurgical response and assess the effect of ore variability and optimize reagent schemes.

The average grade composite (Composite C) used in the second G&T test work program was intended to have gold and silver grades similar to those of the average life-of-mine, however, the resulting sample assays were higher than anticipated.

Drill core and assay reject material has been used in the preparation of metallurgical composite samples. None of this material has been from recently drilled material and is mostly from material that has been in storage (open to the environment) for in excess of 12 months. It has not been determined if oxidation had occurred with this material and, if it has, what the effect is on metallurgical response.

3.9.4 Tailings Storage Facility

The tailing storage facility adjacent to the processing plant (TSF-0) was selected as the primary tailing storage site with TSF-1 as an alternative. The TSF will provide storage for approximately 500,000 metric tons of tailing. The preferred deposition method for a conventional thickened tailing product will be spigotting from the embankment perimeter, with the objective to maintain the reclaim water pond in the southeast corner of the impoundment. No fatal flaws were identified that would prohibit the development of the project.

The preferred dam construction is a zoned rockfill embankment, to be constructed using borrow materials from within the TSF impoundment and adjacent areas. The impoundment will contain a liner system including a double geomembrane with a leak detection system. This liner system will cover the impoundment basin and upstream slope of the embankment. The intent of the liner system is to minimize seepage to groundwater. A blanket drain will be constructed under the impoundment to control groundwater flow from below the TSF.

Surface water diversion channels will be constructed around the TSF. These channels will collect and divert stormwater, thus limiting the amount of runoff that will be collected in the impoundment. During its operational phase, the TSF embankment will not have a spillway to avoid discharge of potentially contaminated water from the impoundment. Therefore, the TSF is conservatively designed to contain the runoff of the PMP from the entire catchment area above the TSF (excluding any reduction available from use of the surface water diversion channels).

Due to the relatively short project life and small TSF size, phased construction of the TSF was not considered. The TSF would be constructed in one phase during mine facilities construction, and would be completed just prior to the time the processing plant is commissioned. In order to optimize the construction process, the TSF and water diversion structure should be constructed during the dry period.

Assuming that construction of the TSF will be completed prior to the start of the wet season, an adequate amount of water for the start of the operations will be accumulated in the impoundment. The start-up water pond should not exceed 40,000m3. The water balance analysis was performed for three scenarios (wettest, driest and average years on record). The results of the water balance for the TSF using the wettest years on the record indicate that the reclaim water pond will grow with time. To maintain adequate freeboard during the later months of operation, measures to reduce the reclaim water pond volume may have to be taken early in the operational phase.

The overall strength values for the rock mass at the TSF embankment location are expected to represent favourable foundation conditions for the proposed design in terms of stability (for failure and sliding) and bearing capacity for the rockfill embankment. Further information on the foundation conditions and borrow materials is presented in the Geotechnical and Hydrogeologic Investigation & Engineering Report.

The results of the stability analyses indicate that the stability of the dam under static conditions is acceptable. Stability analyses indicate that the estimated factors of safety exceed the minimum required factors of safety for the embankment configuration with an upstream slope of 3H:1V and downstream slope of 2H:1V. Based on the simplified deformation analyses presented, the displacement of the embankment from anticipated seismic events is estimated to be less than 30cm. The available freeboard after this assumed deformation is considered adequate to preclude release of tailing due to a seismic event.

The results of static ABA tests on tailing and waste rock samples indicated no potential to generate acid rock drainage. From the kinetic HCT of the waste rock, it is reasonable to assume that it is unlikely that these samples will ever consume all of the available alkalinity and produce acidic leachate. Preliminary results for the tailing samples show similar results for the waste rock, with neutral to alkaline leachate with very low metals concentration. The results of the SPLP tests indicated that the maximum allowable concentrations are exceeded in 4 of the 5 tailings samples for cyanide, all of the tailings samples and one of the waste rock for antimony, all of the tailings samples for arsenic, one tailings sample for cadmium, and four tailings samples for lead. The aesthetic objective for aluminum was exceeded for all samples. Whole rock chemical analyses results indicate that all of the samples are elevated in most metals and metalloids (e.g., antimony, arsenic, copper, lead, molybdenum, selenium, and silver) relative to ranges normally found in average crustal rocks.

3.9.5 Water Supply Structure

A number of alternatives were considered for the WSS. The WSF-1 site was selected as the preferred location for the WSS. Both a diversion structure and a water storage dam were considered at this site. Constructing a diversion structure was selected as a more practical and economical option considering the relatively small amount of required fresh water supply.

3.9.6 Other Infrastructure

The infrastructure components required for the project are each proven processes and technologies.

3.9.7 Costs

The capital cost for the projects is estimated at US$90.4 million to a target accuracy of ±15%. The operating cost is estimated at US$160.83/t ore processed.

3.10 RECOMMENDATIONS

Recommendations for further work are presented in the Sections below.

3.10.1 Geology

SSR should consider exploration drilling of the Ayelén vein at depth, to check for a possible "telescoped" precious metal horizon. The success of the project as outlined in this technical report is not contingent on the success of further drilling.

3.10.2 Mining

An alternate plan to mine the near surface portion of the Ayelén vein from surface should be investigated. In addition to cost, the permitting and social perception implications must be considered for such an approach. The estimated cost for this investigation is $10,000.

Tradeoff studies should be completed to evaluate the opportunity to improve project economics through the use of mining contractors during the development and/or production phase of the mine. The estimated cost for these tradeoff studies is estimated to be $15,000.

Given the short mine life, the use of leased or used equipment should be investigated during the procurement process. The estimated cost for this investigation is $5,000.

The mine ventilation system should be modeled further to assess the advantages and disadvantages of using one or more large main fans instead of several smaller fans. The estimated cost for this investigation is $5,000.

A cost benefit analysis of diesel versus electric powered LHDs should be completed prior to the procurement process. This analysis should consider operating and capital costs, electrical costs, ventilation costs, and maintenance labor availability. The estimated cost for these tradeoff studies is estimated to be $15,000.

3.10.3 Processing

It is recommended a geological model for this project be developed. The combination of the geological model and additional metallurgical testwork needs to be aimed at defining the metallurgical characteristics of the project and establishing a geo-metallurgical model for the project. A budget cost for this metallurgical testwork is estimated to be $500,000.

It is recommended that optimization metallurgical tests be conducted on a life-of-mine composite that closely reflects deposit grades and characteristics. A budget cost for this metallurgical testwork is estimated to be $100,000.

It is recommended that viscosity/rheology testwork be performed on a range of samples, particularly on samples close to the surface where significant weathering may have occurred. A budget cost for this metallurgical testwork is estimated to be $50,000.

It is recommended that selection of composites for ongoing metallurgical testwork is coordinated with SSR's geological staff to ensure representative samples of the different zones and rock types are correctly chosen. It is recommended fresh core is drilled and used for ongoing metallurgical testwork. The testwork goal should be to develop a geo-metallurgical model of the deposit which can be used to reasonably predict metallurgical responses. As mine planning is refined metallurgical testwork samples need to be chosen that are representative of the mine plan and with an emphasis on the early period of production. These samples should be tested to confirm the metallurgical response is consistent with that predicted from the geo-metallurgical model. The cost for obtaining samples will involve determining how many samples and cost to drill them. This cost is undetermined at this point.

The estimated process operating costs for the project are US$56.32/t ore. These costs are based on metallurgical testwork along with budgetary quotations for reagents and consumables. It is recommended reagent and consumable consumption rates are further updated upon completion of additional testwork. The cost for this testwork is included in above estimate.

3.10.4 Tailings Storage Facility

Additional data from the neighboring Pierina meteorological station and the local station at the project site should be collected and analyzed to provide a more detailed precipitation database to be used for final design. The estimated budget for this recommendation is $50,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

The stream flows in the TSF area should be monitored and recorded to provide a better correlation between stream flows and precipitation rates. The estimated budget for this recommendation is $50,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

The material and water balance analyses, the predicted capacity and operating life of the facility, the rate of rise of the impoundment, and the predicted excess water volume should be updated as additional data pertaining to the actual conditions becomes available. The estimated budget for this recommendation is $20,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

Consideration should be given to the preferred method for controlling (reducing) the reclaim water pond volume if precipitation during the operation life of the TSF is similar to the wettest years on record. In this case, installation of a sprinkler system to enhance the rate of evaporation, water treatment or other alternatives should be considered. The estimated budget for this recommendation is $10,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

For final design, additional geotechnical data should be collected to validate the foundation conditions and borrow material at the TSF-0 site. A drill hole should be advanced in the area of the proposed portal and mine access pad to obtain data on soil and bedrock conditions in that area. Monthly water level measurements should be taken in the existing site piezometers. Additional piezometers should be installed on the ridges in selected areas around the mine area to confirm groundwater levels. The estimated budget for this recommendation is $160,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

The mine inflow estimates are steady state flows. The mine tunnel inflow estimates do not consider the possibility of intersecting high permeability fracture zones which may not be continuous, but may provide significant inflow. Numerical modeling based on available drilling information and measured groundwater levels is recommended to provide a better estimation of the mine inflows. The estimated budget for this recommendation is $60,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

During the next stage of the project, ground motion time histories should be developed based on the seismic hazard assessment. The ground motions would be used in the dynamic analysis of the TSF. A simplified deformation analysis was performed to evaluate potential embankment deformations due to the design seismic event. A more robust deformation analysis (e.g., equivalent linear or non-linear methods) is recommended in the next phase of the project to take into account the non-linear response of the foundation soils under large earthquakes. The estimated budget for this recommendation is $40,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

3.10.5 Water Supply Structure

The stream flows in the WSS area should be monitored and recorded to provide a better correlation between stream flows and precipitation rates. The estimated budget for this recommendation is $60,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

3.10.6 Environmental

On-going monitoring in support of the EIA should be continued to extend wet and dry season information. The estimated cost of the ongoing monitoring is $285,000. Successfully completing the monitoring is necessary to completion of the EIA report.

It is recommended that the Environmental Impact Assessment report be complete. The estimated cost to complete the report is $54,000. Successfully completing the EIA report is necessary to advancing the project to construction and operations.

It is recommended that the project permits applications be submitted. The estimated cost to complete the permitting process is $180,000. The project permits must be submitted to advance the project to construction and operations.

ITEM 4 • INTRODUCTION

4.1 INTRODUCTION

The San Luis project is a greenfield exploration and development project owned by Reliant Ventures S.A.C., a joint venture formed between SSR and Esperanza. This feasibility study was commissioned to evaluate the economic viability of the project to produce gold and silver. To this end, the study has been focused on:

- Developing preliminary facility designs.
- Estimating preliminary capital and operating costs.
- Estimating the mineral reserve.
- Investigating environmental and socio-economic aspects.
- Evaluating the project economics, risks and opportunities.

This report documents the work that has been performed to date and presents the conclusions/recommendations for further project development.

4.2 TERMS OF REFERENCE

The San Luis feasibility study was initiated in December 2008 and it was successfully completed in the first quarter of 2010 with the assistance of a variety of organizations forming the San Luis study team. For a list of contributors, see Table 4-1.

4.2.1 Currency and Foreign Exchange

All currency figures in this report are expressed in first quarter 2010 United States funds (US$) except for the mine components which are in third quarter 2009 US funds. Where amounts have been converted to US funds from other currencies, such conversions have been based on foreign exchange rates specified in the economic analysis.

4.2.2 Abbreviations and Units of Measure

Metric units of measure are used in this report, with the following exceptions:

- Metal production quantities and prices are per troy ounce (oz), as appropriate for gold and silver.
- Diameter sizes for piping are in inches, according to ANSI national pipe schedules.
- Capacities and corresponding unit rates for some fuels are customarily indicated in U.S. gallons, throughout Peru.
- Metal grades are reported in g/t. A conversion of 31.103 grams per Troy ounce has been used to convert grams to ounces.

It should be noted that all tonnages are expressed as metric tonnes, written as “tonnes” where 1 tonne equals 1,000kg (2,205lbs or 1.102 imperial tons).

4.2.3 Scope of Qualified Persons

A summary of the qualified persons (QPs) responsible for each section of this report is detailed in Table 4-1. Certificates of QPs are included in Appendix A.

Table 4-1: Scope of Qualified Persons

Organization	QP	Main Scope of Responsibity
Resource Evaluation Inc. (REI)	Donald Earnest	Mineral resource modeling, estimation and drilling validation (with RMI). Mr. Earnest is responsible all of Items 6.1, 6.4, 7.1, 7.2, 7.5, 8, 9, 10, 11, 12, 13, 14, 15 and portions of Items 3.1, 3.2, 3.7, 3.9, 3.10, 4, 6.2, 6.5, 7.4, 20, 21.1, 22.1, and 24, of the Technical Report.
Milne and Associates, Inc.	Steve Milne	Mr. Milne is responsible for editing and approving all of Items 3.2.2, 3.2.3, 3.7.3, 3.9.2, 3.10.2, 19.2, 21.2, 22.2, 25.1, 25.10.2, 25.10.7.1, 25.10.7.2, and 25.11.1 and portions of Items 3.5, 3.7.8, 4, 5, 20, 21.7, 24, 25.4.2.1, 25.4.3.4 and 25.10.1 of the Technical Report.
RR Engineering	Robert Michael Robb	Financial analysis, general study coordination review and report preparation. Mr. Robb is responsible for preparing all of Items 1, 2, 3.1.4, , 3.3, 3.4, 3.6, 3.7.1, 3.7.7, 3.8, 3.10.6, 22.6, 25.4.1.1, 25.4.16, 2.4.17, 25.4.18, 25.8, 23, 25.6, 25.7, 25.9, 25.10.6, 25.10.7, 25.10.8, 25.12 and portions of Items 3.5, 4, 20, 21.7, 24, 25.10.1, 25.10.8.9, and 25.11.3 of the Technical Report.
Mine and Quarry Engineering Services Inc. (MQes)	Christopher Kaye	Mill metallurgical testwork management, mineral processing facility design, power generation design, infrastructure design, cost estimation, project planning and report assembly. Mr. Kaye is responsible for the preparation of all of Items 3.2.4, 3.2.5, 3.2.8, 3.2.10, 3.7.4, 3.9.3, 3.9.6, , 3.10.3, 18, 21.3, 21.6, 22.3, 25.2, 25.4.1.2, 25.4.4, 25.4.5.1,25.4.5.2, 25.4.6, 25.4.7, 25.4.8, 25.4.9, 25.4.10, 25.4.11, 25.4.12, 25.4.13, 25.4.14, 25.4.15, 25.4.19, 25.10.3, 25.10.7.3, 25.10.7.4, 25.10.7.5, and 25.11.2, and portions of Items 3.5, 3.7.8, 3.9.7, 4, 5, 20, 21.7, 24, 25.4.2.1, 25.4.3.4, 25.5, 25.10.1, 25.11.3 of Technical Report.
Resource Modeling Inc. (RMI)	Michael Lechner	Mineral resource modeling, estimation and drilling validation (with REI). Mr. Lechner is responsible for for all of Items 16, 17, and 19.1 and portions of Items 3.2.1, 3.7.2, 3.9.1, 3.10.1, 4, 5, 20, 21.1, 22.1, and 24.
Montgomery Watson Harza (MWH) Americas Inc.	Clinton Strachan	Review of tailings acid-base accounting and characterization testwork, performing slurry consolidation testing, tailings storage facility (TSF) and water collection facility designs and cost estimate, and TSF conceptual closure planning. Geotechnical assessment of ground conditions for project surface facilities. Mr Strachan is responsible for the preparation of all of Items 3.2.6, 3.2.7, 3.2.9, 3.7.5, 3.7.6, 3.9.4, 3.9.5, 3.10.4, 3.10.5, 21.4, 21.5, 22.4, 22.5, 25.3, 25.4.2.2, 25.4.3, 25.10.7.6, 25.10.7.7, 25.10.7.8, 25.10.3 and 25.10.5 and portions of Items 3.5, 3.7.8, 4, 5, 20, 21.7, 24, 25.4.2.1, 25.4.3.4, and 25.10.1.

ITEM 5 • RELIANCE ON OTHER EXPERTS

Milne and Associates, Inc., Mine and Quarry Engineering Services Inc., Montgomery Watson Harza Americas Inc., Resource Evaluation Inc., Resource Modeling Inc. and RR Engineering (the Authors) have reviewed and analysed data provided by SSR and its consultants and have drawn their own conclusions there from, augmented by direct field examination by those participants who have visited the San Luis project site. The Authors have not carried out any independent exploration work, drilled any holes nor carried out any sampling and assaying.

While exercising all reasonable diligence in checking, confirming and testing it, the Authors have relied upon the data presented by SSR and its consultants in formulating their opinions.

The various agreements under which Reliant Ventures holds title to the mineral lands for these projects have not been investigated or confirmed by the Authors. The Authors were provided with a list of tenements by SSR and has relied upon the legal due diligence or title opinion conducted by the legal counsel, Garcia Saya Abogados, and referenced in the letter dated February 5, 2009 for SSR in conjunction with the transaction to confirm the validity of the mineral title claimed by SSR. The description of the property, and ownership thereof, as set out in this report, is provided for general information purposes only.

The Authors are pleased to acknowledge the helpful cooperation of Reliant Ventures and SSR management and staff, all of whom made any and all data requested available and responded openly and helpfully to all questions, queries and requests for material.

ITEM 6 • PROPERTY DESCRIPTION & LOCATION

6.1 PROPERTY LOCATION

The San Luis project is located in the Tambra Annex, the district of Shupluy, the province of Yungay and the department of Ancash, Peru, as shown in Figure 6-1.

Figure 6-1: Project Location

6.2 MINERAL RIGHTS

The project consists of forty mineral concessions covering an area of 33,438 hectares centered at Longitude 77º 47' West and Latitude 9° 23' South, U.T.M. coordinates 8,960,000m North by 195,000m East.

The property lies in the Cordillera Negra of north central Peru, approximately 500 kilometers north northwest of Lima and 100 kilometers east of the city of Casma. The project site is rural and sparsely populated, with approximately 490 inhabitants in each of the nearest towns (Tambra and Pueblo Viejo). Larger population centers near the project site include Casma, Carhuaz and Huaraz. These towns are sources of labour and services required for exploration and development of the project.

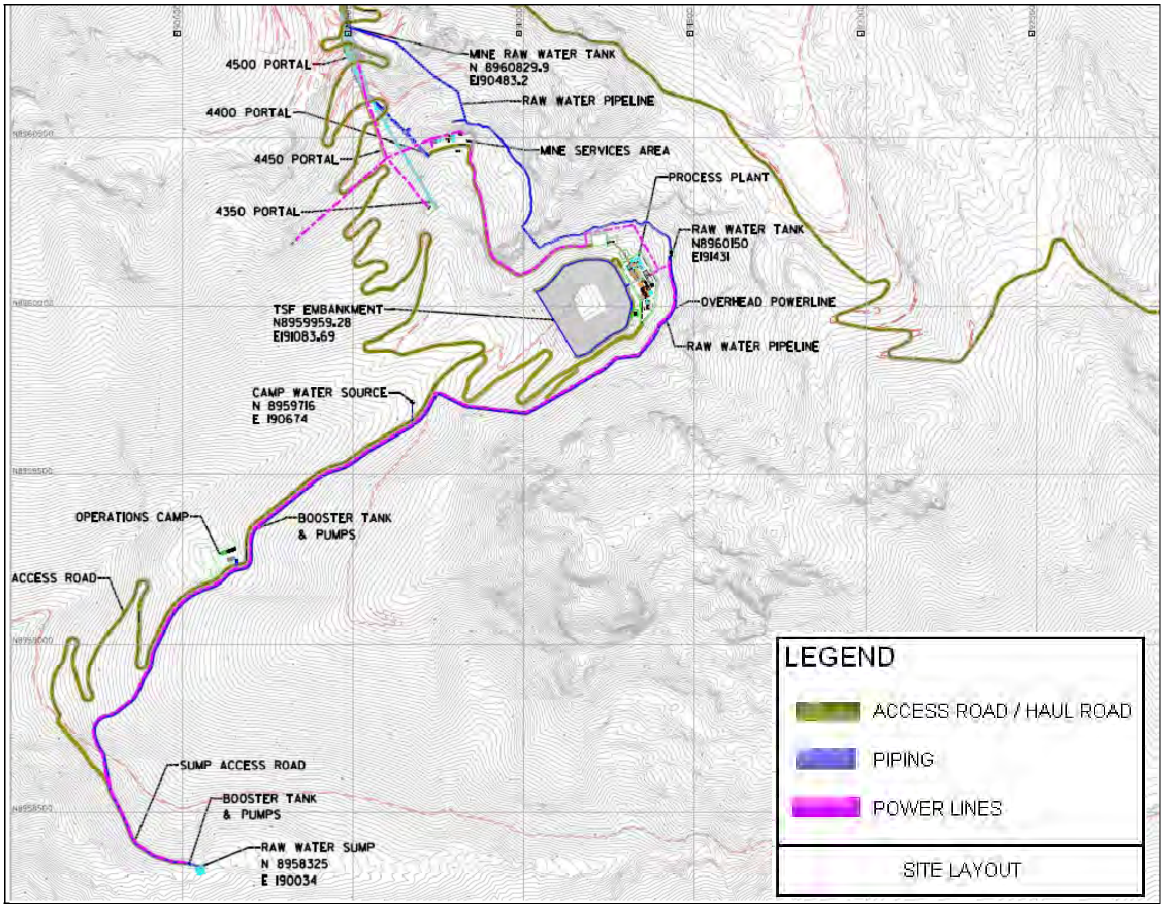
The mining target serving as the subject of this study consists of the Ayelén vein system.

6.3 PROJECT OVERVIEW AND BACKGROUND

This technical report summarizes the report titled Feasibility Study – San Luis Project, Anchash Department, Peru which was commissioned to evaluate the extraction and processing of a mineral resource owned by Reliant Ventures S.A.C. To this end, the current project examines the extraction of resources identified in the Ayelén vein. The main project facilities will be located, as shown in Figure 6-2, and will include:

- An underground mine complete with associated underground infrastructure and mobile equipment fleet.
- A run-of-mine ore stockpile.
- A primary crusher, consisting of a portable crusher, screen and associated conveying components.
- A mill containing facilities for grinding, gravity concentration, cyanide leaching, countercurrent washing/tails thickening, Merrill-Crowe precipitation, refinery, smelter and product storage.
- A tailings storage facility (TSF) adjacent to the mill, and including a short overland pipeline for tailings delivery/deposition.
- A water reclaim system and associated piping.
- Supporting infrastructure facilities, including:
  - A fresh water supply sump, diversion structure and distribution system.
  - On-site roadways.
  - Fuel storage and distribution.
  - Dedicated on-site power generation, distribution and area lighting facilities.
  - Water storage and distribution for plant water, potable water and firewater.
  - Potable water treatment.
  - Sewage treatment.
  - Solid waste disposal.
  - Fire protection.
  - Communication systems.
  - Site buildings (truck shop, mine rescue, mine dry/changehouse, laboratory, medical clinic, administration offices, gatehouses, dining facilities and an expanded existing operations camp).

Figure 6-2: Overall Project General Arrangement



Mining is planned to be executed using the mechanized cut and fill method. The preparation and development will include galleries, ore passes and raises to allow stoping on six levels.

Mineral processing will involve conventional gravity concentration followed by cyanide leaching of the gravity tails. Dissolved metal values will be recovered through Merrill-Crowe precipitation and smelting.

Tailings materials will be delivered from the counter-current washer/thickener circuit to a double-lined TSF. The impoundment area will include features for leak detection and return of leakage to the impoundment. Provision is also included to collect shallow ground water under the TSF in an underdrain system and convey it downstream of the facility.

Considering the availability and reliability of power in the local grid, on-site power generation is planned with diesel generators. Considering the short planned mine life, of approximately 3.5 years, access roads are planned to take maximum advantage of existing roadways that were initially developed to serve exploration purposes. Where modification or improvement of public roads is required, they are assumed to be completed by others using local resources.

Other key infrastructure for the project includes a variety of ancillary buildings, water treatment and sewage treatment facilities. The water and sewage treatment systems are planned as simple skid mounted systems. The buildings are planned as a combination of shipping container-style conversions and sprung structures.

Operating staff will be accommodated in the existing camp facilities with an additional 20 bed unit added to suit the maximum number of company and contractor staff envisioned on site.

6.4 PROJECT LOCATION AND SETTING

The project location is described in Section 6.1 and shown in Figure 6-1.

Geographically, the proposed project facilities would be located at a high altitude in very rugged terrain consisting of steep mountainsides and deep valleys. Specifically, the project site is on the western flank of the central area of the Cordillera Negra. The area’s topography is rugged and abrupt with mountain peaks/ridges having altitudes between 4,000 m and 4,900 masl. The main mine portal and the mineral process plant are planned to be located at an elevation of approximately 4,400 masl.

6.5 PROPERTY AND LAND OWNERSHIP STATUS

The mine property is formed by 40 contiguous concessions entitled to Reliant Ventures S.A.C. and covering a total of 33,438 hectares. These concessions differ slightly from those shown in the resource estimates and reports because, as of this writing, six further concession have been granted to Reliant Ventures S.A.C and they are included here.

The Ayelén Vein falls within the “EPZ Uno” concession which covers approximately 600 hectares of the mining concession. The complete set of mining concessions is shown in Figure 6.3 below and the legal status of each is summarized in Table 6-1.

Figure 6-3: San Luis Mining Concessions and Ownership

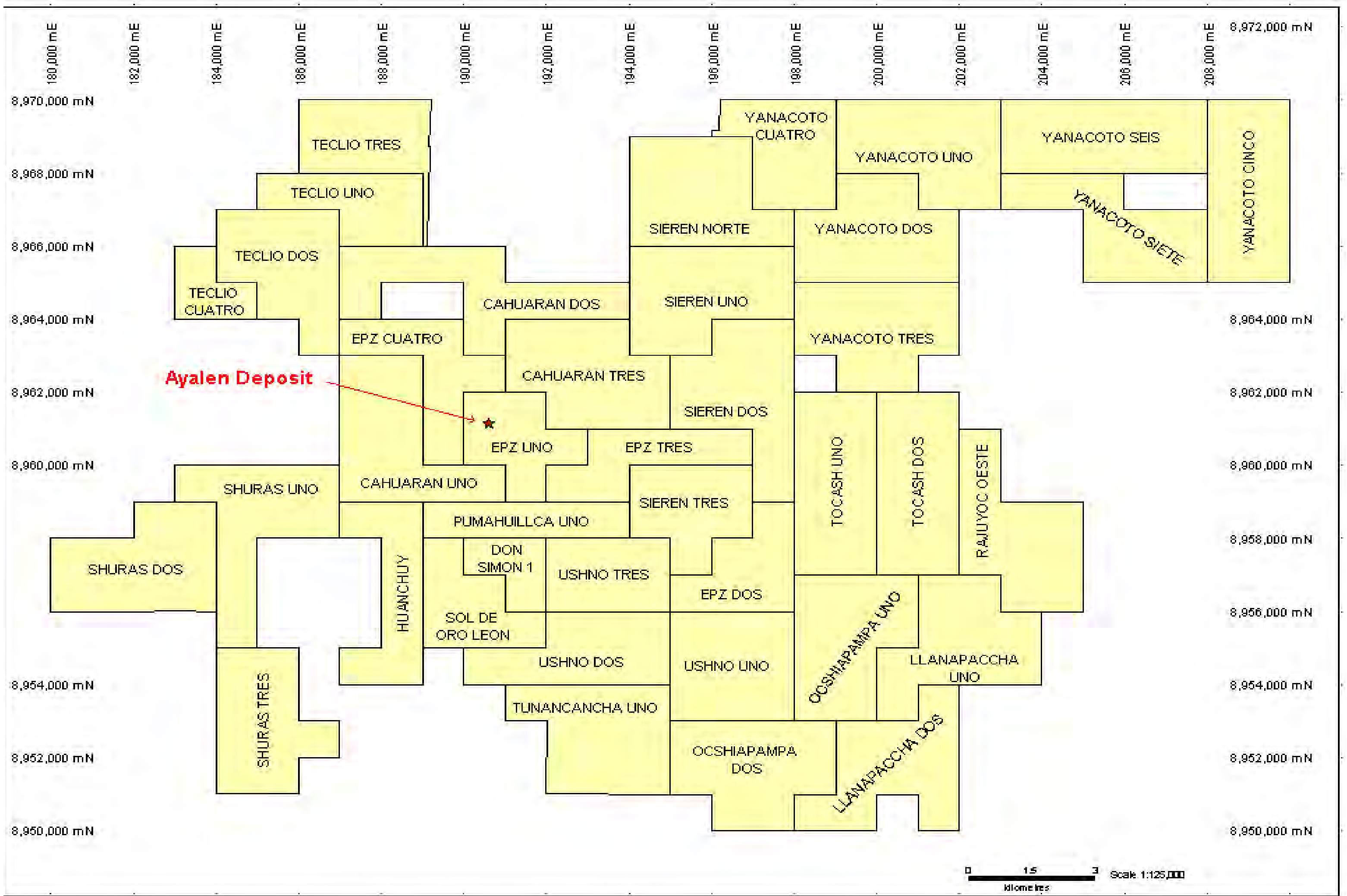


Table 6-1: Size, Ownership and Legal Status of San Luis Mineral Concessions

N°	Name	Code	Area (Ha)	Holder	Title N°	Status
1	CAHUARAN UNO	01-02658-05	1000	Reliant Ventures	4405-2005	Titled
2	CAHUARAN DOS	01-02656-05	882	Reliant Ventures	4409-2005	Titled
3	CAHUARAN TRES	01-02657-05	1000	Reliant Ventures	4534-2005	Titled
4	DON SIMON 1	01-00634-04	300	Reliant Ventures	01955-2004	Titled
5	EPZ UNO	01-01203-05	600	Reliant Ventures	3440-2005	Titled
6	EPZ DOS	01-01202-05	600	Reliant Ventures	3469-2005	Titled
7	EPZ TRES	01-01603-05	600	Reliant Ventures	3311-2005	Titled
8	EPZ CUATRO	01-02455-05	700	Reliant Ventures	4971-2005	Titled
9	HUANCHUY	01-02281-06	700	Reliant Ventures	4237-2006	Titled
10	LLANAPACCHA UNO	01-02355-06	1000	Reliant Ventures	3628-2006	Titled
11	LLANAPACCHA DOS	01-02356-06	1000	Reliant Ventures	3560-2006	Titled
12	OCHIAPAMPA UNO	01-02357-06	1000	Reliant Ventures	3645-2006	Titled
13	OCHIAPAMPA DOS	01-02358-06	1000	Reliant Ventures	3553-2006	Titled
14	PUMAHUILLCA UNO	01-02779-05	500	Reliant Ventures	1236-2006	Titled
15	RAJUYOC OESTE	01-02354-06	1000	Reliant Ventures	3468-2006	Titled
16	SHURAS UNO	01-01970-07	1000	Reliant Ventures	0315-2007	Titled
17	SHURAS DOS	01-01971-07	1000	Reliant Ventures	0388-2007	Titled
18	SHURAS TRES	01-02407-07	900	Reliant Ventures	0709-2007	Titled
19	SIEREN UNO	01-02782-05	1000	Reliant Ventures	5080-2005	Titled
20	SIEREN DOS	01-02785-05	1000	Reliant Ventures	04967-2005	Titled
21	SIEREN TRES	01-00014-06	700	Reliant Ventures	1317-2006	Titled
22	SIEREN NORTE	01-02284-06	992	Reliant Ventures	3690-2006	Titled
23	SOL DE ORO LEON	01-02495-05	600	Reliant Ventures	04550-2005	Titled
24	TECLIO UNO	01-02359-06	600	Reliant Ventures	4326-2006	Titled
25	TECLIO DOS	01-02360-06	900	Reliant Ventures	3521-2006	Titled
26	TECLIO TRES	01-02408-07	664	Reliant Ventures	1586-2007	Titled
27	TECLIO CUATRO	01-00651-08	300	Reliant Ventures	2702-2008	Titled
28	TOCASH UNO	01-02783-05	1000	Reliant Ventures	4964-2005	Titled
29	TOCASH DOS	01-02781-05	1000	Reliant Ventures	4717-2005	Titled
30	TUNANCANCHA UNO	01-02406-07	1000	Reliant Ventures	0895-2007	Titled
31	USHNO UNO	01-02784-05	900	Reliant Ventures	5077-2005	Titled
32	USHNO DOS	01-02780-05	800	Reliant Ventures	869-2006	Titled
33	USHNO TRES	01-00015-06	600	Reliant Ventures	3549-2006	Titled
34	YANACOTO UNO	01-02285-06	1000	Reliant Ventures	3570-2006	Titled
35	YANACOTO DOS	01-02283-06	1000	Reliant Ventures	3513-2006	Titled
36	YANACOTO TRES	01-02282-06	1000	Reliant Ventures	3530-2006	Titled
37	YANACOTO CUATRO	01-02410-07	700	Reliant Ventures	1336-2007	Titled
38	YANACOTO CINCO	01-01922-08	1000	Reliant Ventures	2532-2008	Titled
39	YANACOTO SEIS	01-01923-08	1000	Reliant Ventures	2613-2008	Titled
40	YANACOTO SIETE	01-01924-08	900	Reliant Ventures	4052-2008	Titled
Total Hectares			**33,438**			

6.6 ROYALTIES

Royalties are paid to the Peruvian government at the rate of 1% of the first US$60M of gross revenue and 2% of any gross revenue over US$60M.

6.7 ENVIRONMENTAL LIABILITIES

The only current environmental liabilities are those associated with the removal of the mine camp facilities and reclamation of drill pads and exploration roads.

6.8 AUTHORIZATIONS AND PERMITS

The following permits are required to build and operate the San Luis Project. To date, none of the permits listed have been acquired.

A number of government authorizations and permits are required for the San Luis project. These include:

- Certificate confirming absence of archeological remains (CIRA)
- Explosives
  - Mining operation certificate (MOC)/metallic and non-metallic mining operations
  - ANFO use authorization for underground explosives
  - Six-month-authorization for the use of explosives, goods and related items.
- Chemical goods of customs bonded products
  - IQPF's user's certificate (two years)
  - Registration in the control of chemical goods and customs bonded products
  - Opening authorization in special registries of chemical goods and customs bonded products
- EIA approval for mining
- Mining closure plan and mining environmental debts approval
- Fuel direct consumer
  - Favourable technical report of liquefied propane tank installation
  - Favourable technical report on liquefied propane use and fueling operations
  - Registration in the liquid fuels direct consumer registry and liquefied propane for fixed facilities
- Water use permit
  - Public health authorization of domestic waste water discharge
  - Public health authorization of industrial waste water discharge
  - Public health authorization of drinking water treatment

  - Water use license
- Benefit concession grant
- Mining plan and operations’ start
- Agreement with the community for the use of superficial land

ITEM 7 • ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

7.1 ACCESSIBILITY

The San Luis project site is accessible through a network of public roadways in the area, some of which currently support mining traffic. The main access to the project site is planned to be along public dirt and gravel roadways from the cities of Huaraz and Casma, to the project's west gate. Although this access is public, part of the road from Huaraz passes through property and controlled checkpoints at Barrick's Pierina mine. With this in mind, a short, alternate detour route may be evaluated to avoid potential delays.

A secondary access/egress point is also planned from the project site's east gate to Carhuaz, however the public road in that area is relatively steep and large vehicles are not able to navigate its tight turns.

7.2 CLIMATE

As stated by Pincus[1] and RMI & REI[2]: *"The project area is dry for the greater part of the year, from May to December. During these months the daytime temperature is moderate with a high of 22° Celsius but nights can be as cold as -10° C. The rainy season is from January to April. Temperatures are more moderate but the weather is characterized by sometimes heavy rains, abundant fog, hail and snow at higher elevations. Maximum precipitation is found in February (206mm) while the driest month is July (2.5mm)."*

Historical climatic data has been provided by the Peruvian government (SENAHMI) for the nearby weather stations at Pira and Recuay and by Barrick's Pierina operation, to facilitiate documentation of the project climatic conditions.

7.3 LOCAL RESOURCES

Existing water resources in the project area include several fresh water lagoons and some seasonal surface water. Some of the lagoons are currently used to supply water to the small adjacent communities of Tambra and Pueblo Viejo. This water is, however, subject to seasonal variations and anecdotal evidence indicates that water supplies could disappear during the dry season. As part of the project's EIA activity, baseline data is being collected to measure and observe the patterns of surface water collection and flow as well as the biology of these lagoons and other surface water in the area.

Specifically, four lagoons are located within the project's area of indirect influence. These include the Pacsococha and Orcuncocha lagoons, which drain to the Tocash gorge, as well as the Cotacocha and Yahuarcocha lagoons which drain their waters to the Huanchuy gorge. These lagoons could be used as a potential source of water supply if necessary. Several other lagoons are located within the project's area of direct influence and these include Yanacocha, Azulcocha

[1] "Technical Report on the San Luis Project, Ancash Department, Perú"; William J. Pincus; September 12, 2006.
[2] "Updated Mineral Resource Estimate San Luis Project, Ancash Department, Perú"; Resource Modeling, Inc. and Resource Evaluation, Inc.; January 9, 2009.

Baja, Huancacocha and Sejeacocha lagoons. Yanacocha lagoon drains towards the Iscupampa gorge while the others drain towards the Huanchuy gorge. The Sejeacocha lagoon is planned to be used as a location for the TSF.

The Tocash and Huanchuy gorges join with the Iscupampa gorge to the west of the project to form the Quellaycancha River. This in turn is a tributary of the Yaután River and the Yaután River sub-basin represents approximately 4% of the hydrographic basin of the Casma River[1].

With installation of a water diversion sump in Quebrada Huanchuy, it has been determined that an adequate water supply can be maintained year-round to support the needs of the project and the adjacent communities.

7.4 INFRASTRUCTURE

7.4.1 Regional Centers

The project site is rural and isolated. The nearest population centers are Tambra (located on the western limit of the property) and Pueblo Viejo (situated in the northeast portion of the project concessions). Both villages are located in the Shupluy District, Yungay Province, Ancash Department, and each has approximately 490 inhabitants. The larger population centers near the project are Casma, Carhuaz, and Huaraz. Each of these population centers have labor and services required to support development of a mining project.

7.4.2 Power Supply

Growth of power demand is generally outpacing growth of power generation in Peru and the San Luis project site lies in a region of the country that is currently operating at maximum capacity. Power curtailments for industrial users are not uncommon.

With this in mind, discussions were held with the regional power utility, Distriluz, to determine the availability of grid power for the San Luis project operations. Although Distriluz and its local subsidiary (Hidrandina) have applied for expansion projects that would allow San Luis to draw power from the grid, the utility is unable to offer any assurance that these projects will proceed.

Accordingly, the San Luis project has been estimated on the basis of using self-generated power from diesel generators.

7.5 PHYSIOGRAPHY

Physiographically, the area of study is located on the western flank of the Cordillera Negra, in the lithosolic edaphic region, which is part of the western mountain range (Cordillera Occidental) of Peru. Its morphology corresponds to abrupt gorges with low plains.

The region is mountainous with high elevation rolling hills and valleys surrounded by higher craggy snowcapped mountains and deep valleys, typical of the western side of the Peruvian Andes ranging in altitude between 4,000m and 4,900masl.

[1] "Memoria Descriptiva De La Delimitación Del Distrito De Riego Casma-Huarmey"; Inrena-Irh-Digech, 2005.

"*Vegetation is typical of these high elevations known as the Puna. It is composed principally of Stipa Ichu (grass) and various small bushes*"[1].

Animals such as llamas, sheep, goats, foxes and wild rabbits (vizcacha) as well as resident and migratory birds live within the property as observed by BISA during a site visit, October 20-24, 2008.

7.6 GEOTECHNICAL & SEISMICITY

The San Luis project is located in a high seismicity area dominated by earthquakes occurring along the Peru-Chile subduction zone, which has a history of producing very large earthquakes that could cause significant shaking at the project location.

A feasibility level seismic hazard assessment, considering both deterministic and probabilistic approaches, was developed as part of the present study.

The seismic hazard analysis is based on a seismotectonic model and source characterization of the region surrounding the proposed site. For the purpose of this study, the site region is defined as the area within approximately 200 km of the proposed TSF.

The seismotectonic model identifies three general seismic sources in the site region: subduction zone earthquakes on the Nazca-South America plate interface, earthquakes within the subducting Nazca plate, and shallow crustal earthquakes within the overriding South America plate. The source characterization provides input parameters for both a probabilistic seismic hazard analysis (PSHA) and deterministic seismic hazard analysis (DSHA) for the proposed San Luis tailings dam site. The analyses provide design recommendations by using results obtained from both the PSHA and DSHA.

7.6.1 General Seismicity

The seismicity of the northern region of Peru is associated with various recognized fault systems. Additionally, the Peruvian subduction zone is the source for the majority of the large earthquakes and tsunamis that have occurred in the region. The Peruvian Andes were formed as a result of the collision of the Nazca and South American Plates. The Nazca subduction zone is located about 100 to 200 km offshore of Peru to the west and generally runs parallel to the coastline.

The subduction of the Nazca Plate beneath the South American Plate is responsible for the Quaternary deformations in the western part of the country. Four Quaternary fault systems, based on the publication of Machare et al. (2003), were identified to be the most relevant to the San Luis project site. Based on information provided in Machare et al. (2003), characteristics of the four fault systems are summarized below.

Chaquilbamba Fault: The Chaquilbamba fault is located north of Chaquibamba, about 13km south-southeast of Cajatambo. The normal fault strikes N40°W±20° and dips SW with an approximate length of 16km and an estimated maximum vertical slip of 8 to 10m. The recurrence

[1] "Technical Report on the San Luis Project, Ancash Department, Perú"; William J. Pincus; September 12, 2006.

interval for the fault is unknown. The slip rate is likely less than 1mm/year. A 1937 earthquake is believed to have been caused by the Chaquilbamba fault, forming a 1.5km long rupture.

Eastern Boundary Fault: Located in a remote region of Peru, there is little information available for the potentially active fault that forms the eastern boundary of the Peruvian Andes. The fault was identified during aerial reconnaissance. No field reconnaissance or dating has been conducted and little is understood about the fault.

The normal fault is approximately 217km long with a strike of N50°W±15°; the dip of the fault is unknown. The recurrence interval for the Eastern Boundary fault is unknown. The slip rate is an estimated 1 to 5mm/year. Based on the proximity of the Cordillera Blanca fault, the last movement associated with the Eastern Boundary fault was likely in the late Quaternary.

Quiches Fault: The Quiches fault is located north of Ancash, between Quiches and Chindalgo, west of the Marañon River, north-northeast of Huaraz and northeast of the Western Cordillera. The normal fault is approximately 38.5km in length and has a strike of N45°W±15° and a dip of 42° - 58°.

The Quiches fault was the source of the 1946 Ancash earthquake (M 7.2), during which four distinct scarps were formed along a 20km long deformation zone. The Quiches fault has an estimated slip rate of less than 1mm/yr with an approximate maximum vertical displacement of 3.5m. The recurrence interval for the fault is estimated to be less than 12,500 ± 1,500 years.

Cordillera Blanca Fault Zone: The Cordillera Blanca fault zone is a major active fault system located in northern Peru, extending from Corongo through Chiquian between the Cordillera Negra and Cordillera Blanca. The normal fault system, striking N35°W±20° and dipping west, is approximately 210km in length with a vertical displacement of about 1mm/yr. Based on the geometry and structural features, the fault zone has been divided up into four segments (A through D) with lengths of 42.9km, 76.8km, 54.4km, and 32.4km, respectively, and with an unknown dip, and strikes of N17°W±28°, N36°W±21°, N40°W±15° and N38°W±18°, respectively. The last movement of the fault likely occurred during the Holocene or post-glacial era. The approximate recurrence interval is about 1,000 to 3,000 years.

7.6.2 Deterministic Analysis

The deterministic seismic hazard analysis (DSHA) can be described as a procedure of four steps (Kramer 1996):

1. Identification and characterization of earthquake sources capable of producing significant ground motions at the site.
2. Selection of the source to site distance parameter for each source zone, consistent with the attenuation relationship selected.
3. Selection of the controlling earthquake.
4. Hazard at the site is formally defined, in terms of ground motions produced at the site by the controlling earthquake.

Based on the deterministic analysis, two earthquake scenarios have been identified. The first scenario corresponds to events associated with megathrust events (M_w 9.2) with a PGA of 0.28g. The megathrust events appear to dominate the response for periods above 0.45 seconds.

The second scenario corresponds to events associated with a background source (Zone 4), which controls the response for periods of 0.45 seconds and below. The magnitude for this second scenario is M 6.0 with an associated PGA of 0.47g.

Based on the deterministic analysis, the maximum credible earthquake (MCE) selected for use in the feasibility study is characterized by a PGA of 0.47g

7.6.3 Probabilistic Analysis

The methodology for probabilistic seismic hazard analysis (PSHA) was developed by Cornell (1968), and is used to provide a framework in which uncertainties in size, location and rate of recurrence of earthquakes can be considered to provide a probabilistic understanding of seismic hazard.

A PSHA can be described as a procedure of four steps (Kramer 1996):

1. Identification and characterization of earthquake sources, along with the assignment of a probability distribution to each source zone.
2. Characterization of earthquake recurrence.
3. Determination of ground motion produced at the site by earthquakes of any possible size occurring at any possible point in each source zone.
4. Calculation of the probability that the ground motion parameter will be exceeded during a particular time period given uncertainties in earthquake location, earthquake size and ground motion parameters.

The calculations for this report were performed using the computer software EZ-FRISK 7.37 developed by RISK Engineering, Inc.

Information regarding the geological, tectonic, and seismological setting of the region was evaluated to develop a seismic source model for the calculation of earthquake ground motions at the San Luis site. The seismic source model includes subduction source zones for the plate interface and subducted slab, upper crustal fault sources that represent specific active and potentially active fault zones within the site region, and areal source zones for shallow crustal seismotectonic domains.

The subduction source zones are modeled as dipping faults with source input parameters that include: (1) location, (2) style of faulting, (3) probability of activity, (4) source geometry and depth (dip, depth to top, depth to bottom), (5) activity rate, and (6) maximum earthquake magnitude (M_{max}). The activity of the subduction plate interface is modeled with a truncated exponential distribution, with estimates of seismogenic slip rate and historical events describing recurrence of characteristic "great" earthquakes and exponential Gutenberg-Richter relationships derived from catalog seismicity describing recurrence intervals of lesser magnitude events. The activities of the slab sources are modeled with exponential distributions with recurrence parameters determined by catalog seismicity.

Active and potentially active faults within the site region are modeled as dipping faults with characteristic maximum magnitudes and recurrence parameters based on fault slip rate. The required input parameters include: (1) location, (2) style of faulting, (3) probability of activity, (4) source geometry and depth (dip, depth to top, depth to bottom), (5) slip rate, and (6) M_{max}.

Shallow crustal source zones are modeled as areal (or background) sources with earthquake recurrence based on an exponential distribution and catalog seismicity. The required input parameters include: (1) location, (2) probability of activity, (3) depth interval, (4) activity rate, and (5) M_{max}. The M_{max} distribution characterizes “background” events that may occur within a region that are not included as line sources.

The probabilistic ground motions for various return periods are summarized in Table 7-1**.** Based on the results of the PSHA, the greatest contributions to the hazard is from the background seismic source (periods less than 0.45 seconds) and the Central Peru Subduction Zone (S1, S2 and S3, with S1 dominating at periods greater than 0.45 seconds). The background sources near the site are characterized by low magnitudes (less than M 6.0). For larger amplitudes, the probability density of the subduction zone increases with magnitudes ranging from M 8 to 9 at distances of approximately 100-130 kilometers.

7.7 GENERAL DESIGN BASIS

7.7.1 Introduction

The facilities will be designed to process approximately 400 tonnes/day of ore. The design production rate is based on a balanced design of mine and mill facilities. Given the reserve and resource estimates, at the selected daily production rate, the project life is expected to be approximately 3.5 years.

All aspects of the project will be designed to comply with the Environmental Impact Assessment and with existing Silver Standard health, safety and environmental policies.

Based on the review of historical data and subsequent analysis, the project site characteristics are summarized in Table 7-1.

Table 7-1: Mean Probabilistic Ground Motion on Rock (G’s)

Return Period (Years)	Mean Peak Ground Acceleration	Mean Spectral Acceleration at 1.0 Second Period
475	0.34	0.27
975	0.42	0.34
2,475	0.53	0.46
5,000	0.63	0.57
10,000	0.74	0.70

7.7.2 Results

The design basis earthquake (DBE) is determined to be the maximum credible earthquake (MCE) based on the deterministic approach considering the 84th percentile peak ground accelerations. The design peak ground acceleration (PGA) at top of bedrock is 0.47g, and can be related (in the probabilistic approach) to an event with a return period of about 1,500 years.

Two main scenarios are identified based on the deterministic approach. The first scenario is controlled by an interface (megathurst) event, with a magnitude Mw 9.2 and will control the response for periods of 0.45 seconds and greater. The second scenario identified corresponds to a crustal event with Mw 6.0 and will control the response for periods of less than 0.45 seconds.

Other structures such as building foundations and/or other non-critical structures (non life-threatening if they were to fail or be damaged) may be designed for a PGA of 0.34g which corresponds to 10 percent probability of exceedance in 50 years (return period of 1 in 475 years). This level of design is in accordance with the intent of the Peruvian regulations (2003 Peruvian earthquake resistance design code). The proposed PGA is similar to the value given by the International Building Code (2006). Project site characteristics are summarized in Table 7-2.

Table 7-2: Summary of Site Characteristics

Location	Item	In Cordillera Negro, SW of Huaraz
Elevation (mine portal) (plant site)	4,400 m 4,36 1m	
Average Monthly Temperature (maximum) (minimum)	 8.5 °C 7.6 °C	Mina Pierina (2000-2008)
Extreme Temperature (maximum) (minimum)	 24.6 °C -3.6 °C	Mina Pierina (2000-2008)
Maximum wind Velocity	6.7 m/sec	Pueblo Viejo Station (2007)
Prevailing Wind Direction	North and South	Mina Pierina (2000-2008)
Relative Humidity	50 to 84%	Mina Pierina (2000-2008)
Average Annual Evaporation	1,026 mm	Mina Pierina (2000-2008)
Average Annual Precipitation	1,217 mm	Mina Pierina (2000-2008)
100-year 24-hour Precipitation	86.4 mm	Based on PIRA Station (1965-2008)
200-year 24-hour Precipitation	94.6 mm	Based on PIRA Station (1965-2008)
Probable Maximum Precipitation (PMP)	320.8 mm	Based on PIRA Station (1965-2008)
Probable Maximum Flood (PMF) run-off depth	289.1 mm	Calculated from PMP
Snowfall Depth	Negligible	Based on temperature data
Earthquake (1 in 475 year return period) peak ground acceleration	0.34 g	Used for the design of non-critical structures
Maximum Credible Earthquake (MCE) peak ground acceleration	0.47 g	Magnitude 6 (used for the design of the TSF)

ITEM 8 • HISTORY

The San Luis mineralization was previously described and reported, mainly, in reports prepared by Pincus[1], Blanchflower[2] and Konkin[3]. Some of the following description is referenced from those reports.

As stated in Blanchflower[2] and in Lechner & Earnest[4]: *"According to Konkin (2007) and Pincus and McCrea (2006), there is no evidence of any modern exploration work or previous land tenure in the vicinity of the San Luis vein system in the west-central portion of the property. Six kilometres to the southeast, there are historic test pits, short adits and evidence of past 'highgrading' operations within the BP zone (see Section 8.0 – Mineralization) that were reportedly active in the 1980's and possibly earlier. It appears that the miners extracted manto-hosted pyrrhotite-sphalerite-galena mineralization and transported the hand-cobbed mineralization away for milling. Nevertheless, it appears that the Ayelén (see Section 8.0 – Mineralization) and the other precious metal-bearing vein structures are grass-root discoveries by the current joint venture partners. There are a number of other precious and base metal occurrences in the region that are owned by other operators. Barrick Gold Corporation owns and operates the Pierina gold mine which is situated approximately 25 km east-southeast of the San Luis property."*

There are no reported historical mineral resources or documented production from the San Luis property. Following the discovery of the San Luis quartz vein system in 2005, the joint venture:

- Secured a sizeable land position in the area.
- Obtained exploration work permits.
- Established an exploration camp on the property.
- Completed a definition drilling program.
- Developed a resource estimate.

The property now encompasses over 33,000 hectares of mineral concessions. In 2006 and 2007 a systematic trenching and drilling program was carried out on the westernmost vein, known as the Ayelén Vein. During this time approximately 19,200 meters of drilling was completed in 108 diamond drill holes[4]. Thirty four trenches were also completed. Another 28 holes or approximately 3,200m were drilled in the Inéz vein, bringing the total drilling to 136 holes and approximately 22,400m.

In January 2009 Resource Evaluation Inc. (REI) and Resource Modeling Inc. (RMI) prepared an updated mineral resource estimate of the San Luis project[4].

[1] "Technical Report on the San Luis Project, Ancash Department, Perú"; William J. Pincus; September 12, 2006.
[2] "Technical Report on the San Luis Property District of Shupluy, Yungay Province, Ancash Department, Peru"; Minorex Consulting Ltd., December 28, 2007.
[3] "Annual Report for 2007 San Luis Project, Ancash Department, North Central Peru"; Kenneth James Konkin; 2007
[4] "Updated Mineral Resource Estimate San Luis Project, Ancash Department, Perú"; Resource Modeling, Inc. and Resource Evaluation, Inc.; January 9, 2009.

ITEM 9 • GEOLOGICAL SETTING

9.1 INTRODUCTION AND HISTORY

This section of the technical report is based on work completed by REI and RMI as part of an update of the mineral resources for the San Luis Project that was publicly disclosed in the Canada National Instrument 43-101 Technical Report titled, “Updated Mineral Resource Estimate, San Luis Project, Ancash Department, Perú, January 9, 2009”.

9.2 REGIONAL GEOLOGY

The San Luis property is situated within the Cordillera Negra terrane of the Peruvian Andes. The sedimentary stratigraphy of the region consists (from oldest to youngest) of interbedded mudstones and sandstones of the upper Jurassic Chicama Formation, quartzites, sandstones and shales of the lower Cretaceous Chimú Formation, limestones and calcareous clays of the lower Cretaceous Santa Formation, the Carhuaz Formation (sandstones, quartzites, with interbedded mudstones), the Farrat Formation (fine quartzites with interbeds of red mudstone), and a Cretaceous sequence of calcareous rocks of the Pariahuanca, Chulec, and Pariatambo Formations. An angular unconformity separates the sediments from the overlying volcanic Paleocene Calipuy Formation (tuffs, coarse pyroclastics, agglomerates, lavas and sub volcanic intrusions of andesitic/dacitic/rhyolitic composition). Overlying the Calipuy Formation is the Mio-Pliocene Yungay Formation, which is composed of dacitic tuffs and ignimbrites.

Intrusive bodies in the region consist of the Cretaceous – Paleocene Coastal batholith and the Mio-Pliocene Cordillera Blanca batholith (both composed of granodiorites and tonalites). Recent surficial deposits in the region are mostly fluvial – glacial in character. Regional structure is related to four stages of crustal deformation – uplift of the Andean belt, followed by the Andean Orogenesis characterized by folding and thrust faulting (which affected the Jurassic and Cretaceous sediments), block faulting (large vertical displacements of basement rocks), and Plio-Pleistocene renewed uplift of the Andean belt. Figure 9-1 illustrates the aerial relationship of these stratigraphic units and major structures.

9.3 LOCAL GEOLOGY – STRATIGRAPHY AND STRUCTURE

Local stratigraphy in the San Luis project area is dominated by the Paleocene Calipuy Formation which occurs mainly as flat-lying andesite volcanic sediments consisting of interlayered tuffs, coarse pyroclastic flows, volcanic breccias, agglomerates and lava flows. Fine to coarse-grained andesites overlain by beds of interlayered andesite and volcanic breccia that strike northwesterly and dip from sub-horizontal to 30° southwest are the main hosts for the mineralized structures at San Luis.

Figure 9-1: Regional Geology

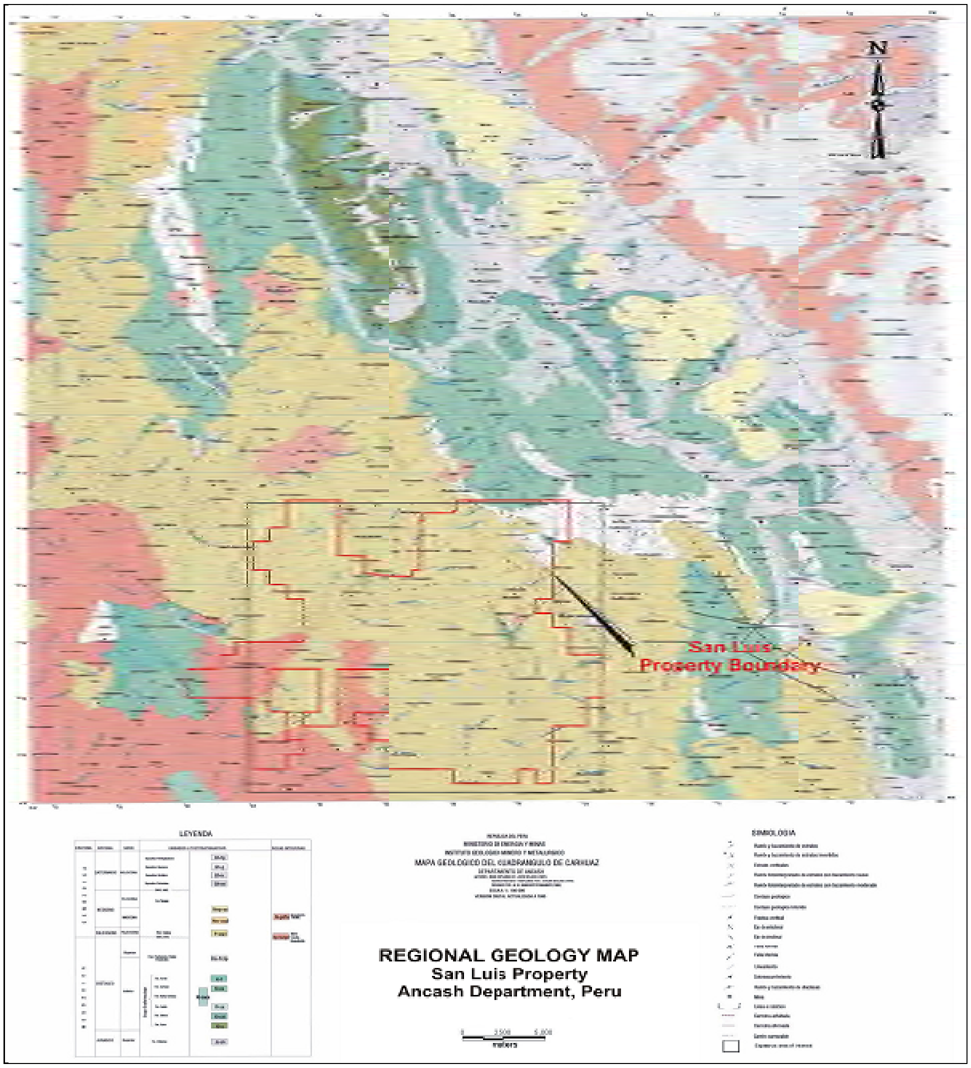


Figure 4 of "Technical Report on the San Luis Property District of Shupluy, Yungay Province, Ancash Department, Peru"; Minorex Consulting Ltd., December 28, 2007.

Other local lithologies include sandstones and fine-grained quartzites interlayered with some clay stones of the lower Cretaceous Carhuaz Formation (exposed as windows in the overlying volcanics) and granodiorites of the Puscao unit of the Coastal batholith, lesser outcrops of granodiorite and tonalite of the Cordillera Blanca batholith, and a series of rhyolitic sills, dikes, and irregular bodies that occur in intimate association with the San Luis veins. Figure 9-2 illustrates the relatively simple aerial relationships between surficial lithologies in the project area.

Structurally, local northwest and north-northwesterly faults have been active since the majority of volcanism ceased but prior to emplacement of the main epithermal veins. However, faulting also occurred during and after vein emplacement, which resulted in the well-developed vein breccia textures, tensional openings that allowed subsequent emplacement of rhyodacite dikes, and later brecciation and displacement of both the dikes and veins.

Figure 9-2: Property Geology

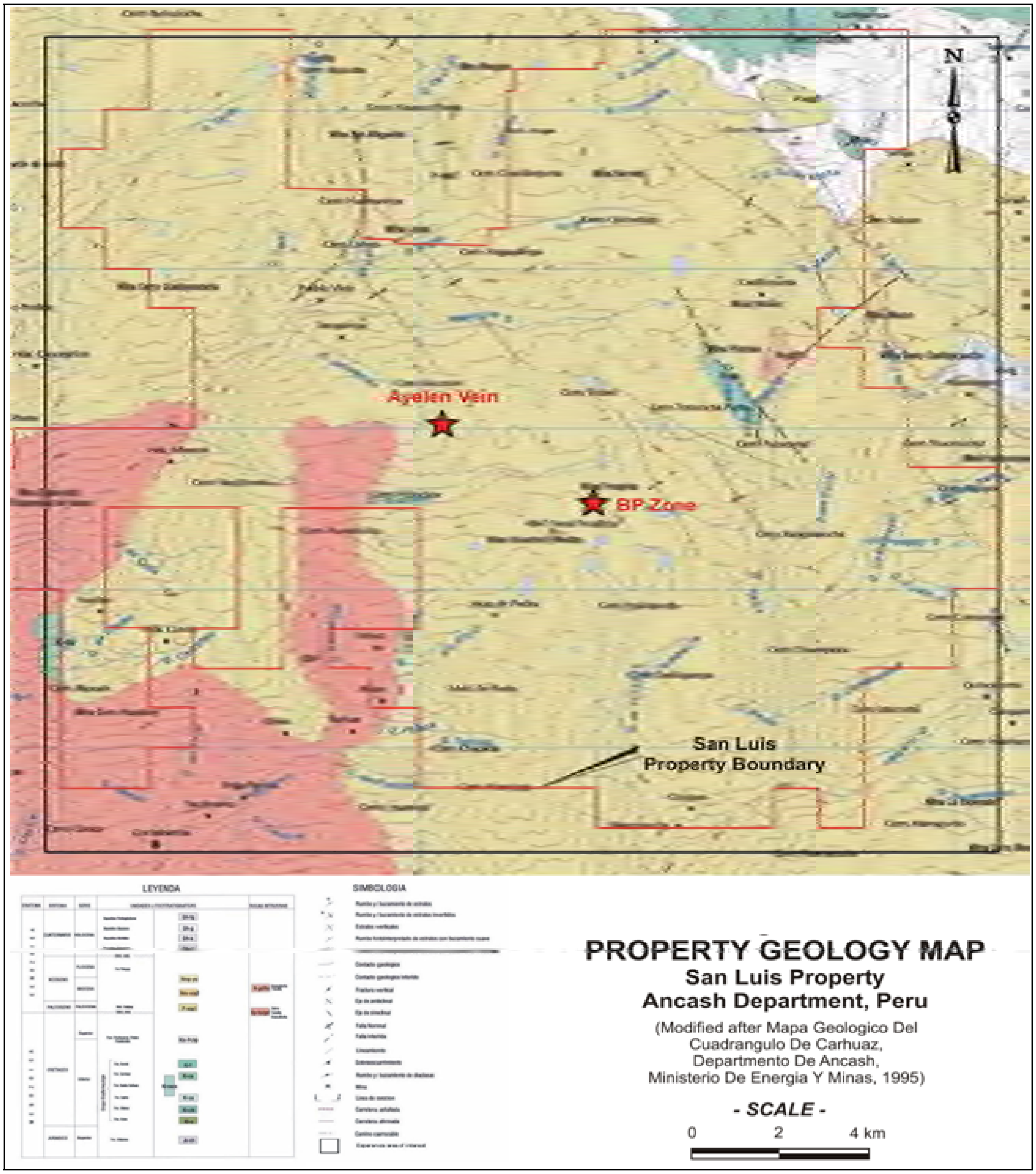


Figure 5 of "Technical Report on the San Luis Property District of Shupluy, Yungay Province, Ancash Department, Peru"; Minorex Consulting Ltd., December 28, 2007.

ITEM 10 • DEPOSIT TYPES

The San Luis vein system is a typical volcanic-hosted low sulphidation epithermal quartz/precious metal deposit. The deposit is very similar to numerous other volcanic-hosted epithermal vein deposits that occur in the Rocky Mountain, Sierra Nevada, Sierra Madre, and Andean cordilleras, including those in districts in Ouray/Telluride and Creede, Colorado, Silver Peak, Nevada, Guanajuato, Fresnillo, and Pachuca, Mexico, and Morococha, Quirivilca, and Casapalca, Peru.

As noted by Pincus and McCrea (2006) "Interpretations of the epithermal model indicate that ore-bearing fluids typically travel along structural pathways at high temperatures with sufficient hydrostatic pressure to prevent boiling. When the pressure drops suddenly through faulting or rupture, boiling occurs and the fluids quickly deposit their mineral load in available open spaces. Deposition of minerals, particularly quartz will typically occur in these open spaces with bands growing from either wall inward. Open spaces are eventually sealed by this growth until ruptured once again by underlying fluid pressure or new faulting and the process begins over again. This repeated rupturing results in the interrupted banded texture typical of epithermal veins. Structural features, particularly faulting and fracturing, are a key element in controlling the location of ore deposition. Ore "shoots" will typically occur in dilational zones, which in turn result from a variety of local stresses. Often these stresses are repeated along the length of a vein structure resulting in multiple ore-shoots."

ITEM 11 • MINERALIZATION & ORE TYPES

Vein gangue mineralization in the San Luis deposit consists of quartz, chalcedony, calcite and minor adularia. Gold occurs as electrum and silver is present as acanthite, other silver sulphosalts, and electrum. Other sulphides include trace amounts of pyrite, chalcopyrite, galena and sphalerite The veins display common epithermal textures including crustiform banding (see Figure 11-1) that often display the interlayers of quartz and sulfide minerals described in Item 10. Bands are frequently disrupted, indicating repeated pulses of mineralization. Lattice textures in which calcite crystals have been replaced by quartz and brecciation are also common characteristics. Faulting and fracturing are key elements in the localization of mineralization in the vein system. Mineralized “shoots” typically occur in dilational zones that are the result of a variety of local stresses, and often these stresses are repeated along the length of a vein structure, resulting in multiple mineralized-shoots.

Figure 11-1: Multi-Stage Quartz Banding

Photograph 10 of “Technical Report on the San Luis Property District of Shupluy, Yungay Province, Ancash Department, Peru”; Minorex Consulting Ltd., December 28, 2007.

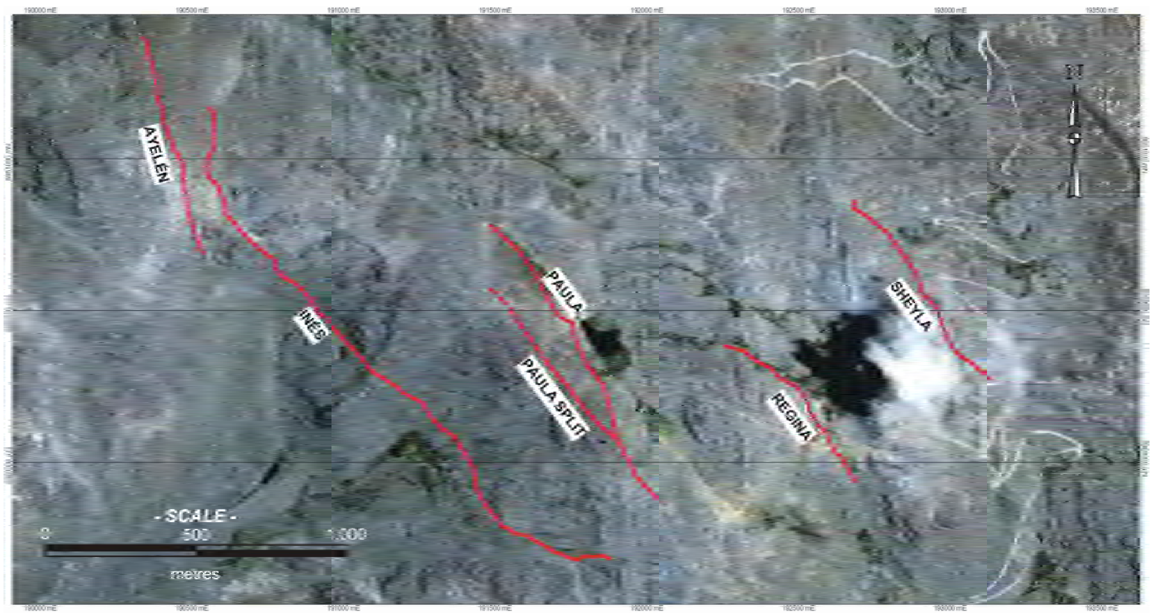
The Ayelén vein is the better mineralized of the known vein structures. Trenching and diamond drilling have traced this structure along a strike of 340˚ to 345˚ for a length of over 720m, with down-dip extensions of more than 325m. Surface mapping shows that the vein structure dips -75˚ to -85˚ west-southwesterly, but drilling results show that the controlling fault structure(s), sub-surface individual vein segments and post-mineral dikes dip vertically to -80˚ west-southwesterly.

True thicknesses of individual vein segments vary from ten's of centimeters to more than 10m, with an average of 1.5m to 3.0m wide.

The Inéz vein, which is situated approximately 110m east of the Ayelén vein, strikes northwesterly at 320˚ to 340˚ and dips -50˚ to -75˚ to the northeast. The vein outcrops as a series of discontinuous resistant ridges for more than 2,200m along strike, with apparent widths of 2.0m to 7.5m. However, only a relatively short section of the Inéz vein contains significant amounts of gold and silver, and this section occurs where the Inéz vein is closest to the Ayelén vein.

Figure 11-2 illustrates the spatial relationship of the Ayelén vein, Inéz vein, and other veins which together comprise the San Luis epithermal system.

Figure 11-2: San Luis Vein System



Photograph 9 of "Technical Report on the San Luis Property District of Shupluy, Yungay Province, Ancash Department, Peru"; Minorex Consulting Ltd., December 28, 2007.

Figure 11-3 is a surface outcrop map of the Ayelén and Inéz veins, which host all of the precious metal (gold-silver) mineral resources that are currently defined at San Luis.

Figure 11-3: Ayelén and Inéz Vein Outcrop Map

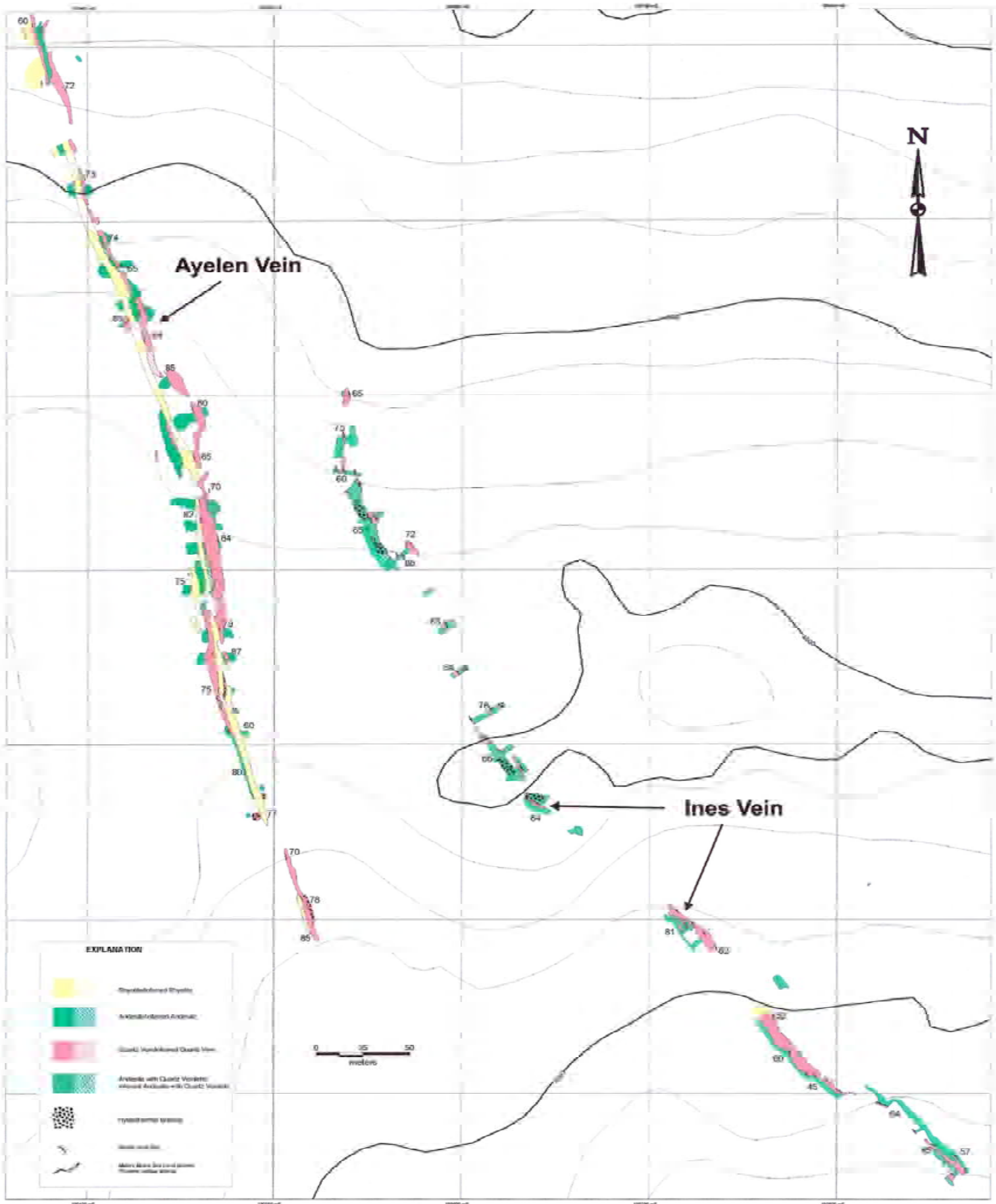


Figure 6 of "Technical Report on the San Luis Property District of Shupluy, Yungay Province, Ancash Department, Peru"; Minorex Consulting Ltd., December 28, 2007.

ITEM 12 • EXPLORATION

No significant additional exploration work has focused on the Ayelén and Inéz veins since the completion of the NI 43-101 Technical Report on the San Luis Project by Blanchflower in 2007. Brief summaries of the 2005 through 2007 exploration work that discovered and delineated the mineralization contained in the Ayelén and Inéz veins can be found in the NI 43-101Technical Report by Lechner and Earnest (2009). Thorough descriptions of this work can be found in earlier NI 43-101 Technical Reports authored by Pincus and McCrea (2006) and Blanchflower (2007). This Technical Report discusses only the surface exploration activities (trenching and channel sampling) that provided some of the basic data for the mineral resource estimate that is the basis for the mineral reserves defined by the San Luis Project Feasibility Study. This Technical Report does not address the exploration work performed by SSR on the BP Zone located southeast of the general vicinity of the Ayelén and Inéz veins since 2008, as that work has no bearing whatsoever on the San Luis Project Feasibility Study that is the focus of this Technical Report.

12.1 2005 EXPLORATION PROGRAM

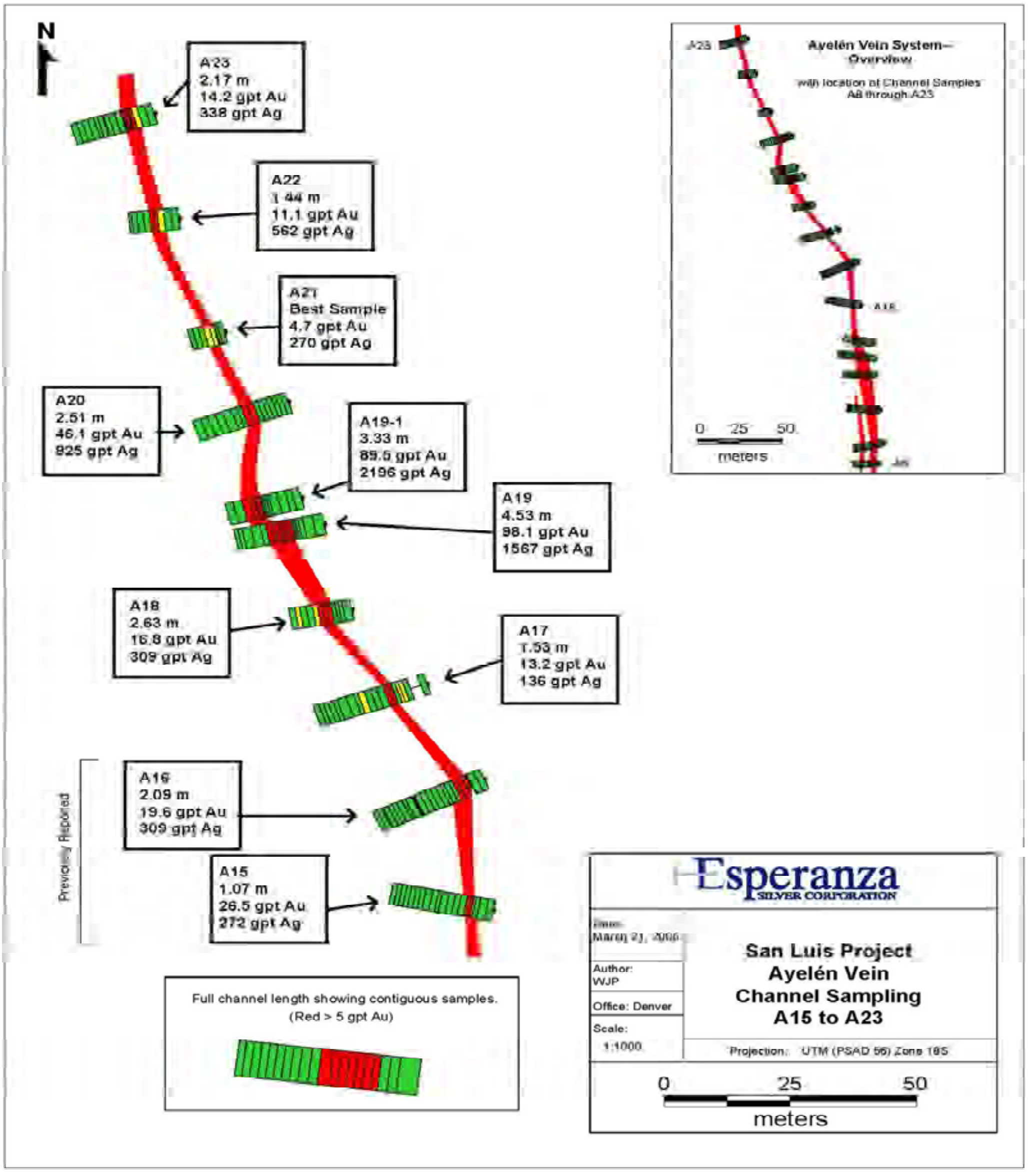
As summarized by Blanchflower (2007): "The San Luis property was first visited by Esperanza field personnel in June 2005. They discovered and sampled the Inés vein structure which returned values ranging from trace to 1.56 gpt gold and 0.8 to 100 gpt silver. During a later property visit in July 2005 the Ayelén and Paula vein structures were discovered, prospected and channel sampled. The channel samples from the Ayelén vein returned values ranging from 0.026 to 173.8 gpt gold and 23 to 2,504 gpt silver (Pincus and McCrea, 2006). Prospecting work by Esperanza later in the year led to the discovery of the nearby Sheyla and Regina vein structures."

12.2 2006 EXPLORATION PROGRAM

As described by Pincus and McCrea (2006), "Thirty-two separate trenches were excavated along the trend of the Ayelén vein structure at 25-metre intervals and 403 channel samples were collected along these trenches from the vein structure and wall rock. True thicknesses of the samples were calculated using the vein dips at each sample site. These vein dip angles varied from -70° to -85° west-southwestwardly. The results of the channel sampling showed that the Ayelén vein structure hosted values ranging from trace to 134 gpT gold and 5.3 to 2,246 gpT silver over true vein thicknesses of 0.98 to 7.14 m (Pincus and McCrea, 2006)." Figure 12-1 shows the locations of channel sample results along the central portion of the Ayelén vein.

"Twenty-five trenches were excavated across the Inés vein structure at 25-metre intervals and 90 channel samples were collected along these trenches from the vein structure and wall rock. Vein structure dip angles varied from -65° to -75° northeastwardly from which true thicknesses of the structure were calculated. Pincus and McCrea (2006) reported precious metal values for samples that graded greater than 1 gpT gold or represented longer true widths across the Inés vein structure. The reported results ranged from trace to 21.07 gpT gold and trace to 1969 gpT silver across true widths of 0.66 to 3.43 m. The results of this work identified the high gold and silver grades hosted by the Ayelén and Inés vein structures and showed that these veins have features commonly associated with typical low sulphidation epithermal vein systems. A detailed description of the trenching and rock geochemical sampling work is documented in the technical report by Pincus and McCrea (2006)."

Figure 12-1: Locations of 2006 Channel Samples on Central Portion of Ayelén Vein



On September 25, 2006, Esperanza commenced the first drilling program on the San Luis property. A detailed description of the drilling completed in 2006 and 2007 is provided in the following Item 13.

ITEM 13 • DRILLING

According to SSR personnel, all drilling of the Ayelén and Inéz veins was performed by Boart Longyear ('Longyear') of Lima, Peru, using LF-70 and LY-44 rigs. All holes were reportedly collared with HQ-size drilling tools that recovered 63.5mm diameter core, with reductions to NQ-size tools (47.6mm diameter core) if poor drilling conditions were encountered.

Drill hole collar sites reportedly were first surveyed using GPS instrumentation and subsequently using Distamat surveying equipment. Upon completion, all drill hole locations were immediately surveyed, and surveyors marked the location of the collars of the drilled holes with labelled rock cairns. Drill holes were surveyed down-hole using an Easyshot® survey tool that collected azimuth and inclination data at regular 50m intervals down-hole, with the final shot taken near the bottom of each drill hole. All down-hole survey data were recorded on the field geologic logs and subsequently in the electronic drill hole database.

Based on a review of core from ten diamond core holes (A-SL-087, A-SL-082, A-SL-079, A-SL-064, A-SL-069, A-SL-104, SL07-001, SL-07-002, SL06-01, and SL06-04), the core appears to be in excellent condition and order, with clearly labeled run blocks in place and sample breaks clearly noted. In the holes examined the core had been well-sampled using a diamond saw.

Logging of diamond drill core included the recording of lithologic, structural, alteration and mineralogical features observed by the project geologists onto conventional logging forms. These data were then transcribed onto drill hole-specific spreadsheet-type files that could be imported directly into a Gemcom® database. The logging of lithologies was very detailed and well done, with numerous (over 50) individual lithologic designations. Structure, alteration and mineralization logging was generally well done. The logging of geotechnical data was not extensive, with only core recovery (%), rock quality designation (RQD), and a general description of core quality recorded.

For the mineral resource estimate on which this Feasibility Study is focused, only diamond core holes that tested the Ayelén and Inéz veins were used. Table 13-1 summarizes the number of diamond core holes and total number of meters drilled (by vein area) that define the current mineral resource. Most of the diamond core holes were drilled at steep angles to the east, resulting in both acute and obtuse intersection angles with the steep westerly-dipping vein.

Table 13-1: Summary of Diamond Drill Holes by Vein Area

Area	Number	Meters
Ayelén	108	19,196.15
Inéz	28	3,157.80
Total	**136**	**22,353.95**

In addition to the diamond core holes, a series of chip-channel samples were collected from surface trenches that were excavated on approximately 25m spaced intervals along and perpendicular to the strike length of the outcropping Ayelén and Inéz veins. These trench data were treated like sub-horizontal drill holes for constructing vein wireframes and estimating mineral resources (see 4.6.2- Vein Wireframe Construction). Table 13-2 summarizes the number of trenches and total meters of trenching by vein area. Most of the trench samples were assayed.

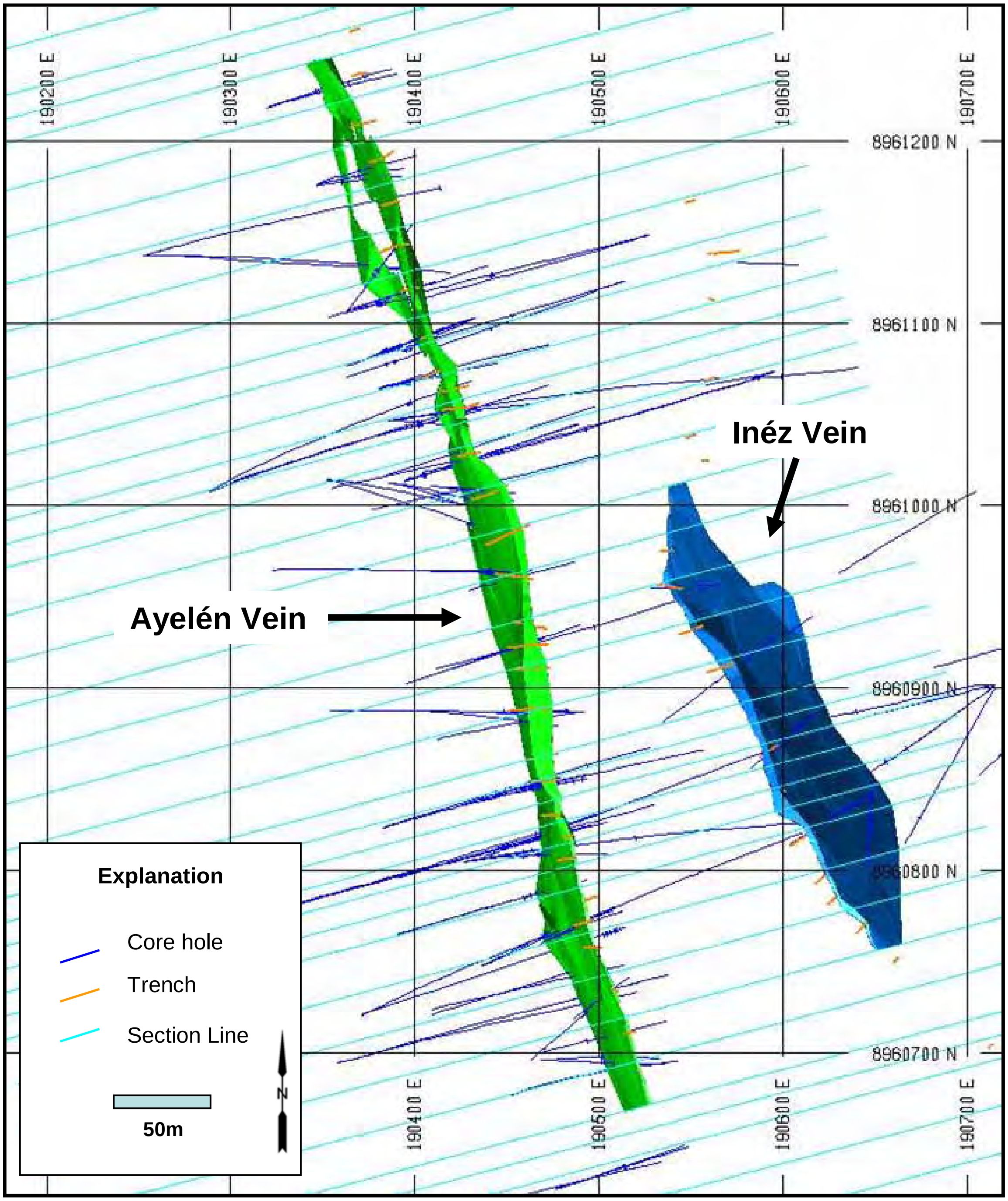
Table 13-2: Summary of Surface Trenches by Vein Area

Area	Number	Meters
Ayelén	34	506.21
Inéz	62	441.00
Total	**96**	**947.21**

The surface chip-channel samples were oriented perpendicular to the strike length of the Ayelén and Inéz veins and closely represent the true thickness of the veins at the sampled locations. The true thicknesses of the veins range between ten's of centimeters to over 10m and average between 1.5m to 3.0m in width. The veins appear to be thicker near the surface and become thinner and lower grade with depth.

Figure 13-1 is a plan map of the Ayelén and Inéz vein areas that shows the areal extent of the surface drilling and trench sampling data that were used to prepare the resource estimate for this Feasibility Study. The plan map distinguishes between surface diamond core holes (shown as dark blue lines) and surface trenches (shown as orange lines), and also shows the traces and orientations of the cross sections (shown as light blue lines). The green and blue-shaded areas on the map represent the main Ayelén and Inéz three-dimensional wireframes that were constructed as part of the geologic interpretation used to restrain estimation of block grades in the resource model.

Figure 13-1: Plan Map Showing Drill Holes and Surface Trenches



ITEM 14 • SAMPLING METHOD & APPROACH

14.1 DRILL CORE SAMPLING

Diamond core recovery for the various drill campaigns was excellent. Fifty-six percent of the assayed intervals had a core recovery of 100% and 95 percent of the total data had core recoveries in excess of 90%. The San Luis project geologists who logged the core determined the core intervals to be sampled. Sample intervals averaged approximately 1.0m in length, with individual samples rarely greater than 1.5m or less than 0.5m in length. Definite sample breaks were made at contacts between geologic units (vein/wallrock boundaries and lithologic contacts). Within an individual geologic unit, sample intervals were extended or reduced slightly to allow for a nearby geologic contact. The maximum allowable sample length was 2.0m.

Once the project geologists marked the sample interval breaks, core from each assigned sample interval was individually removed and cut in half lengthwise using a diamond saw. After sawing the core from an individual sample interval, one-half of the drill core was placed in a 6 mil plastic sample bag for assay, and the other half was returned to its original position in the core box. All of the core from the ten holes examined during the June 2008 site visit was found to be well sawn, with the remaining half core carefully placed back in the boxes in good order.

Once sampling was complete, the individual core sample bags were then securely tied with non slip plastic straps, properly labeled and stored in a locked room in the core storage facility under the supervision of the project geologist until they could be shipped to the assay laboratory.

14.2 SURFACE TRENCH SAMPLING

Surface trenches were sampled by cutting material (using a hammer and chisel) from channels in the floor of the trenches after the floors were cleaned by pick and shovel. Based on REI's examinations of the surface trenches during its 2008 site visit, the samples taken were not true channel samples (where material was carefully removed such that well defined, regular rectangular cuts remained). Rather, the surface trench samples appeared to be what is well known and accepted in the industry as "chip channel" samples. Individual sample locations were then marked by spray paint continuously along the entire length of the trench (see Figure 14-1). Each sample was cut along a length of approximately one meter, although the lengths of individual samples varied. Each individual sample generally weighed between one and two kilograms.

Figure 14-1: Surface Trench Showing Locations of Individual Samples

Photograph 17 of "Technical Report on the San Luis Property District of Shupluy, Yungay Province, Ancash Department, Peru"; Minorex Consulting Ltd., December 28, 2007.

ITEM 15 • SAMPLE PREPARATION, ANALYSES & SECURITY

15.1 SAMPLE SECURITY AND CHAIN OF CUSTODY PROCEDURES

Once trench samples were collected, the sample bags were tied shut by the geologist and then transferred to the project field camp and stored in a locked area at the core shed, with access controlled by project geologists. Diamond drill cores were also transported from the drill rigs and stored in the same locked area. After the cores were sampled, the individual 6 mil sample bags of core were securely tied with non-slip plastic straps, properly labeled, and returned to the locked room in the core shed to await shipment to the assay laboratory. Prior to shipping, the individual sacks of samples were placed into large woven nylon rice bags, the contents were marked on each rice bag, and each bag was securely sealed. These rice bags containing the individual samples were then delivered directly to either the SGS del Peru S.A.C in Lima or the ALS Chemex assay laboratory in Lima by project personnel, thus maintaining an uninterrupted chain of custody between the drill rigs and the assay laboratory. Occasional exceptions to this procedure occurred when trench samples were delivered by project personnel to the town of Casma, where the samples were then bused via commercial transport (TEPSA S.A.) to Lima, and picked up by SGS.

15.2 SAMPLE ANALYSIS

During the exploration of the San Luis veins, sample preparation and analysis was done by several laboratories, including SGS del Peru S.A.C. in Lima, Peru (2006), and ALS Chemex Peru S.A. in Lima (2007). Both laboratories are accredited according to the ISO/IEC 172025 standard, which is specifically designed for Mineral Analysis Testing Laboratories. The sample preparation and analytical procedures used by these laboratories (as described in Blanchflower (2007) and Pincus and McCrea (2006)) are appropriate for the type of epithermal gold-silver mineralization present in the San Luis deposit. No aspects of the sample preparation or sample analysis were conducted by an employee, officer, director, or associate of either Silver Standard or Esperanza Silver Corporation, its joint venture partner in the San Luis project.

15.3 METALLURGICAL COMPOSITE SAMPLES

During 2007, preliminary metallurgical testwork was completed on drill core from the Ayelén vein. The testwork was directed by F. Wright Consulting Inc. of North Vancouver, B.C., Canada. The actual testwork was done primarily by PRA of Richmond, BC. Related chemical analyses were performed by iPL Laboratory of Richmond BC. This preliminary metallurgical testwork was completed on five individual composite samples of diamond drill core that were assembled from 15 diamond core holes drilled during the 2006 and 2007 exploration seasons. The core holes that contributed sample material to the composites are summarized in the NI 43-101Technical Report by Lechner and Earnest (2009).

In addition to the preliminary metallurgical composites, four additional composites were assembled in late July/early August 2009 for further testwork by G&T. These composites (which included material from more 17 diamond core holes drilled in 2006 and 2007) are representative of the following:

- The approximate average grade of the projected mill feed based on the Feasibility Study Life of Mine plan.
- Spatially representative high grade portions of the deposit.
- Spatially representative low grade portions of the deposit.
- The approximate average mill feed grade projected for the first three quarters of mine production, based on the Feasibility Study Life of Mine plan.

In the opinion of the authors of this section of the San Luis Technical Report, these metallurgical composites adequately represent (both spatially and from a production standpoint) the material scheduled for mining from the Ayelén deposit during the term of the Life of Mine Plan, and the composites are valid for the metallurgical testwork performed by G&T.

ITEM 16 • DATA VERIFICATION

16.1 ELECTRONIC DATABASE

REI and RMI obtained copies of certified (signed) assay certificates for 31 core holes and/or trenches, covering a representative number of sampling campaigns for verification of the assays in the electronic database. Only three gold and three silver assay data errors were discovered, all in Ayelén trench A16. It appears that these records had been shifted downward one row in the electronic database, and the errors were corrected prior to estimating mineral resources. Table 16-1 summarizes the results of the database review:

Table 16-1: Database Verification Results

Data Type	Total No. Records	No. Records Checked	% of Records Checked	Errors Found	% Error Rate
Au Assays	5,353	916	17%	3	0.33%
Ag Assays	5,353	916	17%	3	0.33%
Total	**10,706**	**1,832**	**17%**	**6**	**0.33%**

The results of this verification of the gold and silver assays in the electronic database are within typical industry accepted tolerances for accuracy (less than 1% errors), which indicates that the electronic database is acceptable to be used to estimate mineral resources.

16.2 QUALITY ASSURANCE/QUALITY CONTROL (QA/QC) PROCEDURES

According to Blanchflower (2007), a rigorous QA/QC program was established to "*monitor accuracy (i.e. sample standards), contamination (i.e. sample blanks), precision (i.e. duplicates) and other possible sampling errors (i.e. sample mis-labelling)*". The QA/QC protocol for this program included the insertion of at least one quality control (QC) sample per 15 regular samples submitted to the primary assay laboratory. The type of QC sample that was inserted alternated between a duplicate sample (sent to a secondary laboratory), a standard sample, or a blank sample. The standard and blank samples were inserted into the sample sequence as the sample shipment was being prepared. Duplicate samples were inserted at the time of collection. The QC samples were assigned sample numbers in the same sequential order as the primary samples in order to help prevent detection of the QC samples by the primary laboratory. Sample results reportedly were routinely monitored for quality control failures or problems, and when these occurred, the laboratory responsible was notified and check analyses (reruns) were performed to resolve the discrepancies.

The various QA/QC reports (McCrea, 2006; McCrea, 2007; Blanchflower, 2007) that pertain to the assay data which are relative to this report were reviewed by the authors of this section of the Feasibility Study. In addition, the authors independently reviewed the results from 138 quarter core duplicate samples that were obtained from the 2006 and 2007 drilling campaigns. Table 16-2 summarizes the basic descriptive statistics for these duplicate samples.

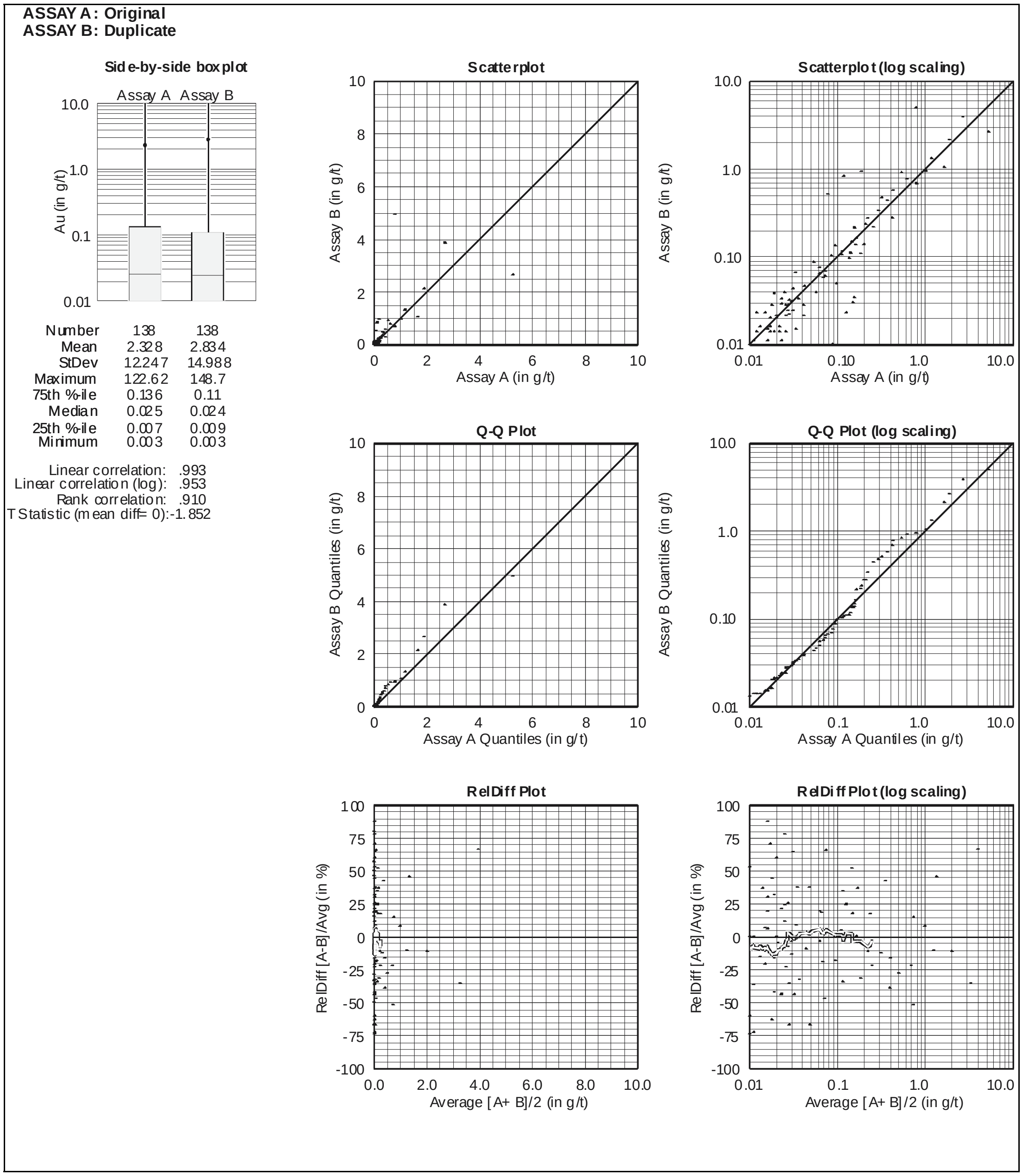
Table 16-2: Quarter-Core Duplicate Sample Statistics

	Gold		Silver	
Statistical Parameter	**Original**	**Duplicate**	**Original**	**Duplicate**
Count	138	138	138	138
Minimum (g/t)	0.003	0.003	0.40	0.40
Maximum (g/t)	122.620	148.700	3793.00	4335.00
Mean (g/t)	2.327	2.834	78.83	89.52
Median (g/t)	0.026	0.024	1.15	1.20
Mode (g/t)	0.003	0.003	1.00	1.00
Standard Deviation	12.292	15.043	382.80	441.13
Coefficient of Variation	5.28	5.31	4.86	4.93

While the duplicate quarter core samples came back with higher average grades for both gold and silver, the authors note that this comparison is based on a limited number of samples. Figure 16-1 illustrates various graphical comparisons between the original and duplicate quarter core samples for gold.

Similar relationships were seen for the 138 duplicate quarter-core silver assays. Based on these reviews of the San Luis QA/QC data, it is the authors' opinion that the assay data have been reasonably verified and are suitable for estimation of mineral resources.

Figure 16-1: Quarter-Core Duplicate Sample Results – Gold



ITEM 17 • ADJACENT PROPERTIES

There are no adjacent mineral properties of any significance, and, to the best of the authors' knowledge, no public disclosures have been made by any owner in the immediate vicinity of the project site.

ITEM 18 • MINERAL PROCESSING & METALLURGICAL TESTING

18.1 INTRODUCTION

Four metallurgical testwork programs have been completed investigating the metallurgical response of mineralized samples from the San Luis project. The first program was performed by PRA. The second program was performed by Plenge. The third and fourth programs were performed by G&T. The programs investigated the recovery of gold and silver using gravity concentration, cyanide leaching and flotation. Comminution tests and settling tests were also performed. Petrographic analyses of core samples have been performed by Dr. Craig Leitch and BISA.

Metallurgical testing to date has been carried out using industry accepted procedures and by reputable facilities. Although considerable effort was made to ensure the representivity of the samples with respect to the entire deposit, nevertheless it is possible that certain mineral assemblages were not identified or tested adequately. This section addresses the results from the metallurgical testwork programs performed to date. Details of samples used for metallurgical testwork programs performed and the results from these programs are summarized below.

18.2 METALLURGICAL TESTWORK SAMPLES

Details of samples used in the various metallurgical testwork programs are presented below. Drill core and assay reject material were used in the preparation of composite samples. None of the samples were prepared from recently drilled material. Most of the material had been in storage (open to the environment) for in excess of 12 months. It has not been determined if oxidation had occurred with this material and, if it had, what the effect was on metallurgical response. Following completion of this feasibility study, a project optimization stage is planned. During this stage confirmatory metallurgical testwork programs are planned in which freshly drilled core will be used. It is planned that these programs will include assessing the effect of oxidation.

18.3 PRA TESTWORK PROGRAM

Five composite samples were prepared for the metallurgical testwork program conducted by PRA. The composites were prepared from assay reject material and were identified as Ayelén #1 to #5. Details of material used in each composite are shown in Table 18-1.

Table 18-1: Details of Metallurgical Samples – PRA Testwork Program

Drill Hole Number	From		To		Composite Label	Au (ppm)	Ag (ppm)
	(m)	Assay Tag	(m)	Assay Tag			
SL06-01	45.00	56189	49.47	56193	Ayelén #1	45.32	997.7
SL06-03	54.25	56378	56.77	56381	Ayelén #1	13.60	184.5
SL06-03	57.77	56383	61.38	56388	Ayelén #1	28.23	615.2
SL06-10	62.00	56614	68.40	56620	Ayelén #1	91.57	2,050.7
SL06-16	42.90	56890	53.75	56901*	Ayelén #1	60.08	1,036.8
SL06-04	77.90	56398	86.07	56408	Ayelén #2	19.53	525.3
SL06-13	88.40	56727	92.90	56731	Ayelén #2	29.90	910.7
SL06-17	62.42	56966	69.00	56972	Ayelén #2	17.97	727.2
A-SL-006	139.3	58074	143.3	58081	Ayelén #3	9.68	368.9
A-SL-010	112.6	58189	114.43	58191	Ayelén #3	6.73	255.5
A-SL-023	96.75	58578	98.60	58579	Ayelén #3	13.25	199.6
A-SL-028	67.8	58694	80.25	58712	Ayelén #4-2	5.42	301.5
A-SL-030	71.15	58754	83.00	58770	Ayelén #4-2	16.18	506.4
A-SL-033	60.60	58836	79.25	58858	Ayelén #4-1	8.87	274.7
SL06-25	118.77	57383	130.15	57398	Ayelén #5	16.02	528.8
A-SL-001	126.10	57868	134.05	57877	Ayelén #5	35.62	1,254.0

Note: * Sample 56900 is a duplicate of Sample 56895; 56895 interval data used, 56900 assay data used.

18.3.1 Plenge Testwork Program

The Plenge metallurgical testwork program was performed on one composite sample. Details of the composite are shown in Table 18-2. The composite was prepared from assay reject material.

Table 18-2: Details of Metallurgical Sample – Plenge Testwork Program

Drill Hole Number	From (m)	To (m)	Au (g/t)	Ag (g/t)
SL06-26	70.85	71.85	2.31	73.5
SL06-26	71.85	72.85	2.62	76.1
SL06-26	72.85	73.55	10.1	324
SL06-26	73.55	74.30	1.97	68.1
A-SL-077	42.10	43.25	1.52	84
A-SL-077	43.25	44.44	0.38	115
A-SL-077	44.44	44.94	94.00	2400
A-SL-077	44.94	45.80	39.40	1740
A-SL-077	45.80	46.80	1.16	158
A-SL-069	228.20	229.00	0.79	67
A-SL-069	229.00	229.67	2.99	109

Drill Hole Number	From (m)	To (m)	Au (g/t)	Ag (g/t)
A-SL-069	229.67	230.67	9.10	279
A-SL-069	230.67	231.67	7.13	215
A-SL-069	231.67	232.67	3.22	136
A-SL-069	232.67	233.67	0.72	34
A-SL-069	233.67	234.67	12.75	802
A-SL-069	234.67	235.67	0.69	61
A-SL-069	235.67	236.67	0.89	61
A-SL-069	236.67	237.67	0.61	54
A-SL-069	237.67	238.67	0.64	40
A-SL-098	70.35	71.49	5.26	154
A-SL-098	71.49	72.64	0.30	19
A-SL-098	72.64	73.8	8.46	287
A-SL-098	73.8	74.96	91.50	1090
A-SL-098	74.96	76.12	133.50	1680
A-SL-098	76.12	77.28	100.00	1220
A-SL-098	77.28	78.44	98.50	1270
A-SL-098	78.44	79.6	53.10	783
A-SL-098	79.6	80.96	0.85	163
A-SL-124	164.49	165.20	0.24	4.00
A-SL-124	165.20	165.91	0.20	3.00
A-SL-124	165.91	166.77	0.16	2.00
A-SL-124	166.77	167.67	0.23	7.00
A-SL-124	167.67	168.58	0.22	6.00
A-SL-124	168.58	169.32	0.10	5.00
A-SL-074	190.75	190.95	0.12	7
A-SL-074	190.95	191.25	0.09	7
A-SL-074	191.25	191.95	0.37	4
A-SL-074	191.95	192.85	0.07	2
A-SL-074	192.85	193.75	0.07	3
A-SL-074	193.75	194.65	0.03	1
A-SL-074	194.65	195.88	0.04	1
A-SL-074	195.88	197.10	0.02	2
A-SL-093	94.50	95.05	0.17	9.00
A-SL-093	95.05	95.65	0.14	11.00
A-SL-093	95.65	96.55	1.22	39.00
A-SL-093	96.55	97.45	0.16	10.00
A-SL-093	97.45	98.30	0.54	18.00
A-SL-093	98.30	99.05	0.11	3.00
A-SL-093	99.05	99.72	0.20	15.00
A-SL-093	99.72	100.93	0.18	10.00
A-SL-093	100.93	102.14	0.07	8.00
A-SL-093	102.14	103.35	0.26	15.00

Drill Hole Number	From (m)	To (m)	Au (g/t)	Ag (g/t)
A-SL-093	103.35	104.56	0.49	24.00
A-SL-093	104.56	105.45	0.25	10.00
A-SL-093	105.45	106.35	0.57	28.00
SL06-21	81.95	82.95	0.59	50.5
SL06-21	82.95	83.55	0.62	59.7
SL06-21	83.55	84.15	0.65	20.8
SL06-21	84.15	84.63	0.47	26
SL06-21	84.63	85.63	0.60	70.1
SL06-21	85.63	86.33	0.70	60.7
A-SL-126	129.20	129.76	49.70	1960
A-SL-126	129.76	130.42	46.90	1840
A-SL-126	130.42	131.12	207.00	8910
A-SL-126	131.12	131.82	169.50	5470
A-SL-126	131.82	132.56	38.90	1090
A-SL-126	132.56	133.61	134.00	3280
A-SL-126	133.61	134.34	2.53	104
A-SL-126	134.34	135.02	4.66	172
A-SL-126	135.02	135.48	39.60	1200
A-SL-064	187.20	187.80	41.60	1855.00
A-SL-064	187.80	189.30	16.90	347.00
A-SL-064	189.30	189.52	72.50	1280.00
A-SL-064	189.52	190.10	1.05	50.00
A-SL-064	190.10	191.00	7.96	205.00
A-SL-064	191.00	191.25	47.30	1450.00
A-SL-064	191.25	192.20	6.88	193.00
A-SL-064	192.20	193.25	26.60	675.00
A-SL-064	193.25	194.25	30.10	1065.00
SL06-28	93.85	94.50	34.60	722.00
SL06-28	94.50	95.50	33.47	742.00
SL06-28	95.50	96.46	56.06	978.00
SL06-11	63.10	64.10	26.62	606.00
SL06-11	64.10	64.70	5.86	454.00
SL06-18	49.31	50.00	37.19	369.00
SL06-18	50.00	50.86	83.80	1185.00
SL06-18	50.86	51.36	2.95	185.00

18.3.2 G&T Testwork Program – KM2437

Samples used for G&T's metallurgical testwork program KM2437 were those remaining from the PRA testwork program. The sample details are shown in Table 18-1.

18.3.3 G&T Testwork Program – KM2453

G&T testwork program KM2453 involved developing comminution data. Twin holes were drilled to obtain samples for this program. The sample details are shown in Table 18-3.

Table 18-3: Details of Metallurgical Sample – G&T Program KM2453

Comminution Interval Sampled			Exploration Assay Interval		
Metallurgical Hole No.	**From (m)**	**To (m)**	**Original Hole no.**	**From (m)**	**To (m)**
A-SL-116	210.8	213.9	SL-07-49	216.6	217.8
A-SL-117	131.0	139.5	SL-07-60	146.7	150.4
A-SL-118	59.0	66.0	SL-06-10	62.0	68.4
A-SL-120	71.2	85.0	SL-07-30	71.2	83.0
A-SL-121	60.3	65.0	SL-06-22	63.8	69.2
A-SL-129	44.0	54.5	SL-06-6	42.9	53.8

In addition to the above, material samples of wall rock were also included. Approximately 20kg of each of the following were added:

- Flow Banded sanadine dyke (code 12fb).
- Greenish porphyritic rhyolite dyke (code 12qfp)
- Quartz flooded contact breccia on edge of vein. Sometimes mineralized with low grade values (code H7bx).

18.3.4 G&T Testwork Program – KM2488

G&T metallurgical testwork program KM2488 evaluated primary grind size, addressed the effect of head grade on metallurgical response and identified the metallurgical response of material indicative of the first 3 quarters of the mine plan. In addition, flocculant screening, thickening, rheology and filtration testwork were performed on leach tails samples. Details of the composite samples used in this program are shown in Table 18-4.

Table 18-4: Details of Metallurgical Sample – G&T Program KM2488

Sample Identification	Sample Number	Drill Hole Number	From (m)	To (m)	Au (g/t)	Ag (g/t)
Mine Plan – 1st 3 Quarters	4126	SL06-23	27.70	29.60	3.65	149.02
Mine Plan – 1st 3 Quarters	4127	SL06-11	63.10	68.30	8.91	259.33
Mine Plan – 1st 3 Quarters	4605	SL06-19	76.00	77.00	26.79	632.00
Mine Plan – 1st 3 Quarters	4530	A-SL-085	36.65	37.32	7.36	233.00
Mine Plan – 1st 3 Quarters	4539	A-SL-098	73.80	74.96	91.50	1,090.00
Mine Plan – 1st 3 Quarters	4542	A-SL-098	78.44	79.60	53.10	783.00
Mine Plan – 1st 3 Quarters	4569	SL06-16	51	51.85	3.10	40.90
Mine Plan – 1st 3 Quarters	4570	SL06-16	51.85	52.85	16.04	416.00
Mine Plan – 1st 3 Quarters	4571	SL06-16	52.85	53.75	5.97	292.00
Mine Plan – 1st 3 Quarters	4523	A-SL-028	77.10	77.60	21.10	747.00
Mine Plan – 1st 3 Quarters	4524	A-SL-028	77.60	78.00	30.30	1,270.00
Mine Plan – 1st 3 Quarters	4526	A-SL-028	78.65	79.54	14.55	708.00
Mine Plan – 1st 3 Quarters	4136	SL06-05	47.05	48.65	4.98	118.81
Mine Plan – 1st 3 Quarters	4565	SL06-16	44.55	45.37	183.54	3,777.00
Mine Plan – 1st 3 Quarters	4566	SL06-16	46.25	47.00	48.56	1,030.00
Mine Plan – 1st 3 Quarters	4567	SL06-16	47.00	50.00	34.69	688.00
Mine Plan – 1st 3 Quarters	4615	A-SL-060	171.71	172.71	37.3	1,150
Mine Plan – 1st 3 Quarters	4506	A-SL-077	44.94	45.80	39.4	1,740
Mine Plan – 1st 3 Quarters	4619	A-SL-069	230.67	231.67	7.13	215
Mine Plan – 1st 3 Quarters	4622	A-SL-104	277.36	278.40	7.34	308
Mine Plan – 1st 3 Quarters	4508	A-SL-107	117.10	117.89	2.02	91
Mine Plan – 1st 3 Quarters	4626	SL06-21	53.10	54.10	17.43	524
Mine Plan – 1st 3 Quarters	4631	SL06-22	67.20	68.20	24.51	775
High Grade sample	4540	A-SL-098	74.96	76.12	133.50	1,680.00
High Grade sample	4520	A-SL-028	74.50	75.50	4.88	418.00
High Grade sample	4146	A-SL-010	112.6	114.8	5.70	214.86
High Grade sample	4618	A-SL-069	229.67	230.67	9.10	279.00
High Grade sample	4625	SL06-21	52.50	53.10	6.53	218.00
High Grade sample	4128	SL06-10	64.2	76.95	26.88	685.73
High Grade sample	4142	SL06-28	93.15	93.85	22.85	491.56
High Grade sample	4544	A-SL-127	141.67	142.31	27.5	526
High Grade sample	4547	A-SL-127	143.50	144.55	46.1	853
High Grade sample	4511	A-SL-001	127.70	128.40	43.53	2,583
High Grade sample	4512	A-SL-001	128.40	129.40	46.8	1,234
High Grade sample	4514	A-SL-001	131.10	131.93	19.25	1,036
High Grade sample	4582	SL06-17	62.42	63.42	29.78	1,169
High Grade sample	4583	SL06-17	63.42	64.42	46.55	1,665
Average Grade sample	4600	SL06-18	49.31	50.00	37.19	369.00
Average Grade sample	4603	SL06-19	74.00	75.00	11.92	292.00
Average Grade sample	4604	SL06-19	75.00	76.00	1.80	58.50
Average Grade sample	4534	A-SL-087	43.68	44.79	5.29	94.00

Sample Identification	Sample Number	Drill Hole Number	From (m)	To (m)	Au (g/t)	Ag (g/t)
Average Grade sample	4535	A-SL-087	44.79	45.83	7.94	301.00
Average Grade sample	4555	SL06-08	53.58	54.58	1.45	19.90
Average Grade sample	4556	SL06-08	54.58	55.58	19.87	202.00
Average Grade sample	4557	SL06-08	55.58	56.58	7.57	74.90
Average Grade sample	4146	A-SL-010	112.6	114.8	5.70	214.86
Average Grade sample	4616	A-SL-060	172.71	173.71	7.89	585.00
Average Grade sample	4507	A-SL-077	45.80	46.80	1.155	158.00
Average Grade sample	4623	A-SL-104	278.40	279.44	7.34	211.00
Average Grade sample	4132	SL06-01	45	56	23.93	589.87
Average Grade sample	4131	SL06-13	88.4	92.9	33.14	1,004.85
Average Grade sample	4549	A-SL-127	145.20	146.00	25.7	559.00
Average Grade sample	4550	A-SL-127	146.00	146.73	53.7	1,380.00
Average Grade sample	4572	SL06-17	53.60	54.60	125.17	2,400.00
Average Grade sample	4577	SL06-17	58.22	59.22	27.61	474.00
Low Grade Sample	4600	SL06-18	49.31	50.00	37.19	369.00
Low Grade Sample	4601	SL06-18	50.86	51.36	2.95	185.00
Low Grade Sample	4528	A-SL-085	35.27	36.00	3.32	205.00
Low Grade Sample	4529	A-SL-085	36.00	36.65	1.05	96.00
Low Grade Sample	4531	A-SL-085	37.32	38.45	6.49	175.00
Low Grade Sample	4612	A-SL-066	273.45	274.15	14.4	317.00
Low Grade Sample	4138	SL06-07	37.4	41.5	2.34	45.01
Low Grade Sample	4145	A-SL-006	138.95	142	11.48	388.28
Low Grade Sample	4616	A-SL-060	172.71	173.71	7.89	585
Low Grade Sample	4617	A-SL-060	173.71	174.65	3.93	348
Low Grade Sample	4620	A-SL-069	231.67	232.67	3.22	136
Low Grade Sample	4624	A-SL-104	279.44	280.50	5.24	162
Low Grade Sample	4502	SL06-04	84.55	85.10	3.916	208.5
Low Grade Sample	4503	SL06-04	85.10	85.50	0.785	41
Low Grade Sample	4504	SL06-04	85.50	86.07	6.13	394.2
Low Grade Sample	4131	SL06-13	88.4	92.9	33.14	1,004.85
Low Grade Sample	4560	SL06-14	33.30	34.31	0.112	7
Low Grade Sample	4561	SL06-14	35.50	36.54	0.205	16
Low Grade Sample	4543	A-SL-127	141.50	141.67	11.1	118
Low Grade Sample	4546	A-SL-127	143.10	143.50	20.1	627
Low Grade Sample	4554	A-SL-127	148.76	149.51	5.02	214
Low Grade Sample	4510	A-SL-001	127.00	127.70	32.86	1,011
Low Grade Sample	4579	SL06-17	60.10	61.00	0.684	23.3
Low Grade Sample	4580	SL06-17	61.00	61.57	0.868	40.8
Low Grade Sample	4581	SL06-17	61.57	62.42	1.317	32.3
Low Grade Sample	4595	SL06-17	73.50	74.10	20.62	371

18.4 MINERALOGY/PETROGRAPHY

Two petrographic analyses have been performed on samples for San Luis. One was prepared by Dr. Craig Leitch and the other was prepared by BISA.

18.4.1 Report by Dr. Craig Leitch, Ph.D, P. Eng.

Dr. Leitch reports that he analyzed nine samples of sawn core to determine:

- Mode of occurrence of gold and silver.
- Locking of metal values with quartz or adularia.
- Secondary enrichment of gold or silver.

Dr. Leitch reports that "*the samples mostly consist of epithermal-looking quartz-carbonate-sericite vein material with textures ranging from breccia to crustiform, colloform banded, comb- or cockade-textured, to locally vuggy or very fine-grained ("chalcedonic"), with only minor sulfides. Clasts in breccia veins are mainly intensely altered, to quartz-sericite-carbonate-local chlorite, rutile, and only rarely retain vestiges of former (felsic to intermediate, volcanic to hypabyssal?) origin. Carbonate likely includes calcite and dolomite (plus local Fe-calcite and ankerite?). Sulfides are mainly fine-grained pyrite, commonly partly to wholly replaced by limonite, but in several samples trace to significant base-metal sulfides, including chalcopyrite, sphalerite (ranging from colourless or pale yellow, low Fe, to almost opaque, due to minute oriented inclusions of chalcopyrite), galena (rarely intergrown with acanthite or tetrahedrite?), and rare possible marcasite (?). In 5 (possibly 6) of the 7 samples from drill holes SL609, 610, 613 and 702, rare small (<15 to 100 micron) particles of yellow-brown, partly tarnished (?) possible native Au or electrum occur, mostly associated with either relatively late carbonate, sericite, quartz veinlets or fractures; they only locally appear to be locked within carbonate or pyrite (or limonite after pyrite), not quartz or adularia. In the two samples from SL01 and 02, traces of highly tarnished, possible native Ag/electrum (?) as ragged particles or aggregates up to 200 microns across are also associated with late quartz, sericite veinlets or open fractures. No evidence of enrichment of Au or Ag was noted in the sections"*.

Samples analyzed by Dr. Leitch are shown in Table 18-5 below.

Table 18-5: Sample for Petrographic Analysis by C. Leitch

SL 609	39.7
SL 609	40.5
SL 610	66.70
SL 610	68.6
SL 610	70.75
SL 613	91.2
SL 702	142.8
SL 01	48.6
SL 02	76.4

18.4.2 Laboratory Report COD Project: 470LA0005A – BISA

BISA performed mineralogical analyses on 11 samples collected from the Ayelén vein area. The samples were from diamond drill core and surface trenches. Details of the samples are shown in Table 18-6 and approximate locations are shown in Figure 18-1.

Table 18-6: Mineralogical Assessment – Sample Details

Drill Hole ID	Sample ID	Depth form Surface (m)	Description
A-10	196255	Surface	Vein with bands of gray and white quartz.
SL06-09	56569	40.00	Quartz with oxides and dissolved sulphides.
SL06-16	56894	45.67	Fractured quartz, filled with Oxides.
SL06-03	56382	57.25	Banded quartz with gray strips.
ASL-089	61032	82.40	White quartz with chlorites and dissolved pyrite.
SL06-13	56732	90.68	Altered volcanic rock with dissolved pyrite.
ASL-107	61660	116.31	Quartz with sulphides dissolved.
SL06-25	57387	120.85	White quartz with dissolved pyrite.
ASL-060	59443	149.40	Wall-Vein contact – dissolved pyrite.
ASL-053	59265	171.50	White quartz fractured, filled with oxides.
ASL-104	4317	281.30	Breccia with dissolved sulphides.

BISA reports that vein material mostly consists of quartz, calcite, sericite and illite. Quartz is present in different forms such as hyaline quartz, gray and chalcedonic with banded, crustiform and brecciated textures.

Microscopic evaluation indicates three common groups of minerals:

- Native minerals (gold, silver, electrum).
- Sulphides (pyrite, chalcopyrite, sphalerite, galena, argentite, pyrrhotite, marcasite).
- Oxides (Hematite, Limonite, Goethite).

Bisa reports that gold is mainly present as electrum and in smaller amount as free fine-grain gold (0.002mm to 0.04mm). Native gold appears to contain some silver giving it a pale color and, in some cases, causing it to be confused with electrum. Silver, in addition to electrum, is present as argentite and argento-jarosite associated to pyrite and gray quartz. The amount of silver in electrum varies and often it is difficult to identify if silver is native silver or electrum.

Pyrite is the most abundant sulphide. It is often associated with occurrences of native gold, native silver and electrum as indicated in Figure 18-2. Chalcopyrite, galena and sphalerite are present in trace amounts. Their amounts tend to increase with depth. Hematite, goethite and limonites are observed in the upper part of the mineralized system.

Figure 18-1: Mineralogical Assessment – Approximate Sample Locations

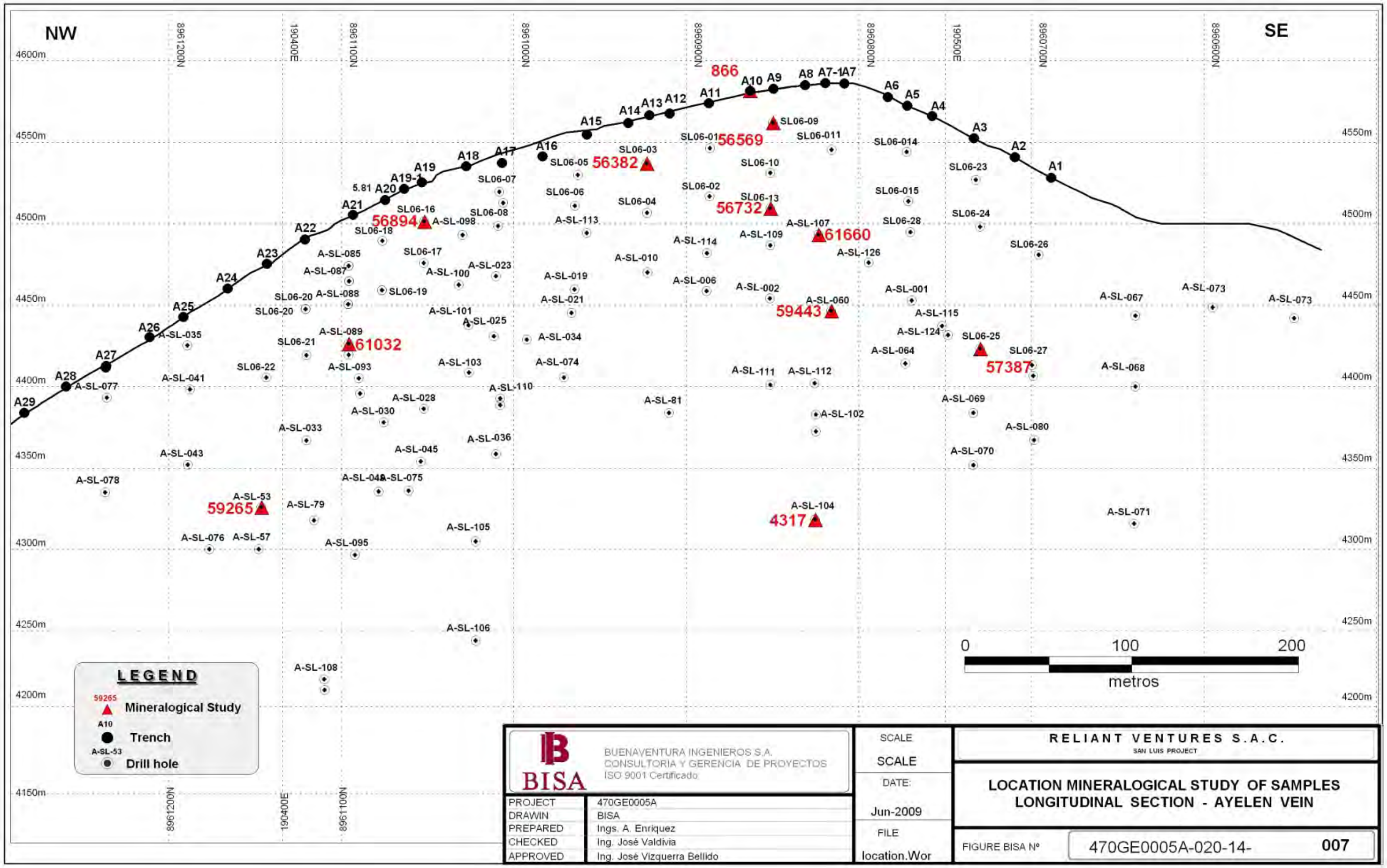
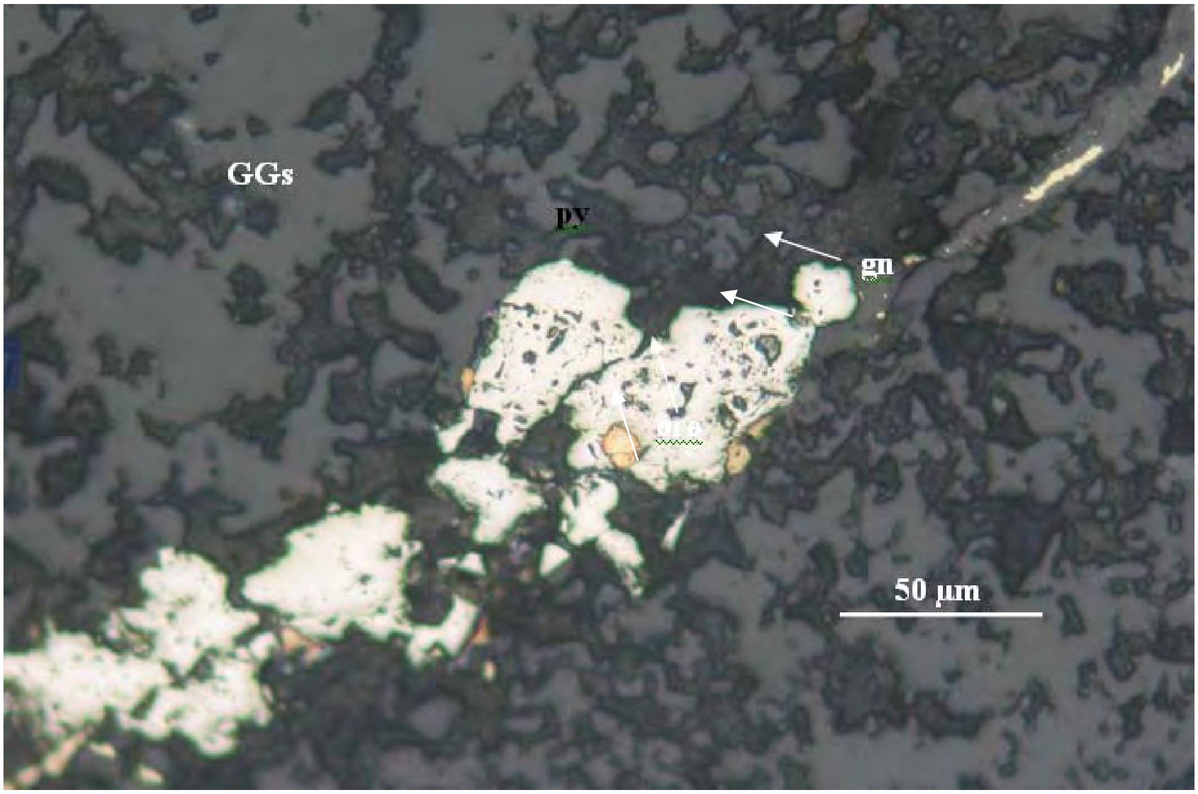


Figure 18-2 Example of Mineralogical Occurrences



Native gold grain filling up porosities in the pyrite (py); the biggest grain measures 0.02mm. Pyrite is present filling up a fracture in the gangue (GGs). In addition to gold, there are some grains of galena (gn) filling porosities in the pyrite.

Table 18-7 summarizes the results of the mineralogical analysis for each sample.

BISA performed X-Ray diffraction analysis on 5 samples in order to quantify and characterize the mineralogy as well as identify the minerals of alteration. Details of the samples are shown in Table 18-8 and the approximate locations where they were collected are shown in Figure 18-3.

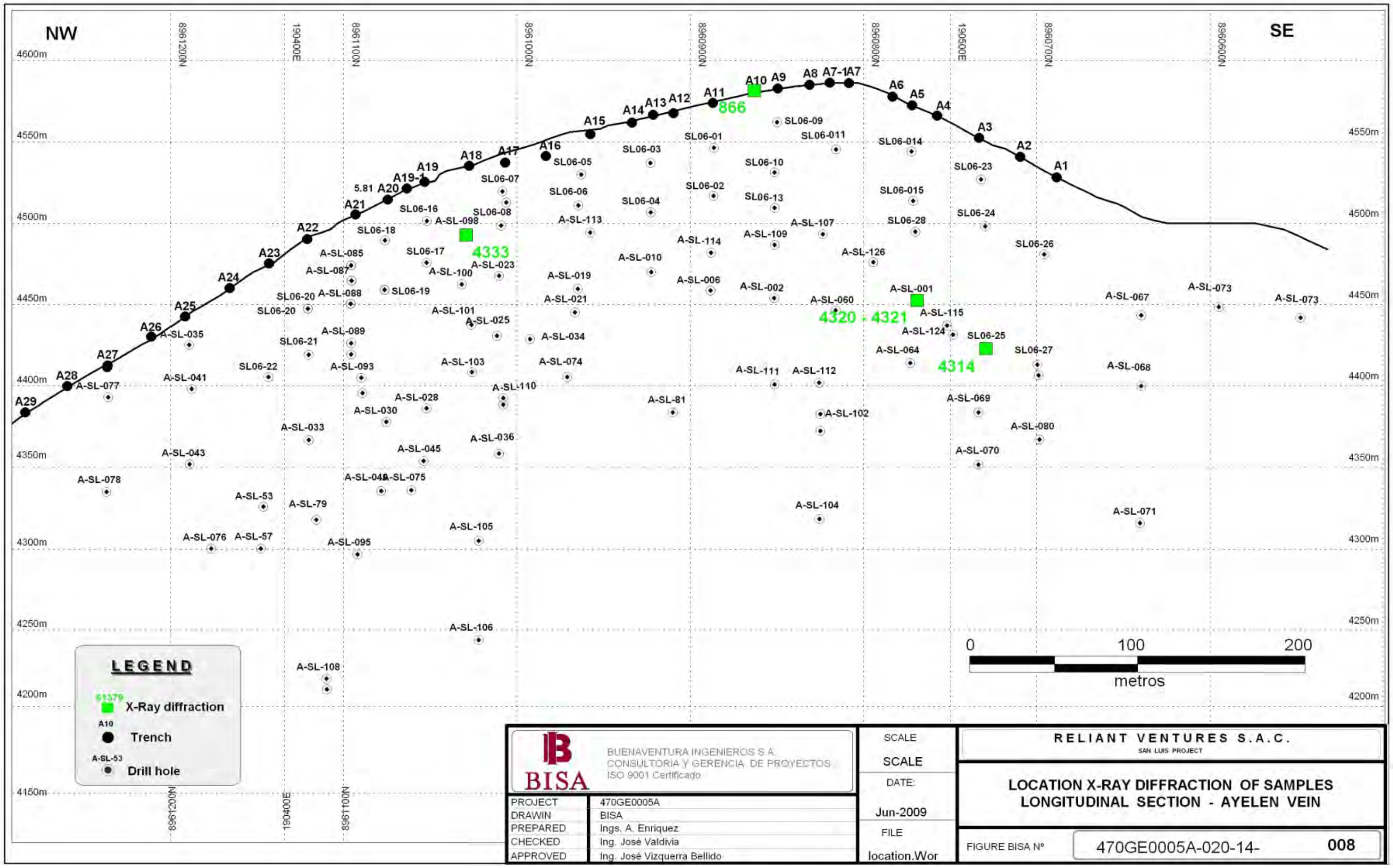
Table 18-7: Mineralogy Description

Sample ID	Description – Mineralogy
196255	Native gold (0.002mm – 0.04mm) disseminated and associated to argentite, native silver, argento-jarosite, electrum, goethite, hematite, limonites. Filling fractures and disseminated.
56569	Pyrite, sphalerite, chalcopyrite, galena, argentite, native gold and/or electrum, native silver, hematite and limonites. Disseminated and filling fractures.
56894	Sphalerite, argentite, pyrite (1%), native gold (0.02mm), native silver, electrum? hematite (20%), limonites. Disseminated and filling fractures.
56382	Pyrite (Tzs), sphalerite - chalcopyrite I, chalcopyrite II, galena, argentite, native gold and/or electrum, associated to pyrite. Disseminated and filling fractures.
61032	Pyrite, chalcopyrite, galena, argentite associated to pyrite and hematite. Disseminated.
56732	Rutile, pyrite (1%), sphalerite? , chalcopyrite. Disseminated.
61660	Rutile, pyrite, phyrrotite, sphalerite, marcasite, chalcopyrite, galena. Disseminated and filling fractures.
57387	Rutile, pyrite, sphalerite, chalcopyrite, galena. Disseminated.
59443	Pyrite, sphalerite, phyrrotite, chalcopyrite, argentite, native gold (< 0.045mm); electrum and native silver in the gangue and pyrite. Disseminated and filling fractures.
59265	Pyrite, chalcopyrite, galena, argentite, native gold and/or electrum, native silver, hematite and limonites. Disseminated and filling fractures
4317	Pyrite (1%), sphalerite - chalcopyrite I, chalcopyrite II, argentite, native silver, native gold. Disseminated and filling fractures.

Table 18-8: Samples Details - X-Ray Diffraction Analysis

Drill Hole ID	Sample ID BISA/SPP	Depth (m)	Description
A-10	866/196283	Surface	Vein with bands of gray and white quartz.
SL06 – 25	4314/57386	120.77	White quartz with pyrite diss and sulphides.
ASL – 001	4320/57868	126.10	Banded quartz with gray strips.
ASL – 001	4321/57874	130.15	Banded quartz with gray strips.
ASL – 098	4333/61379	79.60	White quartz with gray bands.

Figure 18-3: X-Ray Diffraction Analysis – Approximate Sample Locations



X-Ray diffraction indicates the samples mostly consist of quartz – calcite – muscovite – clinochlore - anatase. The quartz (SiO_2) content of the samples analyzed varies from 51.63% to 86.84%. The calcite ($CaCO_3$) content ranges from 0% to 43.37% while the muscovite ($KAl_2(Si_3Al)O_{10}(OH,F)_2$) content from 0% to 41.49%. The results are summarized in Table 18-9.

Table 18-9: X-Ray Diffraction Results

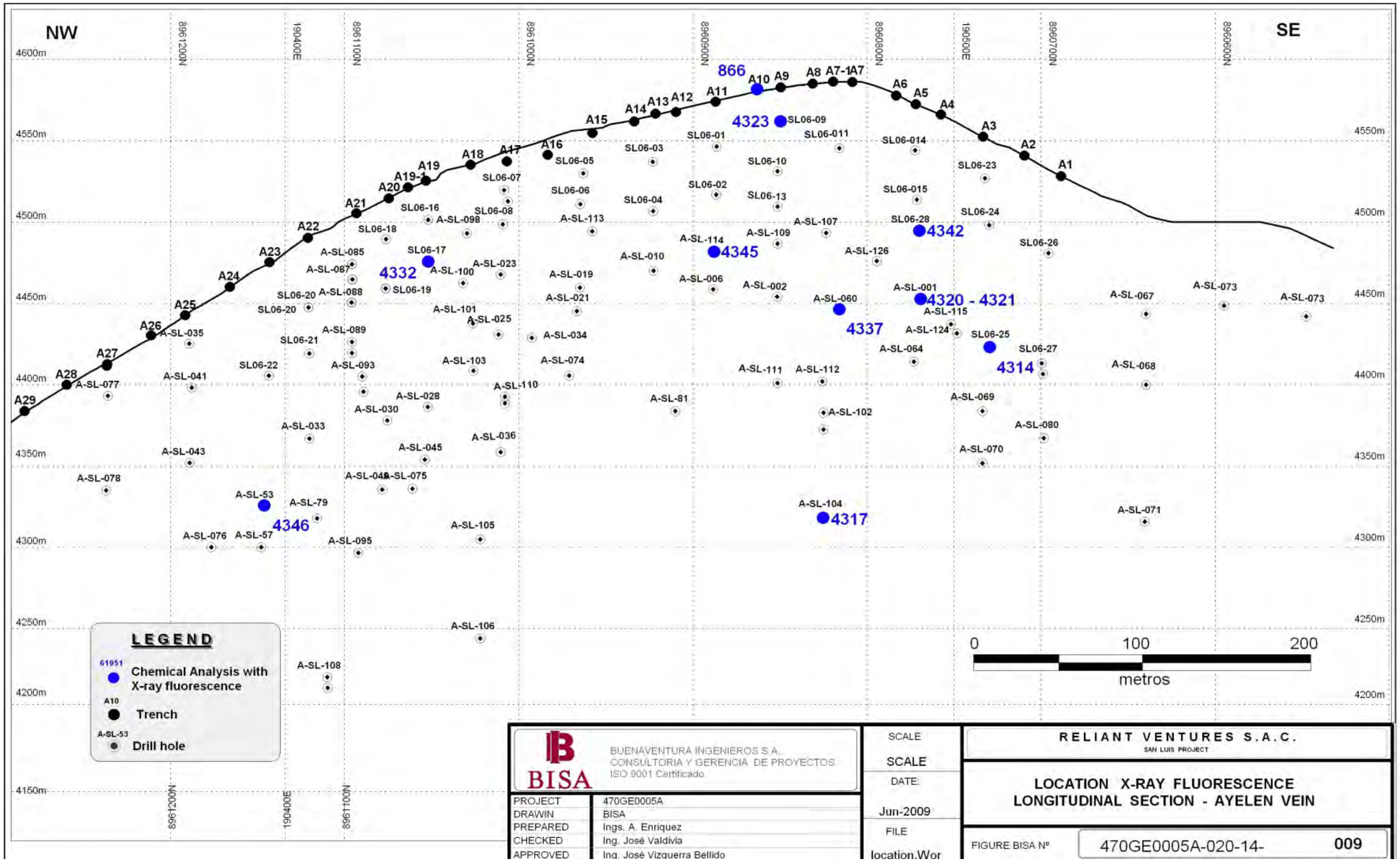
Sample ID BISA / SSP	Quartz %	Calcite %	Clinochlore %	Muscovite %	Anatase %
4314 / 57386	51.63	43.37	5.01	-	-
4320 / 57868	77.87	7.54	-	14.58	-
4321 / 57874	78.51	12.27	2.93	6.29	-
4333 / 61379	58.17	-	-	41.49	0.34
866 / 196255	86.84	-	-	13.15	-

BISA performed X-Ray fluorescence analysis on 11 samples to quantify the amounts of silica, sulphur and iron oxides. Details of the samples are shown in Table 18-10 and their approximate locations are shown in Figure 18-4.

Table 18-10: Sample Details – X-Ray Fluorescence Analysis

Hole	From	To	Interval (m)	SSP Sample	BISA Sample
A-10	**Surface**		**0.60**	**196283**	**866**
SL06 – 25	120.77	121.77	1.00	57386	4314
ASL – 104	280.94	281.83	0.89	61533	4317
ASL – 001	126.10	127.00	0.90	57868	4320
ASL – 001	130.15	131.10	0.95	57874	4321
SL06 – 09	39.58	40.58	1.00	56569	4323
SL06 – 17	76.10	76.65	0.55	56983	4332
ASL – 060	170.71	171.71	1.00	59467	4337
SL06 – 28	94.50	95.50	1.00	57798	4342
ASL – 114	117.79	118.91	1.12	61951	4345
ASL – 053	171.10	172.15	1.05	59265	4346

Figure 18-4: X-Ray Fluorescence Analysis – Approximate Sample Locations



Results from X-Ray fluorescence analysis are shown in Tables 18-11 and 18-12. They indicate the silica content varies from 18.97% to 41.58%. Sulphur varies from 0.053% to 0.357%. Iron oxides content varies from 0.79% to 2.57%.

Table 18-11: Semi-Quantitative Chemical Analysis STANDARDLESS

Element (%)	**Sample**										
	4320	**4314**	**4342**	**4317**	**4321**	**866**	**4332**	**4323**	**4345**	**4346**	**4337**
Ag	0.055	0.170	0.076	0.060	0.428	1.453	0.159	0.594	0.121	0.105	0.090
Al	3.471	1.825	2.555	2.854	2.137	2.589	2.966	2.269	3.102	4.145	1.751
Ca	3.076	18.49	1.040	3.235	5.055	0.030	7.591	9.038	2.083	1.659	1.992
Co	0.009	0.006	0.010	0.013	0.009	0.024	0.007	0.008	0.007	0.007	0.007
K	1.590	0.661	1.083	2.116	0.807	1.090	1.357	0.547	1.946	1.680	0.724
Mg	0.168	0.350	0.101	0.111	0.423	0.147	0.119	1.115	0.206	0.380	0.076
Mn	0.043	0.087	0.028	0.034	---	---					
Na	0.037	---	---	0.057	---	---	---	---	0.045	---	---
P	0.019	---	---	---	---	---					
S	0.357	0.208	0.113	0.109	0.412	0.186	0.060	---	0.157	0.053	0.316
Se	0.002	0.004	0.002	---	0.011	0.040	0.003	---	0.004	0.002	0.003
Si	34.82	18.97	38.19	35.37	33.26	41.58	30.18	27.41	36.53	34.86	38.04
Ti	0.084	0.027	---	---	---	---					
W	0.074	0.050	0.066	0.100	0.075	0.156	0.053	0.060	0.053	0.055	0.084
As	---	0.004	---	---	0.001	0.022	---	---	0.005	---	---
Re	---	0.004	---	---	---	0.011					
Rh	---	0.002	---	---	---	0.005					
Ba	---	---	0.084	---	---	---					
Au	---	---	---	---	0.001	0.017	0.003	0.007	---	---	---
LOI	---	17	---	---	---	---	5	8	---	---	---

Table 18-12: Quantitative Chemical Analysis GEO-QUANT

Element %	Unit	Sample											Limit	
		4320	4314	4342	4317	4321	866	4332	4323	4345	4346	4337	Min	Max
Sc	ppm	---	---	---	---	5	--	5	8	2	4	2	1.5	100
TiO2	%	---	---	---	---	0.02	--	0.01	---	0.01	0.16	0.04	0.001	2.6
V	ppm	---	---	---	---	17	---	28	11	7	43	16	1.6	500
Cr	ppm	---	---	---	---	10	---	4	---	11	16	21	2.8	3000
MnO	%	---	---	---	---	0.09	---	0.07	0.19	0.05	0.04	0.04	0.001	0.5
Fe2O3	%	1.53	1.65	1.45	0.79	2.33	2.26	1.48	2.57	1.32	2.13	1.15	0.001	20
Ni	ppm	5	4	3	2	3	3	4	3	3	4	---	1.6	2500
Cu	ppm	7	68	12	211	348	8	129	238	184	97	40	2.1	1000
Zn	ppm	50	114	71	197	265	73	---	---	---	11	46	1.6	3000
As	ppm	48		47	2	---	---	74	36	92	102	39	2.3	350
Rb	ppm	81	33	64	99	54	81	43	89	67	17	36	0.8	3500
Sr	ppm	44	122	20	57	58	10	1	4	---	4	3	0.8	1500
Y	ppm	4	3	1	1	3	5	6	4	3	30	9	0.9	150
Zr	ppm	30	12	4	6	5	5	6	11	5	6	5	1.4	1000
Nb	ppm	5	8	5	4	9	16	6	7	4	3	31	0.9	1000
Mo	ppm	15	14	9	3	18	41	144	67	455	127	73	0.7	150
Ba	ppm	---	---	---	---	199	---	28	25	25	26	25	4.9	2500
La	ppm	---	---	---	---	---	---	159	651	256	196	114	4.5	350
Ce	ppm	---	---	---	---	31	---	2	2	3	3	---	4.6	2500
Pb	ppm	92	277	98	237	563	1024	55	296	189	183	27	1.4	2500
Th	ppm	2	4	---	---	4	---						1.2	1000

18.5 BOND WORK INDEX AND CRUSHER ABRASION INDEX

Bond Ball Mill Work Index tests have been performed by PRA and G&T. G&T also performed Crusher Work Index and Abrasion Index tests. The results from the PRA tests are shown in Table 18-13 while the results from the G&T tests are shown in Table 18-14.

Table 18-13: Bond Ball Mill Work Index Tests – PRA

Sample	Wi (kWh/tonne)
Ayelén #1	14.9
Ayelén #2	16.0
Ayelén #3	16.8
Ayelén #4	15.4
Ayelén #5	15.3

Table 18-14: Bond Ball Mill Work Index Tests – G&T

Sample	Test Program	Bond Ball Mill Wi (kWh/tonne)	Crusher Wi (kWh/tonne)	Abrasion Index
A-SL-116	KM2453	10.7	-	0.51
A-SL-117	KM2453	14.9	14.2	0.76
A-SL-118	KM2453	13.2	10.8	0.54
A-SL-120	KM2453	14.4	11.0	0.22
A-SL-121	KM2453	9.5	8.9	0.23
A-SL-129	KM2453	13.3	6.2	0.57
H7BX	KM2453	16.5	8.6	0.26
12FQ	KM2453	16.5	-	0.47
12QFP	KM2453	16.3	17.4	0.29
A	KM2488	14.7	-	0.38
B	KM2488	15.3	-	0.40
C	KM2488	14.8	-	-
D	KM2488	15.0	-	0.45

The Bond ball mill work index ranged between 9.5 and 16.5kWh/tonne for the samples tested. Samples A-SL-116 and A-SL-121 were both relatively soft with Bond ball mill work indices of about 10kWh/tonne. The samples H7BX, I2FP and I2QFP were the hardest with Bond ball mill work indices of about 16kWh/tonne. All the samples, other than A-SL-116 and 121, can be considered of moderate hardness based on the Bond ball mill work index data.

The crusher work index ("Cwi") results indicate that samples A-SL-117 and I2QFP have relatively high Cwi values at 14 and 17.4kWh/tonne, respectively. The remaining samples produced Cwi values between 6 to 11kWh/tonne.

Abrasion indices typically range from as low as 0.026 to about 0.6. Over half of the San Luis samples in program KM2453 generated abrasion indices of 0.5 or greater and consequently indicate a high degree of steel wear rates can be expected with processing these materials. Samples A-SL-118, 120, 129 and H7BX produced much lower abrasion indices; around 0.20. Composite samples (A to D) that were tested in program KM2488 indicated an intermediate abrasion index of approximately 0.4.

MQes has used a Bond Ball Mill Work Index of 16.0kWh/t, a crusher Work Index of 14kWh/t and an abrasion Index of 0.5 for this study.

18.6 GRAVITY CONCENTRATION TESTS

Gravity concentration tests have been performed by PRA, Plenge and G&T. The PRA results are shown in Table 18-15. The G&T results are shown in Table 18-16.

Table 18-15: Gravity Concentration Results – PRA

Sample	Test No.	Grind P_{80} (μm)	Pan Conc. Grade (g/t)		Recovery (%)	
			Au	Ag	Au	Ag
Ayelén #1	GC1	114	6,673	26,173	17.8	3.6
Ayelén #1	GC2	79	7,605	29,435	22.0	4.5
Ayelén #1	GC3	63	9,011	35,049	22.3	4.6
Ayelén #1	GC10	39	10,635	34,479	24.1	4.2
Ayelén #1	F8	70	24,971	55,953	19.0	2.0
Ayelén #4	GC4	99	1,423	7,214	14.9	2.9
Ayelén #4	GC5	82	1,632	7,682	21.1	3.2
Ayelén #4	GC6	60	2,609	10,221	17.7	2.2
Ayelén #4	GC13	40	1,670	8,376	14.9	2.5
Ayelén #4	F11	59	4,122	11,969	20.1	1.8
Ayelén #5	GC7	102	2,224	12,150	10.3	2.3
Ayelén #5	GC8	77	3,531	14,321	9.3	1.6
Ayelén #5	GC9	45	4,319	13,982	11.5	1.5
Ayelén #5	GC15	37	4,957	22,535	14.9	2.5
Ayelén #5	F14	58	11,413	37,262	12.0	1.3

The gravity rougher recovery tests used a Knelson centrifugal concentrator operated in an open cycle single pass. The Knelson concentrate was then hand panned. The test results show that 10% to 20% of the gold can be expected to report to the cleaned gravity concentrate, and potentially higher with increasing feed grades, such as with Ayelén 1 sample. The gold and silver grade and recovery in the cleaned concentrate improves with finer grinding. PRA attributes the higher concentrate grades for tests F8, F11 and F14 to panning technique.

Plenge performed a gravity concentration test on approximately 12 kg of material ground to a P_{80} of 65 mesh (210 µm). Gold and silver recovered to the concentrate was 33.7% and 9.1% respectively. The concentrate grade was 2,265 g/t Au and 13,865 g/t Ag. It is not indicated if this is a rougher or pan concentrate. The calculated head grade of the sample was 18.01g/t Au and 410.6 g/t Ag.

Table 18-16: Gravity Concentration Results – G&T

Composite	Test Program	Nominal K80 (microns)	Calc. Feed Grade (g/t)		Recovery (%)					
					Pan Conc.		Pan Tails		Knelson Tails	
			Au	Ag	Au	Ag	Au	Ag	Au	Ag
Ayelén #1	KM2437	45	39.4	919	24.1	0.9	14.4	6.0	69.0	95.3
Ayelén #1	KM2437	45	38.6	904	23.7	4.0	16.7	6.7	63.3	91.6
Ayelén #4	KM2437	55	9.3	341	17.8	2.6	7.9	3.2	72.4	92.8
Ayelén #4	KM2437	55	10.0	324	26.6	3.0	13.6	4.1	59.9	99.0
A	KM2488	74	23.1	624	27.7	5.1	8.5	3.8	59.2	81.1
B	KM2488	74	26.0	663	22.7	4.6	21.1	7.5	63.4	76.1
C	KM2488	140	20.6	534	19.0	4.1	15.9	10.1	70.2	95.9
C	KM2488	74	21.0	568	27.9	5.8	11.9	6.8	66.5	83.9
C	KM2488	74	22.5	570	24.0	4.4	12.5	3.5	65.8	91.7
C	KM2488	45	22.1	584	34.5	5.0	14.5	2.8	57.1	85.4
C	KM2488	45	24.2	548	33.8	4.9	15.6	8.1	52.6	90.5
D	KM2488	74	5.54	198	23.9	5.6	12.6	5.5	61.9	82.8

Note: Calculated feed grades include leach feed assay data

Results from the above table are shown in Figures 18-5 and 18-6.

Figure 18-5: Gravity Recovery for Gold Versus Feed Grade

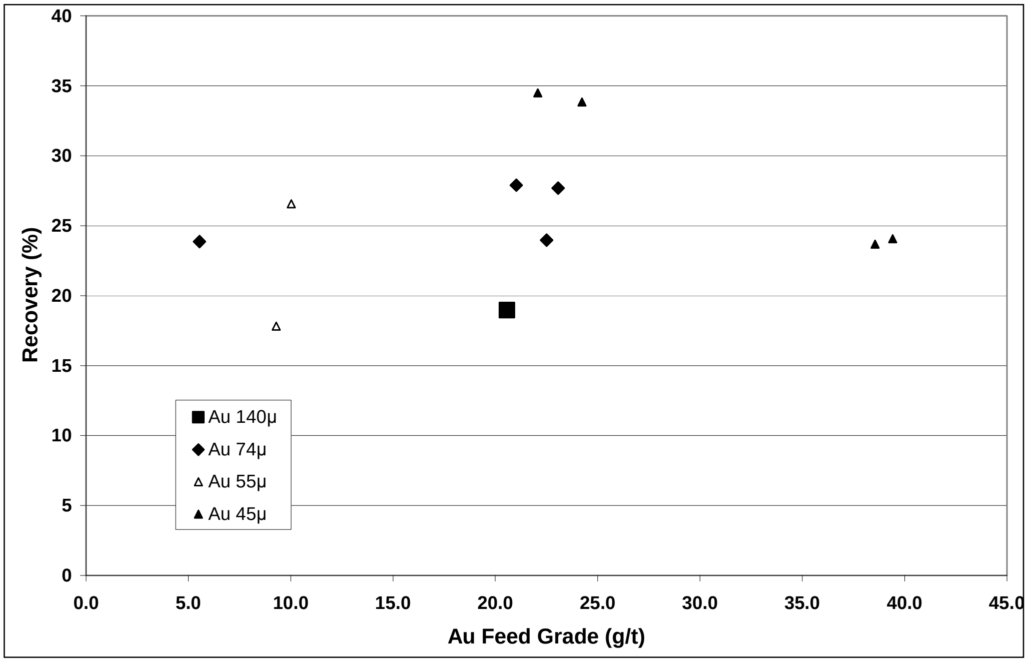
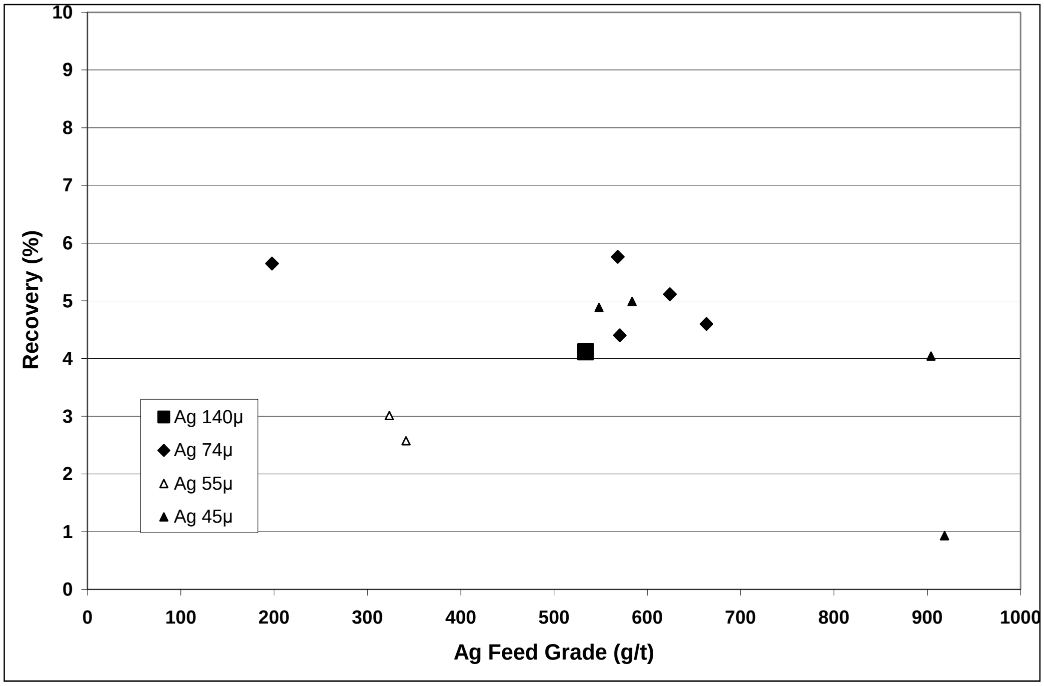


Figure 18-6: Gravity Recovery for Silver Versus Feed Grade



Gold recoveries by a gravity circuit increase as primary grind becomes finer. Silver recoveries remain consistently near or below 5% regardless of silver head grade or primary grind size.

To reduce the chance of coarse gold losses in subsequent treatment following the gravity circuit, MQes has included gravity recovery in the process design to capture any coarse gold and silver and also to assist in reducing cyanide leach retention time. At a primary grind of 80% passing 74 microns, test results suggest that gold and silver recoveries in a gravity circuit should be in the range of 25% and 5% respectively. Although test results indicate a slight improvement in gravity gold recovery in going from a P_{80} of 74 microns to one of 45 microns, a 45 micron grind is quite fine and has downstream effects on criteria such as settling rates, rheology, reagent consumption and other factors. Consequently, MQes has chosen a primary grind size of 80% passing 74 microns for purposes of this study.

18.7 CYANIDE LEACHING TESTWORK

Cyanide leach tests have been performed by PRA, Plenge and G&T. Their results are summarized below.

18.7.1 PRA Testwork

PRA performed cyanide leach tests with and without prior gravity recovery. The results are summarized in Table 18-17.

Table 18-17: Cyanidation with and without Gravity Concentration1

Sample ID	Test No.	Test Condition	Tails Grade (g/t)		Recovery (%)			
			Au	Ag	Au 24h	Au End	Ag 24h	Ag End
Ayelén #1	GC2	72h gravity	2.28	169	85.8*	94.3	58.7*	78.0
Ayelén #1	C6	48h No grav	2.57	212	88.0	94.3	62.1	77.4
Ayelén #4	GC5	72h gravity	0.41	45.5	87.2*	95.4	69.5*	83.4
Ayelén #4	C9	48h No grav	0.45	54.9	92.9	95.0	74.9	84.4
Ayelén #5	GC8	72h gravity	1.38	154	77.7*	95.2	61.7*	77.4
Ayelén #5	C10	48h No grav	1.76	173	85.6	94.2	59.7	76.8

Notes: * = 24h recovery for tests GC 2,5,8 does not include gold recovered to gravity circuit. 1. = Reproduced from Tables 4.5 to 4.7 in PRA report

It is noted that in tests where gravity concentration was used prior to leaching, the intermediate recoveries (24 hour) do not include these recoveries. Ultimate recoveries (“end” as indicated in Table 18-18) for the tests are similar.

Using a 72 hour leach time, PRA performed a series of cyanidation leach tests over a range of grind sizes. The results are summarized in Table 18-18.

Table 18-18: Effect of Grind Size on Recoveries2 – PRA

Sample	Test No.	Grind P_{80} (µm)	Tails Grade		Recovery* (%)	
			Au (g/t)	Ag (g/t)	Au	Ag
Ayelén #1	GC1	114	2.37	188	92.8	75.1
Ayelén #1	GC2	79	2.28	169	92.7	77.0
Ayelén #1	GC3	63	1.26	151	96.1	79.8
Ayelén #1	C10	39	0.86	84	97.2	88.0
Ayelén #1	C1**	1,100	12.95	384	73.4	59.1
Ayelén #4	GC4	99	0.47	46.4	95.2	84.3
Ayelén #4	GC5	82	0.41	45.5	94.2	82.8
Ayelén #4	GC6	60	0.34	48.5	96.0	84.9
Ayelén #4	GC13	40	0.35	35.2	96.2	88.8
Ayelén #4	C4**	1,398	3.26	149	70.8	57.5
Ayelén #5	GC7	102	2.53	163	89.7	75.4
Ayelén #5	GC8	77	1.38	154	94.8	77.0
Ayelén #5	GC9	45	1.02	101	95.6	83.9
Ayelén #5	GC14	37	0.59	63.8	97.1	89.3
Ayelén #5	C5**	1,140	8.64	400	65.3	48.9

Notes: ***** = Recovery does not include gravity recovery. ****** = Test did not include gravity pretreatment. **1.** = Reproduced from Tables 4.8 to 4-10 in PRA report.

The test results listed in Table 18-18 indicate that gold and silver recoveries increase as primary grind size is reduced from a P_{80} of approximately 100µm to a P_{80} of approximately 40µm. These preliminary results indicated that a study of grind size optimization was needed. Tests C1, C4 and C5 had very coarse grind (crush) sizes. Such sizes are not typical for leaching. The results demonstrate, however, that a processing alternative such as heap leaching is not suitable. Treating such high grade material via heap leaching is not standard practice.

Leach kinetic data suggests that, as well as a fine primary grind size and gravity concentration, extended leach times are required to maximize gold and silver recoveries. Although the majority of the gold and silver recoveries are achieved in 24 hours, a 72 hour retention time is indicated to be required given the samples tested and conditions used.

18.7.2 Plenge Testwork

Plenge performed 3 cyanide leach tests at different grind sizes. The results are shown in Table 18-19.

Table 18-19: Effect of Grind Sizes on Recoveries – Plenge

Test No.	Grind P_{80} (µm)	Tails Grade		Recovery (%)		Consumption (kg/t)	
		Au (g/t)	Ag (g/t)	Au	Ag	NaCN	Lime
2	125	1.8	108.9	90.9	74.0	1.5	1.1
3	74	1.2	112.0	94.0	74.0	1.8	1.4
4	44	1.0	31.1	95.3	78.1	2.7	1.3

Plenge reports that the recovery of gold does not significantly increase in decreasing the grind size from a P_{80} of 74µm to a P_{80} of 44µm, however the NaCN consumption increases significantly. This tends to contradict the PRA results, however, it should be noted that the Plenge tests are very preliminary tests (as are the PRA tests) and on only one sample. A leach test performed by PRA on material with a head grade similar to that tested by Plenge and at a similar primary grind size (P_{80} of 77µm for PRA and P_{80} of 74µm for Plenge) produced gold and silver recoveries of 94.8% and 77% respectively. The comparable Plenge test produced gold and silver recoveries of 94% and 74% respectively.

Plenge performed 10 cyanide leach tests on gravity concentrate tails. These tests investigated the effect of pH, primary grind size, and cyanide concentration. The leach time for each test was 72 hours. The results are summarized in Table 18-20.

Table 18-20: Effect of pH, Grind Size and NaCN Concentration on Recoveries

Test No.	Variable Tested	Tails Grade		Recovery (%)		Consumption (kg/t*)	
		Au (g/t)	Ag (g/t)	Au	Ag	NaCN	Lime
6	pH=9.5	0.71	115.1	94.0	69.7	2.2	0.2
7	pH=11.5	0.70	77.8	94.2	79.4	0.9	2.3
8	P_{80}=230µm	1.3	121.3	89.1	68.1	1.3	1.0
9	P_{80}=125µm	1.4	130.6	88.8	66.4	1.6	1.1
10	P_{80}=74µm	0.81	112.0	93.1	68.9	1.6	0.9
11	P_{80}=44µm	0.49	136.9	95.9	67.9	3.1	0.9
12	NaCN=0.025%	1.82	245.7	85.1	32.9	0.5	1.7
13	NaCN=0.050%	1.08	183.5	91.1	47.1	0.9	1.8
10	NaCN=0.100%	0.81	112.0	93.1	68.9	1.6	0.9
14	NaCN=0.250%	0.69	65.3	94.4	82.4	2.0	0.7

Note: * = Consumption based on tonne of gravity tails.

Metallurgical recoveries in Table 18-20 are for those achieved from the gravity concentrate tails. Recoveries to the gravity concentrate are not included. The results indicate that for the variables tested, better metallurgical recoveries are achieved at:

- pH of 11.5.
- Primary grind of P_{80} = 44µm.
- NaCN concentration of 0.250%.

Plenge also performed a CIL test on gravity concentrate tails ground to a P_{80} of 74µm. At a solution pH of 10.5 and a NaCN concentration of 0.10%, recoveries of gold and silver are reported to be 95.4% and 88.4% respectively. A similar test without carbon produced recoveries for gold and silver of 93.1% and 68.9%. Indications are that the presence of carbon appears to assist in the recovery of silver. The reason for this is uncertain. It is noted this is only one test and repeat tests are needed to confirm this observation. The use of carbon in the final process selection is not considered practical due to the amount of silver in the material at the San Luis project.

Plenge performed a single intensive sodium cyanide leaching test on a gravity concentrate sample assaying 2,265g/t Au and 13,866g/t Ag. Recoveries for gold and silver were 97.2% and 86.4%. Sodium cyanide consumption was 34.2kg/t concentrate.

Size analysis by Plenge on testwork feed material indicated that 73.8% of the gold and 84.6% of the silver was in the minus 53µm fraction. Size analysis of the tails from test No. 7 indicated 61.8% of the gold and 65.2% of the silver was in the minus 53µm fraction.

18.7.3 G&T Testwork

The results of the G&T leach tests are presented in Tables 18-21 to 18-23.

Table 18-21: Leach Test Conditions - G&T

Composite	Test Program	P80	Calculated Feed (g/t)		Source Gravity Test	Leach Feed (g/t)		Leach Test No	Solution NaCN (ppm)	$PbNO_3$ (kg/t)
			Au	Ag		Au	Ag			
Ayelén #1	KM2437	43	39.7	902	1	30.2	894	3	2,000	0.0
Ayelén #1	KM2438	43	39.1	935	1	29.7	927	4	5,000	0.0
Ayelén #1	KM2441	43	38.1	906	10	29.0	870	12	5,000	1.0
Ayelén #1	KM2442	43	39.0	902	10	29.9	866	13	10,000	0.0
Ayelén #4	KM2439	55	9.3	342	2	7.7	334	5	2,000	0.0
Ayelén #4	KM2440	55	9.2	340	2	7.6	332	6	5,000	0.0
Ayelén #4	KM2443	55	10.2	329	11	7.5	319	14	5,000	1.0
Ayelén #4	KM2444	55	9.9	319	11	7.2	309	15	10,000	0.0
A	KM2488	73	23.1	624	13	16.7	593	19	5,000	0.5
B	KM2489	70	26.0	663	14	20.1	634	20	5,000	0.5
C	KM2490	140	20.6	534	1	16.7	513	4	5,000	1.0
C	KM2491	74	21.0	568	2	15.2	537	5	5,000	1.0
C	KM2492	74	23.4	541	7	18.0	516	10	5,000	0.5
C	KM2493	74	22.9	561	7	17.5	536	9	2,000	1.0
C	KM2494	74	21.3	610	7	15.9	586	11	2,000	0.5
C	KM2495	45	22.1	584	3	14.5	556	6	5,000	1.0
C	KM2496	45	24.2	548	8	16.8	522	12	5,000	1.0
D	KM2497	74	5.5	198	15	4.2	187	21	5,000	0.5

Table 18-22: Leach Test Results – G&T

Composite	Nominal K80 (microns)	Leach Test No.	Extraction (%)				Reagent Consumption (kg/t)			
			24 hour		72 hour		24 hour		72 hour	
			Au	Ag	Au	Ag	NaCN	CaO	NaCN	CaO
Ayelén #1	43	3	92.2	58.3	97.1	77.8	1.5	0.62	2.24	0.74
Ayelén #1	43	4	97.3	74.1	97.7	90.5	3.24	0.52	4.02	0.56
Ayelén #1	43	12	93.7	95.2	97.3	96.4	2.26	0.46	3.12	0.52
Ayelén #1	43	13	92.7	85	97.6	96	4.04	0.46	5.16	0.5
Ayelén #4	55	5	94	72.4	94.8	87	1.08	0.7	1.34	0.88
Ayelén #4	55	6	96.3	89.3	98.5	94.6	2.86	0.56	3.28	0.56
Ayelén #4	55	14	92.7	92.1	96.3	95	1.28	0.72	2.24	0.82
Ayelén #4	55	15	99.2	95	97.1	95.4	3.34	0.64	4.36	0.64
A	73	19	93.9	79.4	97.7	93.9	2.08	0.36	3.00	0.44
B	70	20	92.2	79.6	95.2	95.2	2.14	0.46	3.12	0.46
C	140	4	92.2	88.8	94.9	93.5	1.92	0.48	2.60	0.52
C	74	5	96.4	89.0	96.1	94.8	1.78	0.52	2.52	0.54
C	74	10	95.7	92.8	97.2	95.2	1.84	0.36	2.40	0.42
C	74	9	92.0	89.3	96.6	94.5	1.16	0.58	1.48	0.76
C	74	11	97.6	84.1	96.5	94.4	1.00	0.50	1.28	0.66
C	45	6	97.5	92.1	97.6	96.4	1.74	0.60	2.36	0.68
C	45	12	96.2	94.3	97.6	96.5	2.48	0.40	3.24	0.44
D	74	21	90.8	83.3	95.8	93.1	1.02	0.08	2.18	0.10

Table 18-23: Total Test Results - G&T

Composite	Nominal K80 (µm)	Average Feed (g/t)		Recovery (%)									
				Gravity (average)		24 Hour Leach		72 Hour Leach		Total (24 hr. leach)		Total (72 hr. leach)	
		Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag
Ayelén #1	43	39.0	911	23.9	2.5	70.2	56.9	73.9	75.9	94.1	59.3	97.8	78.4
Ayelén #1	43	39.0	911	23.9	2.5	74.1	72.3	74.4	88.2	97.9	74.7	98.2	90.7
Ayelén #1	43	39.0	911	23.9	2.5	71.3	92.8	74.1	94.0	95.2	95.3	97.9	96.5
Ayelén #1	43	39.0	911	23.9	2.5	70.6	82.9	74.3	93.6	94.4	85.4	98.2	96.1
Ayelén #4	55	9.7	333	22.2	2.8	73.1	70.4	73.8	84.6	95.3	73.2	96.0	87.4
Ayelén #4	55	9.7	333	22.2	2.8	74.9	86.8	76.6	92.0	97.1	89.6	98.8	94.8
Ayelén #4	55	9.7	333	22.2	2.8	72.1	89.5	74.9	92.3	94.3	92.3	97.1	95.1
Ayelén #4	55	9.7	333	22.2	2.8	77.2	92.3	75.5	92.7	99.4	95.1	97.7	95.5
A	73	23.1	624	27.7	5.1	67.9	75.3	70.6	89.1	95.6	80.5	98.3	94.2
B	70	26.0	663	22.7	4.6	71.2	75.9	73.6	90.8	94.0	80.5	96.3	95.4
C	140	22.3	555	19.0	4.1	74.7	85.1	76.9	89.7	93.7	89.3	95.9	93.8
C	74	22.3	555	25.0	4.8	72.3	84.8	72.1	90.3	97.3	89.5	97.1	95.0
C	74	22.3	555	25.0	4.8	71.8	88.4	72.9	90.7	96.8	93.1	97.9	95.4
C	74	22.3	555	25.0	4.8	69.0	85.1	72.5	90.0	94.0	89.8	97.4	94.8
C	74	22.3	555	25.0	4.8	73.2	80.1	72.4	89.9	98.2	84.9	97.4	94.7
C	45	22.3	555	32.6	4.9	65.7	87.6	65.7	91.6	98.3	92.5	98.4	96.6
C	45	22.3	555	32.6	4.9	64.8	89.6	65.7	91.7	97.4	94.6	98.4	96.7
D	74	5.5	198	23.9	5.6	69.1	78.6	72.9	87.8	93.0	84.2	96.8	93.5

G&T tested gravity concentration followed by cyanidation of the gravity tailings to evaluate gold and silver extractions under variable reagent addition rates. Two test programs were performed; one with Ayelén 1 and 4 composites (KM2437) and another with Composites A, B, C and D (KM2488). The key findings from these tests are:

- Combined gravity plus 72 hour cyanidation of the gravity tailings produced overall gold recoveries ranging between 97 and 98%. Approximately 25% of the gold values were recovered into the gravity concentrate with a grind P_{80} of 74 microns. Five percent or less of the silver reported to gravity concentrates.
- Relatively fine grinds with P_{80}s of 45 microns resulted in high gravity recoveries for gold but increased total gold recoveries increased only slightly. With a P_{80} of 45 microns, gravity gold recoveries were in the range of 32%, compared to 74 micron P_{80} recoveries of around 25%. Total gold recoveries for 45 micron P_{80} grind, however, were in the range of 98.4% compared to 74 micron P_{80} recoveries of 97.1% to 97.9%.
- Twenty four hour leach times produced variable results and slightly lower recoveries compared to 72 hour leach times, especially for silver. Twenty four hour leach times extracted

65% to 77% of total gold and 57% to 92% of total silver, while 72 hour leach times extracted 71% to 77% of total gold and 76% to 94% of total silver.

- Gold and silver recoveries were very similar for low-grade, medium-grade and high-grade samples at a given primary grind size. Total gold recoveries were in the range of 96% to 98% for the lowest grade samples and 97% to 98% for the highest. Other factors, including coarse grind size, reagent concentrations and leach retention time had significantly more influence on total recovery than the feed grade of the samples tested.
- For 24 hour leach time tests, high sodium cyanide concentrations of 5,000ppm with 0.5kg/t $PbNO_3$ produced improved total gold and silver recoveries compared to 2,000ppm sodium cyanide with 0.5kg/t $PbNO_3$. For 72 hour leach tests, 5,000ppm sodium cyanide concentrations with 0.5kg/t $PbNO_3$ produced only slightly higher total gold and silver recoveries than 2,000ppm tests with 0.5kg/t $PbNO_3$.

18.8 FLOTATION TESTWORK

PRA performed 9 bulk flotation tests on gravity concentrate tails to investigate the effect of primary grind sizes on gold and silver recoveries. The results are shown in Table 18-24.

Table 18-24: Bulk Flotation Recovery Versus Grind Size3

Sample	Test No.	Grind P_{80} (µm)	Tails Grade			Recovery (%)*			
			Au (g/t)	Ag (g/t)	S (%)	Au	Ag	S	Mass
Ayelén #1	F6	123	10.5	309	0.03	77	69	89	3.0
Ayelén #1	F7	84	7.3	255	0.02	85	75	95	4.4
Ayelén #1	F8	70	7.0	207	0.02	86	81	93	4.3
Ayelén #4	F9	114	2.7	185	0.06	77	51	83	3.3
Ayelén #4	F10	80	2.8	111	0.05	73	63	84	2.3
Ayelén #4	F11	59	1.9	115	0.05	83	68	86	6.0
Ayelén #5	F12	108	4.6	285	0.04	82	65	89	3.8
Ayelén #5	F13	77	6.5	266	N/A	78	69	N/A	N/A
Ayelén #5	F14	58	4.3	211	N/A	85	75	N/A	N/A

Notes: * = Recovery includes gravity circuit. **3.** = Reproduced from Tables 4.12 to 4.14 in PRA report. **N/A** = Not available.

Recoveries of the precious metals improve with finer grind sizes, however, tailings losses are significant even at the finest grind sizes evaluated. Given the low sulphur contents in the tails, the losses of precious metals are expected to be associated with non-sulphide gangue. Subsequent bulk flotation tests were performed where the gravity circuit was excluded from the circuit. The results indicated that for a flotation circuit, gravity concentration is not required. A series of flotation tests (open circuit cleaner) at a target primary grind of P_{80} = 44µm, without gravity concentration, were also performed. The results showed that significant gold and silver are still lost to tails. In these tests the bulk concentrate was subsequently upgraded via three stages of cleaning. This resulted in concentrates with precious metal grades ranging from 990g/t to 4,340g/t for gold and 18,375g/t to 91,600g/t for silver. Detailed analysis of the concentrates indicated elevated levels of arsenic and antimony which may have potential penalty issues for smelting.

PRA flotation and cyanidation results indicate that higher precious metal recoveries are achieved with cyanide leaching. Gravity concentration has not been shown to be beneficial in a flotation circuit, however, it should be included prior to cyanidation to assist in reducing leach residence time.

G&T performed two rougher flotation tests to investigate the potential for recovering gold and silver into a high grade low mass flotation concentrate. A single cyanidation test was also carried out on rougher tailings produced in one of the flotation tests. The following comments summarize the results of these tests:

- Approximately 89% of the gold and 84% of the silver in the feed were recovered into a bulk sulphide rougher concentrate. The mass recoveries to the bulk rougher concentrate were approximately 10%.
- The rougher concentrate had gold and silver contents of approximately 1,200g/t and 23,000g/t, respectively.
- Cyanide leaching of the flotation rougher tails (72 hours) resulted in gold and silver extractions of 92% and 87% respectively (based on feed to the cyanidation leaching). The combined recoveries (flotation plus cyanidation of flotation tails) for gold and silver recoveries were 99% and 97% respectively.

18.9 SETTLING TESTWORK

The results of settling tests and related work, such as flocculant screening and rheology studies, are needed to determine the number and size of thickeners used for leach slurry counter-current decantation and for use in sizing process pumps and piping. Settling and rheology tests can also indicate potential conditions that could be troublesome in plant operations.

18.9.1 Settling Testwork - Plenge

Plenge performed flocculant screening tests to determine a suitable flocculant. The tails from test numbers 3 and 4 were used for this work. The following flocculants were evaluated:

- Preatsol 2500
- Magnafloc 351
- Sedipur AF-404.

Plenge reports the flocculant Sedipur AF-404 gave the best results. The slurry pH was 11.5 and dose rate was 30g/t of leach tails.

Plenge also evaluated the settling rate of tails materials with at grind sizes of P_{80} = 74µm and 44µm. Using the flocculant Sedipur AF-404 (30g/t of leach tails at a 0.02% flocculent solution concentration) in a slurry at a pH of 11.5 the results are:

- Settling rate = 0.158 m^2/tpd for a grind size of P_{80} = 74µm.
- Settling rate = 0.333 m^2/tpd for a grind size of P_{80} = 44µm.

18.9.2 Settling Testwork - G&T

G&T performed settling tests on tails samples from cyanide leaching tests performed on composites Ayelén 1 and 4. The results are shown in Table 18-25.

Table 18-25: Settling Test Results – G&T

Test	Composite	CN Test	Floc. A-130 (g/t)	Underflow (% Solids)	Area Requirement (m^2/tpd)
A	Ayelén 1	18	5	55	2.65
B	Ayelén 1	18	10	55	2.67
C	Ayelén 1	18	15	55	2.32
E	Ayelén 4	19	5	55	3.49
F	Ayelén 4	19	10	55	2.72
G	Ayelén 4	19	15	55	3.57

Both samples had very poor settling properties under the conditions tested. At an underflow density of 55% solids by weight, the thickener area requirements for Ayelén 1 and 4 composites were 2.32 and 3.57m^2/tpd respectively. Additional settling tests are required to confirm these results and/or optimize settling rates.

18.9.3 Settling Testwork - Pocock Industrial

Pocock received two samples from G&T, which were cyanide tailings from Composites A (test 17) and C (test 18). The Composite A sample was determined by Pocock to have a P_{80} of 70 microns and the Composite C sample to have a P_{80} of 64 microns. Thus both samples were slightly finer than the G&T target grind of 80% passing 74 microns.

Pocock screened both samples with a series of commercial polyacylamide flocculants with a range of charge densities and molecular weights using simple small-scale mixing tests. Their report did not list specific flocculants tested. Anionic polyacylamides in general were effective flocculants for both composites and a medium to high molecular weight nonionic polyacylamide, Hychem AF 301, gave the best overall performance and was used for further settling tests. It is likely that a number of commercial flocculants besides Hychem AF 301 would prove to be acceptable flocculants for San Luis material.

A total of 24 static settling tests were performed on the two composites to determine the underflow slurry densities and overflow solution clarities. Both samples types readily produced settled pulp densities in the range of 65% to 70%. Composite A proved to settle somewhat better than Composite C, requiring an area of 0.178 – 0.243m^2/MTPD (including scale-up factor) as opposed to 0.209-0.299m^2/MTPD for Composite C. These rates are well above the minimum recommended design area of 0.125m^2/MTPD for a conventional thickener. Using these criteria, each conventional thickener would have a preliminary design diameter up to 12.3 meters for a 400 tonne/day operation.

CCD washing tests were also performed on samples from the two composites, to see if settling characteristics varied on previously washed materials. The only significant change was a reduction in the flocculant requirement for previously washed material. The flocculant requirements were approximately half of those needed for fresh samples.

Two series of dynamic settling tests determined hydraulic design bases for the composite samples. As with the static tests, Composite A samples settled more quickly than Composite C material. Feed loading rates ranged from 3.1 to 4.2m^3/m^2/hr for the Composite A sample and 2.6 to 3.7m^3/m^2/hr for the Composite C sample. This translates to a 5.5m diameter to 6.0m diameter high rate thickener for a 400 tonne/day operation.

For the purposes of this feasibility study MQes has chosen the following criteria for the process design:

- Underflow density = 65% solids.
- Thickener type = High Rate.
- Thickener design area = 2.6m^3 slurry/m^2/hr.

18.10 FILTRATION TESTWORK

A filtration step is not currently included in the San Luis mill design. It is possible, however, that tailings filtration could be utilized to prepare material for paste tailings disposal or for use in mine backfill. Preliminary filtration testwork was therefore performed.

18.10.1 Filtration Testwork - G&T

G&T performed filtration tests on the cyanidation tails produced in tests 18 and 19. The results are summarized in Table 18-26.

Table 18-26: Filtration Test Results

Sample Composite	Test No.	Stream	Primary Grind P_{80} µm	Total Time (sec)	Cake Moisture (%)	Cake Thickness (cm)
Ayelén 1	18	CN Residue	43	6,466	32.7	2.0
Ayelén 4	19	CN Residue	55	5,807	32.6	1.5

Both filtration tests indicate poor filtration properties. The sample from the Ayelén 4 composite had the poorest filtration properties. This was consistent with the results from the settling tests where Ayelén 4 composite also exhibited poorer settling properties. These results indicated that additional filtration testing was required.

18.10.2 Filtration Test Results - Pocock Industrial

Pocock performed both vacuum and pressure filtration tests on thickened slurries from both Composite A and Composite C. Vacuum tests were run with and without addition of the same flocculant used in the settling tests. Final cake moisture was 20% in both composites and in both types of test but the samples which included flocculant filtered over 6 times faster than the samples without. Preliminary estimates indicated that Composite A material, with flocculant added, would have a vacuum filtration rate of 358 dry kg feed/m^2 filtration area per hour, while Composite C material would have a rate of 486 dry kg feed/m^2 filtration area per hour.

Pressure filtration tests, run on both composite types without flocculant addition, showed that final moistures of approximately 15% were achievable.

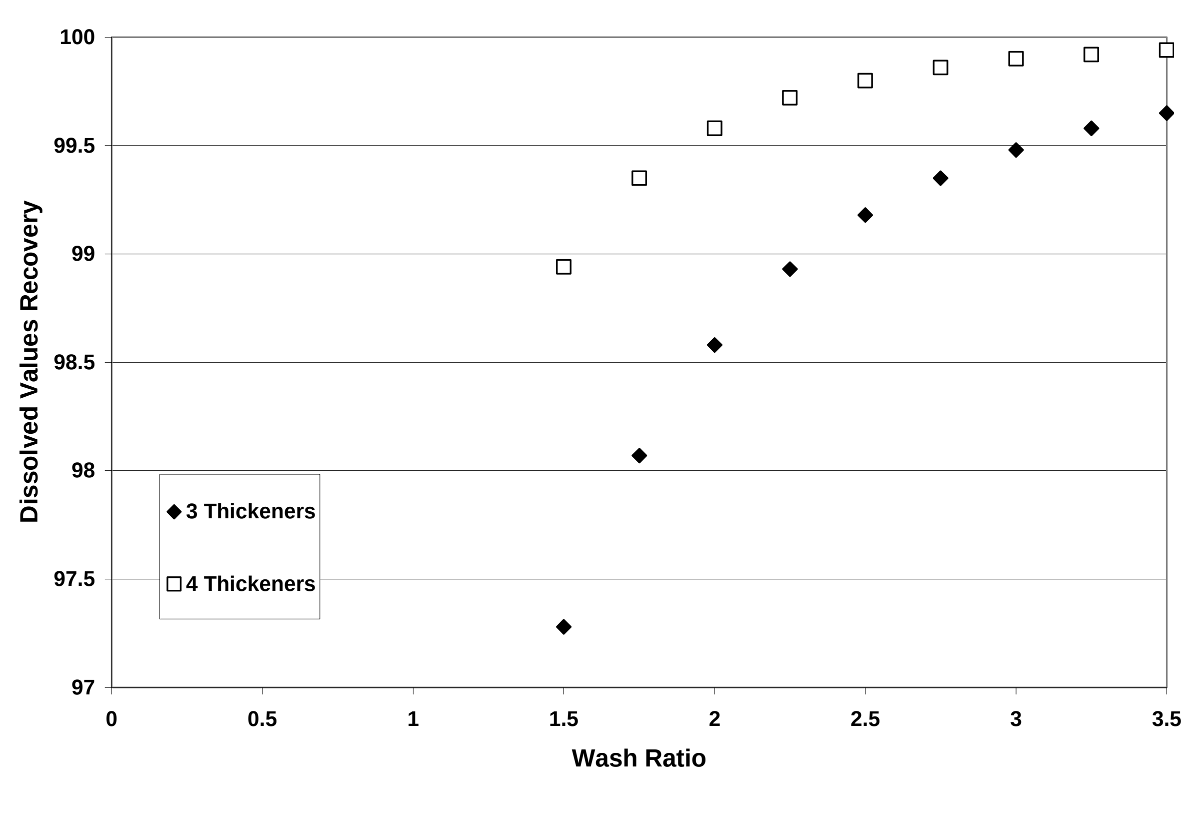
18.10.3 Rheology Test Results - Pocock Industrial

Apparent viscosities of thickened slurries over a range of pulp densities and shear rates were measured using a viscometer. The pulps behaved as non-Newtonian fluids and demonstrated increasing apparent viscosities as both pulp densities and shear rates increased. Apparent viscosities increased significantly for slurries with pulp densities above about 70%, indicating that thickener underflows should not be produced at higher pulp densities. At a 67.3% solids underflow, the coefficient of rigidity for Composite A was determined to be 0.0738Pa-sec with a yield value of 1.7Pa. For Composite C, the coefficient of rigidity was determined to be 0.0561Pa-sec with a yield value of 2.7Pa. MQes has used values associated with Composite A in this feasibility study.

18.10.4 Counter Current Decantation (CCD) Results

The Pocock report includes a set of numerical simulations of counter current decantation (CCD) wash circuits showing the effects on soluble metals recovery using varying numbers of thickeners and wash ratios. Figure 18-7 shows the simulation results for three and four thickeners over a range of wash ratios. It shows that a 99% recovery of solubles can be achieved using three thickeners at a wash ratio of approximately 2.32:1 or by four thickeners at a wash ratio of approximately 1.54:1. A 99.5% recovery can be achieved by three thickeners at an approximately 3.05:1 wash ratio or by four thickeners at a 1.91:1 wash ratio. A wash ratio of 3:1 and three thickeners has been used in this feasibility study to minimize the number of thickeners required, while maintaining a 99.5% solubles recovery.

Figure 18-7: CCD Simulation Results



18.10.5 Oxidation Testwork

Sample material from San Luis contains small amounts of sulphide minerals. To investigate the effects of an oxidation treatment prior to cyanidation, three tests were performed by G&T. In the KM2437 program, two composite types were subjected to an agitated leach in a float cell, with oxygen bubbled into the slurry. The NaCN concentration was maintained at 5000 ppm. Bottle roll leaches on the same material types were performed using the same cyanide concentration. Comparative results are listed in Table 18-27.

Examination of the results would suggest that aeration during leaching significantly hinders gold and silver extraction while increasing cyanide consumption. Agitated leach and a bottle roll are quite different test procedures and direct comparison of results between the two is not possible.

Table 18-27: Oxidation Test Results

Composite	NaCN (ppm)	24 Hour Agitated Leach				72 Hour Agitated Leach			
		Recovery (%)		Consumption (kg/t)		Recovery (%)		Consumption (kg/t)	
		Au	Ag	NaCN	CaO	Au	Ag	NaCN	CaO
Ayelén #1	5000	71.2	66.9	4.26	0.62	84.4	78.2	12.75	0.62
Ayelén #4	5000	95.9	94.1	3.88	0.48	96.3	94.5	10.32	0.52

Composite	NaCN (ppm)	24 Hour Bottle Roll				72 Hour Bottle Roll			
		Recovery (%)		Consumption (kg/t)		Recovery (%)		Consumption (kg/t)	
		Au	Ag	NaCN	CaO	Au	Ag	NaCN	CaO
Ayelén #1	5000	97.3	74.1	3.24	0.52	97.7	90.5	3.28	0.56
Ayelén #4	5000	96.3	89.3	2.86	0.56	98.5	94.6	1.64	0.56

In program KM2488, two parallel tests were performed on Composite A samples, one with and one without pre-aeration. Pre-aeration was performed by rolling the slurry bottle for 12 hours prior to introduction of sodium cyanide and lime. For both tests, the sodium cyanide concentration was 5000 ppm with 0.5 kg/t lead nitrate added. Results are listed in Table 18-28.

In these tests, 24 hour pre-aerated leaches showed lower cyanide and higher lime consumptions than nonaerated leaches. Gold recovery was higher in the pre-aerated leach but silver recovery was lower. The 72 hour leaches showed little difference in metals recoveries or lime consumption but the cyanide consumption was lower in the pre-aerated leach. These results are in contradiction with those achieved in program KM2437. Additional test work is required to more fully assess the effects of pre-aeration. Consequently, pre-aeration is not included in the process design for this feasibility study.

Table 18-28: Effect of Pre-aeration

Composite A	NaCN (ppm)	24 Hour Bottle Roll				72 Hour Bottle Roll			
		Recovery (%)		Consumption (kg/t)		Recovery (%)		Consumption (kg/t)	
		Au	Ag	NaCN	CaO	Au	Ag	NaCN	CaO
Pre-aerated	5000	96.9	78.8	1.78	0.44	97.8	93.2	2.47	0.45
Non-aerated	5000	93.9	79.4	2.08	0.36	97.7	93.9	3.00	0.43

18.11 FURTHER METALLURGICAL TESTWORK

Metallurgical testwork performed on the San Luis project to date indicates grinding to a P_{80} of approximately 74 micron followed by gravity separation and cyanide leaching of gravity tails is a suitable processing route. A flotation processing route has also been investigated by means of preliminary tests. The results indicate that gravity concentration followed by cyanide leaching produced better recoveries than flotation.

Although considerable effort has been made to ensure the representivity of the samples to the entire deposit, nevertheless it is possible that certain mineral assemblages have not been identified or tested adequately. Composite samples used in metallurgical testwork have been identified by REI/RMI. The most recent composites were assembled in late July/early August 2009 and included material from more than 9 diamond core holes that were drilled in 2006 and 2007. REI/RMI considers the composites are representative of:

- The approximate average grade of the projected mill feed based on the Feasibility Study Life of Mine plan.
- Spatially representative high grade portions of the deposit.
- Spatially representative low grade portions of the deposit.
- The approximate average mill feed grade projected for the first three quarters of mine production, based on the Feasibility Study Life of Mine plan.

Composite samples that have been tested to date have addressed the effect of head grade variability in the range of approximately 5.4g/t Au (196g/t Ag) to 28.3g/t Au (574g/t Ag). They have also investigated the effect of primary grind size on recovery of gold and silver by gravity separation followed by cyanide leaching. Other parameters such as reagent concentration, leach time, settling/filtration characteristics and rheology have also been addressed.

The assembled medium composite (Composite C) had higher head grades than anticipated. It is recommended that an additional composite is assembled to better represent the life-of-mine feed grade for confirmatory testing. As well, samples representative of the mine plan for each quarter of the first year and a sample representative of each year after that should also be generated.

A further testwork goal should be to develop a geo-metallurgical model of the deposit which can be used to reasonably predict metallurgical responses. As mine planning is refined, metallurgical samples should be chosen representative of the mine plan with an emphasis on samples in the early period of production. These samples should be tested to confirm the metallurgical response is consistent with that predicted from the geo-metallurgical model.

ITEM 19 • MINERAL RESOURCE & MINERAL RESERVE ESTIMATE

19.1 MINERAL RESOURCE ESTIMATE

This section of the San Luis Project Feasibility Report provides a summary of the mineral resource estimate that was completed by RMI and REI and described in detail in the Canada National Instrument 43-101 Technical Report by Lechner and Earnest (2009).

19.1.1 Geologic Data Analysis and Statistics

Gold and silver assay data were initially collected from HQ core holes and surface trench chip channel samples as described in Section 4.4.3. These data were provided to RMI and REI in the form of Excel spreadsheets and contained collar coordinate locations, down-hole surveys, assay grades, and lithologic codes. Logged lithologic codes were provided for the drill hole and chip-channel samples which were used to construct the mineralized wireframes that were subsequently used in the grade estimation process. In addition, RMI and REI relied heavily on a set of 1:500 scale geologic cross sections that were prepared by Silver Standard's Ken Konkin, Senior Geologist for the San Luis project.

19.1.1.1 Comparison of Diamond Core and Trench Assays

Of all data used to estimate the mineral resources used for the San Luis Project Feasibility Study, 73% were derived from HQ diamond drill holes, with the remaining 27% obtained from surface trench samples. A review by RMI of the two data sets indicated an apparent disparity between the mean grades of the two sampling methods when all data were compared, with the trench samples having a much higher mean grade than the samples from the diamond core holes. The reason for this apparent bias is that nearly all of the trench samples were collected from within the walls of the mineralized veins, while a significant portion of the drill holes that cut the veins at an angle also sampled barren hanging wall and footwall waste lithologies.

In order to eliminate the influence of the wallrock waste, a total of 145 Ayelén trench composites were spatially paired with mineralized composites from nearby core holes so that meaningful statistics could be calculated. Both sample types were composited after the raw assay data were capped at 130g/t and 3800g/t for gold and silver, respectively.

The results are shown in Table 19-1, which indicates that the means of the gold and silver assays from the core samples are about 10 percent higher than the surface trench chip channel samples, based on these paired statistical comparisons. The surface trench sample data do not display significant high biases that are typical of data generated by manual sampling methods. Both sample types have similar standard deviations and coefficients of variation. Based on these comparisons, the trench sample data were judged to be representative and suitable to be used to estimate mineral resources.

Table 19-1: Paired Core vs. Trench Sample Grades

Sample Type	No. Composites	No. Meters	Gold Grades (g/t)			Std. Dev.	CV	GT
			Min	Mean	Max			
Chip Channel	145	145.5	0.03	23.03	130.00	34.12	1.48	3,350
HQ Core	145	145.5	0.00	25.75	130.00	35.10	1.36	3,746
% Difference	0%	0%	n/a	-11%	0%	-3%	9%	-11%

Sample Type	No. Composites	No. Meters	Silver Grades (g/t)			Std. Dev.	CV	GT
			Min	Mean	Max			
Chip Channel	145	145.5	2.00	564.9	3800.0	839.9	1.49	82,161
HQ Core	145	145.5	1.0	621.2	3800.0	816.1	1.31	90,351
% Difference	0%	0%	n/a	-9%	0%	3%	14%	-9%

19.1.1.2 Basic Descriptive Assay Statistics

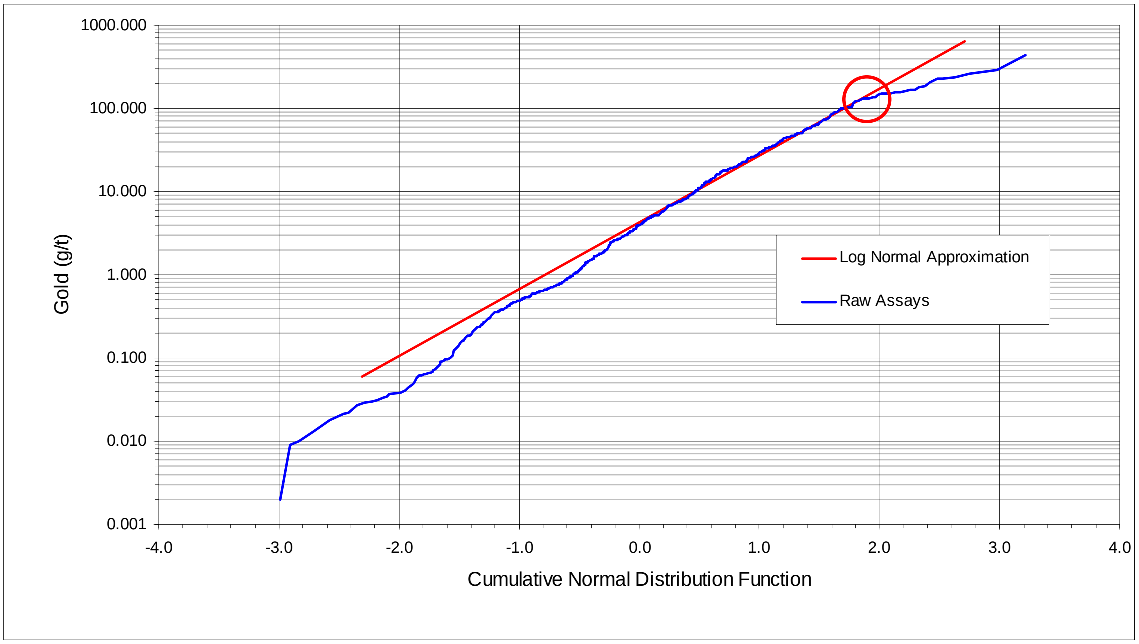
Basic descriptive statistics were calculated for raw and capped precious metal assays by area, sample type, rock type, and mineralized zone. These statistics tabulate the number of meters, mean grades, grade thickness products, standard deviations, and coefficients of variation at four cutoff grades. The results indicate that both gold and silver have relatively high coefficients of variation (standard deviation/mean) which is not unusual for epithermal precious metal deposits. In addition to these analyses, box plots were constructed for both gold and silver which show pertinent information such as minimum grade, maximum grade, first and second quartile grades, median grade, and mean grade. These box plots indicate that the preferential vein host rocks ("H" series lithologic codes) possess significantly higher grades than other lithologic units. Based on the relatively similar metal distributions of the vein material represented by these various "H" codes, all of the logged vein lithologies were combined and modeled for the purpose of estimating mineral resources.

A complete set of the tables and figures summarizing the basic descriptive statistics for gold and silver can be found in Section 17.5 of the NI 43-101 Technical Report by Lechner and Earnest (2009).

19.1.1.3 Grade Capping

High grade gold and silver outlier values were capped (cut) to lower values to minimize the risk of overestimating contained metal. Cumulative probability plots and decile analyses were used to determine gold and silver capping limits. Figures 19-1 and 19-2 are cumulative probability plots for gold that were generated for the Ayelén vein by transforming the grades using the cumulative normal distribution method so that the higher grade population could be examined more closely.

Figure 19-1: Ayelén Au Cumulative Probability Plot



The cumulative probability distribution of gold starts becoming erratic around 130g/t as shown by the area within the red circle in Figure 19.2. This break in slope is one of the primary reasons for the selection of a 130g/t gold cap. This cap was confirmed by decile/percentile analysis, which also indicated a gold capping limit of 130g/t Au.

The capping limit for Ayelén silver assays was set at 2,400g/t based on the cumulative probability plot shown in Figure 19-2, although a decile/percentile analysis like that done for gold indicated a higher cap of approximately 3,800g/t Ag could be considered. Table 19-2 summarizes the gold and silver capping limits that were used for the Ayelén and Inéz veins.

19.1.1.4 Assay Compositing

Approximately one third of the raw drill hole and surface trench assay intervals are less than 1.0m in length, a third are exactly 1.0m in length, and the remaining third are greater than 1.0m in length. RMI composited the raw assays into one meter long intervals based on the vein codes that were assigned to the raw assays. If the last drill hole interval for a given vein composite was less than 0.51m in length, it was added to the last composite inside of the vein wireframe for that hole. This resulted in vein composites ranging in lengths between 0.51m and 1.50m long with the majority being exactly 1m long. The grade estimation calculation was weighted by composite length in addition to inverse distance weighting.

Figure 19-2: Ayelén Ag Cumulative Probability Plot

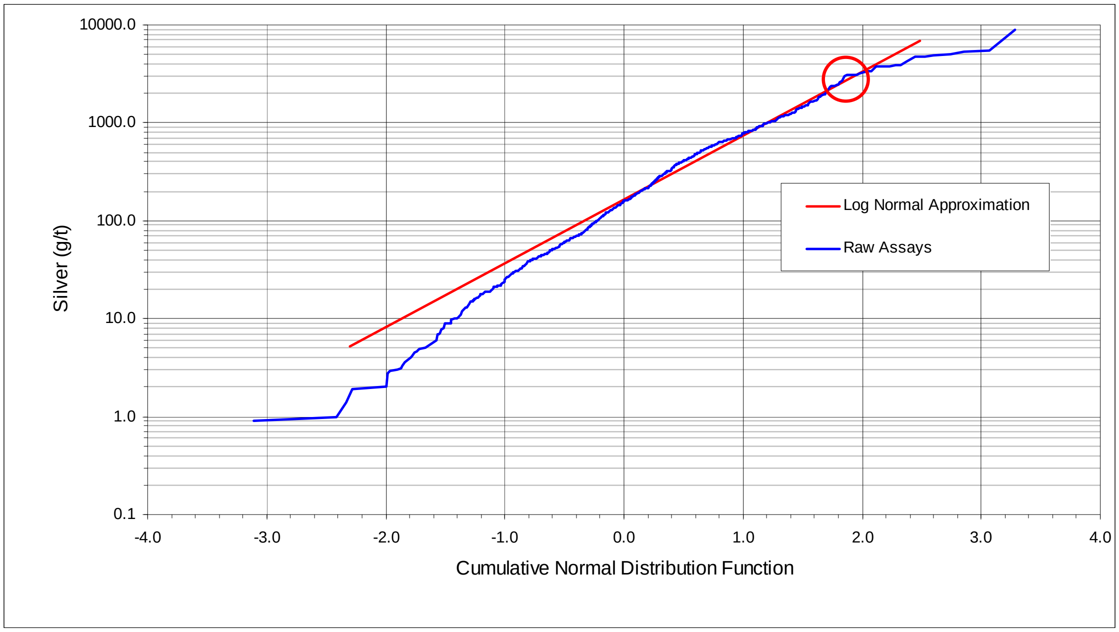


Table 19-2: Summary of Au & Ag Capping Limits

	Capping Limits (g/t)	
Vein	**Gold**	**Silver**
Ayelén	130	2400
Inéz	15	600

Unassayed intervals were set to zero grade prior to compositing the assay data. This ensured that each diamond drill hole was composited in the same manner from top to bottom. Because nearly all of the logged vein lithologies were assayed, very little artificial dilution was incurred by setting unassayed intervals to zero grade. Uncapped and capped assay grades were composited so that various grade models could be estimated to determine the actual effect of the grade capping.

19.1.1.5 Variography

Downhole gold and silver correlograms were generated using uncapped raw assays to establish the nugget effect for each metal. These correlograms showed ranges of about 6.5m, as illustrated in Figures 19-3 and 19-4, with nuggets of 0.45 and 0.40 for gold and silver respectively. Directional gold and silver correlograms also were generated for the Ayelén vein using one-meter-long composites.

Figure 19-3: Ayelén Down-hole Gold Correlogram

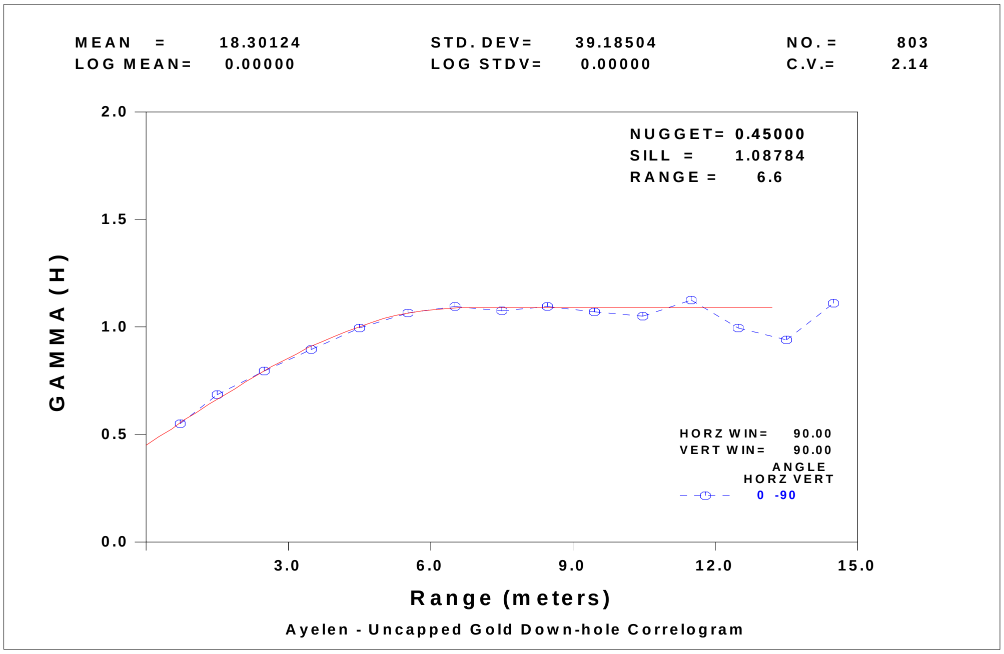
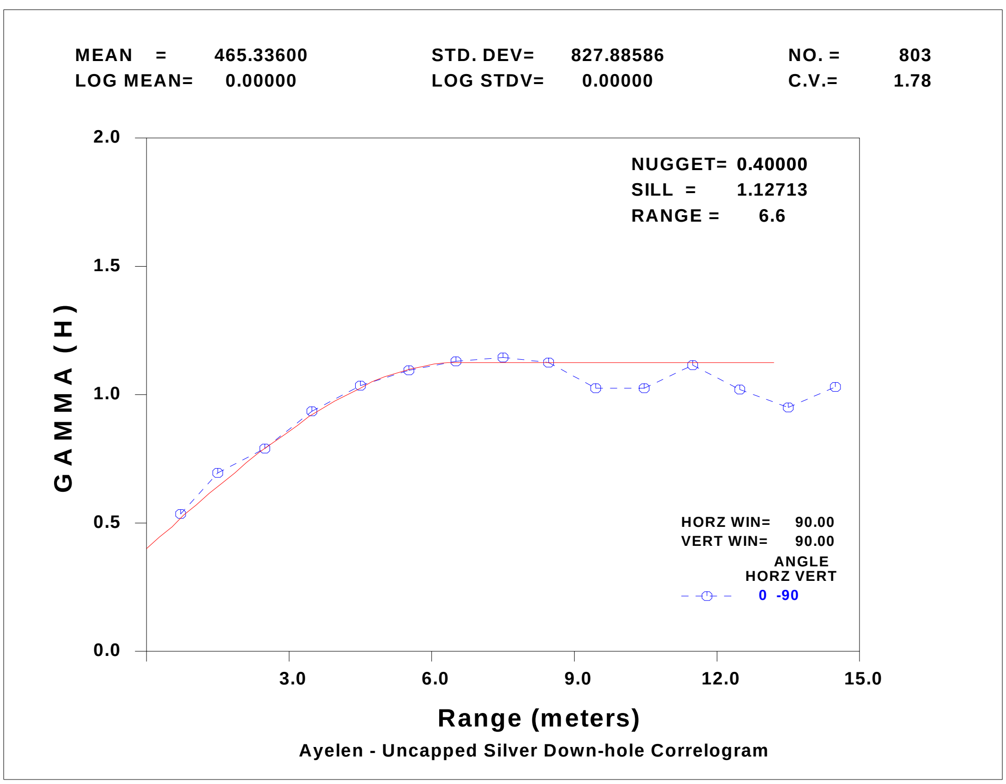


Figure 19-4: Ayelén Down-hole Silver Correlogram



19.1.2 Vein Wireframe Construction

RMI constructed three dimensional wireframe solids for use in constraining grade estimation in the mineral resource block model by assigning a vein code of 10 through 20 to drill hole assay intervals based on the following information:

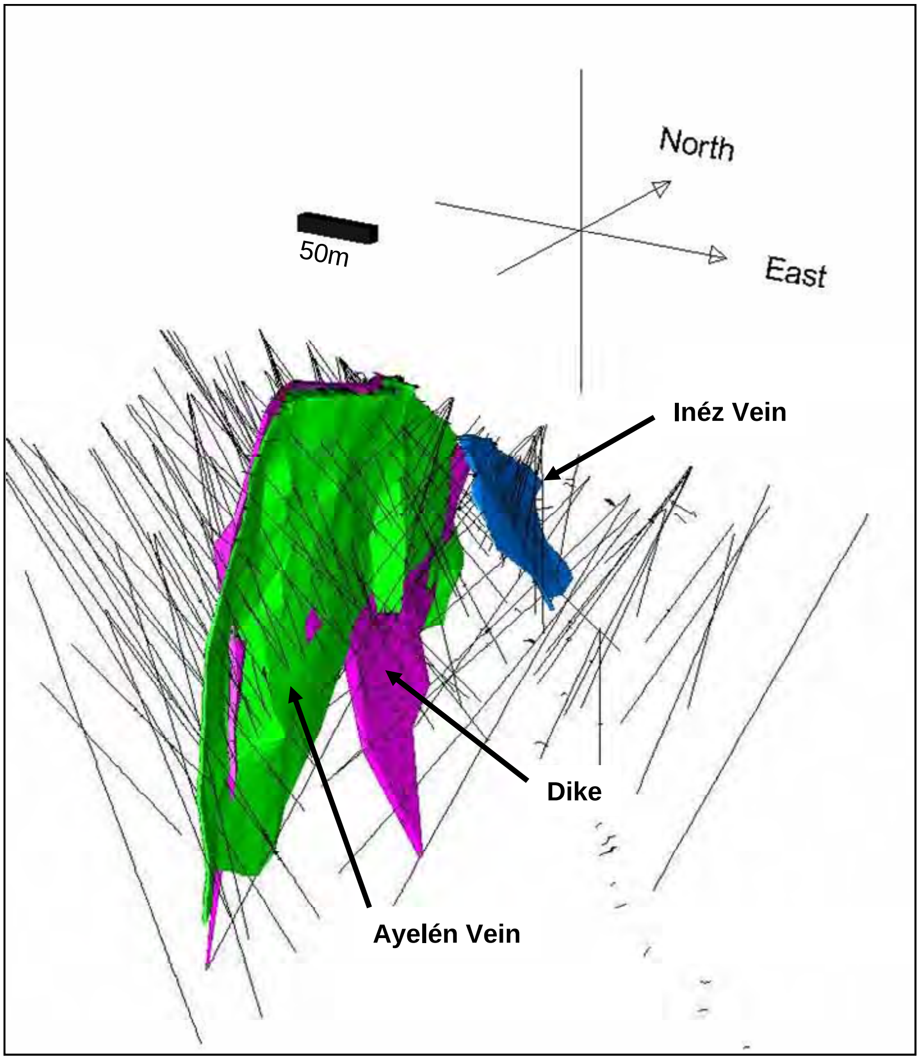
1. A nominal gold cutoff grade of 0.50g/t.
2. Drill hole lithology/mineralization logging.
3. Silver Standard's geologic interpretation.

The main Ayelén vein was assigned a mineral zone (MZONE) code of 10 while various hanging wall and footwall splays of the main Ayelén vein were assigned MZONE codes 11 through 16. The Inéz vein was assigned an MZONE code of 20. Gold and silver grades are noticeably higher near the upper portions of the deposit and clearly decrease with depth. RMI chose to extend the vein wireframes approximately 30 meters beyond the lowest drill hole intercepts despite the fact that many of the assayed intervals were below the 0.50g/t Au cutoff grade. The lower grades from these intercepts helped to establish the base of the mineralized system in the grade estimation plan.

The XYZ drill hole coordinates for the hanging wall and footwall pierce points for each vein were used to develop those respective surfaces. RMI also digitized poly line strings between drill hole "fences" to provide additional control in constructing the hanging wall and footwall surfaces. A three-dimensional "band" was constructed that merged the hanging wall and footwall surfaces into a cohesive solid. Using logged lithologic codes RMI also constructed a three-dimensional wireframe which represents an intrusive dyke that is located along the western side of the Ayelén vein. Locally this dyke cross cuts and disrupts the Ayelén vein. The dyke wireframe was intersected with the Ayelén vein solid resulting in a finalized vein wireframe. Figure 19-5 is a perspective view showing the two main vein wireframes (i.e., Ayelén and Inéz) looking N30°W.

In addition to the vein wireframes described above, two near-surface high grade gold/silver zones were identified and modeled as separate three dimensional wireframes. These two high grade zones are small in volume (together totaling approximately 9,000 m^3 or about 23,500t). Each high-grade zone was based on both surface chip-channel samples and diamond core holes using a nominal gold cutoff grade of 130g/t.

Figure 19-5: Perspective View of Ayelén and Inéz Vein Wireframes



19.1.3 Block Grade Estimation

RMI constructed a three-dimensional block model using MineSight® software. Table 19-3 summarizes the dimensions and extents of the block model.

Table 19-3: San Luis Block Model Extents

Parameter	Minimum	Maximum	Block Size (m)	Number of Blocks	Areal Extent (m)
Easting	190,300	190,700	1	400	400
Northing	8,960,500	8,961,280	3	260	780
Elevation	4,275	4,600	5	65	325

The vein wireframes (see previous section) were used to assign an integer code to the model blocks that identified the various veins along with the percentage of the vein contained in each block. This allowed for a more accurate estimate of vein tonnage than whole or sub-blocking methods. The percentage of "topo" or rock contained in each block was also calculated and stored in the model.

Gold and silver grades were estimated using several techniques including inverse distance ($1/d^x$), ordinary kriging, and nearest neighbor methods. RMI used a dynamic anisotropy method for determining which composites were used to estimate block grades. With this method, prior to estimating metal grades the perpendicular distance between each block and the Ayelén hanging wall and footwall surfaces was calculated and stored in the model. Those distances were then back tagged to the drill hole composites. The distance between each block and the vein contact was used to match composites located at similar distances from the vein for the purpose of estimating gold and silver grades using a three-pass inverse distance interpolation plan that incorporated progressively longer search distances to find eligible composites. If a block was located closer to the hanging wall contact of the vein, only composites located at similar distances from the hanging wall contact were used to estimate block grades. The same function was used for blocks located near the footwall contact.

Block gold and silver grades were estimated using composites derived from capped raw assays. Uncapped grades were used to estimate block grades inside of the two high-grade zones that were described in Section 19.1.2.

Table 19-4 summarizes the basic inverse distance cubed parameters that were used to estimate gold and silver grades for the main Ayelén vein (MZONE 10).

Table 19-4: Inverse Distance Grade Estimation Parameters

Estimation Pass	Composite Selection			Search Distance (m)		
	Min	Max	Max/hole	Along Strike	Down-Dip	Across Strike
1	1	3	1	12.5	12.5	Variable[1]
2	1	3	1	37.5	37.5	Variable[1]
3	1	3	1	75.0	75.0	Variable[1]

Notes: 1. For the main Ayelén vein (MZONE = 10) the cross strike distance for accepting composites was a function of the block/composite distances from either the footwall or hangingwall vein surface. For all other veins a strict matching method was used based on MZONE codes. The cross strike search distance was essentially limited to the width of the vein wireframe.

The same three pass inverse distance cubed estimation strategy was used to estimate gold and silver grades for the remaining Ayelén and Inéz veins (MZONE 11 through 20). The number of composites and drill holes used to estimate block grades was captured along with the Cartesian distance to the closest composite used in the estimation process.

Approximately 37% of the blocks were estimated by the first pass (12.5m search), 58% by the second pass (37.5m search), and the remaining 5% of the estimated blocks by the longer 75m search.

19.1.4 Topographic Data

Silver Standard provided an AutoCAD DXF file which contained topographic contours on 2-meter intervals. According to Silver Standard personnel, the contour data were obtained from Ikonos satellite stereo pair photographs. RMI constructed a three-dimensional surface from the contours and used this surface to code block topography (rock) percentages.

19.1.5 Bulk Density Data

Bulk density (specific gravity) data were provided for 353 drill core samples from the Ayelén and Inéz veins. These determinations were made by taking the average of weighing the sample in air five times (each time after being dried in an oven so as to remove moisture), followed by the average of five measurements of the core sample length and the average of five measurements of the core diameter. These determinations were completed for approximately 30 different logged lithologies, which were combined into four categories (vein, dike, upper and lower volcanic sequences) so that average values for the lithologies to be encountered during production mining could be estimated. Table 19.5 summarizes those averages.

Based on the averages shown in Table 19-5, bulk density values of 2.61g/cm^3 for vein material and 2.65g/cm^3 for all other lithologies were used for tabulating mineral resource tonnages.

Table 19-5: Bulk Density Values by Combined Lithologic Units

Combined Lithologic Units	Count	SG (g/cm^3)
Dike	32	2.60
Lower Volcanics	76	2.66
Upper Volcanics	52	2.65
Vein	193	2.61
Grand Total	**353**	**2.63**

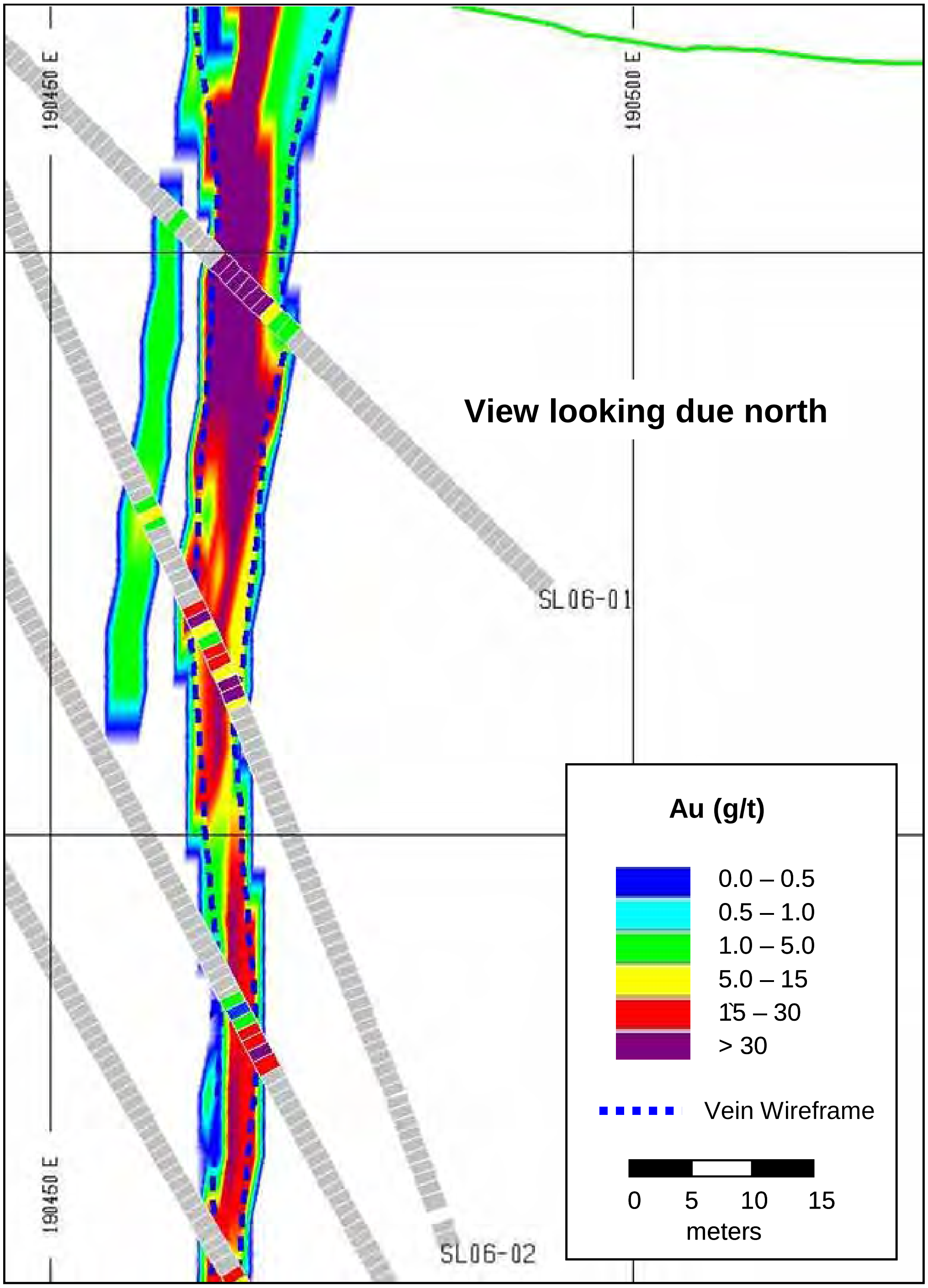
19.1.6 Model Verification

Block grades were validated by both visual and statistical methods. RMI notes that the dynamic anisotropy method generated a distribution of metal grades which is similar to the observed distribution of high grade mineralized shoots and internal waste selvages that are commonly observed in epithermal precious metal vein deposits. For example, Figure 19-6 shows block gold grade correlations for cross section 8,960,885 North which demonstrate feathery zones of various grade ranges and good correlations between drill hole composite assays and block grades.

As part of the verification of the mineral resource model, the distribution or frequency of drill hole composite grades versus estimated block grades was compared, and the composites were declustered using the cell method (1m x 3m x 5m). The composite and block grades were then transformed using the cumulative normal distribution function. Comparisons were made for gold and silver for all vein composites and all estimated blocks regardless of resource category. The results of these comparisons can be found in the NI 43-101 Technical Report by Lechner and

Earnest (2009). Based on the visual and statistical comparisons made to validate the mineral resource model, it is the opinion of RMI and REI that the estimated block gold and silver grades are reasonable and honor the sample data.

Figure 19-6: Block Au Grade Contours – Section 8,960,885 North



19.1.7 Mineral Resource Classification

The blocks in the mineral resource model were classified into Measured, Indicated, and Inferred categories using the distance to data method. The Measured mineral resource classification was only assigned to blocks situated within 15m of surface trenching along the Ayelén vein. The rationale for this assignment is that the continuity of the mineralized system was both visually and analytically confirmed by mapping, sampling and assaying of the surface trenches. Ayelén vein blocks within 25m of sample data (primarily diamond core samples) were classified as Indicated mineral resources. The remaining estimated Ayelén blocks that were not classified as either Measured or Indicated material were classified as Inferred mineral resources. Blocks within 15m of trenching data along the Inéz vein were classified as Indicated resources because of less confidence in the continuity of grade within this vein. The remainder of the estimated Inéz blocks was classified as Inferred mineral resources.

19.1.8 Summary of Mineral Resources

Mineral resources were tabulated using a 6.0g/t Au equivalent cutoff grade. Gold and silver metal prices of US$600 and US$9.25 per troy ounce were used to establish a gold to silver ratio of 65:1, and no metal recoveries were considered. Table 19-6 summarizes the total Ayelén and Inéz mineral resources.

Table 19-6: San Luis Mineral Resources

Measured Mineral Resources						
Tonnes	**Au (g/t)**	**Ag (g/t)**	**AuEQV (g/t)**	**Contained Au Ounces**	**Contained Ag Ounces**	**Contained AuEQV Ounces**
55,000	34.3	757.6	46.0	61,000	1,345,100	81,700
Indicated Mineral Resources						
Tonnes	**Au (g/t)**	**Ag (g/t)**	**AuEQV (g/t)**	**Au Ounces**	**Ag Ounces**	**AuEQV Ounces**
429,000	20.8	555.0	29.3	287,000	7,658,200	404,800
Measured and Indicated Mineral Resources						
Tonnes	**Au (g/t)**	**Ag (g/t)**	**AuEQV (g/t)**	**Au Ounces**	**Ag Ounces**	**AuEQV Ounces**
484,000	22.4	578.1	31.2	348,000	9,003,300	486,500
Inferred Mineral Resources						
Tonnes	**Au (g/t)**	**Ag (g/t)**	**AuEQV (g/t)**	**Au Ounces**	**Ag Ounces**	**AuEQV Ounces**
20,000	5.6	270.1	9.7	3,600	174,900	6,300

Based on further engineering, a mine plan has been developed for the San Luis project and a reserve has been estimated. This mineral reserve is described in Section 19.2 of this report.

19.2 MINERAL RESERVE ESTIMATE

In order to determine the portion of the measured and indicated resources that might qualify for proven and probable mining reserves, it is necessary to plan mineable shapes along the vein(s), based on minimum mining widths and the deposit geometry that can be economically extracted. When the economic portion of the resource has been defined, factors for mining recovery and mining dilution are applied. These factors are based on the “mechanized overhand cut and fill” method selected to mine the deposit and the expected ground conditions.

19.2.1 Cutoff Grade

The economic portion of the resource is typically determined by the application of a breakeven cutoff grade that considers the total operating cost (mine, plant plus general and administration), metal prices, plant recoveries, applicable royalties, and forward costs for freight, insurance, and refining. These parameters are equated to determine the minimum grade, or “equivalent minimum grade” that will need to be mined in order to cover these total operating costs.

At the San Luis Project there will be two payable metals: gold and silver. Therefore for economic analysis purposes it is easier to express the breakeven cutoff as an “equivalent gold grade” (Au_Eq) which represents a composite grade of gold and silver in ore expressed in terms of gold.

Since the breakeven cutoff grade represents the minimum grade that will be mined, the average grade delivered to the plant will always be higher. This increment, between the breakeven cutoff grade and the head grade, provides the return of capital investment and profit.

Incremental cutoff grades, which incorporate opportunity and capital costs, may be employed later in the mine planning process by the mine planners, to handle situations where mineralized material below the breakeven cutoff grade must be mined in order to optimize the NPV. However, these incremental cutoff grades have not been used for calculating the minable reserves at the San Luis Project.

At the San Luis Project, the economic parameters presented in Table 19-7 have been used to estimate the gold equivalent breakeven cutoff grade.

Table 19-7: Economic Variables Related to Cut-off Grade (COG) Calculations

Item	Value	Unit	Source
Production rate	417	tpd	
Operating Costs			
Mining	59.00	US$/t Ore	By BISA
Plant	56.32	US$/t Ore	By MQes
G & A	45.51	US$/t Ore	By SSR
Metallurgical Recoveries			
Gold	94	%	By MQes
Silver	90	%	By MQes
Metal Prices			
Gold	800.00	US$/oz	By SSR
Silver	12.50	US$/oz	By SSR
Refinery			
Recovery of Gold	99.0	%	By SSR
Recovery of Silver	99.5	%	By SSR
Charge – Gold	1.00	US$/oz	By SSR
Charge – Silver	0.50	US$/oz	By SSR
Doré Transport Marketing & Sales Cost	8	US$/oz	Assumed
Royalty (% of Metal Produced)	2	%	By SSR

With these parameters the Ag contribution (or equivalence) in terms of Au grade is calculated.

$$\text{Ag Credit} = \frac{\text{Plant\_AgMetRecov} \times \text{Refiniery\_AgMetRecov} \times (\text{AgPrice} - \text{Reinery\_AgCharge})}{\text{Plant\_AuMetRecov} \times \text{Refinery\_AuMetRecov} \times (\text{AuPrice} - \text{Refinery\_AuCharge})}$$

Equating these parameters provides the following breakeven silver equivalent grade for a 417tpd production rate:

Silver as a credit to Gold	0.014	oz/t ore	(By BISA)
Gold Equivalent value per ounce	710.12	US$/oz	(By BISA)

The following equation illustrates the typical relationship, between the various parameters, to calculate at the breakeven cutoff gold equivalent grade (Au_Eq):

The breakeven cutoff is expressed in US$/t, it is the sum of the mine, plant, and administrative costs.

$$\text{COG (g/t Au\_Eq)} = \frac{(\text{MiningCost} + \text{ProcessingCost} + \text{G\&ACost})}{\text{Plant\_AuMetRecov}\times\text{Refinery\_AuMetRecov}\times[(\text{AuPrice}-\text{Refinery\_AuCharge}-\text{SalesCost})\times(1-\text{Royalties})]}$$

Therefore, COG = 6.9g/t Au-Equivalent

In the time between completion of the resource and reserve estimates, metal prices and operating costs changed due to market fluctuations. Along with metal prices of US$800/oz Au and US$12.50/oz Ag; metallurgical recoveries of 94.0% Au and 90.0% Ag and an operating cost of US$160.83/tonne of ore processed, a breakeven cutoff grade of 6.9g/t AuEq was calculated.

The stope shapes were originally designed to a breakeven cutoff grade of 6.0 g/t AuEq based on preliminary estimates of operating costs. The final economic parameters result in a 6.9 g/t AuEq breakeven cutoff grade. The difference in the reserve between the two cutoff grades is economically insignificant and accounts for 4.5% of the total reserve, well within the accuracy of a feasibility study. This is primarily due to the nature of the vein system which provides a distinct boundary between mineralized and nonmineralized material. Any below cutoff material in the mine plan is treated as planned dilution in that normal operating costs and block model grades are applied.

This breakeven cutoff grade is used to separate and report the mineable reserves (economic part of the diluted resources), from the undiluted measured and indicated resources. Mineable shapes (stopes) are then designed to cover the reserve's volume. This shape may include some sub-economic material that must be taken in the mine planning and stoping process. The sub-economic material is called internal dilution. In addition, some part of the reserve blocks may have to be dropped due to their location outside of the mining shapes, or because they would require excessive development to access and prepare the block for mining. Such occurrences may cause actual results to differ slightly from planned.

19.2.2 Mining Recovery

The mining method chosen for safely and selectively mining the veins at the San Luis Project is "mechanized over hand cut and fill", using unconsolidated backfill (from surface colluvial material and higher level development in waste within the mine) for ground stabilization and as a working platform for personnel and equipment operating within the stope. This mining method is very selective and generally allows for a high percentage recovery of the ore. The primary causes for ore loss are:

1. Irregularities in the vein walls and stope boundaries, both horizontally and vertically, causing ore to be left in the stope walls during the mining process.

2. Unplanned ground failures limiting access to the ore.

3. 1.5m horizontal pillar left in the upper limit of each stope.

Based on a review of the wireframe outlines of the vein cross section, the ore loss due to irregularities in the vein walls was estimated at 3%, an additional 2% was estimated for unplanned ground failures and pillar, for a total ore loss of 5%, or a mining recovery of 95%.

19.2.3 Mining Dilution

Internal waste dilution from expanding the vein widths to minimum mining widths in the stope design process, and waste included in the drillhole compositing process has been accounted for in the resource block model preparation.

External waste in the mining process originates from three principal sources:

1. Irregularities in the vein walls, both horizontally and vertically.
2. Miners over-drilling in the stope, both in the ribs and the back.
3. Mucking up backfill from the floor.

Irregularities in the vein walls generally account for about 2% of the dilution; over-drilling by the miners can vary widely with miners' experience and supervision, but typically adds another 3%; another 5% results from scoop operators mucking up waste backfill from the floor during the ore loading phase and can be controlled by using long timber planks on the last backfilled floor. A 10% allowance for external dilution has been added to the mineral reserve at zero grade.

19.2.4 Mineral Reserve Summary

After the application of the breakeven cutoff grade and factors for mining recovery (95%) and mining dilution (10% in average), the total minable reserves are estimated as shown in Table 19-8. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 19-8: Total Proven and Probable Reserves

Category	tonnes	Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
Proven	56,000	28.3	604.5	51,000	1,088,000
Probable	447,000	16.7	426.2	240,000	6,125,000
Total	503,000	18.0	446.1	291,000	7,213,000

ITEM 20 • OTHER RELEVANT DATA & INFORMATION

To the best of the authors' knowledge, all relevant data and information have been discussed elsewhere in this report.

ITEM 21 • INTERPRETATIONS AND CONCLUSIONS

Conclusions and recommendations from this technical report are presented below.

21.1 GEOLOGY

The geology of the San Luis deposit (specifically the Ayelén and Inéz veins and the associated host rocks) is sufficiently understood to support the geologic interpretations that form the basis for the current mineral resource estimate.

The Mineral Resource estimate that forms the basis of the Mineral Reserves for this Feasibility Study is based on valid assay, specific gravity, lithologic, and alteration, and structural data. These data are supported by a sufficient QA/QC program, such that the resulting current Mineral Resource is a reasonable estimate of the in situ undiluted gold and silver mineralization present in the San Luis deposit, as it is defined by the existing drill holes.

The nine composite samples collected for the metallurgical testing completed as of the date of this Feasibility Report adequately represent (both spatially and from a production time standpoint) the material scheduled for mining from the Ayelén deposit during the term of the life of mine plan. Furthermore, these composites are valid for the metallurgical testwork performed by Mine and Quarry Engineering Services Inc. (MQes) and Process Research Associates Ltd. (PRA) that provided gold and silver recoveries that were used in tabulation of the Feasibility Study mineral reserves.

A significant portion of the Inéz vein currently is classified as Inferred mineral resources. SSR should assess the value of completing additional drilling on the Inéz vein in order to upgrade these Inferred resources to Measured or Indicated status. Based on drilling completed to date, the gold and silver grades within the Inéz vein are significantly lower and less consistent than those found in the Ayelén vein.

21.2 MINING

The mining method and extraction rate selected for the exploitation of the Ayelén vein is appropriate for the deposit's width, dip and length. It is a proven mining method commonly used in Peru.

The productivities used to calculate the mine production rate are reasonable for narrow vein mining in Peru.

21.3 PROCESSING

The process flowsheet to treat San Luis material has been developed based on results of metallurgical testwork to date. Additional metallurgical testwork will better define the metallurgical response and assess the effect of ore variability and optimize reagent schemes.

The average grade composite (Composite C) used in the second G&T test work program was intended to have gold and silver grades similar to those of the average life-of-mine, however, the resulting sample assays were higher than anticipated.

Drill core and assay reject material has been used in the preparation of metallurgical composite samples. None of this material has been from recently drilled material and is mostly from material that has been in storage (open to the environment) for in excess of 12 months. It has not been determined if oxidation had occurred with this material and, if it has, what the effect is on metallurgical response.

21.4 TAILINGS STORAGE FACILITY

The tailing storage facility adjacent to the processing plant (TSF-0) was selected as the primary tailing storage site with TSF-1 as an alternative. The TSF will provide storage for approximately 500,000 metric tons of tailing. The preferred deposition method for a conventional thickened tailing product will be spigotting from the embankment perimeter, with the objective to maintain the reclaim water pond in the southeast corner of the impoundment. No fatal flaws were identified that would prohibit the development of the project.

The preferred dam construction is a zoned rockfill embankment, to be constructed using borrow materials from within the TSF impoundment and adjacent areas. The impoundment will contain a liner system including a double geomembrane with a leak detection system. This liner system will cover the impoundment basin and upstream slope of the embankment. The intent of the liner system is to minimize seepage to groundwater. A blanket drain will be constructed under the impoundment to control groundwater flow from below the TSF.

Surface water diversion channels will be constructed around the TSF. These channels will collect and divert stormwater, thus limiting the amount of runoff that will be collected in the impoundment. During its operational phase, the TSF embankment will not have a spillway to avoid discharge of potentially contaminated water from the impoundment. Therefore, the TSF is conservatively designed to contain the runoff of the PMP from the entire catchment area above the TSF (excluding any reduction available from use of the surface water diversion channels).

Due to the relatively short project life and small TSF size, phased construction of the TSF was not considered. The TSF would be constructed in one phase during mine facilities construction, and would be completed just prior to the time the processing plant is commissioned. In order to optimize the construction process, the TSF and water diversion structure should be constructed during the dry period.

Assuming that construction of the TSF will be completed prior to the start of the wet season, an adequate amount of water for the start of the operations will be accumulated in the impoundment. The start-up water pond should not exceed 40,000m3. The water balance analysis was performed for three scenarios (wettest, driest and average years on record). The results of the water balance for the TSF using the wettest years on the record indicate that the reclaim water pond will grow with time. To maintain adequate freeboard during the later months of operation, measures to reduce the reclaim water pond volume may have to be taken early in the operational phase.

The overall strength values for the rock mass at the TSF embankment location are expected to represent favourable foundation conditions for the proposed design in terms of stability (for failure and sliding) and bearing capacity for the rockfill embankment. The results of the stability analyses indicate that the stability of the dam under static conditions is acceptable. Stability analyses indicate that the estimated factors of safety exceed the minimum required factors of safety for the embankment configuration with an upstream slope of 3H:1V and downstream slope of 2H:1V. Based on the simplified deformation analyses presented, the displacement of the embankment from anticipated seismic events is estimated to be less than 30cm. The available freeboard after this assumed deformation is considered adequate to preclude release of tailing due to a seismic event.

The results of static ABA tests on tailing and waste rock samples indicated no potential to generate acid rock drainage. From the kinetic HCT of the waste rock, it is reasonable to assume that it is unlikely that these samples will ever consume all of the available alkalinity and produce acidic leachate. Preliminary results for the tailing samples show similar results for the waste rock, with neutral to alkaline leachate with very low metals concentration. The results of the SPLP tests indicated that the maximum allowable concentrations are exceeded in 4 of the 5 tailings samples for cyanide, all of the tailings samples and one of the waste rock for antimony, all of the tailings samples for arsenic, one tailings sample for cadmium, and four tailings samples for lead. The aesthetic objective for aluminum was exceeded for all samples. Whole rock chemical analyses results indicate that all of the samples are elevated in most metals and metalloids (e.g., antimony, arsenic, copper, lead, molybdenum, selenium, and silver) relative to ranges normally found in average crustal rocks.

21.5 WATER SUPPLY STRUCTURE

A number of alternatives were considered for the WSS. The WSF-1 site was selected as the preferred location for the WSS. Both a diversion structure and a water storage dam were considered at this site. Constructing a diversion structure was selected as a more practical and economical option considering the relatively small amount of required fresh water supply.

21.6 OTHER INFRASTRUCTURE

The infrastructure components required for the project are each proven processes and technologies.

21.7 COSTS

The capital cost for the project is estimated at US$90.4 million to a target accuracy of ±15%. The operating cost is estimated at US$160.83/t ore processed.

ITEM 22 • RECOMMENDATIONS

Recommendations for further work are presented in the Sections below.

22.1 GEOLOGY

SSR should consider exploration drilling of the Ayelén vein at depth, to check for a possible "telescoped" precious metal horizon. The success of the project as outlined in this technical report is not contingent on the success of further drilling.

22.2 MINING

An alternate plan to mine the near surface portion of the Ayelén vein from surface should be investigated. In addition to cost, the permitting and social perception implications must be considered for such an approach. The estimated cost for this investigation is $10,000.

Tradeoff studies should be completed to evaluate the opportunity to improve project economics through the use of mining contractors during the development and/or production phase of the mine. The estimated cost for these tradeoff studies is estimated to be $15,000.

Given the short mine life, the use of leased or used equipment should be investigated during the procurement process. The estimated cost for this investigation is $5,000.

The mine ventilation system should be modeled further to assess the advantages and disadvantages of using one or more large main fans instead of several smaller fans. The estimated cost for this investigation is $5,000.

A cost benefit analysis of diesel versus electric powered LHDs should be completed prior to the procurement process. This analysis should consider operating and capital costs, electrical costs, ventilation costs, and maintenance labor availability. The estimated cost for these tradeoff studies is estimated to be $15,000.

22.3 PROCESSING

It is recommended a geological model for this project be developed. The combination of the geological model and additional metallurgical testwork needs to be aimed at defining the metallurgical characteristics of the project and establishing a geo-metallurgical model for the project. A budget cost for this metallurgical testwork is estimated to be $500,000.

It is recommended that tests be conducted on a life-of-mine composite that better reflects deposit grades and characteristics. A budget cost for this metallurgical testwork is estimated to be $100,000.

It is recommended that viscosity/rheology testwork be performed on a range of samples, particularly on samples close to the surface where significant weathering may have occurred. A budget cost for this metallurgical testwork is estimated to be $50,000.

It is recommended that selection of composites for ongoing metallurgical testwork is coordinated with SSR's geological staff to ensure representative samples of the different zones and rock types are correctly chosen. It is recommended fresh core is drilled and used for ongoing metallurgical testwork. The testwork goal should be to develop a geo-metallurgical model of the deposit which can be used to reasonably predict metallurgical responses. As mine planning is refined metallurgical testwork samples need to be chosen that are representative of the mine plan and with an emphasis on the early period of production. These samples should be tested to confirm the metallurgical response is consistent with that predicted from the geo-metallurgical model. The cost for obtaining samples will involve determining how many samples and cost to drill them. This cost is undetermined at this point.

The estimated process operating costs for the project are US$56.32/t ore. These costs are based on metallurgical testwork along with budgetary quotations for reagents and consumables. It is recommended reagent and consumable consumption rates are further updated upon completion of additional testwork. The cost for this testwork is included in above estimate.

22.4 TAILINGS STORAGE FACILITY

Additional data from the neighboring Pierina meteorological station and the local station at the project site should be collected and analyzed to provide a more detailed precipitation database to be used for final design. The estimated budget for this recommendation is $50,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

The stream flows in the TSF area should be monitored and recorded to provide a better correlation between stream flows and precipitation rates. The estimated budget for this recommendation is $50,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

The material and water balance analyses, the predicted capacity and operating life of the facility, the rate of rise of the impoundment, and the predicted excess water volume should be updated as additional data pertaining to the actual conditions becomes available. The estimated budget for this recommendation is $20,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

Consideration should be given to the preferred method for controlling (reducing) the reclaim water pond volume if precipitation during the operation life of the TSF is similar to the wettest years on record. In this case, installation of a sprinkler system to enhance the rate of evaporation, water treatment or other alternatives should be considered. The estimated budget for this recommendation is $10,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

For final design, additional geotechnical data should be collected to validate the foundation conditions and borrow material at the TSF-0 site. A drill hole should be advanced in the area of the proposed portal and mine access pad to obtain data on soil and bedrock conditions in that area. Monthly water level measurements should be taken in the existing site piezometers. Additional piezometers should be installed on the ridges in selected areas around the mine area to confirm groundwater levels. The estimated budget for this recommendation is $160,000.

Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

The mine inflow estimates are steady state flows. The mine tunnel inflow estimates do not consider the possibility of intersecting high permeability fracture zones which may not be continuous, but may provide significant inflow. Numerical modeling based on available drilling information and measured groundwater levels is recommended to provide a better estimation of the mine inflows. The estimated budget for this recommendation is $60,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

During the next stage of the project, ground motion time histories should be developed based on the seismic hazard assessment. The ground motions would be used in the dynamic analysis of the TSF. A simplified deformation analysis was performed to evaluate potential embankment deformations due to the design seismic event. A more robust deformation analysis (e.g., equivalent linear or non-linear methods) is recommended in the next phase of the project to take into account the non-linear response of the foundation soils under large earthquakes. The estimated budget for this recommendation is $40,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

22.5 WATER SUPPLY STRUCTURE

The stream flows in the WSS area should be monitored and recorded to provide a better correlation between stream flows and precipitation rates. The estimated budget for this recommendation is $60,000. Advancing to a subsequent phase of work is not contingent on positive results from this phase of work.

22.6 ENVIRONMENTAL

On-going monitoring in support of the EIA should be continued to extend wet and dry season information. The estimated cost of the ongoing monitoring is $285,000. Successfully completing the monitoring is necessary to completion of the EIA report.

It is recommended that the Environmental Impact Assessment report be complete. The estimated cost to complete the report is $54,000. Successfully completing the EIA report is necessary to advancing the project to construction and operations.

It is recommended that the project permits applications be submitted. The estimated cost to complete the permitting process is $180,000. The project permit applications must be submitted to advance the project to construction and operations.

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ITEM 24 • DATE AND SIGNATURE PAGE

The individuals listed below are "qualified persons" as defined by CSA NI 43-101 and are responsible for this technical report. Certificates of qualifications are provided in Appendix A.
The effective date of this technical report is June 4, 2010.

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"Clinton Strachan"

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"Steve L. Milne"

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ITEM 25 • ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

25.1 MINING OPERATIONS

25.1.1 Introduction

The Ayelén vein system at the San Luis Project consists of a series of fracture infilling, epithermal veins containing high-grade gold and silver in low sulphidation mineralization. The vein system is located parallel to and near to a main structure and is made up of three main bands of mineralization. Drilling results show that the controlling fault structure(s), sub-surface individual vein segments and post-mineral dykes dip vertically, to minus 80°, west-southwesterly. True thicknesses of individual vein segments vary from ten's of centimetres to over 10m, averaging 1.5m to 3m wide.

Rock quality in the area varies from good to bad. However, in general, the rocks away from faults, and the vein (quartz filled) and vein walls (andesite) are generally strong. The principal vein in the series is the Ayelén vein. The veins are located at an altitude of between 4,200 and 4,600 m above sea level.

The vein system crosses a hill named Cahuarán in a Northwest-Southeast direction. Cahuarán Hill is located between two small valleys named Huanchuy and Tocash. The Huanchuy valley would allow surface access of the deposit up to 4,300m, and the Tocash valley to the deeper levels. This surface access will allow development and mining of most of the deposit by tunnels, or adits, connected to the surface. Some internal ramping may be necessary to access deeper parts of the vein, where a tunnel to the surface could not be justified.

During diamond drilling exploration, a few fractured and oxidized zones and minor fault zones were encountered, suggesting the presence of underground water. The quantity of water is unknown, but assumed to be small due to the strong, tight vein and very low hydraulic conductivity of the andesite. Any water encountered should be easily handled in ditches constructed in the access/haulage drifts. Due to the low sulphidation genesis of the deposit, no acid rock drainage is expected.

25.1.2 Production Schedule

Production is scheduled to be initiated in the first two stopes on Level 4550, and continue downward until the deposit is exhausted. Figure 25-1 presents a longitudinal section of the Ayelén vein along with the respective stope locations and extraction sequence. Stopes with a minimum width of less than two meters are planned to be mined using captive electric micro scoops (0.7yd^3 bucket capacity), which will limit the production from these stopes to about 1,700 tonnes/month. Stopes wider than two meters are planned to be mined by 2.5yd^3 captive electric scoops, which would result in a production rate of approximately 417 tonnes/day. In both cases, the load-haul distance is limited to between 40 and 60m.

The detail mine production, by stope, is presented in Table 25-1.

Figure 25-1: Stope Locations and Extraction Sequence

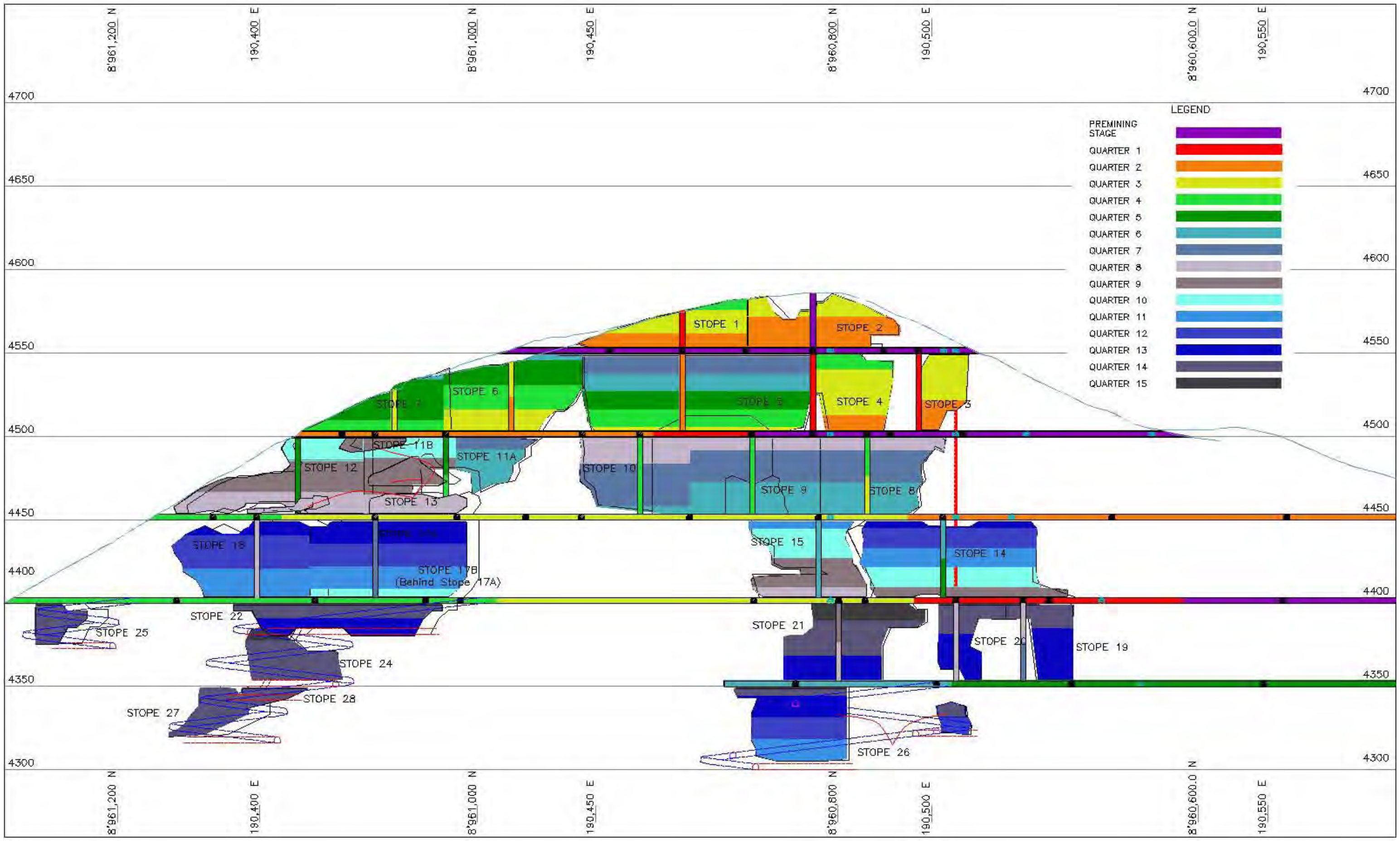


Table 25-1: Mine Production Schedule

Source	Au	Ag	Development	Year 1				Year 2	Year 3	Year 4	Total
	g/t	g/t		Q1	Q2	Q3	Q4				
Stope 1	24.04	663.81		0	7,513	5,622	0	0	0	0	13,135
Stope 2	38.89	884.58		0	7,733	13,257	7,003	0	0	0	27,993
Stope 3	3.79	186.99		0	1,652	2,548	0	0	0	0	4,200
Stope 4	14.21	416.44		0	1,703	5,109	1,537	0	0	0	8,349
Stope 5	31.45	757.43		0	0	2,202	13,209	40,687	0	0	56,098
Stope 6	7.53	115.79		0	0	4,560	5,472	6,605	0	0	16,637
Stope 7	40.5	770.49		0	0	0	3,119	14,490	0	0	17,609
Stope 8	33.86	753.31		0	0	0	0	26,816	0	0	26,816
Stope 9	10.1	329.17		0	0	0	0	12,727	0	0	12,727
Stope 10	12.02	372.28		0	0	0	0	16,123	0	0	16,123
Stope 11A	7.06	84.13		0	0	0	0	3,909	0	0	3,909
Stope 11B	32.83	615.78		0	0	0	0	6,387	19,511	0	25,898
Stope 12	12.48	312.92		0	0	0	0	5,688	14,576	0	20,264
Stope 13	6.85	345.53		0	0	0	0	4,224	6,740	0	10,964
Stope 14	9.88	154.51		0	0	0	0	0	44,131	3,801	47,932
Stope 15	17.49	469.58		0	0	0	0	1,614	10,858	0	12,472
Stope 17 A	5.98	344.97		0	0	0	0	0	20,853	8,670	29,523
Stope 17 B	5.27	197.82		0	0	0	0	0	8,551	0	8,551
Stope 18	9.07	349.54		0	0	0	0	0	23,427	4,962	28,389
Stope 19	7.85	234.82		0	0	0	0	0	0	4,460	4,460
Stope 20	5.04	240.33		0	0	0	0	0	0	6,099	6,099
Stope 21	4.52	173.58		0	0	0	0	0	0	18,446	18,446
Stope 22	7.96	270.21		0	0	0	0	0	0	10,432	10,432
Stope 24	6.9	319.85		0	0	0	0	0	0	7,437	7,437
Stope 25	18.9	644.27		0	0	0	0	0	0	1,831	1,831
Stope 26	4.72	97.65		0	0	0	0	0	11,184	7,432	18,616
Stope 27	14.66	355.66		0	0	0	0	0	0	2,991	2,991
Stope 28	5.33	275.43		0	0	0	0	0	0	1,949	1,949
Preprod Dev.	33.88	772.91	8,475	0	0	0	0	0	0	0	8,475

Expensed Dev. Y1Q1	17.63	473.31		5,303	0	0	0	0	0	0	5,303
Expensed Dev. Y1Q2	27.79	608.39		0	5,199	0	0	0	0	0	5,199
Expensed Dev. Y1Q3	22.81	539.36		0	0	4,379	0	0	0	0	4,379
Expensed Dev. Y1Q4	11.67	408.17		0	0	0	7,025	0	0	0	7,025
Expensed Dev. Year 2	8.85	295.17		0	0	0	0	10,739	0	0	10,739
Expensed Dev. Year 3	10.16	352.92		0	0	0	0	0	2,343	0	2,343
Expensed Dev. Year 4	0	0		0	0	0	0	0	0	0	0
Totals	**17.95**	**446.14**	**8,475**	**5,303**	**23,800**	**37,677**	**37,365**	**150,009**	**162,174**	**78,510**	**503,313**
Au (g/t)			33.88	17.63	27.57	24.85	25.67	24.05	12.05	6.9	17.95
Ag (g/t)			772.91	473.31	672.64	600.37	608.69	563.76	307.18	251.32	446.14

Waste		15,968	15,057	24,672	30,150	34,080	42,910	5,606	0	168,443
Total Material		**24,443**	**20,360**	**48,472**	**67,827**	**71,445**	**192,919**	**167,780**	**78,510**	**671,756**

All of the ore extracted from the mine each period will be hauled to a surface stockpile located near the process plant. At the stockpile, the newly mined ore will be blended with the existing stockpile inventory. This blended result will then be fed to the processing plant.

Since newly-mined ore is continuously being added and blended with the stockpile ore, and blended stockpile ore is continuously being sent to the mill, the stockpile tonnage and grade will also be changing continuously.

Table 25-2 presents the sequence in each production period that results in an ending stockpile inventory (tonnage and grade) at the end of each period.

Table 25-2: Process Stockpile and Plant Feed Schedule

Source	Development	Year 1				Year 2	Year 3	Year 4	Total
		Q1	Q2	Q3	Q4				
Stockpile at Start									
- Tonnes	-	8,475	1,778	78	1,255	2,120	6,129	22,303	
- Au (g/mt)	-	33.88	27.63	25.57	27.86	25.74	24.07	12.49	
- Ag (g/mt)	-	772.91	657.6	671.53	600.53	608.43	564.38	316.54	
Added from Mine									
- Tonnes	8,475	5,303	23,800	37,677	37,365	150,009	162,174	78,510	503,313
- Au (g/mt)	33.88	17.63	27.57	24.85	25.67	24.05	12.05	6.90	17.95
- Ag (g/mt)	772.91	473.31	672.64	600.37	608.69	563.76	307.18	251.32	446.14
Fed to Mill									
- Tonnes		12,000	25,500	36,500	36,500	146,000	146,000	100,813	503,313
- Au (g/mt)		27.63	27.57	24.85	25.74	24.07	12.49	8.14	17.96
- Ag (g/mt)		657.60	671.59	600.52	608.42	564.38	316.55	265.75	446.14
Stockpile at End									
- Tonnes	8,475	1,778	78	1,255	2,120	6,129	22,303	0	
- Au (g/mt)	33.88	27.63	25.57	27.86	25.74	24.07	12.49	-	
- Ag (g/mt)	772.91	657.6	671.53	600.53	608.43	564.38	316.54	-	

The scheduled full production rate was established at 417tpd, for 350 days per year, or 146,000tpy of feed to the process plant, based on the available minable reserve. Five producing stopes are required to produce the scheduled 12,170 tonnes per month.

The currently delineated minable reserve will provide a mine life of approximately 3.5 years. However, on-going exploration in the area during the mining phase, as well as evidence of Ayelén's projection and other veins resources, may extend the mine life.

Since it is typical to experience an S-shaped learning curve at the startup of new mining operations, there is no ore processing scheduled for the first three months. Initial production in the plant has been scheduled to begin at 33% of capacity, then ramping up to 100% of capacity through month eight. Full production is scheduled to commence in the 9th month, at a rate of 12,170t per month, or 146,000tpy, until the currently-delineated minable reserve has been exhausted. This approach is illustrated in Figure 25-2 below.

Figure 25-2: Plant Production for the first year

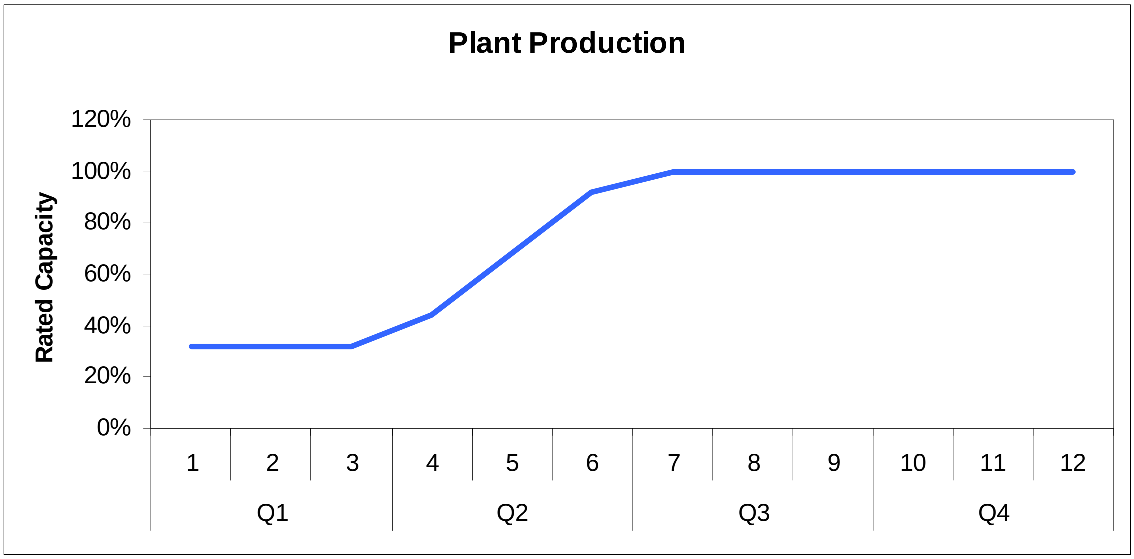
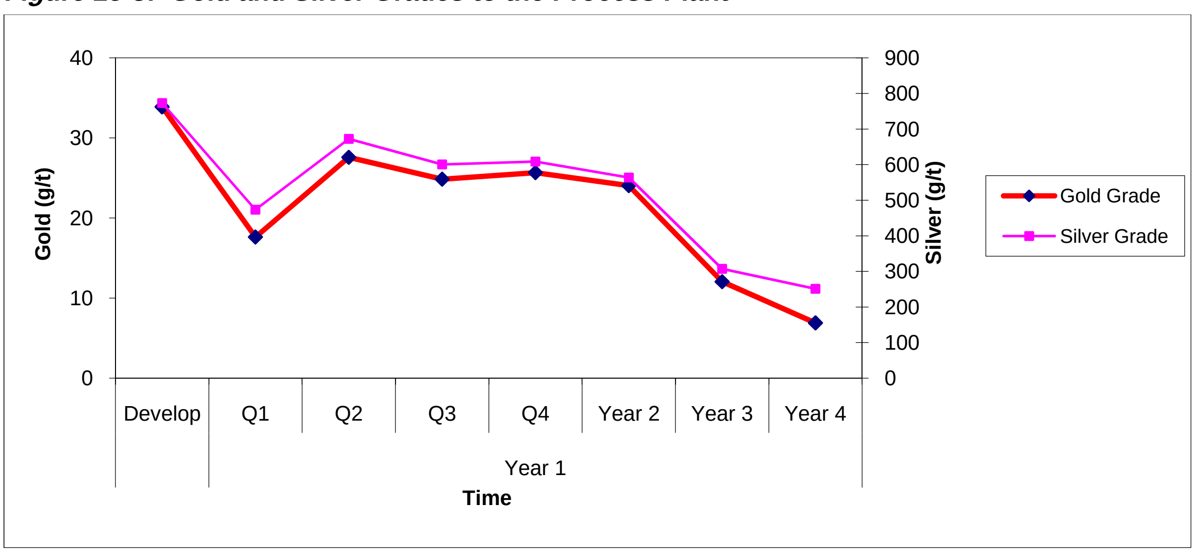


Figure 25-3 below illustrates the estimated gold and silver grades delivered to the mill, through the life of the mineable reserve.

Figure 25-3: Gold and Silver Grades to the Process Plant



25.1.3 Preproduction Development

25.1.3.1 Mine Access

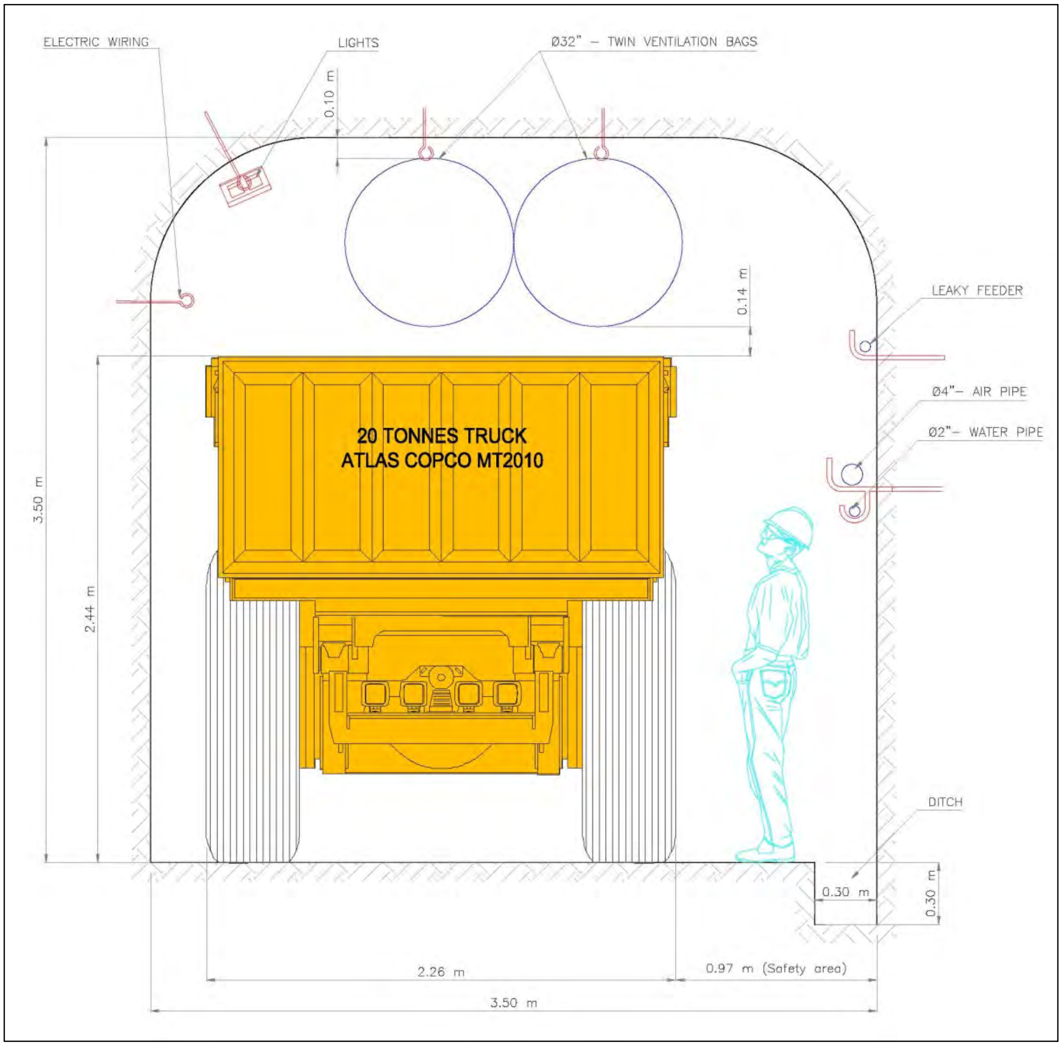
Since the local topography at the deposit site will permit tunnel access to the Ayelén vein from the Huanchuy and Tocash valleys, primary access/haulage levels have been designed along the Ayelén vein at elevations of 4,350 4,400, 4,450, 4,500, and 4,550m above sea level. Access to the Ayelén vein from the Huanchuy valley can be made from the surface down to elevation 4,300m, while the Tocash will allow access to the vein from the surface to deeper elevations.

Concrete lined portals will be constructed at each of the access drifts. These portals are estimated to take one month each to construct.

Current mine planning has the four access/haulage ways being developed simultaneously at an average advance rate of 120 meters per month, each. The total construction time for the access drifts is estimated to take approximately 14 months, although only 5 months will be needed to have enough accesses to start getting ore from stopes.

Each haulage way will be driven at a +0.3% inclination, and include a 0.3m x 0.3m water drainage ditch. Figure 25-4 shows a typical level drift cross section.

Figure 25-4: Typical Access Drift Cross Section



25.1.3.2 Mine Development

After the initial access/haulageways have been constructed, lateral drifts are planned in the footwall of the vein on each access level. These lateral drifts (by-passes) will be driven along the strike of the vein at intervals of between 50 and 100 meters and at least 12 meters into the footwall, with 3.5mx3.5m crosscuts connecting the by-pass (footwall) to the drifts on the vein.

All lateral and crosscuts openings will have the same cross-sectional design and gradient as the primary access/haulageways. Advance rates in the lateral drifts and crosscuts are also planned at 120 meters per month.

Figures 25-5 to 25-9 show plan view sections of the 4350, 4400, 4450, 4500 and 4550 levels respectively.

Figure 25-5: 4350 Level Development

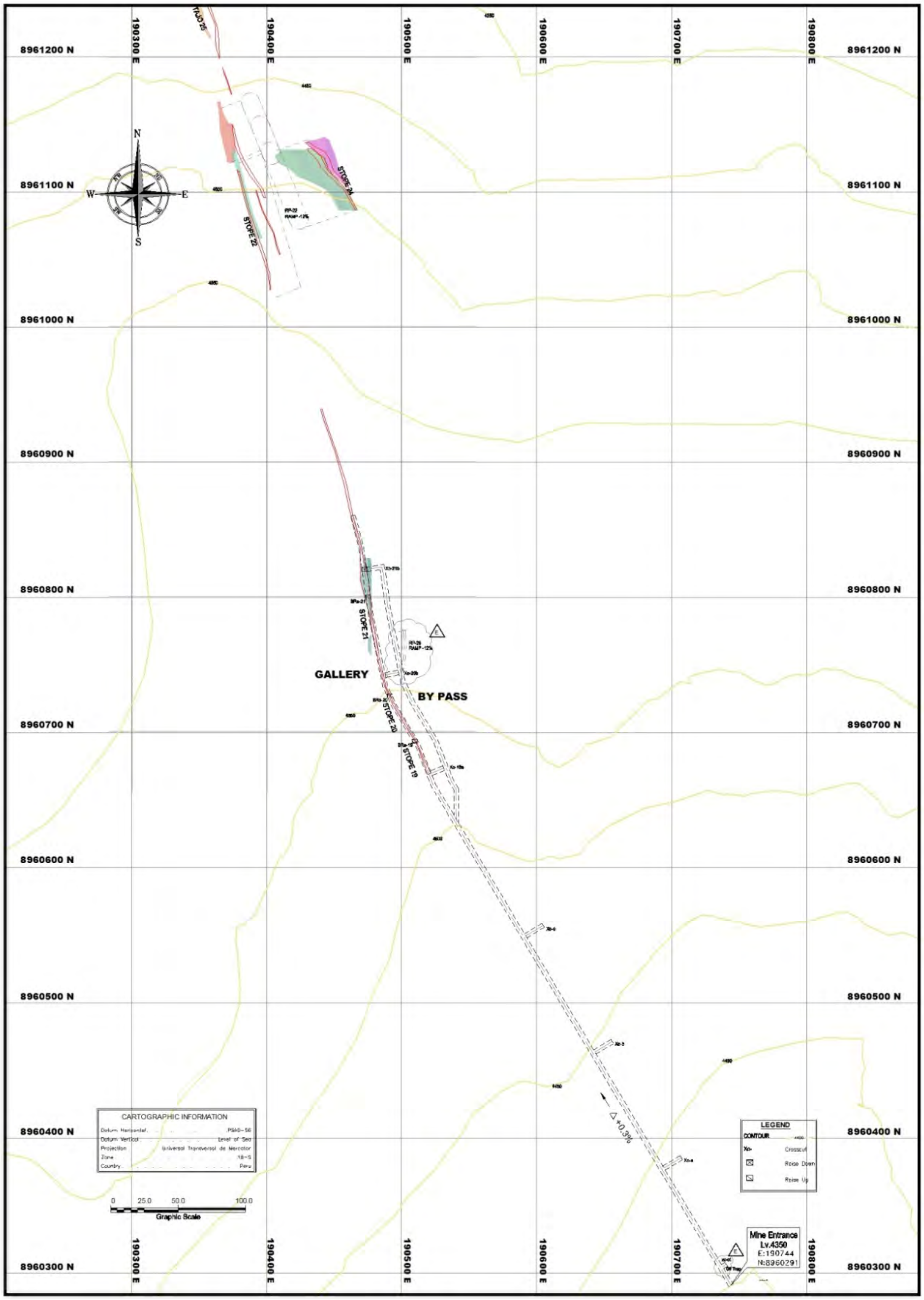


Figure 25-6: 4400 Level Development Plan

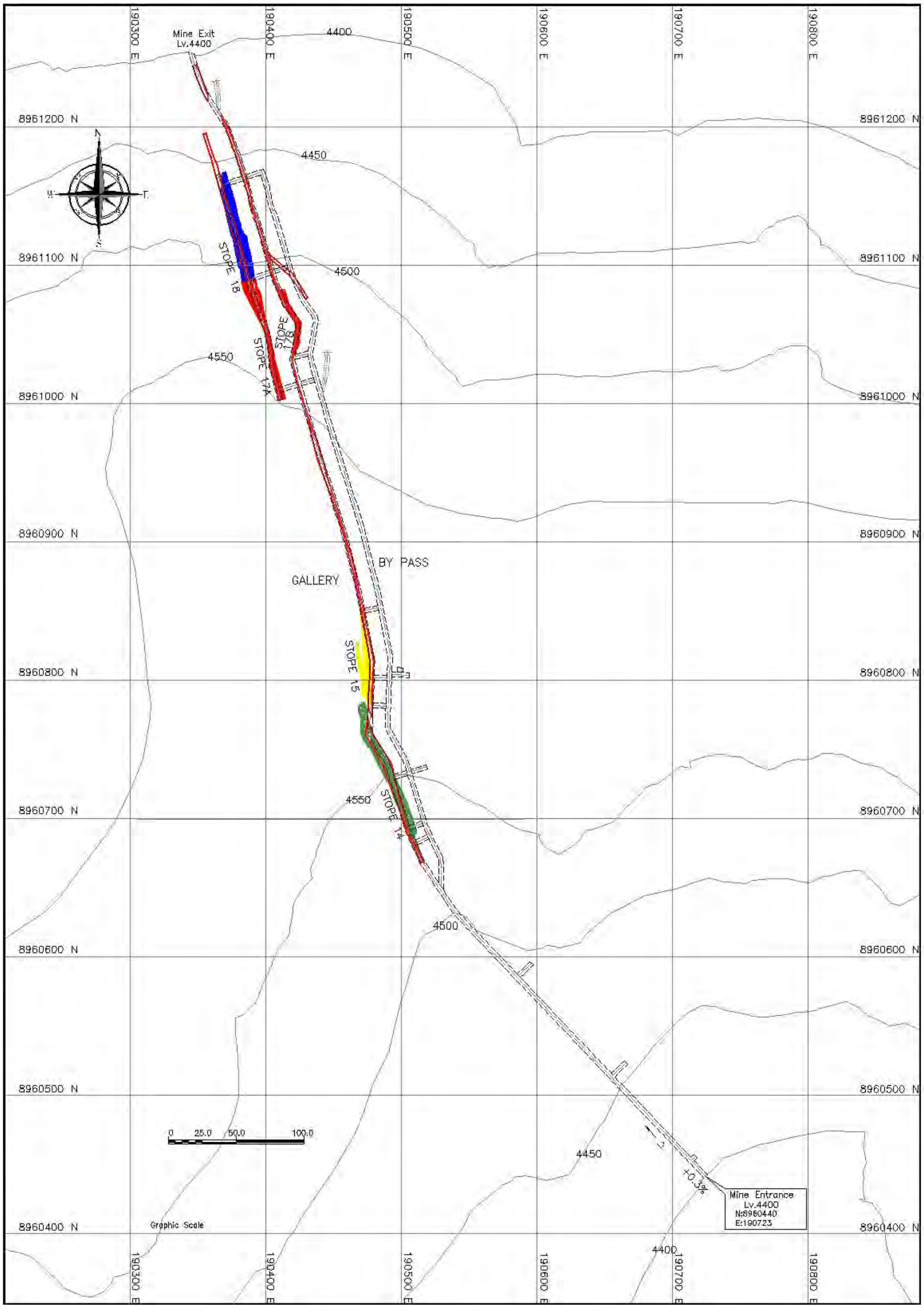


Figure 25-7: 4450 Level Development Plan

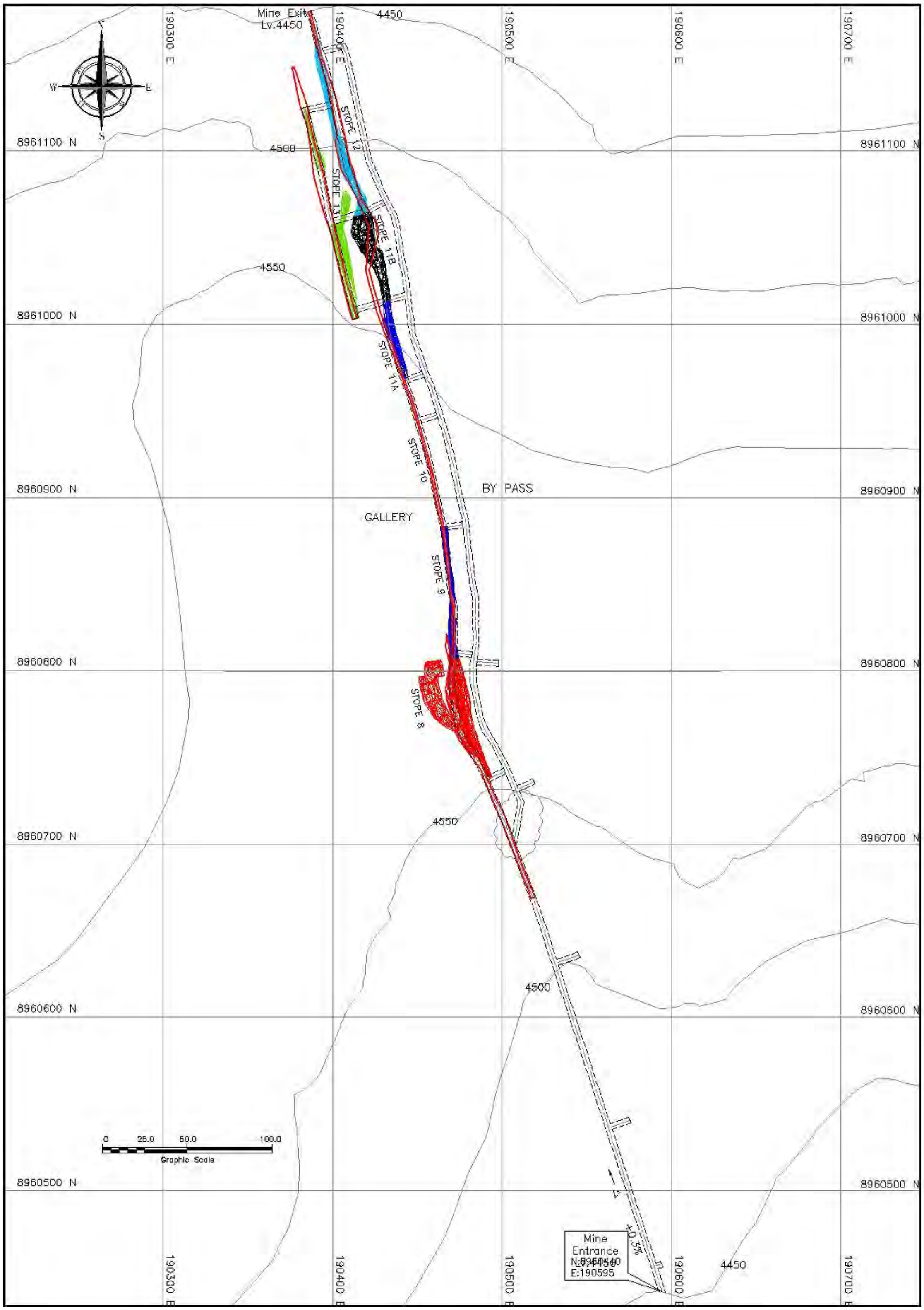


Figure 25-8: 4500 Level Development Plan

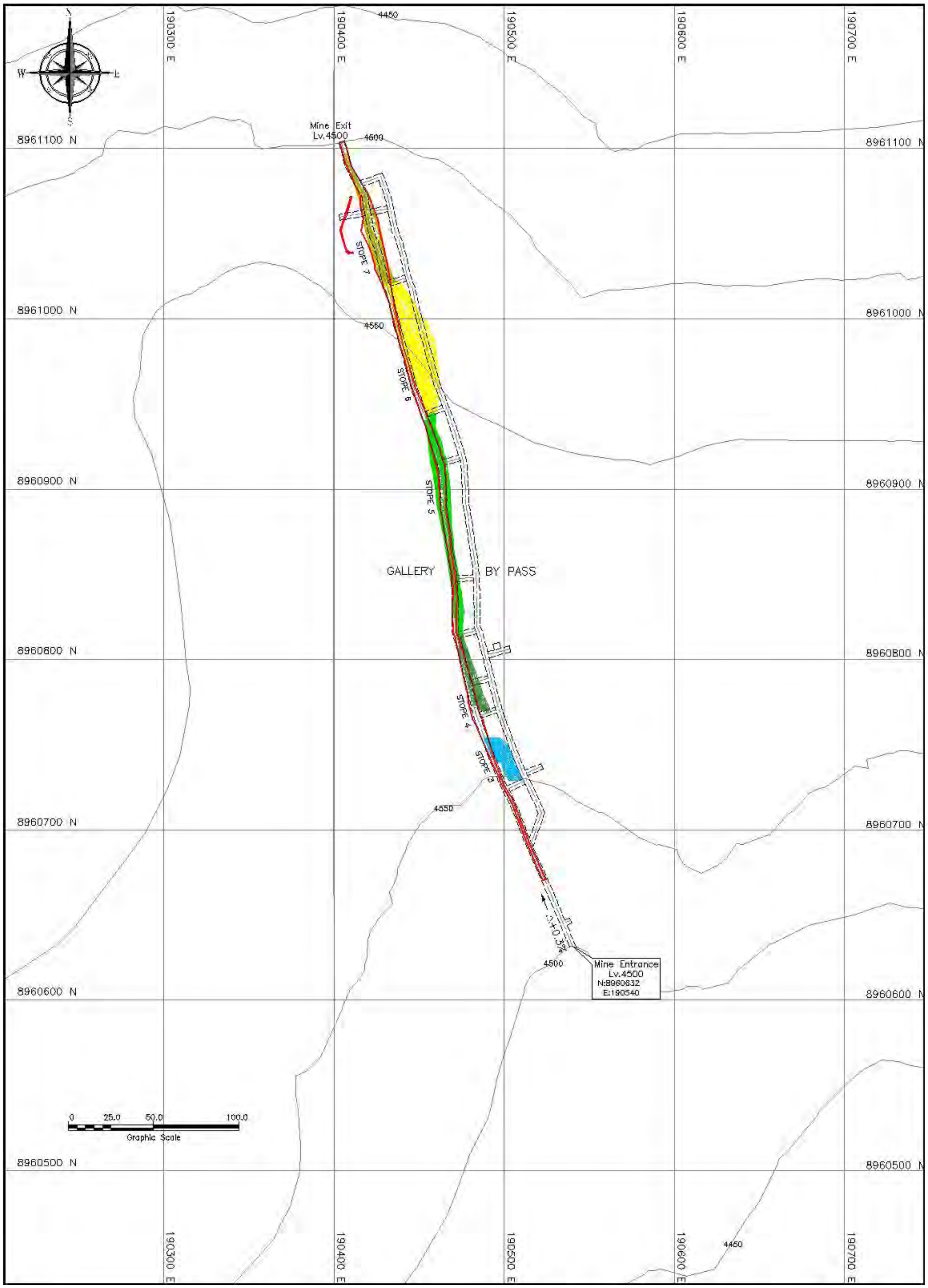
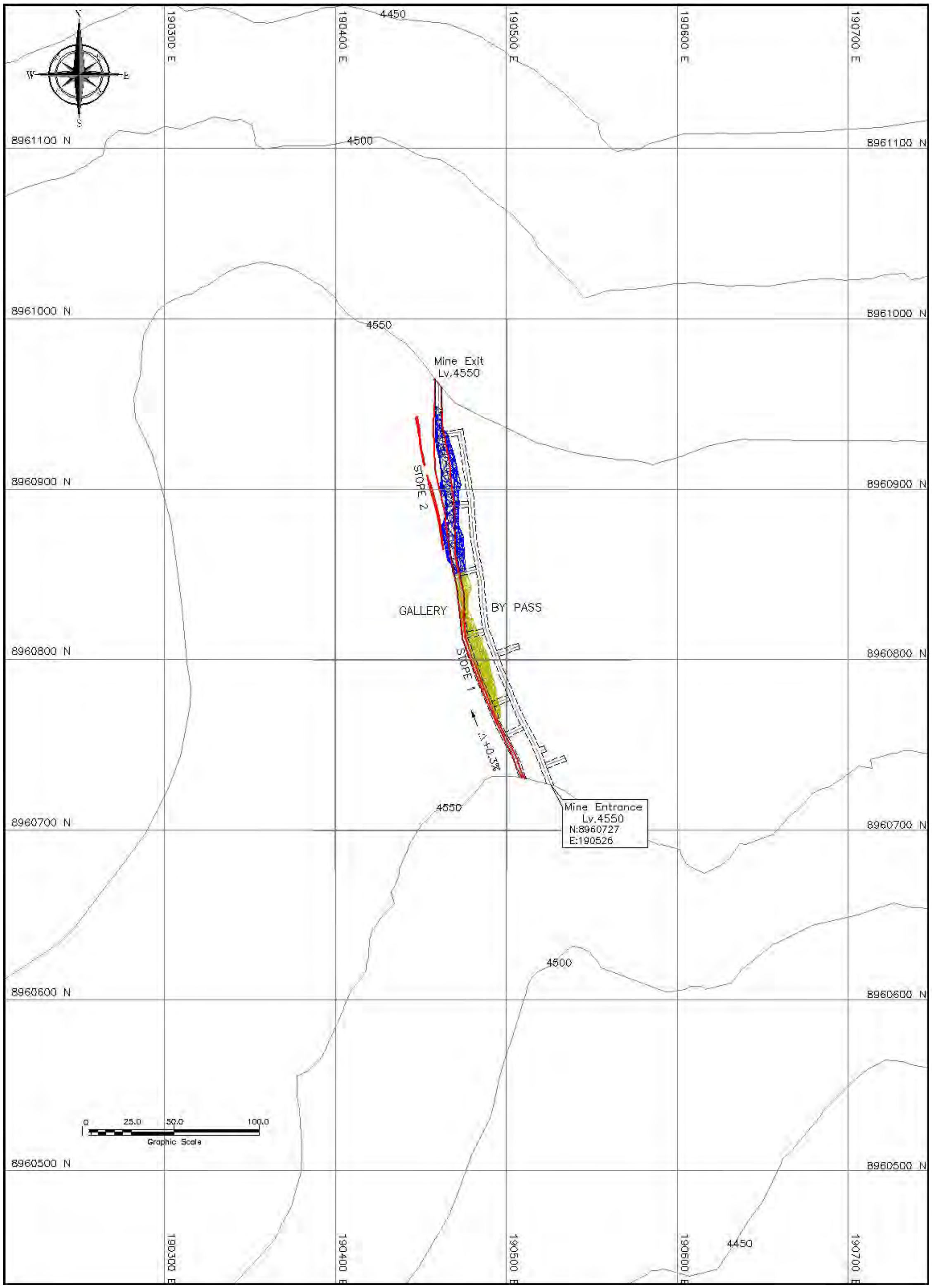


Figure 25-9: 4550 Level Development Plan



All preproduction development drift openings are planned to accommodate a 32 inch twin ventilation bags. Primary ventilation fans will be located at each access portal. The planned volume of ventilation air for each heading is 1,275m^3/min (45,000cfm). The ventilation air will be delivered by five 42,000cfm primary fans.

Electric-hydraulic jumbos, using 3.67m drill rods and 45mm button bits, will perform the 42 holes drilling in each heading. Blasting agents for drifts will include ANFO, nonelectric caps, detonator cord, assembled Carmex guide, and igniter cord. Mucking will be performed with 4.2yd^3 scooptrams, loading into 20 tonne low profile haulage trucks. These trucks will transport the ore directly to the ROM stockpile, or to the waste dump near each mine portal.

In this premining stage, ore coming from galleries and vein raises will be hauled to the ROM stockpile which will be located 1km from the mine entrances on average. When the ore pass is completed, ore will be dumped through and hauled from the 4,400 level by a 20 tonne low profile truck to the ROM stockpile.

In compliance with Peruvian Safety regulations, all stope preparation will include a two-compartment raise (1.5m x 3.0m) driven in the vein. This central raise will provide exhaust ventilation from the stope and for passing down the backfill material. Intake ventilation to the stopes will be through raises extended from the haulage level below to the stope extremities.

A summary of the preproduction development meters required is shown in Table 25-3. A factor of 10% has been added to all take-off quantities for access drifts, laterals, and crosscuts to allow for turnouts and other miscellaneous openings.

Table 25-3: Preproduction Development

Item	Length (Meters)
Backfilling Raises	40
By Passes	288
Crosscuts	168
Galleries	751
Service Raise	40
Total	1,287

The estimated total time to construct the preproduction development necessary for initiating production in Stopes 1 and 2 is approximately 6 months. Stopes 3 and 4 will be prepared by month 8; and Stopes 5 and 6 by month 9. This will allow the full production rate of 12,170 tonnes per month to begin by month 9.

25.1.4 Mining Method

Since the payable metals are contained in irregular, steeply-dipping, narrow veins, mining methods such as caving and room-pillar, were discounted immediately.

Shrinkage stoping, which is relatively flexible and widely used in Latin American for narrow veins, requires strong vein walls and allows only the swell of the broken ore to be extracted until the stope is completed. In addition, it is relatively more expensive to mine than a cut and fill method.

The various forms of longhole stoping, in general, require wider, more competent veins, along with more regular vein walls. In addition, this method is not as selective as other methods, such as shrinkage, or cut and fill. With reasonably strong vein and wall rock, the system is highly productive in wider veins and results in a lower mine operating cost. At the San Luis Project, where there are irregular, narrow veins, in potentially weaker rock, this mining method is considered unacceptable until better geomechanical information on the vein is available that would assure acceptable dilution and recovery of the ore, and meet safety standards.

The selected mining method is “mechanized overhand cut and fill”. This method is very selective, both in the horizontal and vertical directions, handles weak ground well, and allows a reasonable productivity (generally about 3 to 5 tonnes per manshift in the mine) in the mining process.

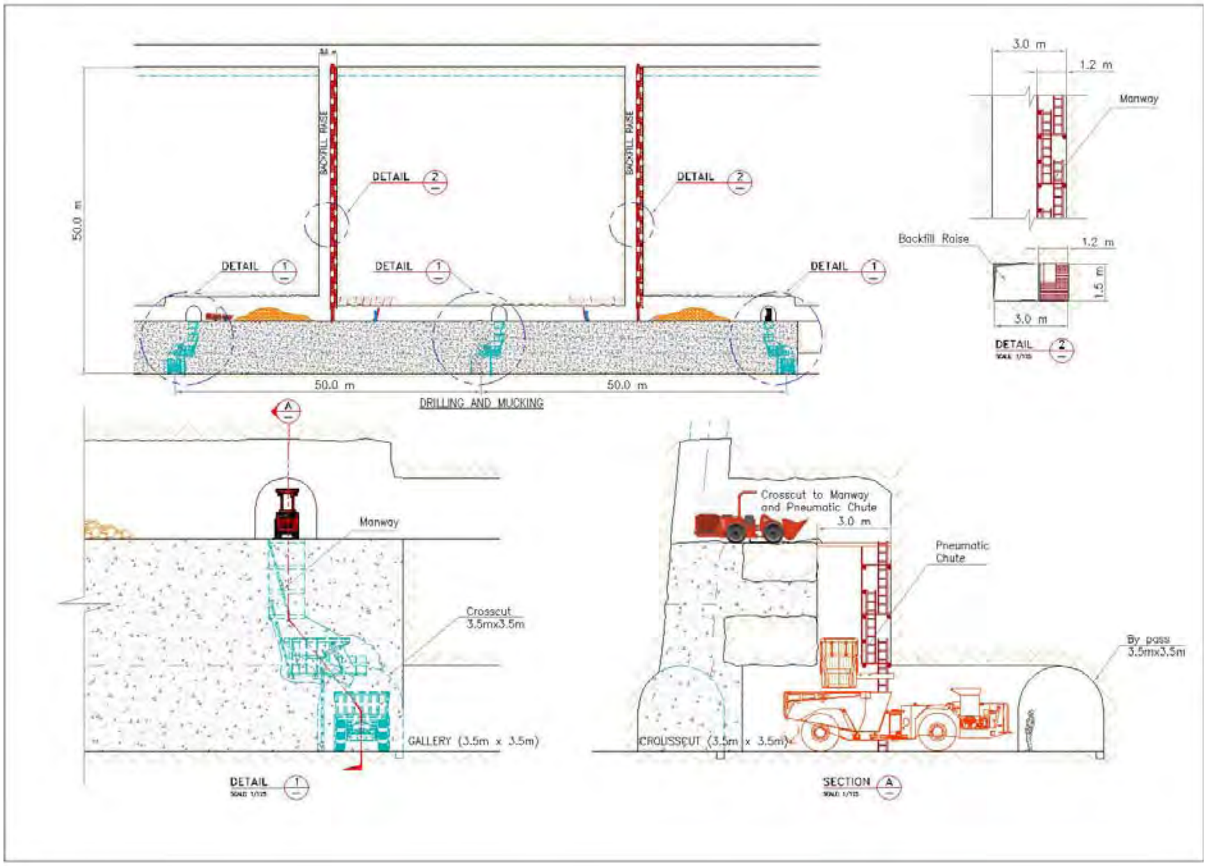
Mining will be performed conventionally with stoper drills and small, captive, electric scoops. There is the potential to use mill tailings for hydraulic backfill but this requires further evaluation. Waste rock can be economically hauled to the stopes from mine development (when available), or from the broad colluvium deposits located near each mine entrance.

25.1.5 Stoping

The stope is prepared with a backfill/ventilation raise in the center of the stope, and two, vertical access raises (1.5m x 3.0m) on the strike of the vein, located at the stope extremities. These access raises are as wide as the mineralization, or the minimum mining equipment width requirement, and about two meters in depth.

At the San Luis Project, the second cut will be accessed by ramping from the first cut inside the stope. This succession is planned for each new cut, leaving the equipment used for drilling and mucking captive in the stope until the level above is reached. This can lower productivity in the stope considerably, since all equipment repairs and maintenance must be performed in the stope. It also requires that extra stopes be prepared at all times to provide the programmed plant feed when scheduled stopes are down for equipment repairs. The advantage of this method is quick access to ore and substantially less capital development in waste rock. Due to the relatively short mine life and ease of access to additional stopes, the captive equipment approach is appropriate for this deposit. Figure 25-10 shows cross sections of the stoping method.

Figure 25-10: Diagram of Planned Mining Method



All stope blasthole drilling is to be performed with stopers, drilling 85º inclined up-holes in the back. The blasted ore will be removed by electric 0.7yd^3 micro-scoops when the vein is less than two meters in width, and electric 2.5yd^3 scoops, when the vein is over two meters width. The blasted ore will be deposited in a raise driven along the vein from the lower haulage to the upper haulage. From there, it will be loaded into low profile 20 tonne trucks, which will haul the ore to the ROM stockpile. A second raise, located at the stope extremities, will provide access for personnel, materials, and ventilation.

Auxiliary services like water for drilling, compressed air and electricity will be supplied both from the higher level through the manway in the backfill raise or from the stope level thru the manway in the chutes.

Once the ore pass is completed from Level 4,400 and 4,550, ore from the upper levels will be passed through it to Level 4,400, where it will be hauled by a 20-tonne low-profile truck to the ROM stockpile.

Once a stope cut has been emptied, it will be backfilled with waste rock to about two meters from the cut back. This backfill will be passed down the backfill raise from the level above into the cut, and distributed within the cut with the corresponding electric scooptram.

The expected production from cuts (stopes) having a width of less than two meters is 1,700 tonnes per month, and 4,200 tonnes per month from cuts (stopes) having a width of two meters, or more.

25.1.6 Backfilling

Backfilling in the stopes is planned to use surface classified detritus (colluvium). This material will be screened to a maximum of 6” in a portable, mechanical screening plant using a stacker. The screening plant will have sufficient capacity to be operated only on day shift. On night shift, the backfill truck driver will use a front-end loader to load his own truck from the stockpile. A 20-ton low-profile truck, or a dedicated surface truck, will transport the screened, waste backfill to the underground, where it will be transferred, or dumped directly into the backfill raises. The backfill raise is a two-compartment raise that will always provide a ventilation exit in the event the waste pass becomes plugged with backfill material. This double compartment backfilling raise can also be used to pass auxiliary services to the stope.

When the stope cut has been completed, the backfill will be dropped from the level above the stope, through the raise into the stope cut, where it will be distributed by the captive electric scooptram. A clearance above the backfill, of about two meters, will be left as a floor for the next cut.

An average of five stopes will be in production to meet the production demands (approximately 90 tonnes per stope, per day, in average). This equates to a backfill rate of approximately 115m^3 per day of classified dry waste coming from development or surface quarry. At any one time, two to three stopes will be in the backfilling phase.

The total voids to be backfilled (stopes) are calculated as 185,000m^3, and the available waste coming from waste development is around 95,000m^3 of loose waste. Therefore, the additional waste required from surface quarries is the difference of 90,000m^3 of classified material, which means that the waste dumps will be temporary.

25.1.7 Mine Working Schedule and Personnel

The mine is scheduled to operate two 12-hour shifts per day, seven days per week and 350 days per year. This schedule will require a mine production rate of 417 tonnes per day to meet the plant capacity of 400 tonnes per day, 365 days per year (146,000 tonnes per year).

Peruvian regulations allow miners to work 12-hour shifts for 14 consecutive days in a row. After that, seven days of rest must follow. A third crew will fill in while miners are on days off. An effective 9-hr work shift is assumed after deductions for travel, breaks, etc.

The overall mine productivity for cut and fill mining, using captive equipment in the stopes, is estimated to be approximately four tonnes per manshift. At a production rate of 417 tonnes per day, this equates to a total mine payroll of approximately 84 direct (hourly working), and 21 indirect (salaried) per day. The mine dry will be designed to accommodate approximately 260 hourly and salaried employees per day. Hourly employees, rotated off, will return to their respective houses away from the mine site. Camp housing will accommodate personnel working on site.

Table 25-4 summarizes the mine personnel required to operate the mine at 417 tonnes per day.

Table 25-4: San Luis Project Estimated Mine Workforce

Position	Shift A	Shift B	Shift C - Off	Total	Total - On Site
Exploration and Development					
Jumbo operator	2	2	2	6	
Jumbo operator's helper	2	2	2	6	
Rock support	1	1	1	3	
Rock support helper	1	1	1	3	
Scoop operator	2	2	2	6	
Subtotal	8	8	8	24	16
Extraction					
Stope miners	6	6	6	18	
Stope rock support	1	1	1	3	
Stope rock support helper	1	1	1	3	
Raise miners	3	3	3	9	
Raise miners' helpers	3	3	3	9	
Raise timberers	1	1	1	3	
Raise timbers' helpers	1	1	1	3	
Scoop operators	6	6	6	18	
Subtotal	22	22	22	66	44
Operations Support					
Explosive loader (development)	1	1	1	3	
Explosive loader (development) helper	1	1	1	3	
Explosive loader (stopes)	1	1	1	3	
Explosive loader (stopes) helper	1	1	1	3	
Backfill Dumper operator	1	1	1	3	
Backfill Loader operator and helper	2	2	2	6	
Warehouse	1	1	1	3	
Mine services	2		2	4	
Subtotal	10	8	10	28	18
Ore and Waste extraction					
Dumper operators (for extraction)	3	3	3	9	
Subtotal	3	3	3	9	6
Subtotal Direct	43	41	43	127	84
Indirect & Supervision					
Mine Manager	1			1	
Shift Foreman	2	2	2	6	

Position	Shift A	Shift B	Shift C - Off	Total	Total - On Site
Chief Engineer	1			1	
Mine Engineers	1	1		2	
Surveyors	2			2	
Chief Geologist	1			1	
Junior Geologist	1	1		2	
Samplers	2	2		4	
Clerks	2			2	
Mine Dry Attendants	1	1		2	
Subtotal Indirect	14	7		23	21
Total	57	48	45	150	105

25.1.8 Mine Equipment

The major mine equipment at the San Luis Project (see Table 25-5) will consist of two single boom electric-hydraulic jumbos for drilling in the major horizontal access/haulageways, laterals and crosscuts. Mucking in the major access/haulageways, laterals and crosscuts will be performed by two 4.2yd^3 diesel scooptrams, loading the broken rock into four, 20 tonne low profile haulage trucks. A mechanical availability of 70% for main access equipment, and a mechanical utilization of 40% for captive electric scooptrams have been used to estimate the spare equipment requirements.

Pneumatic, hand-held stoper drills will be used in the stopes for blasthole drilling. Mucking in the stope cuts will be with 0.7yd^3 and 2.5yd^3 electric scooptrams. The scooptrams will deposit the broken muck in ore passes located in the limits of each stope cut. Low-profile 20-tonne trucks will carry the ore from the bottom of the ore pass to the surface or to the main ore pass.

No underground maintenance shops are planned and all stoping equipment will be captive in the stope until it is completed to the level above. Underground captive equipment will have to be repaired and maintained within the stope. Therefore, the stope access raises will have to accommodate scooptram tire sizes. Surface and diesel equipment will be maintained in a central surface shop approximately 1km from the mine entrances.

With at least two access/haulageways under construction at the same time, two jumbos will be needed. Two diesel 4.2yd^3 scooptrams will be required for mucking in the access/haulage ways, and three 20-tonne low-profile trucks, plus a spare partially used for hauling backfilling material, will be required for this ore and waste extraction and backfill haulage.

Production from five active stopes will be needed to meet the full production goal of 417tpd. These stopes are categorized as being more than 2m in width (wide stopes) and less than 2m in width (narrow stopes). On average, production will be obtained from 3 narrow stopes and two wide ones. One additional narrow stope and one additional wide stope are also planned to be available for production at any given time. With this in mind, production equipment consisting of three 0.7yd^3 electric scooptrams and two 2.5yd^3 electric scooptrams has been included in the project equipment list. Further, a single spare 0.7yd^3 electric scooptram has also been included in the equipment list.

One haul truck per haulage level, plus a spare, equates to four low profile 20 tonnes haul trucks. In addition, one front-end loader or excavator will be needed to load and transfer backfill from the surface quarries.

With five active stopes, each requiring two stoppers per stope and a two spare stopers, the total number of stopers required is approximately 14; also two jacklegs and two spares are required for cable and pipe installation as well as groud support.

Other underground support equipment includes; three underground load centers, three 4x4 pick-up trucks for supervision and engineering, five 45,000cfm fans, six 15hp air tuggers for material handling in the stopes, and two 1,350cfm surface air compressors.

Table 25-5: Major Mine Equipment

Item	Description	Size	Qty
1	Elec. Hyd Jumbo (Boomer S1D-1838)	1 Boom	2
2	Diesel Scooptram ST-710	4.2 yd^3	2
3	Diesel Haul Trucks (Mt-2010)	20 tonnes	4
4	Electric Scooptram (ST-2D)	2.5 yd^3	2
5	Electric Scooptram (LH201-D)	0.7 yd^3	4
6	Front End Loader (CAT 950H)	4.5 yd^3	1
7	Surface Compressors	1350 cfm	2
8	Fan	42000 cfm	5
9	Stoper Drill (BBC 34w S6)	Handheld	14
10	Jack Leg Drills (BBC 16w)	Handheld	4
11	Submersible Pump (Master H)	18 l/s	2
12	Total Station		1
13	Utility Truck		1
14	Anfo Loader	55 lb	3

25.1.9 Temporary Waste Dumps

Overall calculations show that there will be a net waste requirement from surface quarries of 90,000m^3 of loose and classified material, which means that the waste dumps will be temporary.

Based on the chemical characteristics of host rock (volcanic andesite) and mineralization (quartz and oxides-low sulphidation), no Acid Rock Drainage (ARD) is expected when waste is exposed to a rainy environment after temporary storage on the surface.

The waste generated from driving underground development will not be enough to provide all the backfill required for the stopes. In addition to the waste, 90,000m^3 of material from surface quarries will be required.

These temporary waste dumps can be located in areas close to the mine entrances in branches created from the access road in the colluvium (surface andesite) areas. This will create longitudinal lifts, less than 4 meters high that will be removed during the mine closure.

25.1.10 Underground Water Seepage

Water discharge expected from the underground mine was estimated based on exploration drilling information in two main categories: quantity and quality.

The water quantity was based on the low hydraulic conductivity of the local andesite and no presence of water springs in the surrounding area. It was estimated that the rock mass will not add more than 2.50l/s, to the estimated 1.50l/s of fresh water from drilling, for a total of 4.0l/s.

Regarding water quality, all freshwater generated by the rock mass (2.50l/s) and drilling (1.50l/s) is expected to be neutral, with no metals in solution. Water from the mine is expected to be slightly contaminated with oil residues and nitrates.

Small crosscuts (15m to 20m) will be driven from each mine entrance to allow water to flow through ditches, be settled, and then collected by pipes to a common final treatment plant located on the surface. The treatment plant will be located below the lowest level (Level 4350), where a final treatment of oil residues and nitrates will be performed. The final waste water product will conform to Peruvian environmental regulations.

25.1.11 Mine Closure

Environmental liabilities at mine closure will be: underground excavations, underground water seepage, waste dumps, access roads and surface quarries.

Because the neutral, or non acid rock generating (ARG), characteristics of the low grade ore, host rock and quarry material, the closure strategy will only provide for physical stability and fines sedimentation in water seepage. This will be accomplished by revegetating the slopes/benches at the waste dump and quarry, and installing concrete bulkheads in all mine entrances.

With regards to chemical stability, acid rock drainage (ARD) is not expected from surface facilities. Ditches have been designed to prevent water from the surface facilities having any contact with waste dumps, roads, and quarries.

25.1.12 Mine Utilities

25.1.12.1 Compressed Air

Compressed air for drilling and other underground needs will be supplied by two, 1,350cfm electric compressors. The compressed air will be distributed throughout the mine in 4 inch pipes.

25.1.12.2 Pumping

Any water encountered is planned to drain by gravity through the ditches constructed in the access/haulageways to sediments ponds near each mine entrance, where sludge will be separated and disposed of. Therefore, pumps will be needed only on the 4300 Level.

25.1.12.3 Electric Power

Electric power will be required by the electric-hydraulic jumbos, captive scoops, mine compressors, interior mine lighting, and ventilation fans.

Electric power for the site will be supplied by diesel generator sets located on the surface. The total mine electric power demand is estimated at 1.3MW as summarized in Table 25-6.

Table 25-6: Mine Power Consumption

Equipment	Qty	kW	Use	Total (kW)
Jumbos	2	55	50%	55
0.7 yd^3 Scoops	4	85	40%	136
2.5 yd^3 Scoops	2	150	40%	120
Compressors	2	250	80%	400
Ventilation fans	5	100	100%	500
Mine Lighting	1	50	100%	50
Miscellaneous	1	50	100%	50
Total				**1,311**

25.1.12.4 Service Water

The water required for drilling is estimated at 1.5 liters per second. This quantity is readily available from water sources in the area.

Industrial water supply for the mine operations will be pumped from the process plant raw water tank to a mine raw water tank and then conducted by piping to a mine raw water tank located above the mine entrances.

The mine's raw water tank will have a capacity of $35m^3$, and will feed the mine through the service raise with 1 inch hoses.

Prior to the service raise being completed, these hoses will be temporarily located on the surface.

25.1.12.5 Ventilation

All ventilation is designed to be mechanical, using surface fans, drawing clean air into the mine through the primary access drifts located on Levels 4400, 4450, 4500 and 4550, and exhausting through interior raises developed within the mine to the surface. The ventilation design maintains airflow velocities in the main accesses to less than 350 meters per minute to avoid raising dust from the floors. The maximum ventilation quantity requirements, which are summarized in

Table 25-7, are estimated to be 4,476m^3/min (≈150,000 cfm) during the 4th quarter of preproduction.

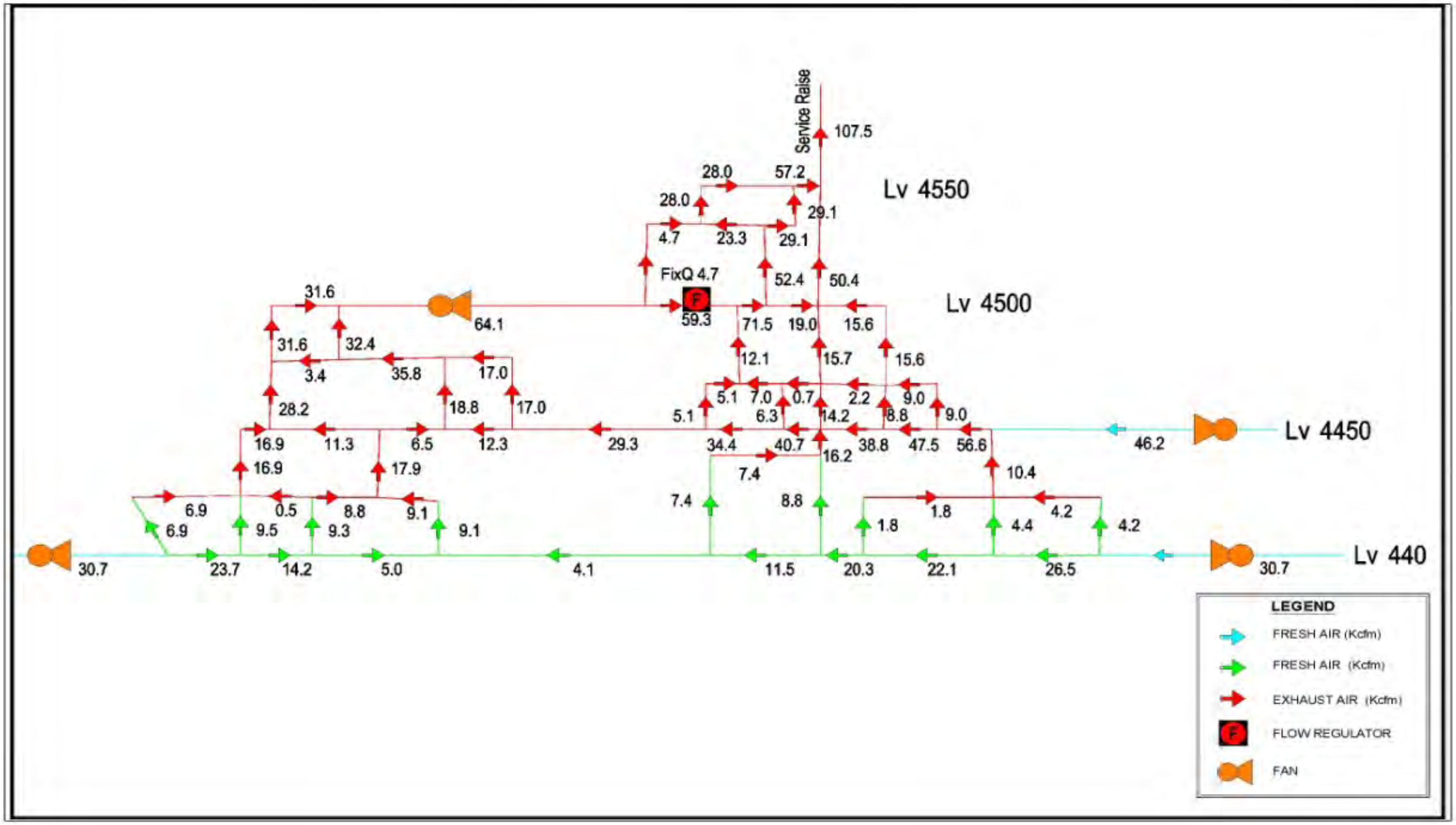
Table 25-7: Mine Ventilation Requirements

Fresh Air required	Source	Qty	Hp (ea)	Total (m^3/min)
3 m^3/min/hp	4.2 yd^3 diesel scoops	2	85	510
3 m^3/min/hp	20-tonne Trucks	4	300	3,600
6 m^3/min/person	Personnel	61	1	366
			Total	**4,476**

Before levels are connected with backfill and service raises, ventilation will be provided and regulated by small fans, vent-bags, and air doors. When the levels have been connected by raises, the fan previously installed at lower level will be moved to the top of the raise to become an extractor fan.

The operational flows within the mine ventilation system have been calculated to determine the flow conditions that will exist during preproduction development and steady state production. This calculation consisted of a network analysis taking into consideration the various mine ventilation parameters such as; personnel and equipment operating within the mine, friction factors in the airways, and acceptable air velocities to arrive at the required quantities of air in each airway, the air regulation required, and the various fan sizes. The results of this analysis, including the network and the calculated flows during full production in the 10th quarter of operation, are shown in Figure 25-11. The ventilation airflow required during this production period is calculated to be approximately 3,060 m^3/min (108,000 cfm).

Figure 25-11: Ventilation Circuit – Operating Stage



25.2 PROCESS FACILIITES

25.2.1 Introduction

This section presents a description of the processing plant planned for the San Luis project.

25.2.2 Site Selection and Alternatives Analysis

Two locations for the process plant (excluding ancillary facilities) were evaluated. Alternative 1 was located 250m southeast of the proposed 4,400 level portal (approximately E190900 N8960300). Alternative 2 was located 750m southeast of the same 4,400 level portal (approximately E191330 N8960100). MQes prepared civil models using parametric software to determine cut and fill requirements for both locations. Costs for civil works were then estimated. The cost for alternative 1 was estimated to be US$2.8 million and cost for alternative 2 was US$1 million. A comparison of the alternatives is shown in Table 25-8.

Table 25-8: Comparison of Process Plant Site Alternatives

Item	Alternative 1 (US$)	Alternative 2 (US$)
Mass Excavation, Drill & Blast	$2,450,000	$710,000
Mass Fill, Spread & Compact	$180,000	$60,000
Waste Haul & Dump	$30,000	$10,000
Hillficker Retaining Wall	$120,000	
Haulage		$220,000
TOTAL	**$2,780,000**	**$1,000,000**
Potential for Expansion	None	Limited
Haul Distance	100m	780m
Distance to Tailings Dam	500m	200m

Based on cost and physical parameters, alternative 2 was selected as the process plant location.

25.2.3 Process Description

Using the metallurgical test results to date, MQes has assembled a series of process flowsheets to treat material from the San Luis project via gravity concentration and cyanide leaching. In developing this information results from metallurgical testwork performed to date have been used and, where necessary, supplemented by assumptions and data from MQes' in-house database. Ongoing metallurgical testwork and planned programs will be used to confirm the bases for the facility design.

25.2.4 Process Overview

The process flow sheet includes two-stage crushing and screening, fine ore storage, grinding and classification, gravity concentration, cyanide leaching, thickening and CCD washing of leach discharge slurries. Leached tailings will be discharged to a TSF near the process plant where cyanide solution will be recycled to the process circuit. Pregnant leach solution will be clarified and deaerated prior to the injection of zinc dust to precipitate gold and silver. Precipitates will be collected in a filter press then dried and smelted to produce doré bars for final refining offsite. The crushing and screening circuits as well as smelting section will operate on day shift only. All other process facilities will operate 24 hours a day.

The following items summarize the processing operation to treat 400tpd of feed and extract gold and silver. The processing criteria consider:

- The mining department will schedule ore deliveries and blend ore types to prevent plant throughput impacts from items such as over size rock at the primary crusher, plugging due to high moisture, etc.
- A primary jaw crusher will reduce the run-of-mine (ROM) feed size to a P_{80} of 55mm.
- Secondary crushing will reduce the size of material to a nominal minus 80% passing 10mm ($^3/_8$").
- Fine crushed material will be screened and fed to a fine ore bin. Material will discharge the fine ore bin onto a belt feeder and then to a conveyor where lime is added before feeding into the ball mill.
- Material will discharge from the ball mill and be cycloned. Cyclone underflow will be fed to a gravity concentration circuit. Gravity concentrator tails will be recycled to the ball mill. Cyclone overflow will be fed to the leach circuit.
- Gravity concentrate will be upgraded via table concentrators, dried and fed to the smelting furnace.
- Gold and silver in the gravity concentrate tails will be leached using sodium cyanide solution. Leach discharge solution will be washed in a CCD circuit to produce a pregnant leach solution. The washed solids material will discharge to a TSF.
- Gold and silver will be recovered from the pregnant leach solution in a Merrill-Crowe plant.
- Zinc precipitate will be dried, mixed with fluxes and smelted to produce silver-gold doré bars, the final product from the processing facility.

The processing circuit will operate as a closed circuit, with barren solution recycled. Plant streams will include pregnant solution, barren solution, process water, fresh water and potable water.

Reagents used in the process include sodium cyanide, flocculant, pebble lime, diatomaceous earth, zinc dust and antiscalant. Storing, preparing and distributing these reagents are included as a part of the process plant.

Process control and monitoring is planned to be accomplished through a system of Programmable Logic Controllers (PLC) interfaced with workstations running supervisor human machine interface (HMI) operator software.

25.2.5 Crushing, Screening, Fine Ore Storage and Grinding

ROM ore will be transported from underground in haul trucks and discharged onto the ROM stockpile pad, adjacent to the crusher. The ROM pad will store up to a nominal 4,000 tonnes of ore (ten days production) which will be reclaimed by a front end loader and fed to the crushing plant.

The primary dump hopper will be fitted with an inclined stationary grizzly to remove oversize. Ore will be reclaimed from the hopper with a vibrating grizzly feeder which will remove undersize material ahead of the primary crusher. Oversize will discharge directly into the crusher. Crushed ore and vibrating grizzly undersize will be conveyed to a vibrating screen. A magnet will remove tramp metal from the conveyor.

Screen oversize will be discharged to a secondary cone crusher with secondary crushed ore being conveyed back to the vibrating screen. Screen undersize will be conveyed to the fine ore bin ahead of grinding.

Ball mill feed will be reclaimed from the fine ore bin by a variable speed belt feeder which discharges onto a conveyor equipped with a belt scale to control and record tonnage. The fixed speed ball mill will operate in closed circuit with hydrocyclones. A trommel screen will remove over size from the mill discharge slurry.

Lime will be added to the feed conveyor prior to the ball mill, for pH control and sodium cyanide solution will be pumped continuously to the ball mill feed chute. Grinding media will be added once per day. The grinding circuit will be designed for a recirculating load of 200%. Mill discharge slurry will be pumped to hydrocyclones. Cyclone feed density will be adjusted by adding barren solution to the mill discharge sump. Cyclone underflow reports to the gravity concentrator and the gravity tails return to the grinding mill. Cyclone overflow will pass through a single deck, high frequency, trash screen and then report to the leach circuit. Trash from the screen will discharge to a tote bin.

The grinding bay will be provided with a 2 tonne capacity, pendant controlled hoist for grinding media handling.

The grinding area will be curbed so that spillage may be collected and pumped to the mill discharge sump.

25.2.6 Gravity Concentration and Leaching

Gravity concentrate will feed into a holding tank. From there, the material will pass over concentrating tables to remove gangue material. The resulting high grade concentrate will be collected in a container, dried and mixed with smelting furnace feed. Table concentrator tails will be pumped to the ball mill feed.

Cyclone overflow discharges to the first leach tank. Three mechanically agitated leach tanks will operate in series to provide a total residence time of 72 hours. Leach solutions will be separated from leached solid residues with three CCD washing thickeners.

Compressed air, necessary to support the chemical dissolution of gold and silver will be injected into the slurry below the leach tank agitators. Three blowers (two operating and one standby), operating in parallel, will provide air to the leach tanks. The first leach tank will overflow to the second and the second will overflow to the third. Leach tank feed and discharge launders will be arranged such that any tank can be bypassed for maintenance without interrupting process operations.

Cyanide strength will be adjusted as required throughout the leach circuit by the addition of sodium cyanide solution.

The leaching area will be curbed and spillage can be returned to the circuit.

25.2.7 Counter-Current Decantation

CCD washing will be achieved in three high rate thickeners. The third leach tank discharges to CCD 1 washing thickener. In the CCD circuit the thickener underflow, at 65% solids, will be pumped from CCD 1 to CCD 2 and then from CCD2 to CCD 3. Thickener overflow will flow counter current from CCD 3 to CCD 2 and then to CCD1. A distributor will add barren solution to the feed of CCD 3. Overflow from CCD 1 discharges to the pregnant solution tank. Underflow from CCD 3 is pumped to the tails sump from which it will be pumped to the TSF.

Thickener feed and discharge piping will be arranged such that any thickener can be bypassed for maintenance without interrupting process operations.

The CCD area will be curbed and spillage can be returned to the circuit.

25.2.8 Merrill-Crowe Refinery

Gold and silver is recovered from CCD circuit pregnant solution by zinc precipitation of metal ions using zinc dust and then smelting the precipitate to produce doré. This will be accomplished by use of a conventional Merrill-Crowe process. The recovery of silver and gold by this process includes:

- Clarification of pregnant solution to remove suspended solids.
- De-aeration of pregnant solution to reduce dissolved oxygen.
- Precipitation of gold and silver metal by addition of zinc dust.
- Filtering and drying of precipitate.
- Smelting the precious metal precipitate in a furnace to produce doré bars.

The precious metal recovery circuit is designed to process approximately 79,200 ounces of gold and 1,885,000 ounces of silver annually.

Pregnant solution from the CCD circuit discharges to a pregnant solution tank located in the CCD area. Pregnant solution from the tank is pumped to two (one operating and one standby) self-cleaning pressure leaf filters. The filters are leaf type with a wash sequence that is activated as needed based on differential pressure. Filters are pre-coated using diatomaceous earth as a filter aid. They will also have a continuous body feed of diatomaceous earth to assist filtering. Pressure for filter operations is provided by the pregnant solution pumps. Filtrate (clarified solution) discharges directly to the deaeration tower.

Clarified solution is passed through the deaeration tower to reduce dissolved oxygen to less than 0.5ppm prior to zinc dust addition. The deaeration tower is connected through a barometric seal to a vacuum pump.

The clarified, deaerated, pregnant solution is withdrawn from the bottom of the deaeration tower by a single-stage, vertical, in-line, centrifugal pump, submerged in water to prevent re-entry of air through the pump gland. The pump discharges to the precipitation filter presses. An emulsion of zinc dust and pregnant solution is added to the zinc cone at the pump suction to precipitate the silver and gold.

Zinc dust is hand loaded into a zinc feeder hopper from as delivered 45kg capacity pails, and discharges via a rotary feeder into a mixing cone which emulsifies the zinc dust with pregnant solution. The mixing cone is continuously supplied with pregnant solution to prevent air from entering the suction of the precipitation feed pump. The slurry is then pumped to two (one operating and one standby) plate-and-frame filter presses in parallel, where the precipitated precious metals are collected. The plate and frame presses are pre-coated with diatomaceous earth similarly to the clarifying filters, and are manually opened and cleaned with precipitate being collected in carts.

Barren solution (filtrate) exiting the Merrill-Crowe circuit returns to the barren solution tank from where it is distributed to the CCD washing circuit with any excess returned to the ball mill circuit.

25.2.9 Refinery

Zinc precipitate is placed in carts and dried in ovens along with gravity concentrate. The carts containing dry material are removed from the ovens by forklift and tipped into the furnace feed hopper. A smelting flux mixture is then added.

The smelting furnace is propane fired. The charge is melted and the majority of slag is decanted into conical molds. The decant slag is cooled and checked for any contained metal prior to short term storage and disposal. Following the final decant step, doré bullion comprised of silver and gold, is poured and sinks to the bottom of semi-cylindrical bar molds for subsequent handling. Small quantities of bar slag, containing fused fluxes, impurities, and minor amounts of doré splash and prills are recycled to the next smelting charge.

Bars are cleaned at a bar cleaning station using a sand blasting technique and a needle gun, weighed and stamped with an ID number and weight. Doré bars weighing approximately 110 kilograms are the final product of the operation and stored in a vault until shipment to an off site facility for further refining.

Fumes from the melting furnace are collected through ductwork and cleaned in a dust collector before discharging to atmosphere.

25.3 TAILINGS STORAGE FACILITY

25.3.1 Introduction

The facilities for tailing storage were designed according to the following objectives.

- Store the tailing in an environmentally responsible manner and in accordance with internationally accepted practices.
- Satisfy internationally accepted stability criteria for embankment construction in areas of high seismicity.
- Utilize proven and feasible engineering and technology for the design, construction, operation, and closure; in order to minimize any discharges from the facility to the environment.
- Satisfy the relevant Peruvian regulatory requirements associated with construction of the facility.

From 2007 Canadian Dam Safety Guidelines, the San Luis TSF embankment would be in the "High" to "Very High" dam safety classification, based on the potential impact the failure of the embankment could have on downstream population and the environment. The project design criteria were selected based on this classification. Design criteria included containment of affected runoff in the TSF for storms up to the probable maximum precipitation and acceptable performance under seismic conditions up to the maximum credible earthquake.

25.3.2 Facility Description

From 2007 Canadian Dam Safety Guidelines, the San Luis TSF embankment would be in the "High" to "Very High" dam safety classification, based on the potential impact the failure of the embankment could have on downstream population and the environment. The project design criteria were selected based on this classification. Design criteria included containment of affected runoff in the TSF for storms up to the probable maximum precipitation and acceptable performance under seismic conditions up to the maximum credible earthquake.

A site selection study was conducted for tailing storage facilities and water storage structures in the immediate site area as part of conceptual design and feasibility design work. Sites for slurried and filtered tailing disposal were identified and evaluated. The tailing storage facility adjacent to the processing plant (TSF-0) was selected as the primary tailing storage site, with TSF-1 as an alternative. The TSF was sized to provide storage for approximately 500,000 metric tons of tailing as well as containment of process water and extreme storm precipitation. The preferred deposition method for a conventional thickened tailing product will be spigotting from the embankment perimeter, with the objective to maintain the reclaim water pond in the southeast corner of the impoundment.

The TSF-0 impoundment will be created by a cross-valley embankment. The preferred design is a zoned rockfill embankment, to be constructed using borrow materials from within the TSF impoundment and from adjacent borrow areas. The impoundment will contain a liner system including a double geomembrane with a leak detection system between geomembranes. This liner system will cover the impoundment basin and upstream slope of the embankment. The intent of the liner system is to minimize seepage to groundwater. A blanket drain will be constructed under the impoundment to control groundwater flow from below the TSF.

Due to the relatively short project life and small TSF size, phased construction of the TSF was not considered. The TSF would be constructed in one phase during mine facilities construction, and would be completed just prior to the time the processing plant is commissioned.

Stability analyses of the TSF embankment were conducted according to accepted methods and compared with standard stability criteria. The results indicate acceptable stability under static conditions, with calculated factors of safety exceeding the minimum required factors of safety for the embankment configuration with an upstream slope of 3H:1V and downstream slope of 2H:1V. Seismic stability was evaluated with methods typically used for highly seismic areas (deformation analyses). Simplified deformation analyses indicate that displacements of the embankment from anticipated seismic events are acceptable (estimated to be less than 30cm). The available freeboard after this assumed deformation is considered adequate to preclude release of tailing due to a seismic event.

25.3.3 Materials Description

The overall strength values for the rock mass at the TSF embankment location are expected to represent favourable foundation conditions for the proposed design in terms of stability (for failure and sliding) and bearing capacity for the rockfill embankment. The site exploration logs and geotechnical testing results for the foundation conditions and borrow materials is presented in the Geotechnical and Hydrogeologic Investigation & Engineering Report.

Representative samples of tailing and mined materials were evaluated for standard geochemical characteristics related to performance when exposed to the environment. The results of static ABA tests on tailing and waste rock samples indicated no potential to generate acid rock drainage. From the kinetic HCT of the waste rock, it is reasonable to assume that it is unlikely that these samples will ever consume all of the available alkalinity and produce acidic leachate. Preliminary results for the tailing samples show similar results for the waste rock, with neutral to alkaline leachate with very low metals concentration. The results of the SPLP tests indicated that the maximum allowable concentrations are exceeded in 4 of the 5 tailings samples for cyanide, all of the tailings samples and one of the waste rock for antimony, all of the tailings samples for arsenic, one tailings sample for cadmium, and four tailings samples for lead. The aesthetic objective for aluminum was exceeded for all samples. Whole rock chemical analyses results indicate that all of the samples are elevated in most metals and metalloids (e.g., antimony, arsenic, copper, lead, molybdenum, selenium, and silver) relative to ranges normally found in average crustal rocks.

Geotechnical testing was conducted on representative samples of tailing for standard geotechnical properties and consolidation characteristics. The consolidation testing and modeling specific to tailings were used to estimate the average density of tailing in the TSF.

25.3.4 Water Management

Climate data used for design was obtained from available meteorological stations within a 50-km radius of the project site. Stations closest to the project site with the longest period of record comprised the preferred data, with extreme precipitation events calculated using standard hydrological techniques.

Surface water diversion channels will be constructed around the TSF. These channels will collect and divert stormwater, thus limiting the amount of runoff that will be collected in the impoundment. During its operational phase, the TSF embankment will not have a spillway. In order to avoid discharge of potentially contaminated water from the impoundment, the TSF is designed to contain runoff of the PMP from the entire catchment area above the TSF (conservatively excluding any reduction from use of the surface water diversion channels).

The water balance analysis of the TSF was performed for three scenarios (wettest, driest and average years on record). The results of the water balance indicate that the reclaim water pond in the TSF will grow with time under conditions of the wettest years on the record. To maintain adequate freeboard during the later months of operation, measures to reduce the reclaim water pond volume may have to be taken early in the operational phase. Assuming that construction of the TSF will be completed prior to the start of the wet season, an adequate amount of water for the start of the operations will be accumulated in the impoundment. The start-up water pond should not exceed 40,000m^3.

25.3.5 Closure

A conceptual closure plan was developed for the TSF, based on Peruvian mine closure law and accepted international guidelines. The plan includes removal of residual process water by evaporation and covering and re-vegetating the TSF impoundment surface.

25.4 INFRASTRUCTURE

The required infrastructure to support San Luis Project operations includes mine operations facilities, process plant, operations camp, on site access roads, raw, fire and potable water supply and electrical power generation and distribution.

The main infrastructure considered in this section is:

- On site access roads to the mine, process plant, tailings facility and water dam.
- Raw, fire and potable water supply.
- Electrical power generation and distribution.
- Buildings and offices for administration, general site services, metallurgical laboratory, warehouse and maintenance facilities.
- Ancillary facilities such as fuel storage, fire protection systems and communications.
- Waste handling and sewage treatment systems.

A layout of the infrastructure and support facilities is shown in the site plan, Figure 6-2.

25.4.1 Access Roads

25.4.1.1 Off Site Access Roads

BISA performed a trade off study to evaluate the technical and economical aspects of the three existing access roads to the San Luis Project. The three routes evaluated were:

- Route 1: Casma – San Luis.
- Route 2: Carhuaz – San Luis.
- Route 3: Huaraz – San Luis.

Of the three existing routes, Route 3 Huaraz to San Luis was selected for the following reasons:

- Routes 1 and 2 have areas of numerous sharp curves, narrow road widths and steep grades.
- Route 1 passes through several communities.
- Route 3 is in better overall condition, would require less upgrading.
- Route 3 leads to Huaraz which is the largest of the three towns and has existing infrastructure to support mining operations.

Another trade off study was performed to evaluate the surfacing possibilities. This study evaluated the capital, operating and maintenance costs of the roadway for life of the mine. Alternatives considered were:

- Alternative A: Asphalt layer 1'' thick.
- Alternative B: Slurry seal treatment.
- Alternative C: Stabilizing the existing dirt road with cement.
- Alternative D: Improvements to the existing dirt road.

Based on this study, it was concluded that road surfacing would not be necessary considering the short life of the project.

25.4.1.2 On Site Roads

The on site access roads begin at the main entrance to the project site which is an extension of the public road near the community of Cochabamba. The on-site roads are designed to provide access to all project facilities.

Existing on site roads will be upgraded to meet the project requirements for safety and efficiency. The roads have been designed with a 5.4m wide roadway surface for the first 6km from the west entrance, and a 4.5m wide roadway for the remaining portion of the access. The areas of 4.5m road width will have a safety turnout every 500m. All roads will be surfaced with a 0. 20m thick gravel layer.

Approximate lengths of the various on site access roads are summarized in Table 25-9.

Table 25-9: On Site Roads - Distances

Road	Length (m)
C1: Main Entrance Control Gate	6,280
C2: Access C1 to Operations Camp	84
C3: Access C1 to Administration Check Point	1,020
C6: Access C1 to Mine Portal at 4350	87
C7: Access C1 to Main Mine Portal	250
C8: Access C1 to Mine Portal at 4450	38
C9: Access C1 to Explosive Magazine	165
C10: Access C1 to Water Tank	12
C11: Water Tank to East Entrance	4,094
C13: Access from main mine portal to ROM Stockpile.	830
Total Length	**12,860**

C1: Main Access Road - The main facility roadway designated road C1 originates at a turn off from the public road at coordinates N 8,958,550, E 189,650. This roadway will be upgraded to 5.4m wide with grades targeted near 11%. Eight other minor access roads originate from this roadway.

C2: Operations Camp Access Road - This is an existing 80m long road that begins approximately 1,580m from the start of road C1 at an elevation of approximately 4,150 masl. It provides access to the camp and requires only minor improvements.

C3: Process Plant Access Road - Road C3 will provide access to the process plant security gate near the administrative offices and process plant. This new road will be 1,020m long, 5.4m wide with grades up to 10%. Road C3 branch begins approximately 3,285m from the start of C1 at an elevation of 4,290masl.

C6: 4,350 Portal Access Road - C6 is a new 80m long by 5.4m wide road with grades to 9%. It will provide access to the 4,350 portal.

C7: 4,400 Portal Access Road - C7 is a new 240m long road that begins approximately 4,920m from the start point of C1 and leads to the main mine portal and the maintenance shop area at 4,400masl.

C8: 4,450 Level Access Road - C8 is an existing 80m long road that originates approximately 4,585m from the start point of C1 and provides access to point C9, 4,450 portal access road.

C9: 4,450 Portal Access Road - C9 is a new 150m long road which begins at road C8 and provides access to the 4,450 portal.

C10: Mine Water Tank Access Road - C10 is an existing 20m long road that starts at 6,040m from the start point of C1 and leads to the mine water tank.

C11: East Entrance On-Site Access Road - C11 is an existing 4,510m long road that starts near the C10 turn off and leads to the East entrance of the site. Improvements to this section of the road include target grades near 10% and smoothing of sharp bends.

C13: Ore Haul Road - C13 is a new road that begins at the main mine portal located at 4,400masl and extends 830m to the ROM stockpile area. This road is the primary haul road used to transport ore from the mine.

25.4.2 Water Supply and Water Balance

This section presents the project fresh water requirements, the design criteria, analysis of alternatives, site selection, and feasibility level design of the Water Supply Structure (WSS) for fresh water supply, and a summary of the overall water balance for the San Luis project.

25.4.2.1 Fresh Water Requirements

A reliable fresh water supply is required for the project's operations. Based on information provided by S. Milne and BISA, the estimated average fresh water requirement for the mine operation (dust suppression and drilling) is approximately 2l/s. The potable water requirement for the plant site is estimated to be 0.52l/s, and for the camp site is 0.26l/s. The estimated fresh water requirements for the mine and mill operations, as well as for potable water for the mine camp and offices are summarized in Table 25-10 below.

Table 25-10: Fresh Water Requirements

	Fresh Water	
Item	**l/s**	**Source**
Mine	2	Huanchuy
Process Plant	1.58 – 2.0	Huanchuy
Potable (plant)	0.52	Huanchuy
Potable (camp)	0.26	Spring

As can be seen in Table 25-9, the average fresh water demand for the project's operations from the Quebrada Huanchuy is 4.52l/s. The fresh water requirement for the camp site from a local spring is 0.26l/s. Information on the selection of the water supply sources is presented in the sections below.

25.4.2.2 Water Supply Alternatives Analysis

A number of water sources (surface and groundwater) as well as alternative locations for a water supply facility were considered for this study. Seven alternative water supply facility sites (WSF) were identified, located entirely within the Cochabamba boundary in Quebrada Huanchuy or its major drainages. A process of elimination was adopted for the alternatives analysis. The proposed sites were ranked and eliminated based on a range of criteria including technical, environmental, social and economic factors. The technical considerations included capacity, embankment/impoundment ratio, geotechnical conditions, size of the catchment, water availability and distance from the plant site. The environmental and social factors included availability of sufficient water to maintain the local community's water supply and the ecological flows. MQes considered capital and operating costs to deliver the water to the project operations.

Hydrologic and geotechnical analyses and investigations were completed for the proposed sites to different levels of detail. Field investigations consisting of drilling and test pit excavation were performed at some of the sites.

As a result of the evaluation, WSF Site 1 (WSF-1) was selected as the preferred location for a water supply facility. Two options were considered for development at the WSF-1 site; a water storage dam and a water supply structure (WSS). The construction of a water supply structure was considered more suitable for the project and was advanced to a feasibility level design, while the construction of a dam was left as an alternative and described at a conceptual level.

25.4.3 Water Supply Structure

25.4.3.1 Site Conditions

The WSF-1 site is located in a relatively tight and narrow portion of the Huanchuy valley downstream of the main areas of alpine glaciations at an approximate elevation of 3950m. The drainage area is primarily used for pasture land. It has a catchment area of approximately 16.1km^2. It is covered with exposed fractured bedrock and gravel sized colluvial soil at the base of the slopes. Granodiorite and andesite bedrock were encountered during drilling at the WSF-1 site. Some of the bedrock was highly decomposed to completely weathered to a depth of about 10m. This deep weathering does not appear to extend across the valley as the bedrock immediately below the overburden cover appears to be slightly weathered to fresh and highly competent.

The San Luis property is drained by three major valleys called Quebrada Huanchuy to the south, Quebrada Tocas to the north, and Quebrada Quellaycancha to the west. Numerous small lakes and ponds contain water throughout most of the year but the smaller ones do dry up during the dry season. The main drainages contain year-round water flow but the flow rate is dramatically reduced near the end of the dry season in November and December. Numerous springs and seepage points were observed during the February field investigations above the valley bottoms along fractures in the bedrock and in the overburden soils.

Streamflow data was collected on a monthly basis in several locations in Huanchuy valley and in other valleys in the general location of the project from September 2008 through January 2010. The available data was evaluated to assess the availability of sufficient flow at the WSS location. It was established that the minimum recorded flow in Huanchuy stream in the general location of the WSS was approximately 10l/s. Due to the limited available data and inherent variations in the climatic conditions, it is difficult to guarantee the potential minimum flow that may occur in the creek during the mine operations, because the historical data has been collected for a relatively short time and natural variation is possible. Given the available historical data, as the best available guide at the time of this writing it is considered reasonable to expect that sufficient flow to satisfy the mine fresh water requirements would be available. The currently permitted diversion rate for the project is 3l/s for exploration purposes. An application to extend this permit and increase the diversion rate to 5l/s is currently being prepared.

25.4.3.2 Construction Materials

Construction materials at the selected WSF-1 site are less abundant than in the TSF site because glacial till has not accumulated in this location. The main valley is covered with thin veneer of colluvium along the side slopes and coarse alluvial materials in the valley bottom. There are very extensive talus deposits on the valley side slopes composed of angular, 3 to 8cm diameter andesite rock fragments that could be processed and crushed to produce concrete or hard shell aggregate, drain, or filter materials.

25.4.3.3 WSS Analyses and Design

The planned WSS on Quebrada Huanchuy would consist of a reinforced concrete weir wall, with upstream and downstream apron slabs, and reinforced concrete side walls. The maximum height of the overflow weir wall would be 2m. Stability of the weir wall was analyzed for overturning and sliding stability assuming water at the top of the weir wall on the upstream side, and no downstream tailwater. The material properties and results of the analysis are summarized in Tables 25-11 and 25-12**,** respectively.

Table 25-11: WSS Strength Parameters

Feature	Density (kN/m^3)	Friction Angle
Concrete	24	-
Bedrock	-	50
Water	9.81	-

Table 25-12: WSS Stability Analysis Results

Case	Minimum Factor of Safety	Calculated Factor of Safety
Overturning	1.5	1.6
Sliding	1.5	2.6

The obtained factors of safety presented in Table 25-11 satisfy the design criteria for such structures. With a low height weir, headwater and tailwater levels will not be significantly different at higher flows. Therefore, additional analysis was not deemed necessary at this stage of project development.

The WSS would consist of a 2m high concrete weir constructed across the channel. Concrete aprons would be constructed 2m upstream (approach apron) and 5m downstream (discharge apron) of the weir. Under-seepage cutoff walls would be provided at the beginning of the approach apron and at the downstream side of the discharge aprons to control uplift pressures for stability and for erosion control. A plan, profile and section of the WSS are shown on Figures 25-13 and 25-14.

Figure 25-12: Water Supply Structure Plan View

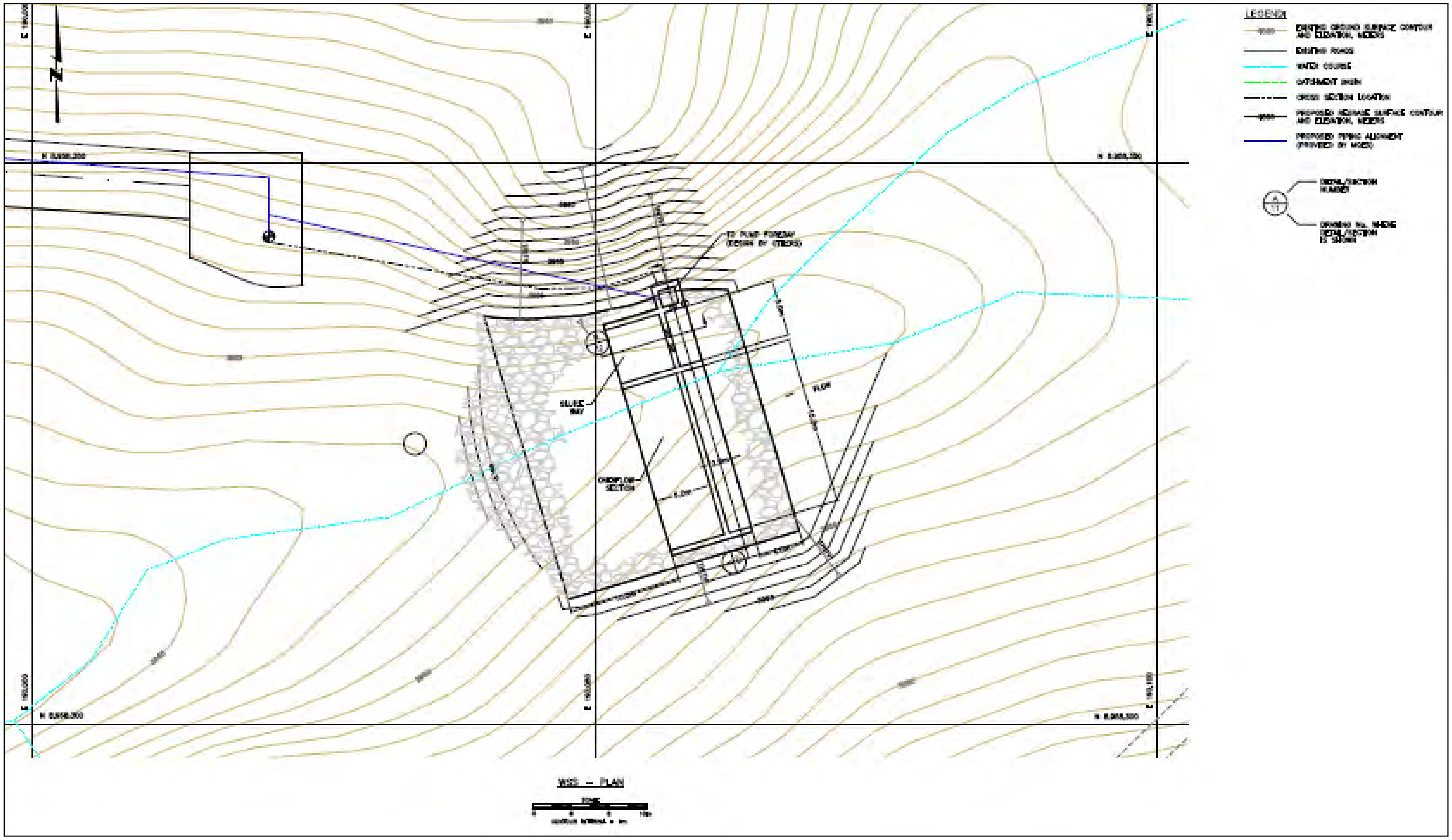
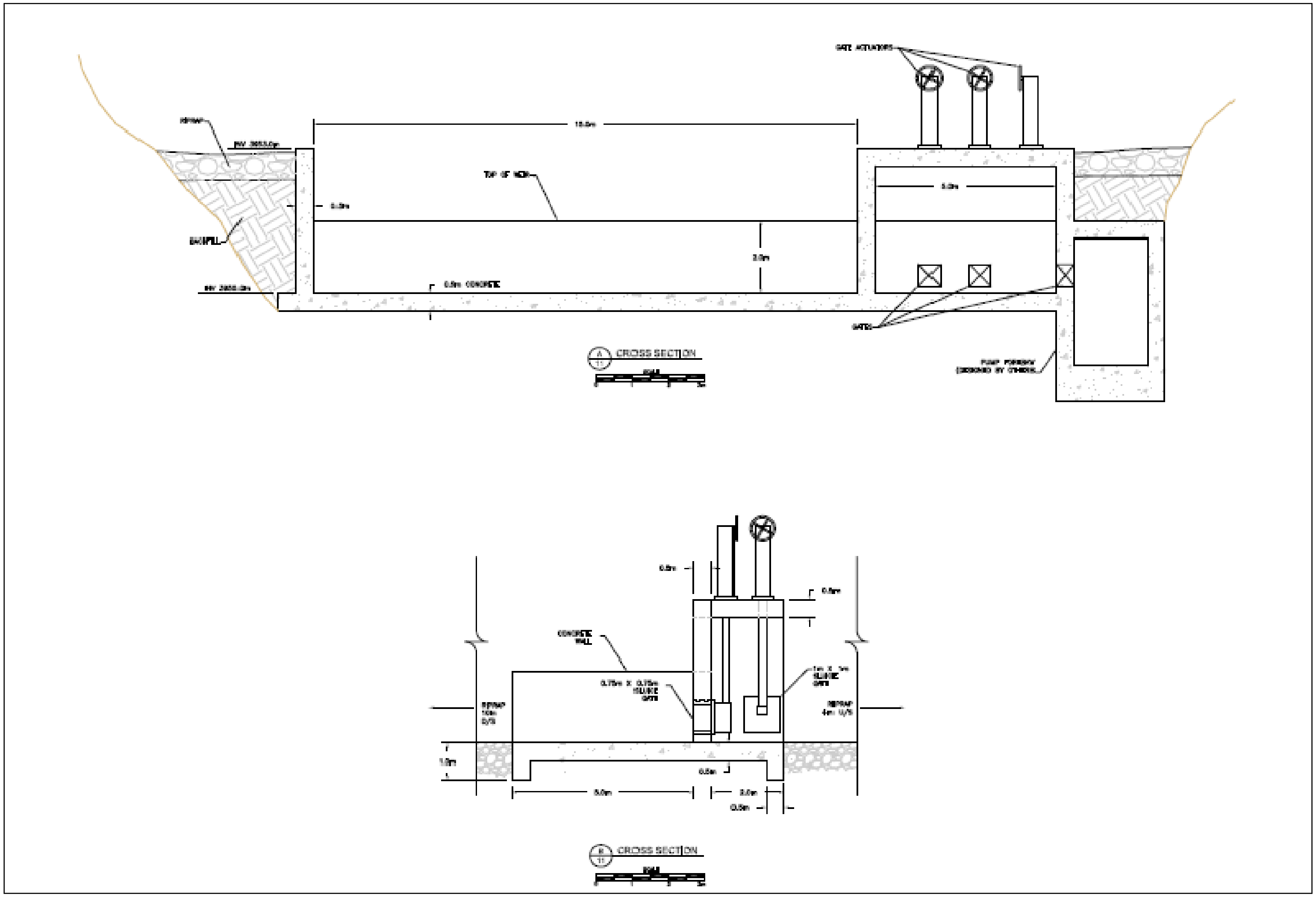


Figure 25-13: Supply Structure Cross-Sections



The WSS is designed to impound water about 2m deep with higher flows overtopping the structure. The abutments and downstream area would be armored with grouted riprap to minimize erosion and maintenance. The adequacy of grouted riprap for this use will need to be evaluated in further studies. Fresh water would be supplied from a pump bay or pipeline at the WSS for storage in a nearby tank for project use.

25.4.3.4 Water Balance Analysis

Underground Mine: Water is required for drilling and dust suppression. Water is expected to be produced in the mine workings due to groundwater inflow (see Section 8.5.5). It is assumed that no water losses will be incurred due to evaporation or wetting. All water used in the mine or produced by the mine will report to a water treatment plant and will be discharged to the environment.

Process Plant and TSF: Fresh water is required for the mill gland and mill reagent. Process water from the TSF pond will be reused in the plant processes. Process water will be deposited to the TSF with the thickened tailing slurry. A significant portion of this water (over 80%) will be trapped in the voids of the deposited tailing and slowly released through the process of consolidation. Water will be added to the impoundment through runoff from precipitation and lost to evaporation. The net water gain (precipitation – evaporation) is expected to vary between 1.3l/s (wet years) to almost zero for dry years.

Plant Area Potable: An allowance of 0.52l/s has been developed for plant and mine dry area potable water requirements.

Camp Site: An allowance of 0.26l/s has been developed for plant and mine dry area potable water requirements.

A summary of the water balance results showing the main inflow and outflow streams is presented in Table 25-13 below.

Table 25-13: Water Balance Results

Item	Inflow, l/s	Outflow, l/s
Mine		
Raw water from Huanchuy	2	
Groundwater inflow	3	
Discharge to environment		5
Process		
Raw water from Huanchuy	1.58 – 2.0	
Ore moisture content	0.26	
Tailing voids loss (average)		2.1
Plant Area Potable		
Raw water from Huanchuy	0.52	
Discharge to environment		0.52
Camp Area		
Spring water used	0.26	
Spring water overflow	0.74	0.74
Discharge to environment		0.26

25.4.4 Water Distribution

Storage and distribution of water is required for raw/service water, fire water, and potable domestic use and will be distributed to the mine, process and administration areas from the WSS.

The mine area will require service water for drilling, dust control, and service water for the maintenance shop. The process and administrative areas require raw water for operations as well as fire and potable water. These water services will be distributed to the various users by means of overland pipelines.

25.4.4.1 Plant Raw Water Storage Tank

The plant raw water storage tank will be located as shown on the site plan drawing. The tank will have 218m^3 of total volume. Of this volume, 114m^3 will be dedicated to plant area fire water and 104m^3 to process water. The fire water supply for the plant area is delivered via gravity in separate firewater main and other piping from the tank is arranged such that fire water is always available.

25.4.4.2 Process Water

A 3.6m diameter x 2.5m high polyethelene tank will be located within the process plant area and will be supplied raw water by gravity from the plant raw tank and reclaim waters pumped from a barge located at the TSF. Distibution of process water is accomplished by means of dedicated process water pumps (1 operating/1 standby).

25.4.4.3 Mine Water Storage Tank

In order to provide the water requirements of the mine and maintenance shop a storage tank of $31.5m^3$ capacity will be located near the mine as shown on the site plan. It has been sized to provide mine water for 6 hours. This tank will provide service water to the portals, maintenance shop, and fire water to the maintenance shop area. The fire water supply for the maintenance shop area is delivered via gravity in separate firewater main, and other piping from the tank is arranged such that fire water is always available.

Two 50hp pumps (1 operating/1 standby) will be installed at the plant raw water tank to provide $23m^3/h$ to the mine raw water tank. The water will be transported via a 3" diameter overland pipeline. Distribution to the mine and maintenance shop will also be via 3" diameter pipeline.

25.4.4.4 Fire Water Storage

The fire water system has been designed to NFPA 30 and DS-42F criteria. The lower $114m^3$ of the main raw water storage tank will be reserved for fire protection. The upper $104m^3$ will be isolated by an elevated stand pipe to provide the other water requirements.

Fire water requirements for the process plant area will be provided by a 6" HDPE SDR 9 pipeline.

25.4.4.5 Potable Water

Potable water will be required at the operations camp, process plant, administrative offices, mine dry and maintenance shop.

Based on the results of water quality analysis, it was determined that the water requires only conventional treatment. Fresh water will be filtered to remove solids and activated carbon will remove organic materials. The water will be treated using hypochlorination.

Two compact potable water treatment plants will be installed, one in each of the following areas:

- Process plant area and administrative offices.
- Operations camp.

Potable water for the Mine Dry building will be supplied from the potable water treatment plant located near the plant raw water tank and it will be pumped to a potable water tank located above the Mine dry building.

Administration and operations camp areas will be fed potable water via gravity.

25.4.4.6 Operational Conditions and Parameters

The potable water plants will operate 365 days per year, 24 hours per day during the life of mine.

The generated potable water will comply with Peruvian water quality standard (DS 002-2008-MINAM).

The water needed for backwashing of the sand filters will be supplied by the feed tanks.

The potable water treatment plant for the administration area, mine dry and maintenance shop area has a design capacity for 120 people per day allowing for 120 liters per person per day.

The operations camp will accommodate an estimated 80 people. This potable water treatment system has been designed for 60 people with a per person allowance of 120 liters per day.

Table 25-14 shows the design criteria for the two potable water treatment plants.

Table 25-14: Potable Water Treatment Plants Design Criteria

Item	Admin Area/Mine Dry	Camp Area
Potable water volume (Avg) m^3/d	14.4	7.2
Population (inhabitants)	120	60
Operating plants (unit)	1	1
Stand by equipment	Pumps only	Pumps only
Operational availability	24 h/d – 365 d/a	24 h/d – 365 d/a
Operation environment	Open site	Open site

25.4.4.7 Potable Water Design Criteria

The potable water treatment plant packages will include:

- Fresh water pumps for automatic operation based on the levels of the feed and potable water tanks.
- Filtration.
- Hypochlorination.

25.4.4.8 Potable Water Distribution System

The potable water distribution system design will comply with specification OS.050 of Reglamento Nacional de Edificaciones (R.N.E.)

25.4.4.9 Materials of Construction

The materials of construction for the main potable water pipe lines will be HDPE ASTM-F714 with secondary distribution lines constructed of Polyvinyl Chloride (PVC) standard NTP-ISO 4422.

The storage tanks for freshwater and potable water will be fabricated from High Density Polyethylene (HDPE), with capacities of $5m^3$ and $10m^3$ respectively.

Gravity feed will provide the necessary end use pressures within the system.

25.4.5 Power Supply

Discussions with the regional power utility, Distriluz, indicated that the current grid capacity is fully utilized and surplus capacity is not available for the San Luis project. Further, although Distriluz is investigating potential capacity expansion projects, Distriluz concluded that they (or their local subsidiary, Hidrandina) could not assure provision of suitable, reliable power for the project. Accordingly, on-site diesel generation of power has been planned and allowed for in the project facilities.

25.4.5.1 General

The total electrical power demand required for the operation is approximately 3.4MW.

The power plant for the Project will consist of three 2,250kW diesel powered generators (2 Operating/1 Standby) with an installed capacity of 6,750kW at the site elevation of 4,400masl.

In order to assure the proper sizing of the electrical equipment, load flow and short circuit voltage studies were prepared by BISA. The load flow study concludes that the system is electrically stable and that a motor starting sequence should be developed in order to avoid high frequency variations. The short circuit voltage study reveals that the electrical current levels are below standard protection levels and therefore special measures are not required.

Step up transformers will increase the voltage from 4.16kV to 13.8kV for overhead powerline distribution. The substations will be located near the process plant to reduce cable distances and line losses.

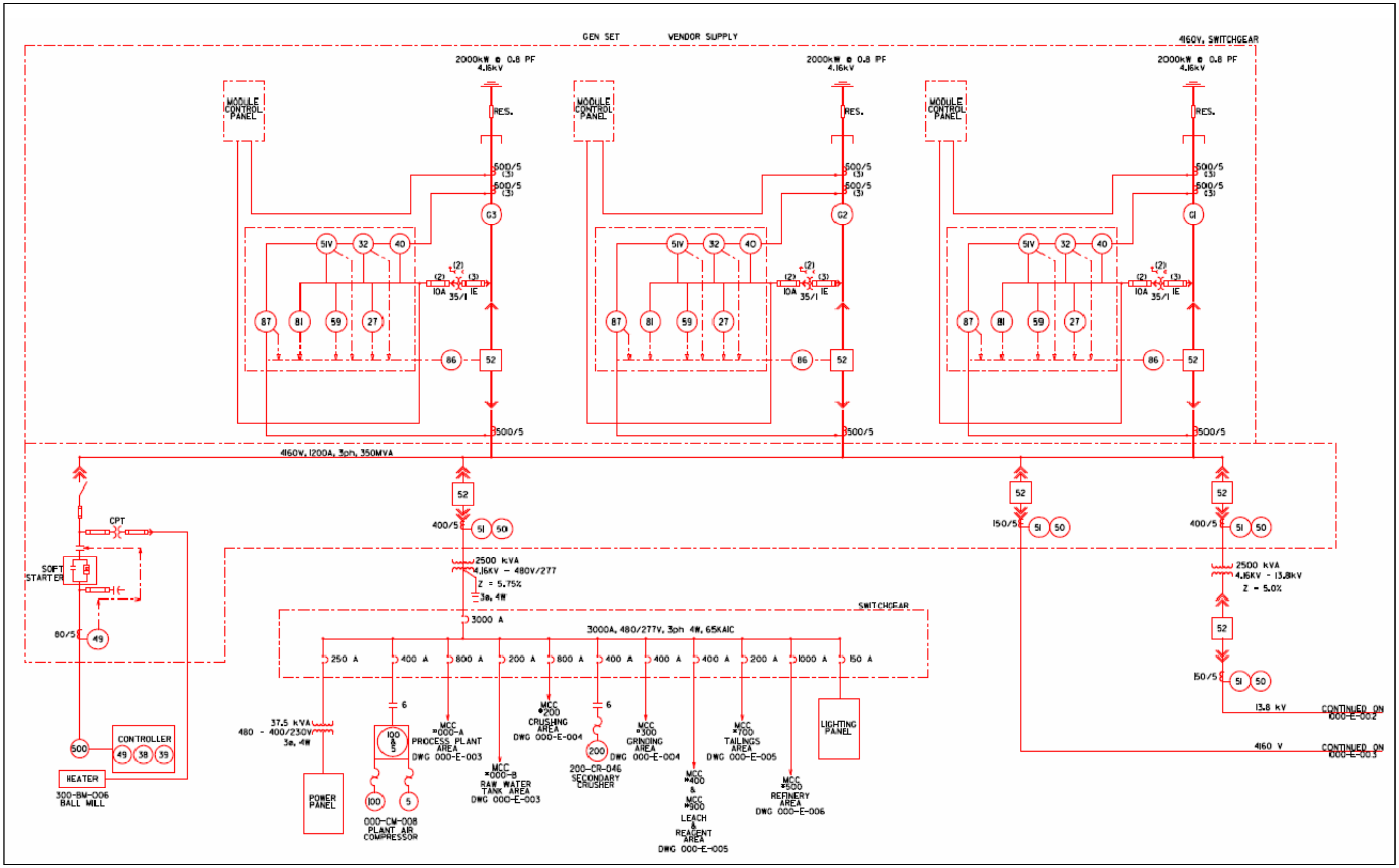
The distribution voltage levels designed for the process plant are indicated in Table 25-15. The system utilizes a radial distribution to reduce line voltage losses.

Table 25-15: Distribution Voltages

Distribution Voltage	Load
13.8kV	Main Substation
4.16kV	Large Loads (Over 186kW)
480V	General Loads
230V	Small Loads (Up to 0.56kW)

According to the design criteria, the power supply distribution will be designed considering a single power feed connected to the switchgear. The distribution system single line diagram is shown in Figure 25-15.

Figure 25-14: Single Line Diagram of the Distribution System



25.4.5.2 Bulk Site Power Distribution

The design of the electrical power distribution system outside of the process plant consists of the main substation, overhead distribution lines, to the mine, Quebrada Huanchuy and the operations camp. Power distribution cabling has been designed for a voltage drop of not more than 3.5%, in accordance with Peruvian standards.

25.4.6 Electrical Equipment

25.4.6.1 Medium Voltage Switchgear

The switchgear for the main substation consists of a 1,250A, 15kV switch and three 1,250A switches for the main feeders. These items are included with the diesel generation supply.

Each switchgear will include current and voltage transformers, a continuous current control system and a “feeder” type numeric generation protective relay, which guarantees the ANSI minimum protection standards: 50/51, 50/51G, 49, 38 and 25.

The switchgear will be coupled to the 2.5MVA power transformer by means of medium voltage cable.

25.4.6.2 Motors

Motors will be NEMA design standard and shall comply with IEEE 481 and NEMA MG1 standards. The motors will comply with the criteria shown in Table 25-16. All motor ratings have been selected such that they are suitable for derated motor cooling at the site elevation.

Table 25-16: Motor Requirements

Motor Size	Motor Voltage	Service Factor	Enclosure
0.56kW (Under 3/4HP)	220 V Single phase	1.15	TENV
0.56 to 186kW (3/4HP to 50HP)	460 V, 3 phase	1.15	TEFC
186kW (larger than 250HP)	4.160 V, 3 phase	1.0	TEFC or WP-II

25.4.7 Ancillary Buildings

Office and warehousing space for the project are of modified shipping containers. Where practical these containers have been arranged into complexes allowing for consolidation, utility and personnel efficiencies. Other structures with higher special demands are of Sprung Structure construction. The ancillary buildings for the project consist of the following:

- Administration complex.
- Medical/First aid building.
- Kitchen and dining building.
- Metallurgical laboratory.
- Process plant offices.

- Warehouse complex.
- Maintenance complex.
- Maintenance shop.
- Mine dry building.
- Security checkpoints.

25.4.7.1 Administration Offices

The administrative and mining offices are of modified shipping containers with dedicated offices, conference room, and additional space for support equipment and document storage. This complex also includes lavatory facilities. The complex is located adjacent to the process plant.

25.4.7.2 Medical/First Aid Building

Located within the administration complex is the medical/first aid building. This building is a modified container with a dedicated office and a first aid room.

25.4.7.3 Dining Room / Kitchen

This building is a Sprung Structure with full kitchen facilities and a dining area with seating for eighty people. It is located just south of the administration complex adjacent to the process plant entrance.

25.4.7.4 Metallurgical Laboratory

This building is located adjacent to the process plant and is of modular construction. This facility will be provided and operated on a contract basis.

25.4.7.5 Process Plant Offices

This complex includes two dedicated offices for operations personnel, a breakroom, and lavatory facilities. These spaces are of modified containers and are centrally located adjacent to the process plant.

25.4.7.6 Warehouse

The warehouse complex consists of four modified containers. The first has three dedicated offices and an area for document storage. The other three containers are retrofitted with rollup doors allowing for side access, storage and retrieval of larger material. This complex is located near the process plant and configured to allow fork truck access.

25.4.8 Mine Area Buildings

25.4.8.1 Maintenance Shop

The maintenance shop is located near the 4,400m portal and is a Sprung Structure with 2 maintenance bays for equipment maintenance. A concrete wash bay is also located beside the shop.

25.4.8.2 Maintenance Complex

This complex is comprised of two modified containers allowing for three dedicated offices, document storage, and a tool crib. This complex is located next to the maintenance shop.

25.4.8.3 Mine Dry Building

The mine dry building is a Sprung Structure building with lavatory, shower area, an equipped change room with locker baskets, a ready room, and lamp storage and charging area. This building also contains the necessary support equipment such as water heaters and ventilation.

25.4.8.4 Mine Rescue Container

A dedicated container for storage of mine rescues equipment will be located next to the mine dry building.

25.4.9 General Site Support Buildings

25.4.9.1 Security Check Points

Four, two room modified containers have been positioned at key points throughout the site. Two of the buildings are located at the entrances to the property, a third is located at the powder magazine, and the forth is located at the entrance to the process plant.

25.4.10 Site Mobile Equipment

Mobile support equipment included in the capital cost estimate for the process plant includes the following:

- 5,000lb. all terrain warehouse/refinery forklift.
- 10,000lb. Gradall forklift.
- 20t boom truck.
- 45t all terrain crane.
- Skid steer (Bobcat).
- 8t flat bed truck.
- Maintenance truck with compressor and welder.
- Ambulance.

- Portable light plants.
- Portable compressors.
- Portable welders, diesel powered.
- Fusion machine for HDPE pipe.
- Portable generator for fusion machine.

25.4.11 Fuel Storage and Distribution

All site mobile equipment and generators will use Type 2 diesel fuel. The fuel will be delivered by the vendor using 9,000 gallon capacity fuel trucks.

Two fuel storage tanks sized for seven day supply will be used. One tank will be located near the generators and have a capacity of 174.1m^3 (46,000 gallons).

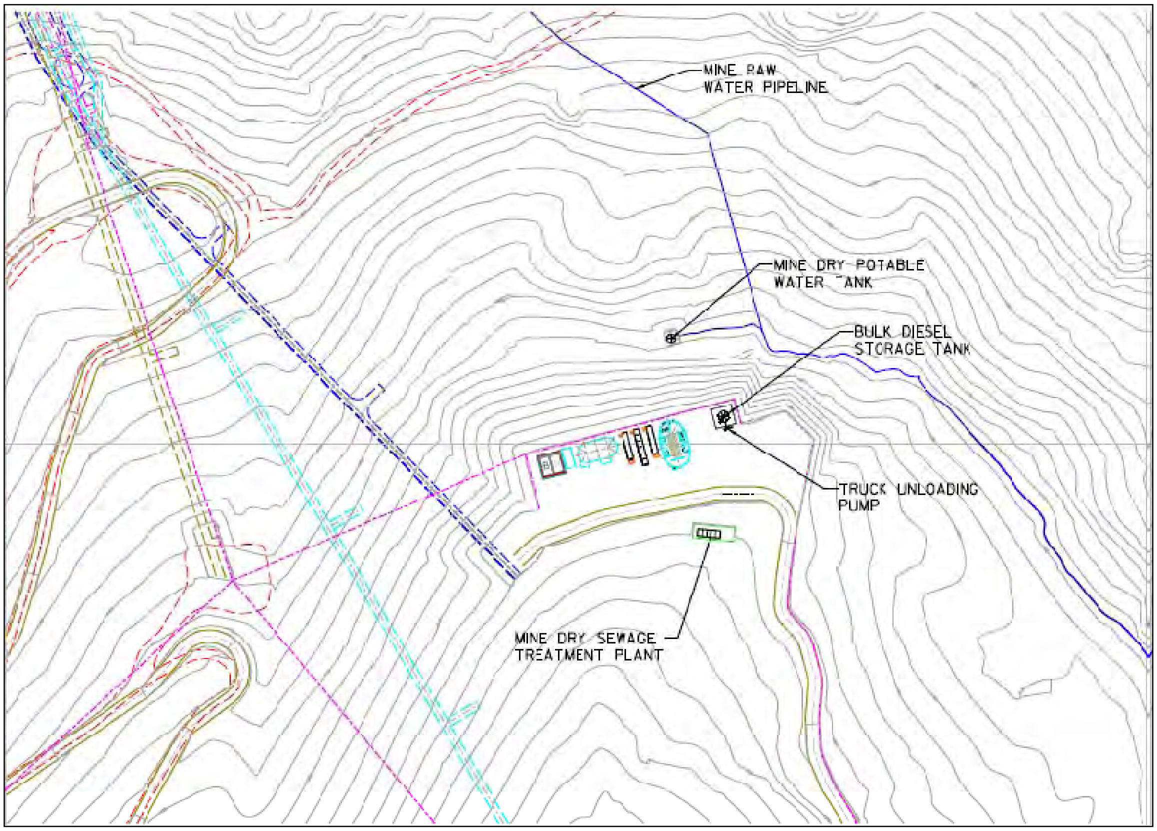
The second storage tank will be located near the maintenance shop. It will have a capacity of 74.8m^3 (19,760 gallons) and be equipped with two dispensing stations, a high volume station for large equipment such as haul trucks, and a low volume station for light vehicles.

The design of the storage tanks will be in compliance with the specifications of DS-052-93-EM (Safety Code for Hydrocarbons Storage-Peru). Additionally the tank designs will comply with API650 and the fire protection system will comply with NFPA 30. Table 25-17 presents the fuel storage and delivery criteria. The distribution system is shown in Figure 25-15.

Table 25-17: Fuel Storage Tanks

Installation	Location	Volume (m^3)	Storage (days)	Auxiliary Installation	Comment
Generators	N8,960,091 E191,478	174.1	7	Day tank- 1200 gal	Feed to day tank pump- 2HP/30gpm
Mobile Equipment	N8,960,381 E190,513	74.8	7	2 Dispatch stations	1 Station for light vehicles. 1 station for heavy vehicles.

Figure 25-15: Maintenance Shop No. 2 Diesel Storage and Delivery System



25.4.12 Fire Protection

25.4.12.1 Fire Suppression

A fire risk analysis performed by BISA shows that the highest risk areas for the facility are the fuel storage tanks and the process plant. The primary fire suppression method will be water. Foam suppression will be used to suppress fires at fuel storage tanks. The suppression systems will comply with the specifications of NFPA 24, 11, DS-42F-Título III – (Prevention and Protection Against Fire - Industrial Safety Code), DS-052-093-EM (Safety Code for Hydrocarbons Storage – Chapter VII- Protection Against Fire) and regulations of National Edifications Code (RNE – Norma A-130-Safety Requirements).

Due to the relatively small areas of other installations such as offices, shops, warehouse, the use of portable dry chemical dust extinguishers will be used according to RNE and NFPA.

Dry chemical fire extinguishers will be mounted on all mobile equipment.

25.4.12.2 Fuel Storage Fire Suppression

Each fuel station will be equipped with a buried fire water distribution main. Each main will include two hydrants and one foam generating monitor. Two 55 gallon drums of foam concentrate will be stored at each monitor.

A mixture of water and foam will be used to create an aqueous film forming foam to suppress fires at the two fuel storage facilities. Foam generating nozzles will produce foam by mixing 60gpm water with the foam concentrate at a ratio of 97% water to 3% foam concentrate. The system will be designed to NFPA 11 standards.

Cooling water available by hydrant will also be sprayed on the tanks in the event of a fire. The system will have a dedicated reserve capacity of 125 gpm for 2 hours.

Table 25-18 shows required fire suppression water volumes for the two fuel storage facilities.

Table 25-18: Fuel Tank Fire Suppression Requirements

Item	Facility	Required Water Volume	Foam Concentrate Volume
1	Generator Fuel Station	17,910 gallons (67.8m^3)	2 x 55 gallons
2	Maint. Shop Fuel Station	17,910 gallons (67.8m^3)	2 x 55 gallons

25.4.12.3 Process Plant

The process plant will be equipped with a perimeter fire water distribution main. This system will include hydrants. Fire cabinets will be located between hydrants. The cabinets will contain 2 ½" and 1 ½" hoses with fog jet water nozzles and foam concentrate feeders.

The dedicated fire water reserve for the process plant will be 30,000 gallons (114.0m^3) assuming two fire hoses at a flow of 125gpm for two hours.

25.4.12.4 Other Installations

The camp, administration offices, warehouse, and maintenance shop will be equipped with 25lb dry chemical ABC type fire extinguishers spaced no more than 25m apart in accordance with NFPA 10 specifications.

25.4.12.5 Mine

The only fuel sources within the mine will be the equipment fuel tanks. There will be no fixed fuel storage underground, therefore no fixed fire suppression system will be required. Each piece of mobile equipment will be equipped with a portable dry chemical ABC type fire extinguisher that meets the requirements of the national mining safety code D.S. Nº 046-2001-EM. Additional wall mounted extinguishers will be placed at key locations underground.

25.4.12.6 Detection and Fire Alarm

Fire detection and alarm systems will be installed in the camp, security guard house, maintenance shop, offices and warehouses. The fire detection systems will be designed according to NFPA 72 specifications and criteria. The features of the fire detection and alarm system for each area are shown in Table 25-19.

Table 25-19: Features and Locations of Considered Areas

Installation	Smoke Detector	Heat Detector	Emergency Switch	Visual Alarm	Audible Alarm
Offices and general warehouse	Yes	No	Yes	Yes	Yes
Maintenance shop	No	Yes	Yes	Yes	Yes
Main control gate	Yes	No	Yes	Yes	Yes

25.4.13 Communications, IT and Business Systems

25.4.13.1 Specific Considerations

Permanent communications services consisting of voice, data, closed circuit television (CCTV) and control system networks will be installed at the site. The various facilities will be connected by fiber optic cable.

A satellite link will be used for communications between the site and the Lima offices. This link will provide both voice and data communications.

A radio communication system for the project will include fixed, and handheld radios, and a relay antenna or repeater station. A leaky feeder system will provide radio communications within the mine.

25.4.13.2 Data Network

An allowance of US$50,000 has been included in the capital cost estimate for the data network. This allowance would include an Ethernet 1000BaseT and Ethernet 100BaseT technology and structure cabling system using ADSS fiber optics and UTP cable. Fiber optics will be installed along the overhead power lines.

The process plant, administrative offices and maintenance shop will be linked through an Ethernet TCP/IP network.

The planned equipment for the data network includes:

- Personal computers.
- Network servers
- Network hardware and structured cabling.

25.4.13.3 Radio System

The radio link system will use a conventional VHF link network. The system will allow half duplex wireless communication for the site using fixed, vehicle mounted, and hand held radios.

To obtain the coverage in all the installations a repeater station will be installed. A leaky feeder system will be installed to provide VHF radio communications within the mine.

25.4.13.4 Closed Circuit Television

A Closed Circuit Television (CCTV) system will be installed as part of the site security system. The CCTV system will transmit over Ethernet TCP/IP communications using fiber optic and UTP cable.

The CCTV will be used for video surveillance at the security gates, Merrill-Crowe plant, and warehouse. Cameras with night vision capability will be placed in strategic locations around the process plant, administrative offices, warehouse, maintenance shop, and mine portals.

25.4.14 Sewage Treatment

Two pre-engineered, modular, sewage treatment plants (STP 1 & 2) will be installed at the San Luis Project: STP 1 located near the process plant and administrative offices and STP 2 near the maintenance shop.

The two sewage treatment plants will operate 365 days per year, 24 hours per day and will be available during the operating life of the mine. The treatment plants will include activated sludge processing with mechanical aeration. During construction the various contractors will provide and operate temporary sewage treatment plants.

The design criteria for each sewage treatment plant are shown in Table 25-20.

Table 25-20: Sewer Treatment Plant Design Criteria

Sewage Treatment Plant	Units	STP 1	STP 2
Location		Process Plant / Admin Offices	Mine Dry / Maintenance Shop
Population	N°	120	60
Sewage daily flow (average)	m^3/day	15	7.5
Flow peak hour	m^3/h	0.90	0.45
Organic daily load	Kg BOD5/day	0.42	0.21
Sewage temperature	°C	13	13
Treatment System (In Operation)		One (1)	One (1)
Treatment System (Stand by)		None	None
Operational cycle		24h/d – 365d/y	24h/d – 365d/y
Operational environment		Outdoors	Outdoors

25.4.15 Effluent Requirements

Sewer treatment plant effluents will be discharged. Table 25-21 presents the criteria for discharging the water to the environment.

Table 25-21: Sewer Treatment Plant Discharge Parameters

Item	Value
BOD_5 mg/l	<35
Phosphorus mg/l	<2
Grease and oil mg/l	<20
Fecal coliform NMP/100 ml	<400
Kjeldahl Total Nitrogen mg/l	<50
Total Suspended Solids (TSS), mg/l	**<25 (Annual average value)** **<50 (Random time value)**
pH Range	6 to 9

25.4.16 Solid Waste Disposal

During operations a total of approximately 85 people will be on site, in all shifts at any given time. These people will be distributed between the process plant and administrative offices, maintenance shop, mine, and camp.

Solid waste will be categorized as domestic waste or industrial waste and will be disposed of in a designated landfill. Hazardous waste will be temporarily stored on site until it is disposed of by an appropriate contractor. Recyclable waste will sorted and stored until it can be taken off site to an appropriate sorting and recycling facility.

25.4.17 Domestic Waste

The domestic waste will be collected and deposited in the on site landfill where it will be covered by an appropriate material.

Transportation of the waste to an external landfill is not possible because none are available nearby.

25.4.18 Industrial Waste

Industrial waste will be classified into non-hazardous and hazardous categories. Non-hazardous industrial wastes will be disposed of in the same facility as the domestic waste. Hazardous industrial wastes will be disposed of in compliance with Peruvian regulations.

25.4.19 Mine Effluent Treatment

The mine effluent treatment facility has been designed to treat the water produced during underground mining activities. The following assumptions have been made:

- Estimated design flow is 5l/s.
- Runoff water is expected to be acid free according to geotechnical testwork on ore, gangue and tailings. Therefore a neutral or basic pH has been assumed.
- Runoff water may contain oil (hydrocarbons) due to diesel consumption by the mining equipment. This amount of oils is estimated to be 250mg/l as its source is operational leakage.
- The water at the portals will pass through a sand filter for removal of the fine solid particles.

The maximum amount of hydrocarbon in water to be discharged is 20mg/l according to the Supreme Decree DS-037-2008-PCM "*Límites Máximos Permisibles de Efluentes Líquidos para el Sector Hidrocarburos*".

Due to the use of ANFO for blasting in the mine, gases will be produced. These gases will be carbon monoxide (CO) and nitrogen oxides (NO_X) which will be evacuated through the ventilation ducts. Safe blasting practices will be used to minimize explosive use, gas emission and possible contamination of the mine water.

Accordingly, a filtration system consisting of hydrocarbon filters and an ion exchange circuit for nitrate removal will be employed to reduce nitrates in the mine drainage water to below the maximum 10mg/l allowed by the National Regulation DS N° 002-2008-MINAM. A portion of the filtered product from this system will receive an additional step of filtration and is recycled to the mine raw water tank, the remainder will be discharged.

25.5 METALLURGICAL RECOVERIES

The processing plant recoveries are based on a series of metallurgical tests which are detailed in Item 18. The refinery recoveries are based on a quotation from Metalor Technology S.A. ("Metalor"). The recoveries are summarized in Table 25-22.

Table 25-22: Metallurgical Recoveries

Metal	Recoveries (%)	
	Plant	Refinery
Gold	94.0	99.0
Silver	90.0	99.5

25.6 MARKETS

The doré from San Luis will be saleable to any number of refiners, bullion banks or brokers in North and South America.

25.7 CONTRACTS

There are currently no contracts in place for the sale of San Luis doré. However, the refining terms used in the financial analysis are based on budgetary quotations from the local operations of international refiners and are within industry norms.

25.8 ENVIRONMENTAL CONSIDERATIONS

25.8.1 Introduction

This section reviews the environmental, archaeological and cultural aspects of the San Luis Project relevant to the exploitation of the Ayelén Vein.

25.8.2 National Environmental Regulations

The regulations applicable to the socio-environmental, archaeological, and cultural aspects are composed of a set of legal rules and regulations that govern the environmental issues relevant to mining in Peru. This set of regulations includes all national and sector rules which focus on the protection, preservation, and sustainable management of natural resources, the application of air, water, noise, soil, flora, and fauna quality standards. They also include Performance Policies and Rules on Social and Environmental Sustainability from the International Financial Institution (World Bank) and the Equator Principles.

25.8.3 Social and Environmental Influence Area

The Area of Direct Influence (ADI) is defined as the area where the San Luis project can potentially directly affect the environment during construction and operation. These areas include:

- Areas of direct influence such as the mine, process plant, tailings facilities, and other ancillary facilities.
- Hydrological areas of influence such as the Huanchuy, Tocash, and Iscupampa ravines
- Meteorological areas of influence such as downwind areas that can be affected by dust.
- Socioeconomic areas of influence such as the communities of Tambra and Pueblo Viejo.
- Land tenure areas such as the EPZ ONE, EPZ THREE, EPZ FOUR, CAHUARAN ONE, and PUMALHILLCA ONE mining concessions.
- Other superficial land assigned to the Communities of Cochabamba and Ecash.

The Area of Indirect Influence (AII) includes areas such as access roads and towns of Cochabamba and Miramar that will be a labor resource to the operation.

25.8.4 Environmental Baseline Conditions

25.8.4.1 Geographic Location

The project location is described in Section 3.1 and shown in Figure 3.1 of this report.

The proposed project facilities will be located in very rugged terrain consisting of steep mountainsides and deep valleys. Specifically, the project site is on the western flank of the central area of the Cordillera Negra. The area's topography is rugged and abrupt with mountain peaks/ridges having elevations between 4,000m and 4,900masl. The main mine portal and the mineral process plant will be located at an elevation of approximately 4,400masl.

The project lies in the political districts of Shupluy and Cochabamba, province of Yungay and the department of Ancash, Peru.

The project's area of influence is not located in a Protected Natural Area (PNA).

25.8.4.2 Site Access

The San Luis project site is accessible through a network of public roadways in the area, some of which currently support mining traffic. The main access to the project site is planned to be the public dirt and gravel roadways from the cities of Huaraz and Casma, to the project's west gate. Although this access is public, part of the road from Huaraz passes through property and controlled checkpoints at Barrick's Pierina mine.

A secondary access/egress point is also planned for the project site's east gate to Carhuaz.

25.8.4.3 Meteorological Data

The meteorological baseline information used for the project was collected from the following meteorologic stations:

- Recuay
- Pira
- Pueblo Viejo & Tambra
- Pierina Mine.

25.8.4.4 Precipitation

Precipitation records obtained from the Recuay station indicate that the maximum precipitation occurs between December and March. March has the highest average precipitation and July has the lowest.

Based on data from the Recuay station, the maximum 24 hour rain fall event in the last 20 years occurred in January 1998 with 53.3mm and the largest historical monthly precipitation event occurred in March 2001 with 259.1mm of rainfall.

The Pierina station records indicate a total average precipitation for a nine year period (2000 to 2008) of 1,217mm and a maximum precipitation in a 24-hour period of 46.20mm.

25.8.4.5 Temperature

Temperature variations in the project area are low with only slight seasonal and hourly fluctuations.

Table 25-23 shows minimum and maximum temperatures for each of the four meteorological stations.

Table 25-23: Average Baseline Temperature Data

Station	Temperature (°C)		
	Minimum	Maximum	Median
Recuay	-0.5	23.6	11.5
Pueblo Viejo	-0.4	5.5	
Tambra	4.4	10.5	
Pierina	-3.6	24.6	8.1

The Pueblo Viejo data is indicative of the temperature ranges that can be found at the mine and processing facility, while the Tambra station is located at the existing camp site.

25.8.4.6 Humidity

All four stations indicate the highest humidity occurs between February and March while the lowest occurs between June and September.

Table 25-24 shows the relative humidity ranges for each of the meteorological stations.

Table 25-24: Baseline Relative Humidity Data

Station	Relative Humidity	
	Minimum	Maximum
Recuay	48.3%	82.8%
Pueblo Viejo	63.0%	91.9%
Tambra	48.3%	75.1%
Pierina	50.3%	84.0%

25.8.4.7 Wind

Wind data from the Recuay station indicates winds are predominantly from the north. Wind directions for Tambra and Pueblo Viejo are predominantly from the East.

Wind speeds range from calm, generally in the early morning hours when there is cloud cover, to greater than 3m/s. High wind speeds occur during winter (May to August).

25.8.4.8 Air Quality

Baseline air quality data was collected in April, 2009 from four stations which monitor for PM_{10} particulates, SO_2, NO_2, CO, H_2S, and O_3. The data collected was below the maximum permissible levels described in the following standards:

- Air Quality National Standards, D.S. No. 003-2008- MINAM.
- Regulation of Air Environmental Air Quality National Standards, D.S. No. 074-2001-PCM.

- Maximum Permissable Level in Gaseous Emissions of Metallurgical Mining Activities, R.M. No. 315-96-EM/VMM,
- United States Environmental Protection Agency, EPA (NAAQS, 1971-1996)

25.8.4.9 Noise

Eleven monitoring stations were set up around the project site to record baseline data. All data collected was below maximum acceptable levels for the following standards:

- The Industrial and Residential Areas from D.S. No. 085-2003-PCM (Noise Environmental Quality National Standards).
- Environmental, Health and Safety Guidelines from the IFC – World Bank - April 2007.
- World Health Organization (Organizacion Mundial de la Salud).

25.8.4.10 Geologic Setting

A description of the project geologic setting can be found in Section 4 of this report.

25.8.4.11 Soil Characteristics

The criteria and methodologies used to determine the edaphic nature of the San Luis study area followed the rules and guidelines established in the Soil Survey manual (1994) and in the Soil Taxonomy manual (2003) from the Department of Agriculture of the United States of America (USDA).

Soil uses around the project site have been classified as urban and/or private facilities lands, permanent cultivation/ farming lands, natural prairie lands and lands without use and/or non-productive soils.

25.8.4.12 Surface Water Quality – Physical and Chemical Properties

A surface water quality monitoring program was completed in April and August of 2009. Water samples were taken from thirteen locations on small lakes and twelve surface locations to assess chemical properties, sediments, and hydrobiological parameters.

The lake monitoring stations were located at Huancacocha, Azulcocha Baja, Sejeacocha, Yanacocha (Jatum), Azulcocha Alta, Cotacocha, Pacsococha (two stations), Torococha, Yahuarcocha, Cutacocha, Macún and Huancayoc. Five surface water sampling stations were located on the Tocash ravine; Chopicancha underwater knoll, Pacsocoha, Azulcocha Alta underwater small lake and Tocash ravine (waters beneath and above the populated town of Pueblo Viejo). Five other stations were located in the Huanchuy ravine. Four of these correspond to waters upstream of the San Luis Project camp and a fifth station corresponds to downstream from the camp. An additional station sampled water from the Iscupampa ravine. The final station sampled the drinking water at the San Luis project camp.

The chemical analysis of the surface water samples compared favorably to the following water quality standards:

- Water Quality National Standard (ECA D.S. 002-2008 MINAM).
- Water General Act (Act number 17752 and the amended D.S. 007-83 S.A.).
- United States Environmental Protection Agency (USEPA).

With the exception of iron, all samples were below the limits for all Total Metals content. The higher than normal iron content is likely due to disturbance of sediments from erosion during the rainy season.

25.8.4.13 Baseline Ground Water Quality

Initial basic investigations and preparations for hydrogeological studies have been made during the site geotechnical assessment. A program of ongoing data collection and evaluation is planned to further examine the sources, flows and quality of underground water.

25.8.4.14 Hydro-Biological Characteristics

In April 2009 (wet season) and August 2009 (dry season) a hydro-biological assessment was realised. Twenty-four monitoring stations were established to monitor superficial water bodies (11 stations) and small lakes (13 stations).

Water at all stations was translucent and contained some microalgae. Results indicate that each station may be classified as moderately polluted environments. None of them reach a value of 3 in relation to the criteria established by Wilhm & Dorris (1968). Seven Macrozoobentos species were recorded. The insect class *Chironomidae* is the most numerous.

At this writing, a limnological study (characterization of lakes) has been completed for the dry season (August 2009) and a wet season (February 2010) program is underway to:

- Assess the bento and plankton (phytoplankton and zooplankton) communities.
- Determine the hydrobiological community.
- Relate the community's species to the physical and chemical properties of the water samples.
- Assess the physical and chemical characteristics of the lake sediments.

25.8.4.15 Flora

Five habitats or vegetation formations have been identified in the project area; vegetation of rocky areas, bofedal, puna pajonal, puna grassland, and riparian zone.

The Flora inventory recorded 54 wild species (*Plantae* Kingdom) distributed in 21 families with a predominance of gramineous plants. The average value of vegetation coverage is 78.3%. The average Shannon-Weiner index value is 2.85. The data indicates an ecosystem with a moderate biodiversity, homogeneous distribution and low dominance of species. Only the *Chuquiraga spinosa* "Huamapinta" species was found to be on the list of species protected by Peruvian legislation. None of the species inventoried are protected by the Convention on International Trade in Endangered Species (CITES) or the International Union for Nature Preservation (IUCN).

25.8.4.16 Fauna

Fourteen species of birds grouped in 7 families have been identified. The average Shannon-Wiener index value for the birds' community is 2.34, indicating a moderately diverse ecosystem.

Llamas are wild and live in herds averaging 16 to 20 individuals. Other mammals present include domesticated cattle, horses and sheep that pasture in the area. One amphibious species, a toad, belonging to the *Bufonidae* family was recorded.

None of the fauna species inventoried are protected by Peruvian legislations, CITES, or the IUCN.

25.8.4.17 Human Interest and Cultural Component – Archaeology

The archeological assessment for the San Luis project is planned to be completed in three stages. The first stage will be a field survey to determine the location and existence of archeological remains. The second stage will include sketches of each identified archeological site for planning the detailed survey and excavation. The thirds stage will establish a survey involving 2 m x 1m grids and excavation of any sites.

The purpose of the archeological assessment is to:

- Identify archeological sites on the project.
- Assess the archeological and cultural significance of the sites.
- Install markers and barriers to protect important sites.
- Obtain the Lack of Archeological Remains Certificate (CIRA).

25.8.4.18 Environmental Management

Environmental management of the San Luis project will be defined in an Environmental Management Plan (EMP). The EMP will be developed as part of the ongoing Environmental Impact Study.

The goals of the EMP will be the protection, recovery and improvement of the environment as well as development of an environmental understanding of the San Luis project area and the surrounding communities. The EMP will include the following:

- Control and mitigation plan.
- Solid waste management plan.
- Environmental monitoring plan.
- Occupational safety and health plan.
- Contingency plan.
- Conceptual closure plan.

25.8.4.19 Environmental Impact Assessment

The Environmental Impact Assessment (EIA) is being developed as part of a procedure that SSR will follow in order to obtain the required permits to develop and operate the San Luis project. The EIA will be developed according to the requirements of the following regulations and agencies:

- Environmental Protection Regulation related to Mining-Metallurgical Activity D.S. No. 016-93-EM amended by D.S. No. 059-93-EM, D.S. No. 029-99-EM, D.S. No. 058-99-EM & D.S. No. 022-2002-EM.
- Citizen Participation Regulation in the Mining Sub-Sector - D.S. No. 028-2008-EM.
- Citizen Participation Regulation in the Mining Sub-Sector - M. No. 304-2008-MEM/DM.
- Mining Environmental Affairs Directorate (DGAAM).
- General Directorate of Mining (DGM) from the Ministry of Energy and Mines (MINEM).

The study will also take into account guidelines from the Guide for the Preparation of Environmental Impact Statements, the Guide for Community Relationships as well as the World Bank and other international environmental guidelines.

25.8.4.20 Solid Waste Management Plan

A waste management plan, in compliance with Ley N° 27314 (General Law of Solid Waste), will be developed to provide procedures for the management of solid wastes generated by the project. The plan will consider waste types, collection, segregation, transportation, storage, and disposal procedures. It will also define record keeping, reporting, and training requirements.

25.8.4.21 Procedures and Requirements

Section 6.8 shows a procedures and requirements matrix along with details requested by the competent authorities in order to obtain the necessary authorizations and permits.

25.8.4.22 Permitting Related Risks

The San Luis project has controllable and non-controllable risks. The controllable risk is the ability to comply with the requirements for obtaining authorizations and permits in a timely manner.

The primary non-controllable risk is the time required for the regulatory authorities to evaluate the permit applications and issue the permits.

Based on similar projects, it is estimated that the review and approval process for the EIA and issuance of the related permits will take approximately six months.

Since permitting approvals form the project critical path, delays in the permitting process could have a negative impact on the project economics.

25.8.4.23 San Luis Project Environmental Permits Status

Using the list of authorizations presented in Section 6.8, the environmental permit status for the San Luis project is summarized below.

CIRA Certificate

SSR's archeological team has:

- Prepared the project's archeological assessment for restricted excavations and presented it to the National Institute of Culture (INC).
- Commenced hand excavation of 25 trial-pits on site. When completed, the results will be inspected by the INC Ancash supervisor.

Upon completion of the hand excavation, an archeological assessment report will be prepared. This report will be submitted to the INC Lima and Ancash offices for approval by the cultural estate technical commission after which the directors' approval resolution will be received. This resolution will allow SSR to apply for and obtain the CIRA.

EIA Approval for the Exploitation and Mining Benefit (Large and Medium Mines)

Activities completed to date include:

- Environmental baseline data collection for the 2009 rainy season including air quality, surface waste quality, soil sampling, sediment sampling and noise level sampling.
- Collection of socioeconomic data through surveys and interviews in 2009 for the social baseline.
- Environmental Baseline data collection for the 2009 dry season including air quality, surface waste quality, soil sampling, sediment sampling, and noise level sampling.
- Completion of the planned hydrological studies in August 2009 and February 2010.
- Limnological study of the small lakes in the area.
- Geodynamic studies.

Additional ongoing activities required to secure this approval include:

- Citizens participation process workshops in the towns of Tambra, Pueblo Viejo, Miramar, and Cochabamba. These workshops are planned for the second quarter of 2010.
- Analysis of ongoing baseline data collection and environmental monitoring.
- Preparation of reports on field investigations.
- Preparation of the project description chapter including the TSF and WSS.
- Impact identification and assessment.
- Environmental management plan.
- Environmental cost/ benefit analysis.
- Sanitary Authorizations

SSR will submit applications to obtain the following approvals:

- Public health authorization of domestic waste water discharge.
- Public health authorization of industrial waste water discharge.
- Public health authorization of drinking water treatment.

It is estimated that these applications will take approximately three months to review and approve.

25.8.4.24 Water Use License

SSR will apply for a water use license from the National Water Authority (ANA) for the construction and use of the water collection facility (sump and diversion) at Quebrada Huanchuy.

The ANA is a new department, recently established by the Peruvian constitution, so approval schedules are somewhat unknown. However, BISA estimates the review and approval process will take a minimum of three months.

25.8.4.25 EIA Development Schedule

Figure 25-16 shows an estimated schedule for Environmental Impact Assessment completion, delivery to MINEM and approval.

Figure 25-16: EIA Development Schedule

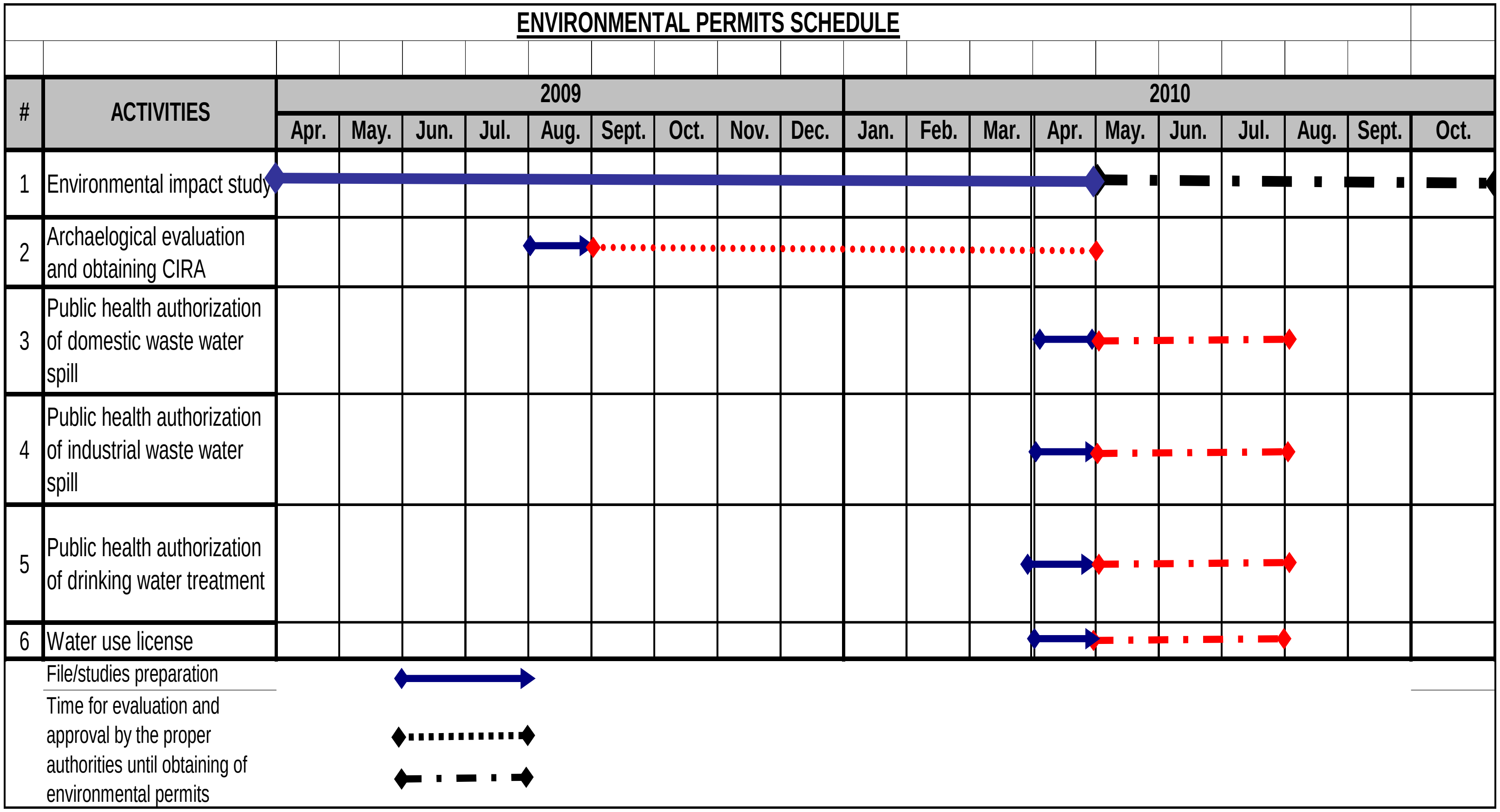
Themes	Jul	Aug	Sept	Oct	Nov	Dec	Jan	Feb	Mar	Apr	May
Environmental baseline		dry season					wet season				
Citizen participation									1st workshop		2nd workshop
Feasibility Study										Mid April	
Impacts identifying					dry season				wet season	Finished	
Environmental Management Plan						dry season			wet season		Finished
ARIC Project request											
ARIC Project development										Approval	
EIA delivery											Finished

Notes: ARIC means: absence of archeological remains certified; NIC means: National Intitute of Culture.

25.8.4.26 Environmental Permitting Schedule

Figure 25-17 shows an estimated schedule for completing the EIA report and government review process.

Figure 25-17: Environmental Permits Schedule



25.8.5 Conceptual Closure Plan

The Conceptual Closure Plan represents a preliminary plan describing dismantling of the San Luis facilitates and reclamation of the project site. Upon completion of the operation, the main closure activities will be:

- Processing vessels and pipe lines containing hazardous materials will be cleaned and residual materials will be removed and disposed of according to Peruvian environmental regulations.
- Process equipment will be dismantled and transported to a storage facility located off site.
- Electrical installations will be dismantled and transported to an off site storage facility.
- Environmentally hazardous materials will be removed and disposed of according to Peruvian environmental regulations.
- Steel structures will be dismantled and transported to an off site storage facility.
- Tanks and pump boxes will be dismantled and transported to an off site storage facility.
- Concrete structures, platforms and footings will be demolished. Debris will be disposed of on site.
- Tanks and equipment for fuel handling will be cleaned, sealed and transported to an off site storage facility. The site will be purged of hydrocarbons.
- Office buildings, which are prefabricated container type buildings, will be transported to an off site storage facility.
- Following the removal of all equipment, concrete, structures and piping, the affected areas will be reclaimed and covered with top soil. These areas will be conditioned; drainage established and re-vegetated with approved native species.
- The mining fixed and mobile equipment will be removed and transported to an off site storage facility.
- The mine opening will be sealed in accordance with the Peruvian regulations. The plan is discussed in detail in Section 4.5.
- The WSS will be left intact and ownership transferred to the local communities for their use and benefit. Further, the access roads associated with the WSS and other surface facilities will also be transferred to these communities or reclaimed, subject to further discussion with the communities.
- TSF closure will begin during the final stages of operation. This preparatory work includes modification of tails discharge points to provide a final tailing beach surface that drains to the planned spillway location. This work also includes time for tailing consolidation, residual process water evaporation and tailings surface drying to develop sufficient bearing capacity of the tailing surface for placement of cover material.
- Following initial TSF closure steps, further activities will include:
  - Placement of geogrid and an interim cover of local borrow material over the final tailing surface.

- Installation of a synthetic liner, geocomposite clay liner (GCL) or similar material over the interim cover
- Placement of a final cover of local borrow material over the liner, with the surface of the final cover will be re-vegetated with approved native species.
- Areas of runoff concentration will be armoured with rock or grouted riprap.

Items at the offsite storage facility will be either sold or transported to other SSR operations for re-use. Items than cannot be re-used or sold will be disposed of according to applicable regulations.

25.9 TAXES

25.9.1 IGV Taxes

While IGV is paid when various material or services are acquired, it is refundable as a tax credit. This results in a net zero sum in the cash flow model and thus it is not considered in the cash flow analysis.

25.9.2 Income Taxes

The Peruvian income tax rate is 30%. The income tax calculations shown in the cash flow have been provided by the accounting firm BDO Peru.

25.10 CAPITAL COST ESTIMATES

25.10.1 Capital Cost Estimate

The capital cost for design, procurement and construction of the San Luis project is estimated to be US$90.4 million. Breakdowns of this cost are presented in Tables 25-25 and 25-26 below, to provide an overview of capital budgeting needs and to facilitate understanding of the estimate components/assembly. The area costs for Mine, Process Plant, Infrastructure, TSF, WSS, Owner's Cost and Contingency are summarized in Table 25-25.

Table 25-25: Estimated Initial Capital Costs

Area[1]	Amount US$ (000's)	% of Total
Mine	8,123	9%
Process Plant & Infrastructure	50,984	56%
TSF and WSS	8,062	9%
Owner's Cost	11,972	13%
Contingency[2]	11,284	13%
Total	90,426	100%

Note: **1.** Component contributions (including contingencies) for the mine, process & infrastructure, TSF & WSS, owner's costs were provided by Milne & Associates (with BISA support), MQes, MWH, and SSR respectively. **2.** The overall contingency presented here is a combination of those provided by each contributor.

Table 25-26 below provides a further breakdown showing the split between direct and indirect cost components, including the labor, materials and equipment associated with each area.

Table 25-26: Estimated Direct and Indirect Costs

Description	Labor and Contract US$ (000's)	Bulk Materials US$ (000's)	Equipment US$ (000's)	Subtotal US$ (000's)	% of Total
Mine	924	472	5,895	7,291	8%
Process Plant & Buildings	14,060	6,824	14,478	35,362	39%
TSF & WSS	632	2,063	3,538	6,233	7%
Subtotal Direct Costs	**15,616**	**9,359**	**23,911**	**48,886**	**54%**
EPCM	10,741			10,741	12%
Freight & Duty	4,912			4,912	5%
Pre-ops, First Fill & Commissioning	888			888	1%
Spare Parts			1,743	1,743	2%
Subtotal Indirect Costs	16,541	0	1,743	18,284	20%
Owner Costs	11,193		779	11,972	13%
Contingency	11,284			11,284	13%
Escalation	n/a			n/a	
Total Estimated Cost	**54,634**	**9,359**	**26,433**	**90,426**	**100%**

All costs presented here are shown in first quarter 2010 US Funds, except for the mine components which are in third quarter 2009 US Funds. The basis for each of these costs is outlined below. With this in mind, the following sections outline the scope and basis for each of the project components.

25.10.2 Mine Capital Costs

25.10.2.1 Summary

Mine capital costs have been estimated by BISA and Milne & Associates. The initial capital cost for the mine and mine infrastructure for the San Luis project is estimated at US$9.3 million (including US$1.1 million in contingency) and is summarized in Table 25-27.

Table 25-27: Estimated Initial Capital Cost – Mine

Area	Cost US$ (000's)
Mine Development	707
Ore Development	22
Mine Mobile Equipment	5,440
Ground Support	168
Mine Portal	133
Mine Infrastructure	821
Subtotal Direct Costs	**7,291**
Spares Parts	295
Commissioning	40
First Fills	73
Third Party Support	67
Vendor Representatives	45
QA/QC	68
SHE	68
Freight	176
Subtotal Indirect Costs	**832**
Subtotal Direct and Indirect Costs	**8,123**
Contingency	1,139
Total Estimated Mine Capital Cost	**9,262**

25.10.2.2 Basis of Estimate

The capital cost estimate includes the direct costs, indirect costs and appropriate project estimating contingencies to develop the San Luis mine facilities. This estimate is based on a project directly operated and managed by SSR, using SSR staff and without the use of a general contractor. The estimated costs to design the project facilities and to manage construction are included in the overall project owner's costs.

25.10.2.3 Sources of Data

The capital cost estimate for the mine is based on the following project data:

- Design criteria.
- Single line diagrams.
- Budgetary quotations for major mine equipment.
- Material quantity take-offs and estimates from general arrangement drawings.
- Preliminary design calculations.

- Historical data and experience.
- Plan views of each level and longitudinal sections.
- Typical sections for drifts and bypasses.
- Geomechanical study of the mine to estimate ground support cost.
- Mine mobile and fixed equipment list.

25.10.2.4 Direct Costs

Direct costs for the mine are those associated with mine development and mine infrastructure. The basis used to estimate these is described below.

25.10.2.5 Mine Development and Bulk Material Quantities

Mine development direct costs were estimated using a variety of built-up unit rates with an estimated crew composition and quantities taken from preliminary mine design drawings. Crew and equipment composition was developed specifically for the San Luis mining operations, to suit the project mine plan and development schedule. The associated labor rates were based on specific Peruvian mining wage data, including benefits, overheads and accommodation applied to other operating staff. Specifically, the various development quantities and unit rates are shown in Table 25-28.

Table 25-28: Estimated Mine Development Quantities and Unit Rates

Item	Units	Estimated Qty	Unit Price US$/Unit	Extended Cost US$(000's)
Preproduction Development				
Galleries	m	751	565.34	425
By-passes	m	288	611.02	176
Crosscuts	m	168	577.95	97
Backfill raises	m	40	533.83	21
Service raise	m	40	228.98	9
Surface Quarry (for Backfill)				
Load and transportation < 1 km	t	5000	0.51	3
Transportation for > 1 km	t	5000	2.12	11
Preproduction Ground Support				
Rock Bolts	each	1,901	28.76	55
Split Set	set	5,538	20.49	113

Considering the mine construction will be self-performed, the project is expected to benefit from cost savings associated with sharing of equipment between development and production. Construction and mine equipment costs for the mine have been estimated based on vendor quotations. The estimated mine equipment costs are summarized in Table 25-29.

Table 25-29: Mine Equipment Costs (Fixed & Mobile)

Description	Size	Qty	Quoted Amount US$(000's)
Electric Hydraulic Jumbo Drill	1 Boom	2	920
Diesel Scooptram	3.2m^3 (4.2 yd^3)	2	816
Diesel Haul Trucks	20 Tonnes	4	1,992
Electric Scooptram	1.9 m^3 (2.5 yd^3)	2	681
Electric Scooptram	0.5 m^3 (0.7 yd^3)	4	675
Front End Loader	3.4 m^3 (4.5 yd^3)	1	244
Stoper Drill	Handheld	14	77
Jack Leg Drill	Handheld	4	22
Total Station	-	1	13
Subtotal Estimated Mine Mobile Equip			5,440
Surface Compressors	1,350CFM	2	186
Fan	42,000CFM	5	171
Mobile Electric Transformer	-	1	82
ANFO Loader	24.9kg (55 lb)	3	3
Submersible Pump	18l/s	2	13
Subtotal Estimated Fixed Equipment			455
Total Estimated Mine Equipment			5,895

25.10.2.6 Mine Infrastructure

Mine infrastructure was estimated using a combination of bulk material quantity take-offs, equipment budgetary quotations, and unit rates for installation. Mine infrastructure labor and other unit rates were developed using the same methodology and information used to estimate other project infrastructure.

The mine infrastructure costs, including the corresponding fixed equipment and surface quarry costs, is presented in Table 25-30 below.

Table 25-30: Estimated Mine Infrastructure Costs

Mine Infrastructure	Amount US$ (000's)
Ventilation	205
Electrical Distribution	233
Compressed Air	238
Dewatering & Drainage	16
Truck Chutes	30
Explosives Storage	79
Explosives Handling / Loading	3
Excavation - Surface Quarry	17
Total Estimated Mine Infrastructure Cost	**821**

In reviewing the above mine infrastructure costs, it should be noted that the costs presented above include the associated cost of fixed equipment estimated at US$ 680,000.

25.10.2.7 Indirect Costs

Design, procurement and construction management of the mine development represent a significant portion of indirect costs associated with the mine capital cost. Since this work is planned to be completed by SSR operating staff, the associated costs have been included in the overall owner's costs. For reference, Table 25-31 outlines some of the main technical elements that have been included in the owner's costs for design and procurement of the mine facilities.

Since owner costs include elements that are shared with other project aspects, the shared costs are not broken out here and they are not included in the estimated mine indirect costs of Table 25-31 above. Other indirect cost items have been estimated to complement the owner's costs for mine development and these are outlined in Table 25-31 above. Table 25-32 below summarizes the basis for the allowances and treatments that have been used to budget those other indirect costs.

Table 25-31: Indirect Design & Procurement Items Included in Owner Costs

Design & Procurement Task	Allowance/Estimate US$(000's)
Mine Production & Development Planning[1,2]	127
Mine Surveying[1,2]	44
Mine Geology[1,2]	138
Mine Engineering[1,2]	137
Estimated Owner Mine Staff Costs[2]	446
Mine electrical allowance	50
Truck chute allowance	20
Ventilation allowance	60
Mine contract development & award allowance	50
Compressed air system allowance	30
Mine equipment	30
Subtotal Supplements	240
Estimated Technical Design & Procurement Costs[2]	686

Notes: 1. The full owner team estimate is based on a staffing plan and other associated owner staffing costs outlined in Section 14.6 of this report. **2.** The amount shown does not include general owner's team costs shared with other aspects of the project such as Project Management, Construction Management, Project Controls, QA/QC, Safety, Quantity Surveying, Environmental/Permitting, Project Accounting, Operations & Maintenance Representatives, Camp Costs and Document Control.

Table 25-32: Indirect Cost Components and Allowances

Description	Allowance
Design, Procurement, Construction Management and Construction Services	By owner, included in owner costs
Third Party Support Services	US$ 67,000
Vendor Representative Support for Commissioning	US$ 45,000 [1]
Temporary Construction Facilities	Included in unit rates
QA/QC and Quantity Surveying	US$ 68,000 for support services
Spare Parts	5% of the total mine equipment costs, based on historical experience with similar Peruvian projects (0.05 x US$5,895,000 = US$295,000)
Commissioning	US$ 40,000
First Fills	1% of direct cost, based on historical experience
Safety, Health and Environment	US$ 68,000 for support services
Freight and Duty	3% of the total mine equipment costs, based on historical experience with similar Peruvian projects (0.03 x US$5,895,000 = US$176,000)

Note: 1. Accommodation for vendor representatives is included in the owner's camp estimation.

25.10.2.8 Support Services

The cost for mine design and management activities are included in the project owner's costs and shown in section 14.6. Further to those costs, an allowance of US$67,000 is included for support services.

25.10.2.9 Vendor Representatives

An allowance of US$45,000 is included to cover the costs of vendor representatives.

25.10.2.10 Temporary Construction Facilities

Temporary construction facilities, temporary fuel facilities, temporary power, temporary communication, construction camp and construction catering are included in the contractor's costs.

25.10.2.11 Assumptions and Clarifications

The mine development portion of the capital cost estimate has been prepared with the following assumptions and clarifications:

- A contingency allowance of approximately 14% has been included in the overall project cost estimate, to cover uncertainties related to in-scope mine development items. This is further described in Section 14.7 below.
- Escalation of initial capital and operating costs, beyond the Q3 2009 timeframe, is excluded.
- Duties for the importation of mining equipment are included in the purchase price of equipment sourced from Peruvian vendors.
- The following items are excluded from the mine development cost estimate presented in this section. They are addressed under the owner's cost and contingency sections of this report.
  - Inclusion of out-of-scope items
  - Force Majeure issues
  - Strikes, labor unrest or changes in labor legislation
  - Delays and outages due to weather conditions
  - All taxes such as Peruvian IGV (VAT) tax, excepting labor payroll taxes
  - Special project incentives related to project schedule acceleration and productivity improvement
  - Exchange rate fluctuations.

25.10.3 Process Plant and Infrastructure

25.10.3.1 Summary

The capital cost estimate for the process plant and infrastructure has been prepared by MQes. This section has been prepared by MQes and presented in a format requested by SSR. In order to present this information in this manner some reclassifications have occurred. The initial capital cost for the process plant and infrastructure to treat 400tpd of ore is estimated at US$58.6 million (including US$7.6 million in contingency) and is summarized in Table 25-33.

Table 25-33: Estimated Initial Capital Cost - Process Plant & Infrastructure

Area	US$ (000's)
General Site and Infrastructure	15,941
Mine	4,005
Crushing and Screening	3,515
Grinding	3,724
Leach and CCD	3,643
Merrill-Crowe, Precipitation & Smelting	3,826
Tailings	315
Reagent Handling	391
Subtotal Estimated Direct Costs	**35,362**
Freight, Ocean	1,553
Freight, Inland	1,065
Import Duties	2,118
Spare Parts	1,448
Vendor Representatives	245
Indirect Field Costs	8,664
Initial Consumable and Reagents	530
Subtotal Estimated Indirect Costs	15,623
Subtotal Estimated Direct and Indirect Costs	**50,984**
Contingency	7,648
Total Estimated Process Plant & Ancillary Building Capital Cost	**58,633**

25.10.3.2 Basis of Estimate

The capital cost estimate for the process plant and infrastructure is based on the following project data:

- Process design criteria.
- Process flowsheets.
- Major equipment list.
- Budgetary quotations for major process equipment.
- General arrangement drawings.
- Architectural drawings.
- Parametric civil model.
- In-house database.
- Material quantity take-offs and estimates.

25.10.3.3 Battery Limits

Capital costs for the process plant and infrastructure area are confined to the battery limits of the 4,400 level portal area and haul road interface, haul road, ROM delivery to the primary crusher dump hopper, fine ore storage, grinding, gravity concentration, leaching, CCD, pregnant solution handling, precipitation, smelting, power generation station, power distribution, raw water supply and buildings as defined in the equipment list, process flowsheets, and Section 7 of this report. These facilities are located within the property boundary.

25.10.3.4 Site Facilities and Process Equipment

Site facilities were estimated using a combination of bulk material take-offs, unit rates, and allowances. Major infrastructure items estimated in this manner include:

- Water supply and distribution.
- Power distribution.
- Fuel storage and distribution.
- Communication and information technology.
- Sewage treatment and solid waste disposal.

Process equipment is supported by budget quotations from qualified vendors. Approximately 87% of total equipment costs are supported by quotation. Costs for the remaining equipment as well as tankage and plate work are based on MQes in-house data.

Estimated costs by prime account are presented in Table 25-34.

Table 25-34: Estimated Costs by Prime Account

Account	Labor Hours	Costs US$(000's)					
		Labor	Const. Equip.	Material	Sub-contract	Equip.	Total
Earthworks	78,612	1,329	3,517	387	651		5,884
Concrete	46,448	859	351	517			1,726
Structural Steel	10,406	205	234	759			1,198
Architectural & Buildings	13,864	276	56	1,266			1,598
Mechanical	71,688	965	681		1,139	13,249	16,033
Piping	36,888	743	629	1,392			2,763
Electrical	51,897	949	629	2,399	119	110	4,206
Instrumentation & Control	23,763	345	223	821	157	404	1,949
Painting, Coating & Insulation					4		4
Total	**333,565**	**5,671**	**7,539**	**6,824**	**2,069**	**13,763**	**35,362**

25.10.3.5 Direct Labor Rates

Crew wages are based in part on labor surveys conducted with local construction companies, specifically, SSK Montajes e Instalaciones S.A.C. and Salfamontajes S.A.

- The direct labor rates are summarized in Table 25-35 and they include:
- Direct labor rate.
- Contractor's temporary buildings (office, warehouse).
- Construction utilities.
- Safety supplies.
- Maintenance of equipment and tools.
- Fuel and lubricants.
- Clean up.
- Material handling.
- Warehousing.
- Construction camp and catering.
- Transportation of employees to and from the job site.
- Contractors overhead, benefits and profits.

Table 25-35: Direct Labor Rates by Discipline

Discipline	Base Labor Rate US$/Direct Hour	Construction Camp Allowance US$/Direct Hour	Crew Rate US$/Direct Hour
Earthworks	16.08	2.00	18.08
Concrete	16.49	2.00	18.49
Structural Steel	17.74	2.00	19.74
Architectural	18.37	2.00	20.37
Mechanical Equip.	18.48	2.00	20.48
Piping	18.14	2.00	20.14
Electrical	18.57	2.00	20.57
Instrument & Control	19.13	2.00	21.13
Painting & Insulation	18.37	2.00	20.37

25.10.3.6 Bulk Material Quantities

Material quantities have been developed from the site layout, general arrangement drawings, discipline drawings, and design calculations. Bulk material prices were obtained from various local and foreign suppliers for the specified materials.

25.10.3.7 Earthworks and General Civil Works

As noted above, earthworks were estimated using a combination of bulk material take-offs, and unit rates for installation. Items estimated in this manner include:

- Mass excavation and backfill
- Structural excavation and backfill
- Trenching for site utilities
- Roads

Topographical data for this work was provided by SSR with contours of 2.0 meter intervals. Geotechnical data was provided by SSR based on a study of the area by MWH. Accuracy of the mass earth work estimate is limited by the precision of the topographic and geotechnical data available.

25.10.3.8 Concrete

Concrete items were estimated using a combination of bulk material take-offs, and unit rates for installation. Cast-in-place reinforced concrete was estimated including formwork, reinforcing steel, inserts, additives, expansion joints and finishing.

It should be noted that unit pricing for concrete is based on the use of a site batch plant.

25.10.3.9 Piping

Piping, including overland pipelines, was estimated using a combination of bulk material take-offs, and installation allowances based on mechanical equipment costs.

25.10.3.10 Electrical

Electrical facilities were estimated using a combination of bulk material take-offs, and installation allowances based on mechanical equipment costs. Items estimated in this manner include:

- Site power distribution.
- Cable tray and supports.
- Wire, cables and terminations.
- Electrical equipment.
- Grounding.
- Lighting.
- Miscellaneous electrical components such as duct banks.
- Overhead power lines.
- Transformers.

25.10.3.11 Instrumentation & Communication

Instrument and communication facilities were estimated using a combination of bulk material take-offs, vendor quotations, and installation factors based on mechanical equipment costs. Items estimated in this manner include:

- Site control and communication.
- CCTV system.
- Fire detection and alarm system.
- Fiber optic cabling.
- Installation of site instruments and interconnection of field devices to a field bus and to a central control room.

25.10.3.12 Indirect Field Costs

Construction Management

An allowance of 11% of the capital cost at the sub-total direct field cost level is included to account for construction management of the process plant and ancillary buildings. This allowance includes:

- Construction staff mobilization and demobilization.
- Salaries and expenses for the field management staff.
- Hook-ups of temporary utilities.

Engineering and Procurement

Allowances of 11% at the sub-total direct field cost level has been used to estimate the cost of engineering and procurement. Salaries and expenses for home office construction support staff are included.

Startup and Commissioning

An allowance of 2.5% of the sub-total direct field cost has been used for startup and commissioning.

Initial Fill of Reagents and Consumables

An allowance equivalent to 1.5% of the subtotal direct field costs has been used for the initial fill of reagents and consumables.

Freight

Ocean freight costs, including export packaging, for process equipment and material has been estimated at 11% of the equipment and material price. Inland freight for all process equipment and materials is estimated at 5% of the equipment and material price.

Import Duties

Import duties and customs clearance fees have been estimated at 15% of the imported equipment and material price.

Taxes

While IGV is paid when various material or services are acquired, it is refundable as a tax credit. This results in a zero sum and thus it is not considered in the analysis.

Spare Parts

An allowance for spare parts is based on 10% of the process equipment costs.

Vendor Representatives

An allowance equivalent to 2% of the process equipment costs is used for vendor representatives for support with construction and start-up.

25.10.3.13 Assumptions and Clarifications

The process plant and ancillary building portion of the capital cost estimate has been prepared with the following assumptions and clarifications.

- A contingency allowance of approximately 15% has been included.
- Escalation beyond Q1 2010, is excluded.
- The following items are excluded from the process plant and ancillary building cost estimate. They are addressed under the owner's cost and contingency sections of this report.
  - Force majeure issues and delays due to labor unrest, weather conditions, legislation changes or similar unforeseen events.
  - All owners' costs such as the following have been excluded:
    - Pre-operations expense.
    - Land acquisition.
    - Environmental impact report and permitting.
    - Owner's project management.
    - Hiring and relocation.
    - Legal.
    - Public relations.
    - Geotechnical investigations.
    - Testwork.
    - Sunk costs prior to and including this study.
    - Off-site facilities.

25.10.4 Tailings Storage Facility and Water Supply Structure (TSF and WSS)

25.10.4.1 Summary

Estimated costs for the Tailing Storage Facility (TSF) and Water Supply Structure (WSS) have been prepared by MWH. The initial capital cost for the TSF and WSS is estimated at US$9.1 million (including US$1.06 million in contingency) and is summarized in Table 25-36.

Table 25-36: Estimated Capital Cost TSF and WSS

Area	Labor and Contract, US$(000's)	Bulk Materials US$(000's)	Construction Equipment US$(000's)	Subtotal US$(000's)
TSF Direct Cost:				
Earthworks	190		2,713	2,903
Liner System	193	1,076	37	1,306
Underdrain	61	138	658	857
Diversion Channel & Basin	161	516	81	758
Perimeter Road & Fence	5	128	17	150
Subtotal TSF Direct Costs	610	1,858	3,506	5,974
WSS Direct Cost:				
Earthworks	5	50	8	63
Concrete	9	83	13	105
Mechanical	8	72	11	91
Subtotal WSS Direct Costs	22	205	32	259
Subtotal Direct Costs	**632**	**2,063**	**3,538**	**6,233**
TSF Indirect Cost:				
Contractor Indirect cost	780			780
EPCM Costs	950			950
Subtotal TSF Indirect Costs	**1,730**			**1,730**
WSS Indirect Cost:				
Contractor Indirect cost	49			49
EPCM Costs	50			50
Subtotal WSS Indirect Costs	99			99
Subtotal Indirect Costs	**1,829**			**1,829**
Contingency	1,060			1,060
Total	**3,521**	**2,063**	**3,538**	**9,122**

25.10.4.2 Basis of Estimate

In preparing the cost estimates for the TSF, the followed methodology and procedures have been used for an Association for Advancement of Cost Engineering (AACE) Class 4 Estimate. The estimate is comprised of three components, direct cost, indirect costs, and markups.

The estimated direct costs include labor, equipment, materials and subcontracts to perform work activities for the project. The estimated indirect costs include those costs for the contractor to manage the work, such as supervision, office trailers, and administration costs. The markups are items such as taxes, bond and insurance, and job profit. The indirect and markup costs are spread over the direct cost to obtain a bid cost for the items identified in the bid schedule of values.

Estimated costs for the construction of the TSF and WSS were estimated from the following information:

- Quantities generated from the feasibility design drawings.
- Unit costs for materials (such as pipe and concrete) developed from vendor quotes and published information.
- Unit costs for earthwork and other construction calculated from hourly construction rate data and hourly production (based on an efficiency factor for work at this elevation in Peru). These calculated unit costs compared closely with unit costs for similar work in Peru (from MWH experience) and similar work in the US (from published information).
- Labor rates were based on knowledge of local labor costs in Peru.
- Equipment rates are for contractor owned equipment (including ownership and operating costs).
- A crew based methodology was used to develop costs for the direct cost portions of the estimate.

25.10.4.3 Battery Limits

The battery limits for the TSF facility are located at the discharge of the tailing distribution pipe system and at the inlet of the reclaim water system at the TSF. It includes the feasibility level design of the tailing embankment and impoundment, perimeter road, diversion channel and stilling basin, and monitoring equipment. Tailing delivery and deposition system (piping and pumping) and water reclaim systems (piping, pumping and barge) from the TSF are included within the process plant battery limits.

The battery limits for the WSS facility are located at the suction of the water pumping and piping system. The battery limits include the design of the WSS at WSF-1 site in Huanchuy valley. Pumping arrangements, power supply and pipeline from the WSS to the raw water tank for project use are included in the process plant and infrastructure scope/estimate. Presented capital costs for the WSS are confined to the battery limits of the WSS.

25.10.4.4 Direct Costs

The estimated direct costs were built up based on a crew-based estimate where enough information was available at this design level. Once the direct cost was calculated, the indirect costs and the markups were spread to the direct cost to obtain a final price for the item. The estimated crew-based cost is comprised of a crew of labour, equipment and materials at an hourly rate to obtain the cost for an activity.

25.10.4.5 Construction Equipment

Construction equipment was estimated using industry standard software for cost estimating. Specifically, the system used is International Project Estimating (IPE) software. The software is updated on an annual basis and has a vast data base for construction equipment costs. This data base is comparable with equipment in other data bases such as the contractors blue book for equipment cost. The estimated construction equipment rates include ownership and operating costs for owned equipment. Ownership cost is the purchase price of the equipment less 10% salvage divided by the useful life of the equipment to arrive at an hourly rate. The hourly operating cost includes major and minor repairs, mechanics cost, parts cost, tires or track repair costs, fuel oil and grease costs for an hour of operation on the piece of equipment.

25.10.4.6 Bulk Materials

The purchase price of material items such as geomembrane, geogrid, piping, concrete, rebar, structural steel and instrumentation is included in this estimate and it includes freight on board (FOB) to the project. Taxes for these materials have not been included in the purchase price. The major material cost items are outlined in Table 25-37 below.

Table 25-37: Material Quantity Summary

Description	Quantity	Unit	Rate	Total Cost
Geomembrane 40 mil	119,733	m^2	4.50	538,801
Install Geodrain	58,929	m^2	5.38	317,039
Leachate 150 mm Slotted HDPE Pipe	58	m	16.40	951
Leachate 150 mm Solid HDPE Pipe	81	m	13.50	1,093
Leak Detection 100 mm Solid HDPE Pipe	84	m	8.00	672
Leak Detection 100 mm Slotted HDPE Pipe	31	m	0.32	10
Underdrain Pipe HDPE 10 in	100	m	12.00	1,200
Underdrain Pipe HDPE 6 in	419	m	4.00	1,676
Wet Screen Supplies	1	LS	6,000	6,000
Bedding Material	4,850	t	18.00	87,313
Rip Rap	12,945	t	15.07	195,112
Yard Gravel	700	t	20.00	14,000
Grout for Rip Rap	1,239	m^3	100.00	123,893
Redi-Mix Concrete	190	m^3	125.00	23,750

25.10.4.7 Construction Labor

The estimated construction labor budget is based on 60 hours of work per week, with no allowance made for overtime premiums. It is assumed that the local labor force is skilled, well equipped, readily available and able to complete the required work within the scheduled time. A shortage of any of these items could adversely affect the construction allowance and schedule. The labor rates are based on daily rates paid to the work force in Peru.

The overall labour cost for the TSF is US$528,000 US Dollars, including benefits, mark-ups and all associated direct labour costs. Similarly the labour cost associated with the WSS is estimated to be approximately US$10,000. The labor rates used to develop the estimate of direct costs are outlined in Table 25-38 below.

Table 25-38: Peru Labor Rates Summary ($US)

Description	Base Rate (US$/Hr)	Rate with Markup (US$/Hr)	Description	Base Rate (US$/Hr)	Rate with Markup (US$/Hr)
Foreman	6.65	7.30	Equipment Operators		
Survey Party Chief	13.00	14.27	Equipment Foreman	7.50	8.23
Equip Oper Cl 3 (Backhoe)	7.50	8.23	Auger Operator	3.75	4.12
Equip Operator CL 3 (Dozer)	7.50	8.23	Backhoe Operator	7.50	8.23
Equipment Operator CL3 (Loader)	7.50	8.23	Grader Operator	7.50	8.23
Survey Rodman	6.50	7.13	Loader Operator	7.50	8.23
Oiler	5.00	5.49	Dozer Operator	7.50	8.23
Labor Cl 5	3.75	4.12	Packer Operator	6.00	6.59
Parts Runner	5.00	5.49	Labourers		
Field Engineer II	24.00	26.34	Labour Foreman	6.00	6.59
Project Manager III	33.50	36.77	Labourer	3.75	4.12
Indirect Labour			Equipment Operators		
General Superintentent	13.50	14.82	Boomtruck Operator	5.00	5.49
Project Office Clerk	24.96	27.39	Auger Operator	3.75	4.12
Safety Manager	22.50	24.69	Grader Operator	7.50	8.23
Labourers			Packer Operator	6.65	7.30
Labourer	3.75	4.12	Engineering Design		
Concrete Foreman	6.00	6.59	Electrical Technologist	8.00	8.78
Concrete Labourer	3.75	4.12	Truck Drivers		
Vibrator Operator	3.75	4.12	Off Hwy Truck Driver	6.00	6.58
Concrete Truck Spotter	3.75	4.12	Mechanics		
Foreman Roadwork	7.50	8.23	Heavy Duty Mechanic	6.00	6.59
Grademan Roadwork	6.08	6.67	Mechanic Welder	6.00	6.59
General Labour Foreman	6.00	6.59	Master Mechanic	16.00	17.56
General Labourer	3.75	4.12	Carpenters		
Rodmen			Carpenter Foreman	6.00	6.59
Rodman Foreman	7.00	7.68	Carpenter	5.00	5.49
Rodman	6.00	6.59			
Rodman Helper	4.00	4.39			

25.10.4.8 Indirect Costs

The estimated contractor's indirect cost for TSF and WSS construction includes the following components:

- Project Management.
- Quality Control.
- Field Engineering.
- Surveying.
- Safety.
- Administration Equipment.
- Administration Costs.
- As-Builts/O&M Manuals.

A break-down of the TSF contractor's indirect costs is provided in Table 25-39.

Table 25-39: TSF Contractor Indirect Costs Summary

Category	**US$ (000's)**
Project Management	166
Contractor QC	130
Field Engineering	62
Surveying	21
Safety	72
Administration	11
Equipment	138
Administration Costs	134
Office Shop Setup	41
As-Builts/O&M Manuals	5
Total	**780**

The WSS contractor's indirect costs were estimated to be US$49,000. EPCM costs for the TSF and WSS include all aspects of engineering, procurement, project management, and construction management. Estimated costs are based on engineering personnel time before construction, and field engineering personnel during the anticipated period of construction. The estimated EPCM cost for the TSF is US$950,000, and US$50,000 for the WSS. These allowances were estimated as a percentage of the cost and is not a built up cost. It is approximately 12% of the total cost for both facilities selected based on experience with similar projects.

25.10.4.9 Assumptions and Clarifications

The TSF and WSS portion of the capital cost estimate have been prepared with the following assumptions and clarifications:

- A project cost contingency appropriate to the scope of work has been included to cover the uncertainty in the cost of in-scope items that have been estimated. A contingency allowance of approximately 15% has been included in the cost estimate for the TSF and WSS. This allowance was selected based on the level of understanding of the project and experience with similar projects. The estimated contingency amount is included in Table 25-39 above.
- Escalation of initial capital and operating costs, beyond the Q1 2010 timeframe, is excluded.
- Taxes are excluded here.
- Processing of filter, drain and bedding materials will be within 2 km of the project.
- Estimated pricing assumes competitive conditions at time of tender (+3 bidders).
- Estimated costs for processing of filter, drain and bedding materials assumes a 25% waste factor for these materials.
- Estimated costs for processing of filter and drain materials assumes a washed screen operation.
- Overall supervision of the project is included in the EPCM and owner cost allowances.
- Bedding material will not be processed but will be hand-picked of larger than 2 cm material.
- Tailing delivery and reclaim water systems for the TSF are excluded here and included with process plant costs.
- Power supply, piping and pumping from the WSS are are excluded here and included with infrastructure costs.
- The following items are excluded from the TSF and WSS cost estimate presented in this section. They are addressed under the owner's cost and contingency sections of this report.
  - Acquisition of land and water rights.
  - Environmental assessment report and facility permitting.
  - Water supply to the battery limits.
  - Changes in scope of work.
  - Force majeure issues.
  - Strikes, labor unrest or changes in legislation.
  - Delays and outages due to weather conditions.

25.10.5 Owner's Costs

Owner's costs have been estimated by SSR and included in the project capital cost estimate to allow for a variety of costs related to the owner's project management team and various expenses expected during the project. Specifically, these items and their estimated values are summarized in Table 25-40 below.

Table 25-40: Owner's Costs

Item	US$ (000's)
Owner's Project Management Team	
Optimization & Design Phase	1,668
Construction Phase (Expat)	1,665
Construction Phase (Local)	967
Subtotal Estimated Owner's Team Cost	**4,300**
Other	
Access	2,000
Accounting system & implementation	300
Community activities & safety awards	60
Consultants - mine design & procurement [note 1]	240
Consultants – third party	200
Legal & Audit	100
Metallurgical Test Work	120
Mine Plan Software, AutoCAD & Computers	221
Office – Lima	620
Office – Project Site [note 2]	116
Operating & Maintenance Manuals	100
Operator Training	153
Misc (incl. group meals, postage & stationary, radios)	294
Permitting	342
Project Insurance	383
Safety & Mine Rescue Equipment	461
Small vehicles [note 3]	779
Staff - Dependent Schooling	123
Staff - Relocations	385
Staff - Salary gross up for local taxes	583
Temporary Site Accomodation	92
Subtotal Other Estimated Costs	**7,672**
Total Estimated Owner Cost	**11,972**
Contingency @ 12%	1,437
Grand Total Estimated Owner Cost	**13,409**

Notes: **1.** Project and construction management support is allowed for through the owner's project management team and staffing plan. **2.** Building is provided for in the process facility and ancillary building portion of the estimate. **3.** Includes a variety of vehicles such as pickups, 5-ton truck, minivans, bus, forklift and front end loader.

25.10.6 Contingency Estimate and Basis

This section has been prepared by SSR using information provided by the contributors identified above. The overall project contingency has been developed through the use of an analysis of the uncertainties in each major area. This calculated contingency essentially provides a weighted estimate of the project contingency, considering the relative magnitude and varying uncertainty in each of the main components. The individual contingencies used for each project area are judged to be adequate allowances such that the cost for each area (with contingency) is expected to be within 15% of the actual project cost.

To this end, given the level of development and design for each of the project components, Table 25-41 summarizes the estimated component uncertainties that were determined for each area.

Table 25-41: Summary of Calculated Contingency Distribution and Amounts

Project Area	Contingency Allowance % of Area Direct & Indirect Cost	Contingency Amount US$(000's)
Mine	14%	1,139
Process Plant & Infrastructure	15%	7,648
TSF & WSS	13%	1,037
Owner's Cost	12%	1,437
Overall, Estimated Total Project Contingency	14%	11,261

In reviewing these contingencies, it should be noted that they do not include coverage for out-of-scope items or activities that may arise during project execution. Examples of such things 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 to date.

25.10.7 Closure

This section has been prepared by SSR using information provided by MQes, MWH, BISA and others. Closure costs for the major cost centers at the San Luis Project have been estimated and summarized in the following Table 25-42. The major cost centers include those costs associated with the mine, process plant, TSF, infrastructure and owner accounts. A brief description of the major items included in each of these cost centers follows below.

Table 25-42: Summary of Estimated Closure Costs

Area	Amount US$ (000's)
Mine	192
Process Plant & Infrastructure	7,567
TSF & WSS	1,773
Owner, Permitting, Environmental & Other	2,914
Total Closure Cost	**12,446**

25.10.7.1 Mine Closure Costs

The major costs associated with the mine closure will include: removal of mine infrastructure and site rehabilitation, sealing of the mine entrances (portals. and ventilation openings) with concrete bulkheads and the preparation, covering and seeding of the mine waste dumps. All of the mine closure work is anticipated to be performed by outside contractors, which will be carried out to Peruvian environmental standards for mine closures.

The estimated direct costs for mine closure were based on the same unit rates, methods and cost structure used for estimation of the mine development and operation. The associated indirect costs for closure were estimated as a percentage of these direct costs as shown in Table 25-43.

Table 25-43: Indirect Mine Closure Allowances

Indirect Cost Item	Allowance (% of Direct Cost)
General and Utility Costs	20%
Engineering, Supervision and Administration	10%
Contingencies	15%

Table 25-44 summarizes the breakdown of the estimated mine direct and indirect closure costs.

Table 25-44: Breakdown of Mine Direct & Indirect Closure Costs

Description	Closure Cost US$ (000's)
Labor/Materials	61
Equipment	38
Backfill	33
Subtotal Direct Costs	132
General Costs and Utilities	26
Engineering, Supervision and Administration	14
Contingency	20
Subtotal Indirect Costs	60
Total Estimated Mine Closure Costs	**192**

25.10.7.2 Assumptions and Clarifications

The mine portion of the closure cost estimate has been prepared with the following assumptions and clarifications:

- A contingency allowance of approximately 15% has been included to cover uncertainties related to in-scope mine items.
- Escalation of initial capital and operating costs, beyond the Q3 2009 timeframe, is excluded.
- The following items are excluded from the mine closure cost estimate presented in this section. They are addressed under the owner’s closure cost section below.

- Force majeure issues.
- Strikes, labor unrest or changes in labor legislation.
- Delays and outages due to weather conditions.
- All taxes such as Peruvian IGV (VAT) tax, excepting labor payroll taxes.
- Special project incentives related to project schedule acceleration and productivity improvement.
- Exchange rate fluctuations.
- Environmental report and permitting.
- Legal.
- Public relations.
- Geotechnical investigations.
- Temporary power supply.
- Temporary water supply.
- Decontamination costs.
- Hazardous materials disposal costs.
- Off-site facilities.
- Salvage value of equipment and materials.

25.10.7.3 Process Plant and Infrastructure

Upon cessation of project operations, the process plant and infrastructure are planned to be dismantled and the project site is planned to be rehabilitated. The closure cost for the process plant and ancillary buildings is estimated at US$7.57 million and is summarized in Table 25-45.

Table 25-45: Estimated Closure Cost for Process Plant & Ancillary Buildings

Accounts	**Cost US$(000's)**
Civil	1,296
Concrete	60
Structural Steel	90
Architectural and Buildings	1,160
Permanent Equipment	1,807
Piping	401
Electrical	457
Instrumentation	155
Subtotal Direct Costs	5,426
General Costs and Utilities	99
Engineering, Supervision and Administration	49
Haulage Allowance (Transport to Huaraz)	340
Contingency	1,655
Total Est. Process Plant and Infrastructure Closure Costs	**7,569**

25.10.7.4 Basis of Estimate

Unit rates developed for the various demolition and rehabilitation tasks utilize the same unit rates basis as is used in preparing the process plant capital and infrastructure cost estimate.

The closure cost estimate for the process plant and infrastructure is based on the following project data:

- Major equipment list.
- General arrangement drawings.
- Architectural drawings.
- Parametric civil model.
- Material take-off's.
- In-house database.

Battery Limits

Closure costs are confined to the battery limits of the process plant and infrastructure as defined in the equipment list, the process facility and infrastructure narrative of this document.

Permanent Equipment

Costs for dismantling process equipment, tankage, and platework are based on the major equipment list and the initial installation estimate. Equipment, tanks and pump boxes will be dismantled and transported off-site.

Concrete

Estimated costs for demolition of process concrete structures and miscellaneous building foundations are based on initial construction material takeoffs and unit rates developed for the various demolition activities. Concrete structures, platforms and footings will be demolished and it is assumed that all concrete debris will be disposed of on site without further treatment.

Structural Steel

Costs for demolition of steel structures are estimated based on initial construction material takeoffs and unit rates developed for the various demolition activities. Steel structures will be dismantled and transported off-site. No salvage value is included in this estimate.

Architectural and Ancillary Buildings

Estimated demolition costs for architectural construction, ancillary buildings, and furnishings are based on initial construction material takeoffs and corresponding unit rates developed for the various activities. It is assumed that all buildings will be dismantled and transported off-site. No salvage value is included in this estimate.

Process Piping

Cost estimate for the demolition of process piping is based on a percentage of the estimated installation cost and assumes that pipe lines will be classified into appropriate categories (steel, synthetic, etc.), dismantled and transported off-site.

Electrical

An estimated cost for demolition and removal of process electrical equipment and materials is based on a percentage of the estimated installation cost and assumes that electrical equipment and materials will be dismantled and transported off-site. Any environmentally sensitive materials will be removed and disposed of according to environmental regulations. No salvage value is included in this estimate.

Process Instrumentation

An estimated cost for the demolition and removal of process instrumentation equipment and materials is based on a percentage of the estimated installation cost and assumes that instrumentation and related materials will be dismantled and transported off-site. Any environmentally sensitive materials will be removed and disposed of according to environmental regulations. No salvage value is included in this estimate.

Civil

Following removal of all equipment, steel, concrete, debris, etc. the process plant site will be prepared and covered with top soil. The top soil cover will be conditioned and drainage slopes will be prepared to prevent erosion.

Estimated costs for site reclamation are based on the initial construction material takeoffs and unit rates developed for the various reclamation activities.

Transportation Allowance

An allowance of US$340,000 for transporting the various equipment and materials off-site is included.

25.10.7.5 Assumptions and Clarifications

The process plant and infrastructure portion of the closure cost estimate has been prepared with the following assumptions and clarifications:

- A contingency allowance of approximately 28% of direct and indirect cost items has been included to cover uncertainties related to in-scope closure items.
- Escalation of initial capital and operating costs, beyond the Q1 2010 timeframe, is excluded.
- The following items are excluded from the process plant and infrastructure closure cost estimate presented in this section. They are addressed under the owner's closure cost section below.

- Force majeure issues.
- Strikes, labor unrest or changes in labor legislation.
- Delays and outages due to weather conditions.
- All taxes such as Peruvian IGV (VAT) tax, excepting labor payroll taxes.
- Special project incentives related to project schedule acceleration and productivity improvement.
- Exchange rate fluctuations.
- Environmental report and permitting.
- Legal.
- Public relations.
- Geotechnical investigations.
- Temporary power supply.
- Temporary water supply.
- Decontamination costs.
- Hazardous materials disposal costs.
- Off-site facilities.
- Salvage value of equipment and materials.

25.10.7.6 TSF Closure

The estimated closure cost for the TSF is US$1,773,000 (including contingency). This estimate consisted of US$1,419,000 for materials and labor and US$354,000 for contingency. The estimated closure costs are summarized in Table 25-46.

Table 25-46: TSF Closure Costs

Description	**Cost US$ (000's)**
Mobilization/Demobilization	60
Excavate Spillway and Channel	26
Placement of geogrid over tailing surface	305
Placement of random fill cover over geogrid	251
Installation of HDPE geomembrane over random fill	315
Placement of soil cover over geomembrane	125
Bedding zone installation	27
Production and placement of spillway/channel riprap	154
Revegetate	26
Markups	130
Subtotal Direct Costs	1,419
Contingency	354
Total Estimated TSF Closure Costs	**1,773**

25.10.7.7 Basis of Estimate

The closure costs were estimated using the same procedures and unit rates used for the estimation of the TSF construction costs described in Section 14.4 above. Closure costs were estimated for the TSF, based on quantities generated from the conceptual closure design.

Closure costs for the TSF were estimated for the following TSF closure and reclamation activities:

- Placement of geogrid and an interim cover of local borrow material over the final tailing surface.
- Installation of a synthetic liner or similar material over the interim cover.
- Placement of a final cover of local borrow material over the liner, with the surface of the final cover amended for approved re-vegetation, and vegetation established on the cover surface.
- Areas of runoff concentration armored with rock or grouted riprap.

The closure tasks listed above would be conducted after preparatory work in the later stages of operation. This preparatory work includes modification of tails discharge points to provide a final tailing beach surface that drains to the planned spillway location. This work also includes time for tailing consolidation, residual process water evaporation and tailings surface drying to develop sufficient bearing capacity of the tailing surface for placement of cover material.

Unit Rates

Earthwork costs for closure of the TSF were estimated from calculated quantities and estimated unit rates of local or regional contractors. Unit rates were estimated from equipment rates, operator hourly rates, and standard efficiencies and cycle times for the haul distances associated with the work. Unit rates were compared with contractor unit rates for similar work in Peru.

Unit rates include excavation and hauling of material from the borrow areas, and placement and compaction in the specific area of the TSF. Key unit rates are presented below.

Cover Material Placement (excavate, haul, place)	US$4.82/$m^3$
Riprap (Produce and Place)	US$65.72/$m^3$

Materials

Materials costs for TSF closure were estimated from calculated quantities and unit prices quoted by qualified vendors. The unit prices include procurement, delivery, placement, and installation of material. Key unit prices are the geogrid and geomembrane for the cover system over the tailing, as shown below.

HDPE geomembrane (2 mm thickness)	US$6.07/$m^2$
HDPE geogrid	US$5.86/$m^2$

25.10.7.8 Assumptions and Clarifications

The TSF portion of the closure cost estimate has been prepared with the following assumptions and clarifications:

- A contingency allowance of approximately 25% has been included to cover uncertainties related to in-scope closure items.
- Escalation of initial capital and operating costs, beyond the Q1 2010 timeframe, is excluded.
- The following items are excluded from the TSF closure cost estimate presented in this section. They are addressed under the owner's closure cost section below.
  - Force Majeure issues.
  - Strikes, labor unrest or changes in labor legislation.
  - Delays and outages due to weather conditions.
  - EPCM
  - All taxes such as Peruvian IGV (VAT) tax, excepting labor payroll taxes.
  - Special project incentives related to project schedule acceleration and productivity improvement.
  - Exchange rate fluctuations.
  - Environmental report and permitting.
  - Public relations.
  - Temporary power supply.
  - Temporary water supply.
  - Decontamination costs.
  - Hazardous materials disposal costs.
  - Off-site facilities.
  - Final tailing disposal in the TSF (to achieve desired slopes and grades).
  - Process water evaporation at completion of process plant operation.
  - Management of the various closure activities and contractors.

25.10.8 Indirect, Owner Team, Environmental & Permitting Closure Costs

Indirect owner closure costs have been estimated by SSR to include the indirect costs that would be associated with project closure. These activities include budget allowances for general activities related to all closure areas and which have been excluded from other portions of the closure cost estimate. Examples of such items include contractor supervision, mobilization, demobilization, environmental monitoring, decontamination and disposal costs expected during closure. These indirect closure cost allowances and their basis are outlined in Table 25-47 below.

Table 25-47: Summary of Estimated Owner's Closure Costs

Item	$US (000's)
Closure Phase	651
Subtotal Estimated Owner's Team Cost	651
Community Relations	52
Consultants - Environmental	100
Consultants - Geotechnical	150
Consultants - Third Party	100
Corporate Charges - Lima office	310
Decontamination	200
Environmental Compliance & Permitting Actvities	130
Environmental Monitoring	90
Hazardous Material Disposal	250
Legal & Audit	50
Project Insurance	100
Re-vegetation of Remediated Areas	100
Test Work - Geochemical, ARD and Other	120
Temporary Power Supply	50
Temporary Water Supply	25
Temporary Site Accomodation	74
Temporary Storage in Huaraz	50
Subtotal Other Estimated Costs	**1,951**
Subtotal Estimated Owner Costs	**2,602**
Contingency @ 12% of costs	312
Total Estimated Owner Costs	**2,914**
Salvage Credit for Mine Equipment	-1,360
Salvage Credit for Process Equipment	-1,581
Salvage Credit for Other Equipment	-213
Sub-Total Salvage Value	**-3,154**
Net Estimated Owner Closure Costs	**-240**

Notes: 1. Project and construction management support is allowed for through an owner's project management team and staffing plan.

25.11 OPERATING COST ESTIMATE

The operating costs for the San Luis project is estimated at US$160.83/t ore processed. The costs broken down into Mining, Processing and G&A are summarized in Table 25-48.

Table 25-48: Estimated Operating Costs

Area	Annual Cost US$	Unit Cost (US$/t Ore)
Mine	8,613,713	59.00
Process	8,222,281	56.32
G&A	6,644,496	45.51
Total	**23,480,490**	**160.83**

In reviewing the operating cost, it should be noted that staffing levels for the mine, process facilities and general administration were developed by Milne & Associates, MQes, and SSR respectively. The roster system (shift staffing) and overall number of personnel was developed by SSR. Specifically, the labor wages for individual operating staff have been based on a 2008 survey of 18 Peruvian mining companies by the Hay Group.

The build-up of these costs is presented below.

25.11.1 Mine Operating Costs

25.11.1.1 Summary

Estimated mine average operating costs broken down into operating labor, maintenance labor, power, consumables and maintenance spares are summarized in Table 25-49.

Table 25-49: Estimated Mine Operating Cost Summary

Category	Annual Cost (US$)	Cost US$/t Processed
Labor - Direct	1,970,826	13.50
Labor - Maintenance	355,891	2.44
Labor - Indirect	1,340,653	9.18
Maintenance Spares	1,129,706	7.74
Power	2,512,517	17.21
Consumables (Expl., Ground Support, etc)	1,304,120	8.93
Total	**8,613,713**	**59.00**

Annual mine operating costs per activity are shown in Table 25-50.

Table 25-50: Annual Mine Operating Costs

Item	Year 1				Year 2	Year 3	Year 4	Total
	Q1	Q2	Q3	Q4				
Direct Operating Cost								
Permanent crosscuts (ore pass, service, etc.)	13,553	48,002	13,553	38,401	49,696	22,589	0	185,794
Raise Simple (Service Raise)	0	10,762	10,762	0	21,524	0	0	43,048
Raise Simple (Ore pass)	154,811	0	0	0	0	0	0	154,811
By Pass	113,650	172,919	281,070	268,850	93,486	0	0	929,975
Drift (Gallery)	121,549	308,113	339,207	306,417	294,545	0	0	1,369,831
Bolts & Plates	118,630	182,216	214,674	199,299	162,476	7,592	0	884,888
Steel Sets	28,732	72,832	80,182	72,431	69,625	0	0	323,801
Stopes' Crosscuts	7,332	29,329	58,658	51,326	54,992	46,438	0	248,075
Stopes' Ramps	0	0	0	0	389,221	426,493	0	815,714
Drift (Sublevel)	0	0	0	0	223,634	119,149	0	342,783
Raise Double (Backfilling)	66,195	37,635	41,105	75,270	250,900	106,499	0	577,603
Steel Sets	20,222	17,620	25,724	33,456	232,405	173,746	0	503,172
Stope (min 1.5m)	0	230,621	378,555	148,742	990,274	1,401,033	1,480,069	4,629,294
Stope (min 2.5m)	0	73,746	147,416	222,488	883,097	894,624	83,561	2,304,933
Steel Sets	0	25,144	45,009	41,012	188,256	216,055	106,123	621,600
Backfill Load & Haul from surface	0	50,524	90,443	82,410	378,285	434,145	213,246	1,249,052
Sub Total	**644,674**	**1,259,462**	**1,726,358**	**1,540,100**	**4,282,416**	**3,848,364**	**1,882,999**	**15,184,373**
Indirect Operating Costs								
Mine Superintendent	51,326	51,326	51,326	51,326	205,304	205,304	91,843	707,755
Mine Foreman	82,653	82,653	82,653	82,653	330,611	330,611	147,899	1,139,733
Chief Engineer	29,362	29,362	29,362	29,362	117,447	117,447	52,540	404,881
Mine Planning Engineers	39,628	39,628	39,628	39,628	158,512	158,512	70,910	546,446
Surveyors	14,561	14,561	14,561	14,561	58,243	58,243	26,055	200,783
Chief Geologists	29,847	29,847	29,847	29,847	119,390	119,390	53,409	411,579
Jr. Geologists	37,608	37,608	37,608	37,608	150,431	150,431	67,295	518,588

Clerks	24,268	24,268	24,268	24,268	97,071	97,071	43,425	334,637
Samplers	13,313	13,313	13,313	13,313	53,252	53,252	23,822	183,578
Dry Attendants	12,598	12,598	12,598	12,598	50,393	50,393	22,543	173,722
Subtotal	**335,163**	**335,163**	**335,163**	**335,163**	**1,340,653**	**1,340,653**	**599,740**	**4,621,700**
Maintenance								
Mechanics	42,363	42,363	42,363	42,363	169,451	169,451	75,804	584,156
Mechanics Helpers	28,421	28,421	28,421	28,421	113,685	113,685	50,857	391,913
UP GO Electricians	18,189	18,189	18,189	18,189	72,754	72,754	32,547	250,810
Sub Total	**88,973**	**88,973**	**88,973**	**88,973**	**355,891**	**355,891**	**159,207**	**1,226,879**
Power	628,353	628,353	628,353	628,353	2,513,412	2,513,412	1,124,373	8,664,609
Grand Total	**1,697,163**	**2,311,951**	**2,778,846**	**2,592,589**	**8,492,371**	**8,058,320**	**3,766,319**	**29,697,562**

25.11.1.2 Mine Operating Labor

Operating labor costs for the mine are summarized in Table 25-51. The operating labor cost is estimated to be US$22.68/t ore processed.

Table 25-51: Mine Operating Labor Costs

Position	Annual Cost US$
Labor – Direct	
Jumbo operators	102,760
Jumbo helper	66,370
Diesel Scoop operators	105,614
Electric Scoop operators	289,440
Front Loader operators	73,074
Ore Dumper operators	230,854
Backfill/waste dumper operators	76,951
Miners (stopes, raises, rock support, timber)	525,538
Miners' helpers	272,855
Utility (warehouse, explosive loader, services)	227,370
Sub-Total Direct	**1,970,826**
Labor – Indirect	
Mine Superintendent	205,304
Mine Foremen	330,611
Chief Engineer	117,447
Mine Planning Engineers	158,512
Surveyors	58,243
Chief Geologist	119,390
Jr. Geologists	150,431
Clerks	97,071
Samplers	53,252
Dry Attendants	50,393
Sub-Total Indirect	**1,340,653**
Totals	**2,681,307**
US$/t Processed	**22.68**

25.11.1.3 Mine Maintenance Labor

Maintenance labor costs for the mine are summarized in Table 25-52. The maintenance labor cost is estimated to be US$2.44/t ore processed.

Table 25-52: Mine Maintenance Labor Costs

	Annual Cost
Position	**US$**
Mechanics	169,451
Mechanics helpers	113,685
UP GO Electricians	72,754
Total	**355,890**
US$/t Processed	2.44

25.11.1.4 Power - Mine

Power costs were estimated based on preliminary load evaluations for individual motors in the major equipment list for the mine. The unit power cost was determined to be US$0.281/kWh. Power consumption is estimated at 8,935,092kW per year. The annual power cost is estimated to be US$2,512,517 (US$17.21/t ore processed). Power consumption is summarized in Table 25-53.

Table 25-53: Power Consumption – Mine

Equipment	Annual Cost (US$)
Jumbo -Single Boom	115,043
Scoop - Electric 0.7yd^3	312,164
Scoop - Electric 2.5 yd^3	258,343
Compressors	704,789
Fan	1,049,518
Lighting	36,329
Miscellaneous	36,329
Total	**2,512,517**

25.11.1.5 Consumables – Mine

Mining consumable prices are based on budget pricing supplied by BISA. The prices are delivered to site. Consumption rates are based on either vendor information or industry standards. The annual mine consumables are estimated to cost US$1,304,120 (US$8.93/t ore processed). Mine consumables are summarized in Table 25-54.

Table 25-54: Mine Consumables

Item	Annual Cost (US$)
Auxillary	51,168
Drill	256,488
Drill Oil	43,037
Explosives	472,223
Piping	12,710
Ground Support	219,656
Safety	15,372
Timber	18,907
Fuel	214,559
Total	**1,304,120**

25.11.1.6 Mine Maintenance Spares

Mining maintenance spares are based on vendor information and budget pricing supplied by BISA. The annual mine maintenance spares cost is estimated to be US$1,129,700 (US$7.74/t ore processed). This cost includes repair parts and tires. Mine maintenance spares are summarized in Table 25-55.

Table 25-55: Mine Maintenance Spares

Item	Annual Cost (US$)
ANFO Loader	1,074
Jumbo -Single Boom	162,900
Drill - Jackleg	33,598
Drill - Stoper	56,782
Front End Loader - Surface	95,157
Lamp	625
Lamp Charger	231
Scoop - Diesel	68,070
Scoop - Electric 0.7yd^3	115,261
Scoop - Electric 2.5 yd^3	85,838
Truck	510,169
Total	**1,129,706**

25.11.2 Process Operating Costs

25.11.2.1 Summary

Estimated process operating costs broken down into operating labor, maintenance labor, power, process consumables and maintenance spares are summarized in Table 25-56.

Table 25-56: Estimated Process Operating Cost Summary

Category	Annual Cost US$	Cost US$/t Processed
Operating Labor	940,641	6.44
Maintenance Labor	609,088	4.17
Power	2,416,825	16.55
Reagents and Consumables	2,866,454	19.63
Spares	1,389,273	9.52
TOTAL	**8,222,281**	**56.32**

25.11.2.2 Process Operating Labor

Operating labor costs are summarized in Table 25-57. The operating labor cost is estimated to be US$6.44/t ore processed.

Table 25-57: Process Operating Labor Costs

Position	Annual Rate US$	Number Required	Annual Cost US$
Processing Superintendent	189,257	1	189,257
Senior Metallurgist+ (process trainer)	101,550	1	101,550
Metallurgical Technician	44,548	1	44,548
Process Supervisor	49,765	3	149,295
Primary Crusher Operator	20,857	2	41,714
Crushing Operator Helper	15,904	2	31,808
Grinding Operator	20,857	3	62,571
Leach Circuit Operator	20,857	3	62,571
Merrill-Crowe Operator	20,857	3	62,571
Merrill-Crowe Operator Helper	15,904	3	47,712
Smelting - Operator	20,857	2	41,714
Smelting - Operator Helper	15,904	2	31,808
Reagent Operator	20,857	2	41,714
Reagent Helper	15,904	2	31,808
Totals		**30**	**940,641**
US$/t Processed			6.44

25.11.2.3 Maintenance Labor

Maintenance labor costs are summarized in Table 25-58. The maintenance labor cost is estimated to be US$4.17/t ore processed.

Table 25-58: Process Maintenance Labor Costs

Position	Annual Rate US$	Number Required	Annual Cost US$
Maintenance Superintendent	121,798	1	121,798
Electrical Supervisor	72,023	2	144,046
Mechanical Supervisor	72,023	1	72,023
Electrician	23,363	3	70,089
Instrument Technicians	25,833	3	77,499
Mechanic/Welder	22,455	3	67,365
Mechanic Helpers	18,756	3	56,268
TOTALS		16	609,088
US$/t Processed			4.17

25.11.2.4 Power

Power costs were estimated based on preliminary load evaluations for individual motors in the major equipment list. The unit power cost was determined to be US$0.281/kWh. Power cost is summarized in Table 25-59. The power cost is estimated to be US$16.55/t ore processed. Power cost for the mine is reported in the mine operating costs.

Table 25-59: Power Costs by Area

Description	Annua kWh	Annual Cost US$	Unit Cost US$/t Processed
000/General Site	2,221,186	624,817	4.28
100/Mine Support	94,297	26,497	0.18
200/Crushing	379,396	100,610	0.73
300/Grinding	3,076,720	864,558	5.92
400/Leaching & CDD	713,609	200,524	1.37
500/Refinery	2,022,953	568,450	3.89
700/Tailings	56,447	15,862	0.11
900/Reagents	36,194	10,171	0.07
Totals	**8,600,802**	**2,416,825**	**16.55**

25.11.2.5 Process Consumables

Process consumable prices are based on budget pricing supplied by BISA. Five percent has been added for transportation from Lima to the San Luis project. Lime is considered to be supplied from Huaraz. A 5% increase in price has been added to cover transportation to site. Consumption rates for sodium cyanide, lime and lead nitrate (leaching) are based on metallurgical testwork. Consumption rates for other consumables are based on calculations. Costs are summarized in Table 25-60. The process consumables cost is estimated to be US$19.63/t ore processed.

Table 25-60: Consumables and Reagents

Item	Annual Usage Kg	Unit Cost US$/Kg	Annual Cost US$	Unit Cost US$/t Processed
Crusher Liners			48,286	0.33
Grinding Balls	291,897	1.20	350,277	2.40
Mill Liners			382,582	2.62
Lime	100,405	0.15	14,659	0.10
Sodium Cyanide	311,898	2.63	820,291	5.62
Flocculant	17,520	4.69	82,169	0.56
Zinc	91,908	3.87	355,963	2.44
Fluxes	53,584	0.66	35,366	0.24
Diatomaceous Earth	52,399	0.65	34,059	0.23
Lead Nitrate - Merrill-Crowe	2,710	3.94	10,679	0.07
Lead Nitrate - Leaching	73,000	3.94	287,620	1.97
Propane	335,750	1.30	436,475	2.99
Antiscalant	2,920	2.75	8,030	0.06
Total			**2,866,454**	**19.63**

25.11.2.6 Maintenance Spares

An annual allowance equal to 10% of mechanical equipment costs has been provided to cover maintenance spares and supplies. The maintenance spares cost is estimated to be US$9.52/t ore processed.

25.11.3 General and Administration Costs

25.11.3.1 Summary

Estimated G&A costs broken down into staff labor, power, maintenance and general expenses are summarized in Table 25-61.

Table 25-61: Estimated G&A Cost Summary

Category	Annual Cost US$	Cost US$/t Processed
Staff Labor	1,795,677	12.30
Power	1,029,818	7.05
Maintenance	30,000	0.21
General Expenses	3,789,000	25.95
Total	**6,644,496**	**45.51**

25.11.3.2 G&A Staff Labor

Staff labor costs are summarized in Table 25-62. The staff labor cost is estimated to be US$12.30/t ore processed.

Table 25-62: Staff Labor Costs

Position	Annual Rate US$	Number Required	Annual Cost US$
General Manager	317,751	1	317,751
Administrator	118,646	1	118,646
Admin Assistant	43,648	1	43,648
Purchaser	71,472	2	142,944
Accountant	104,720	1	104,720
Accountant Assistant	43,648	2	87,296
Information Technology	38,684	1	38,684
Store Keeper	21,036	1	21,036
Receiver & Warehouseman	54,982	2	109,964
Social Services	119,646	1	119,646
Human Resources	61,457	1	61,457
Timesheet Control	56,049	2	112,098
Community and SH&E Supt	109,611	1	109,611
SH&E Technician	69,184	1	69,184
SH&E Assistant	54,314	1	54,314
Health Supervisor	104,152	1	104,152
Nurse	55,668	2	111,336
Community Relations Coordinator	38,684	1	38,684
Community Relations Helpers	15,253	2	30,506
Totals		**25**	**1,795,677**
US$/t Processed			12.30

25.11.3.3 Power

Power costs were estimated based on preliminary load evaluations for ancillary facilities. The unit power cost was determined to be US$0.281/kWh and the power cost is estimated to be US$7.05/t ore processed.

25.11.3.4 Maintenance

An annual allowance equal to 10% of the estimated ancillary facilities costs has been provided to G&A maintenance spares and supplies. The Maintenance cost is estimated to be US$0.21/t ore processed.

25.11.3.5 General Expenses

General Expense costs are summarized in Table 25-63. Costs were generated by MQes in conjunction with SSR. The general expenses cost is estimated to be US$25.95/t ore processed.

Table 25-63: General Expenses

General Expenses	Description	Annual Cost US$
Office Supplies	Allowance	25,000
Safety and First Aid Supplies	Allowance of approx. US$20/yr/Person	4,000
Communications	Allowance	20,000
Annual Audit	Allowance	50,000
Computer Supplies	Allowance	40,000
Insurance, Catastrpohic Loss	Mill replacement	200,000
Insurance, Business Interruption	0.5% of LOM Average profit	250,000
Miscellaneous Consultants	Allowance	200,000
Hiring & Relocation	Allowance	50,000
Licenses, Permits, Rights, Fees	Allowance	50,000
Land Use	Allowance	300,000
Community Relations	Allowance	60,000
Security	Proposal from Forza SA in Lima	480,000
Employee Relations	Allowance	20,000
Vehicle Rentals	Allowance	25,000
Assay Laboratory Operations	Budget proposal from SGS in Lima	820,000
Assay Checks	Allowance	10,000
Misc. Equipment Rental	Allowance	25,000
Catering & Housekeeping	Budget Proposal SODEXHO	648,000
Personnel transportation to/from site	Round trip to Huaraz utilizing company buses	20,000
Garbage and Waste Management	Allowance	20,000
Lima Office Costs	Allowance	372,000
Miscellaneous Expenses	Allowance	100,000
Total		**3,789,000**
US$/t Ore		**25.95**

25.12 ECONOMIC ANALYSIS

The financial evaluation of the San Luis project has been developed using the operating and capital costs shown in Section 25. For the purposes of this evaluation a Base Case has been developed against which the sensitivity of key variables have been compared.

25.12.1 Assumptions

25.12.1.1 Monetary Units

All amounts are in 1st quarter 2010 US$ with the exception of the mining equipment which is 3rd quarter 2009 US$. Amounts shown are in thousands unless otherwise stated. No allowances have been made for escalation.

Certain equipment quotations and cost calculations were done in other currencies and converted to US funds at the exchange rates shown in Table 25-64.

Table 25-64: Currency Conversions

Country	Currency	Designation	Conversion to US$
United States	Dollar	US$	1.00
Peru	Soles	PEN	0.3333

25.12.1.2 Metal Prices

The metal prices used in the base case evaluation are shown in Table 25-65.

Table 25-65: Metal Prices

Metal	Price (US$/oz)
Gold	800
Silver	12.50

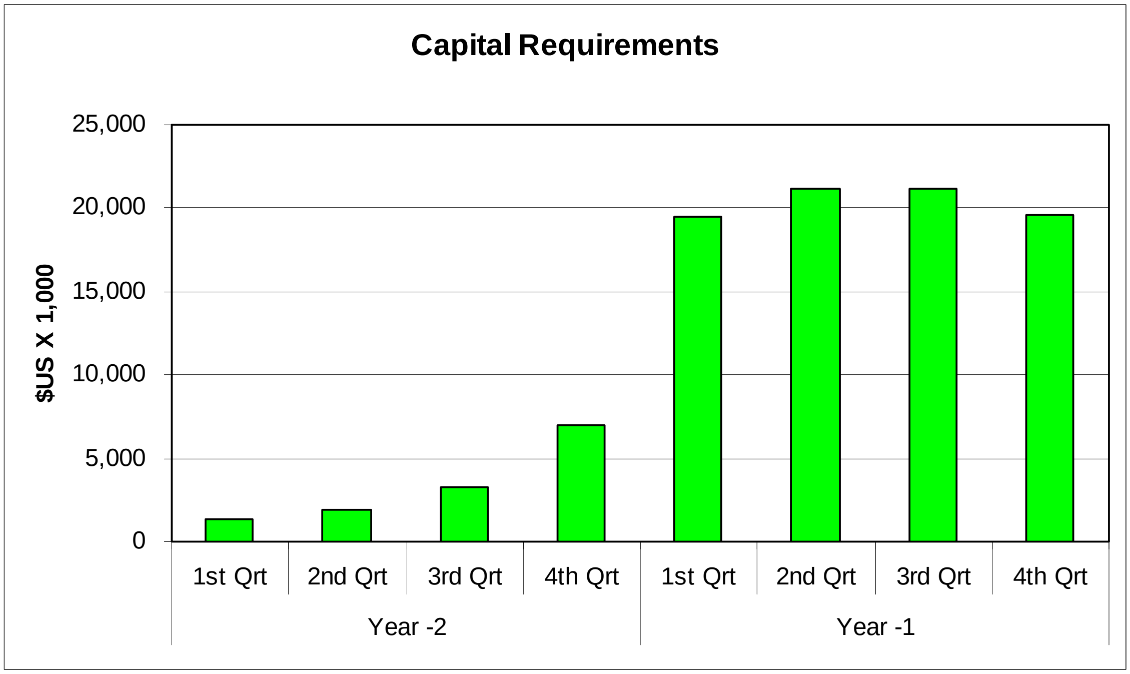
25.12.1.3 Tonnage and Grade

The tonnage and grades of ore to be treated were developed by Milne & Associates Inc. in conjunction with BISA as discussed previously in this report.

25.12.1.4 Forecast Initial Capital Expenditures

The estimated initial capital expenditures for the mine, process facility, infrastructure, and other areas are shown by quarter in the cash flow model summarized in Table 25-70 and shown graphically in Figure 25-18..

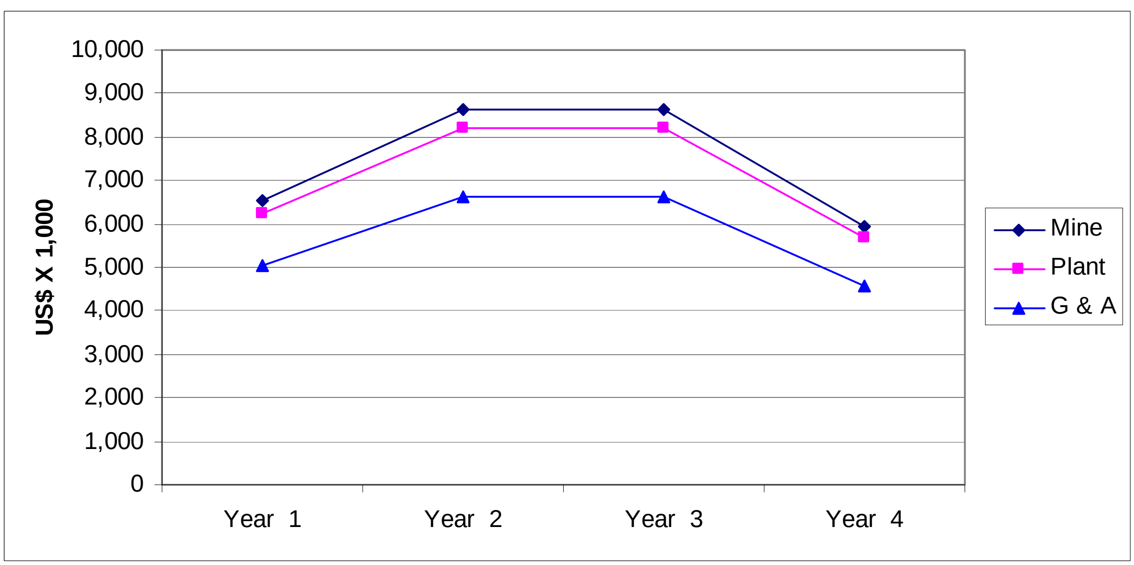
Figure 25-18: Forecast Capital Expenditures



25.12.1.5 Estimated Operating Costs

The estimated operating costs for the mine, process facility, infrastructure, and other costs are shown by year in the economic model (See Table 25-70). Details of the estimated operating costs are shown graphically in Figure 25-20.

Figure 25-19: Estimated Operating Costs Summary



25.12.1.6 Doré Transportation

Quotations were obtained for secure transportation from the San Luis site to the refinery in Lima. On the basis of cost and reliability, Hermes S.A.'s (Hermes) costs were used in the economic analysis. Hermes is a division of Brinks Inc.

25.12.1.7 Refining Charges

Quotations were obtained from various precious metal refineries in Peru. On the basis of cost and reliability, Metalor was chosen for use in this analysis. These charges are summarized in Table 25-66.

Table 25-66: Refining Charges

Item	Cost (US$)	Units
Treatment	0.50	$/oz of Ag Received
Refining	1.00	$/oz of Au Returned

25.12.1.8 Downstream and Sales Costs

Depending on market conditions at the time of sale, the gold and silver will be either deposited in a bullion bank of SSR's choosing or sold directly to Metalor. The marketing and downstream costs are assumed to be nil.

25.12.1.9 Working Capital

Working capital is considered to be three months of operating costs. This cost is also considered to be fully recovered at the end of the project.

25.12.1.10 Depreciation

Under Peruvian law the maximum amount of capital expenditure that can be recovered in one year is 20%. The project does not have a long enough life to allow for complete depreciation of the capital expenditures.

25.12.1.11 Amortization

The capital invested prior to the decision to proceed with the project (sunk costs) is amortized over the life of the project. This amortization is based on saleable ounces of gold produced during the project life. Specifically, SSR provided a sunk cost value of US$15.9M.

25.12.1.12 Royalties

Royalties are paid at the rate of 1% of the first US$60M of gross revenue and 2% of any gross revenue over US$60M.

25.12.1.13 Depletion

Depletion is not an allowable deduction under the tax laws of Peru.

25.12.1.14 Salvage Value

The mining equipment and processing facility are considered to have the salvage values shown in Tables 25-67. The salvage value is recovered in the year following the last production. These values are typical for the mining industry.

Table 25-67: Assumed Salvage Values

Item	Salvage Value
Mine Equipment	25%
Process Plant Equipment	12%

25.12.1.15 IGV Tax

While IGV is paid when various material or services are acquired, it is refundable as a tax credit. This results in a net zero sum in the cash flow model and thus it is not considered in the cash flow analysis.

25.12.1.16 Closure Costs

Closure costs are estimated at US$12.4M. The summary of these costs is shown in Table 25-43 and detailed in Section 25-10.8. This expenditure is assumed to occur in the year following the last production.

25.12.1.17 Income Tax

The Peruvian income tax rate is 30%. The income tax calculations shown in the cash flow have been provided by the accounting firm BDO Peru.

25.12.1.18 Financing

It has been assumed that the project will be financed through use of 100% equity financing.

25.12.2 Base Case Cashflow Analysis

The Base Case is calculated to have an Internal Rate of Return (IRR) of 26.5% and a Net Present Value (NPV) of US$39.2 million using a discount rate of 5%. Further details are shown in Table 25-69. The cashflow is shown in Table 25-70.

Table 25-68: Base Case Cashflow Summary

Item	Value	Units
IRR	26.5	%
NPV 0%	57,478	US$(000's)
5%	39,214	US$(000's)
10%	25,553	US$(000's)
Payback Period	1.2	years
Gold Production	270,031	ounces
Silver Production	6,454,810	ounces
Cash Cost of Gold	313.16	US$/oz

Table 25-69: Base Case Cashflow

31-Mar-10

Rev K

San Luis Project, 400 TPD

$US & Tonne X 1,000

Economic Evaluation

Recovery

Metal	Plant	Refinery
Gold	94.00%	99.00%
Silver	90.00%	99.50%

Selling Price

Metal	$US/oz
Gold	800
Silver	12.50

Item		Unit	Year -2 1st Qrt	Year -2 2nd Qrt	Year -2 3rd Qrt	Year -2 4th Qrt	Year -1 1st Qrt	Year -1 2nd Qrt	Year -1 3rd Qrt	Year -1 4th Qrt	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Production																
Ore		T									110,500	146,000	146,000	100,813	-	503,313
Waste		T									103,958	42,910	5,606	0	-	152,474
Total		T									214,458	188,910	151,606	100,813	-	655,787
Grade - Au		g/t									26.07	24.07	12.49	8.14	-	17.96
Grade - Ag		g/t									625.73	564.38	316.55	265.75	-	446.14
Au Recovered - Plant		oz									86,935	106,038	55,023	24,761	-	272,758
Ag Recovered - Plant		oz									1,997,543	2,380,521	1,335,189	773,994	-	6,487,246
Au Recovered - Refinery	1	oz									86,066	104,978	54,473	24,514	-	270,031
Ag Recovered - Refinery	1	oz									1,987,555	2,368,618	1,328,513	770,124	-	6,454,810
Gross Revenue																
Au		$US									68,853	83,982	43,579	19,611	-	216,024
Ag		$US									24,844	29,608	16,606	9,627	-	80,685
Total		$US									93,697	113,590	60,185	29,238	-	296,710
Operating Cost		$US														
Mine		$US									6,520	8,614	8,614	5,948	-	29,695
Plant		$US									6,223	8,223	8,223	5,678	-	28,347
G & A		$US									5,029	6,644	6,644	4,588	-	22,906
Total											17,772	23,481	23,481	16,214	-	80,948
Transportation		$US									25	25	25	25	-	100
Treatment		$US									999	1,190	668	387	-	3,244
Refining		$US									86	105	54	25	-	270
Sub Total		$US									1,110	1,320	747	437	-	3,614
Total - Cash Op Cost	1	$US									18,882	24,801	24,228	16,650	-	84,561
Net Revenue		$US									74,816	88,788	35,957	12,587	-	212,148
Deductions											74,816	163,604	199,561	212,148		
Royalty -		$US									1,274	1,672	604	600	-	4,149
Depreciation		$US									15,403	15,403	15,403	15,403	-	61,612
Amortization		$US									5,059	6,170	3,202	1,441	-	15,871
Sub Total		$US									21,735	23,245	19,208	17,444	-	81,632
Taxable Income		$US									53,080	65,544	16,748	-	-	135,372
Income tax & others related											20,298	25,064	6,405	-	-	51,767
Add Back																
Depreciation		$US									15,403	15,403	15,403	15,403	-	61,612
Amortization		$US									5,059	6,170	3,202	1,441	-	15,871

31-Mar-10

Rev K

San Luis Project, 400 TPD

$US & Tonne X 1,000

Economic Evaluation

Recovery

Metal	Plant	Refinery
Gold	94.00%	99.00%
Silver	90.00%	99.50%

Selling Price

Metal	$US/oz	
Gold	800	1
Silver	12.50	1

Item		Unit	Year -2 1st Qrt	Year -2 2nd Qrt	Year -2 3rd Qrt	Year -2 4th Qrt	Year -1 1st Qrt	Year -1 2nd Qrt	Year -1 3rd Qrt	Year -1 4th Qrt	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Capital Costs																
Ming		$US	0	0	6	1,818	3,460	2,051	1,337	590						9,262
Plant & Infrastructure		$US	0	0	925	3,450	13,020	15,013	15,470	10,753						58,631
TSF & WSS		$US	0	0	0	310	1,500	2,560	2,760	1,992						9,122
Owners Cost		$US	1,344	1,904	2,335	1,456	1,478	1,758	1,602	1,533						13,411
Total		$US	1,344	1,904	3,266	7,034	19,458	21,382	21,169	14,868	0	0	0	0	0	90,426
Closure															12,446	12,446
Working Capital		$US								4,720					(4,720)	0
Salvage- Mine		$US													(1,360)	(1,360)
Salvage - Plant		$US													(2,158)	(2,158)
Total Capital	1	$US	1,344	1,904	3,266	7,034	19,458	21,382	21,169	19,588	0	0	0	0	4,207	99,353
Cash Flow		$US	(1,344)	(1,904)	(3,266)	(7,034)	(19,458)	(21,382)	(21,169)	(19,588)	53,244	62,053	28,948	12,587	(4,207)	57,478
Cumulative Cash Flow		$US	(1,344)	(3,248)	(6,514)	(13,548)	(33,007)	(54,389)	(75,558)	(95,146)	(41,902)	20,150	49,099	61,686	57,478	
Net Present Value		0%	57,478		IRR	26.5%										
		5%	39,214													
		10%	25,553													

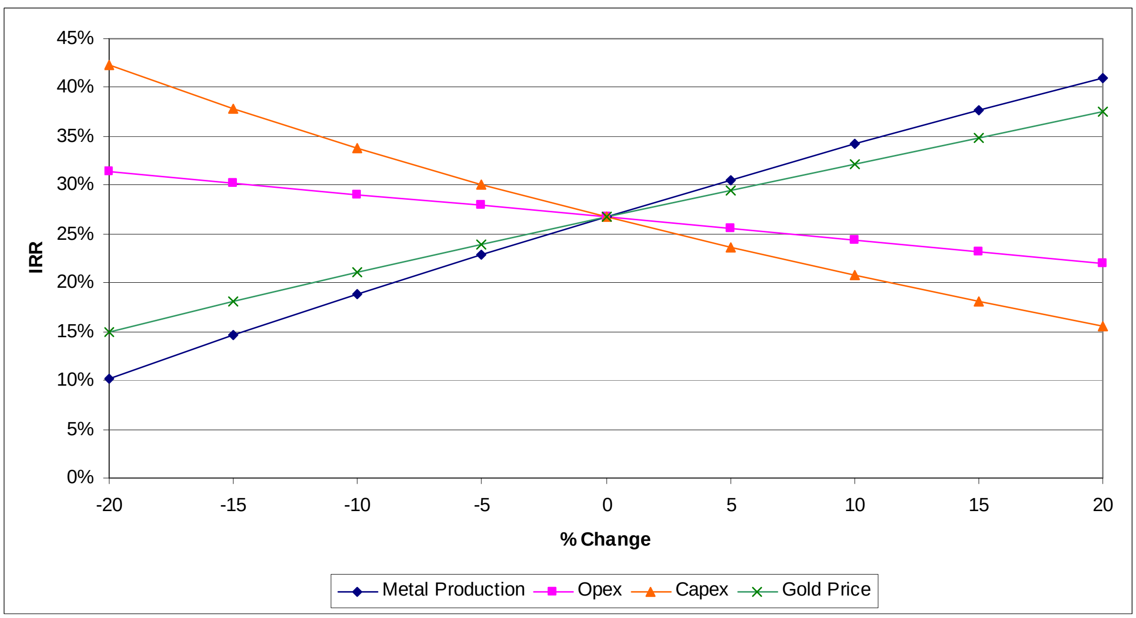
25.12.3 Sensitivity Analysis

A sensitivity analysis was performed for a variety of key parameters and cost elements, to examine the impact of variations within the range of ±20%. The key parameters include grade, which is shown as metal production, the price of gold, the price of silver, capital cost, and operating cost. The results of this analysis are summarized in Table 25-71 and shown graphically in Figure 25-20.

Table 25-70: Sensitivity Analysis - IRR

Item	% Change								
	-20	-15	-10	-5	0	5	10	15	20
Metal Prod'n	9.9%	14.3%	18.6%	22.6%	26.5%	30.3%	34.0%	37.6%	40.9%
Opex	31.2%	30.0%	28.9%	27.7%	26.5%	25.3%	24.2%	22.9%	21.7%
Capex	42.1%	37.6%	33.6%	29.9%	26.5%	23.4%	20.5%	17.9%	15.4%
Gold Price	14.7%	17.8%	20.8%	23.7%	26.5%	29.3%	32.0%	34.7%	37.3%

Figure 25-20: Sensitivity Graph



25.12.4 Alternative Evaluations

25.12.4.1 Spot Price

The economics of the project were evaluated using the April 29, 2010, London Metals exchange (LME) morning fix for gold and silver. The prices were US$1,170 per ounce of gold and US$18.50 per ounce of silver. Using these values, the project has an IRR of 58.0% and a NPV @ 5% of US$109 million.

25.12.4.2 Three Year Average

The economics of the project were evaluated using the three year trailing average as of April 23, 2010. This average is based on the weekly averages as available from Bloomberg Financial Services. The prices were US$995 per ounce of gold and US$14.76 per ounce of silver. Using these values, the project has an IRR of 40.2% and a NPV @ 5% of US$68 million.

25.12.4.3 Summary – Alternative Evaluations

A summary of the alternative evaluations is shown in Table 25-72.

Table 25-71: Summary of Alternative Evaluations

	Gold (US$/oz)	Silver (US$/oz)	IRR	NPV @ 5% (US$ X 1,000)
Base Case	800	12.50	26.5%	39,200
Spot Price	1,170	18.50	58.0%	109,100
3-yr Average	955	14.76	40.2%	68,200

APPENDIX A

CERTIFICATE OF QUALIFIED PERSON

I, **Robert Michael (Mike) Robb**, of Albuquerque, New Mexico, USA, do hereby certify that:

1. I am a Mining Engineer and President of R R Engineering, residing at 6004 Buffalo Grass CT, NE, Albuquerque, New Mexico 87111USA.
2. This certificate applies to the technical report entitled "Technical Report for the San Luis Project Feasibility Study" dated June 4, 2010 (the Technical Report).
3. I am a graduate of the University of Idaho (1963) with a Bachelor of Science degree in Extractive Industries.
4. I am a Registered Professional Engineer (P. E.) in the States of Arizona (29812), Nevada (11986), New Mexico (8647), Maine (6676), and Virginia (19834).
5. I am a member in good standing of the Society of Mining Engineers (Member 2723030RM) and the Mining and Metallurgical Society of America (Member 01306QP). I have been certified as a Qualified Person (QP) by both organizations. I am a member at large of Professional Engineers Exam Committee of the Society of Mining Engineers.
6. I have over 40 years of experience in the mining industry and have held various technical, managerial and executive positions for mining companies in North and South America.
7. I visited the San Luis, Peru, project site on April 29, 30 and May 1, 2009.
8. I am independent of Reliant Ventures S.A.C., Silver Standard Resources Inc. and Esperanza Silver Corporation, applying all of the tests in Section 1.4 of NI 43-101.
9. I have read NI 43-101 and certify that by reason of education and experience that I am a Qualified Person as defined by this act and that this Report has been prepared in compliance with the act.
10. I am responsible for all of Items 1, 2, 3.1.4, , 3.3, 3.4, 3.6, 3.7.1, 3.7.7, 3.8, 3.10.6, 22.6, 25.4.1.1, 25.4.16, 2.4.17, 25.4.18, 25.8, 23, 25.6, 25.7, 25.9, 25.10.6, 25.10.7, 25.10.8, 25.12 and portions of Items 3.5, 4, 20, 21.7, 24, 25.10.1, 25.10.8.9, and 25.11.3 of the Technical Report.
11. I have no prior involvement with the San Luis, Peru, project.
12. As of date of this Certificate, to my knowledge, information, and belief, this Techncal Report contains all scientific and technical information that is require to be disclosed to prevent the Report from being misleading.
13. I consent to the filing of this with any securities regulatory authority, stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 4th day of June, 2010.

Signed and Sealed

"Robert Michael Robb"

Robert Michael (Mike) Robb, P.E.
President
RR Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, **Steve L. Milne**, P. E. do hereby certify that:

1. This certificate applies to the technical report entitled "Technical Report for the San Luis Project Feasibility Study" dated June 4, 2010 (the "Technical Report").
2. I am an Independent Mining Engineering Consultant, and a contributor to a Technical Report as a principal of Milne & Associates, Inc. residing at 1651 Calle El Cid, Tucson, AZ 85718 USA.
3. I graduated from the Colorado School of Mines with a degree of "Engineer of Mines".
4. I am a 50-year member of the Society for Mining, Metallurgical, and Exploration, Inc. (SME #2225200) and am a registered professional mining engineer in Arizona (#12111), Colorado (#25589)and Utah (#2886).
5. I have worked as a supervisor/mining engineer/mining consultant for a total of 50 years since my graduation from the Colorado School of Mines.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
7. I am responsible for editing and approving all of Items 3.2.2, 3.2.3, 3.7.3, 3.9.2, 3.10.2, 19.2, 21.2, 22.2, 25.1, 25.10.2, 25.10.7.1, 25.10.7.2, and 25.11.1 and portions of Items 3.5, 3.7.8, 4, 5, 20, 21.7, 24, 25.4.2.1, 25.4.3.4 and 25.10.1 of the Technical Report.
8. I have not personally visited the property, but have been in contact with several Qualified Persons associated with this report who have visited the property on several occasions.
9. I have not been involved with the preparation of other reports related to the Technical Report.
10. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I am independent of Reliant Ventures S.A.C., Silver Standard Resources Inc. and Esperanza Silver Corporation, applying all of the tests in Section 1.4 of NI 43-101.
12. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.

Dated this 4th day of June, 2010.

Signed and Sealed

"Steve L. Milne"

Steve L. Milne, P.E.
President
Milne & Associates, Inc.

CERTIFICATE OF QUALIFIED PERSON

I, **Michael J. Lechner**, of Stites, Idaho do hereby certify:

1. That I am an independent consultant and owner of Resource Modeling Incorporated, an Arizona Corporation with it's office located at 124 Lazy J Drive, Stites, ID 83552.
2. This certificate applies to the technical report entitled "Technical Report for the San Luis Project Feasibility Study" dated June 4, 2010 (the "Technical Report").
3. That I am a registered professional geologist in the State of Arizona (#37753), a P. Geo. with British Columbia (#155344) and a Certified Professional Geologist with the AIPG (#10690).
4. That I am a graduate of the University of Montana (1979) with a Bachelor of Arts degree in Geology.
5. That I have practiced my profession continuously since 1977.
6. That I have worked as an exploration geologist, mine geologist, Engineering Superintendent, resource modeler, and consultant on a wide variety of base and precious metal deposits throughout the world.
7. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education and professional registration (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101
8. That I, Michael J. Lechner, performed various statistical and geostatistical analyses of the drill hole data and independently estimated gold resources for the San Luis deposit. I'm responsible for all of Items 16, 17, and 19.1 and portions of Items 3.2.1, 3.7.2, 3.9.1, 3.10.1, 4, 5, 20, 21.1, 22.1, and 24.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. That I have written this report as an independent consulting geologist and have no material interest, direct or indirect, in the property discussed in this Technical Report and have not had any prior involvement with this property prior to working with Silver Standard Resources and Esperanza Silver Corporation.
11. I have read NI 43-101 and fully believe that this report has been written in complete compliance with NI 43-101.
12. I am independent of Silver Standard Resources Inc., Reliant Ventures S.A.C., and Esperanza Silver Corporation, applying all of the tests in Section 1.4 of NI 43-101.

Dated this 4th day of June, 2010.

Signed and Sealed

"Michael J. Lechner"

Michael J. Lechner, P. Geo
President
Resource Modeling Incorporated

CERTIFICATE OF QUALIFIED PERSON

I, **Donald F. Earnest**, P.G. do hereby certify that:

1. I am a Mining Geologist and President of Resource Evaluation Inc., residing at 11830 N. Joi Drive, Tucson, Arizona 85737 USA.
2. This certificate applies to the technical report entitled “Technical Report for the San Luis Project Feasibility Study” dated June 4th, 2010 (the “Technical Report”).
3. I am a graduate with a Bachelor of Science, Geology degree from The Ohio State University, 1973.
4. I am a Registered Professional Geologist (P.G.) in the States of Arizona (#36976) and Idaho (#746), and a member of the Society of Mining Engineers (SME).
5. I have 35 years experience in mining and exploration geology, mineral resource and mineral reserve estimation, mine management, and consulting, which includes over 20 years directly related to vein-hosted precious metal deposits.
6. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education and professional registration (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
7. I am responsible for preparation of all of Items 6.1, 6.4, 7.1, 7.2, 7.5, 8, 9, 10, 11, 12, 13, 14, 15 and portions of Items 3.1, 3.2, 3.7, 3.9, 3.10, 4, 6.2, 6.5, 7.4, 20, 21.1, 22.1, and 24, of the Technical Report.
8. I visited the San Luis project site on June 25 and 26, 2009.
9. I have not had prior involvement with the San Luis project that is the subject of the Technical Report.
10. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I am independent of Reliant Ventures S.A.C., Silver Standard Resources Inc. and Esperanza Silver Corporation, applying all of the tests in Section 1.4 of NI 43-101.
12. I have read NI 43-101 and the Technical Report has been prepared in accordance with NI 43-101.

Dated this 4th day of June, 2010.

Signed and Sealed

“Donald F. Earnest”

Donald F. Earnest, P.G.
President
Resource Evaluation Inc.

CERTIFICATE OF QUALIFIED PERSON

I, **Clinton Strachan**, Principal Geotechnical Engineer of Fort Collins, Colorado USA do hereby certify that:

1. I am a Principal Geotechnical Engineer with Montgomery Watson Harza (MWH) Americas Inc. with a business address at 2629 JFK Parkway, Suite 206, Fort Collins, Colorado USA.
2. This certificate applies to the technical report entitled "Technical Report for the San Luis Project Feasibility Study" dated June 4, 2010 (the "Technical Report").
3. I am a graduate of Colorado State University (BS, Agricultural Engineering in 1972; MS, Civil Engineering 1979).
4. I am a registered professional engineer in Colorado (License # 19302) as well as Arizona, Idaho, Montana, Nevada, New Mexico, Oklahoma, Texas, Washington, and Wyoming. I am a member of the U.S Society on Dams, the American Society of Civil Engineers, and the Society of Mining Engineers.
5. I have practiced my profession continuously for over 30 years since my graduation from university.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
7. My relevant experience with respect to geotechnical engineering and mine tailings facilities includes over 30 years of continuous work in the discipline.
8. I am responsible for the preparation of all of Items 3.2.6, 3.2.7, 3.2.9, 3.7.5, 3.7.6, 3.9.4, 3.9.5, 3.10.4, 3.10.5, 21.4, 21.5, 22.4, 22.5, 25.3, 25.4.2.2, 25.4.3, 25.10.7.6, 25.10.7.7, 25.10.7.8, 25.10.3 and 25.10.5 and portions of Items 3.5, 3.7.8, 4, 5, 20, 21.7, 24, 25.4.2.1, 25.4.3.4, and 25.10.1.
9. I visited the San Luis Property during the period June 12-13, 2009.
10. I have had no prior involvement with the Property that is the subject of the Technical Report.
11. As of the date of this Certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12. I am independent of Reliant Ventures S.A.C., Silver Standard Resources Inc. and Esperanza Silver Corporation, applying all of the tests in Section 1.4 of NI 43-101.
13. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.

Dated this 4th day of June, 2010.

Signed and Sealed

"Clinton Strachan"

Clinton Strachan, P.E.
Principal Geotechnical Engineer

CERTIFICATE OF QUALIFIED PERSON

I, **Christopher Edward Kaye** do hereby certify that:

1. I am a Principal Process Engineer, with the firm of Mine and Quarry Engineering Services, Inc. (MQes) of 1730 S. Amphlett Blvd. Suite 200, San Mateo, CA 94402, USA. I carried out this assignment for MQes.
2. This certificate applies to the technical report entitled "Technical Report for the San Luis Project Feasibility Study" dated June 4, 2010 (the "Technical Report").
3. I am a member of Australasian Institute of Mining and Metallurgy in Australia. I graduated from the University of Queensland, Australia, with a B. Eng. in Chemical Engineering in 1984.
4. I have worked as a process engineer in the minerals industry for over 20 years. I have been directly involved in the mining, exploration and evaluation of mineral properties internationally for gold and base metals.
5. I have not personally visited the San Luis project site.
6. I am responsible for the preparation of all of Items 3.2.4, 3.2.5, 3.2.8, 3.2.10, 3.7.4, 3.9.3, 3.9.6, , 3.10.3, 18, 21.3, 21.6, 22.3, 25.2, 25.4.1.2, 25.4.4, 25.4.5.1,25.4.5.2, 25.4.6, 25.4.7, 25.4.8, 25.4.9, 25.4.10, 25.4.11, 25.4.12, 25.4.13, 25.4.14, 25.4.15, 25.4.19, 25.10.3, 25.10.7.3, 25.10.7.4, 25.10.7.5, and 25.11.2, and portions of Items 3.5, 3.7.8, 3.9.7, 4, 5, 20, 21.7, 24, 25.4.2.1, 25.4.3.4, 25.5, 25.10.1, 25.11.3 of Technical Report.
7. I am independent of Silver Standard Resources Inc., Reliant Ventures S.A.C., and Esperanza Silver Corporation as independence is defined by Section 1.4 of National Instrument 43-101 ("NI 43-101").
8. I have had no prior involvement with the San Luis Property that is the subject of this Technical Report.
9. I have read NI 43-101 and, by reason of education and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101. This Technical Report has been prepared in compliance with NI 43-101.
10. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 4th day of June, 2010.

Signed

"Christopher Edward Kaye"

Christopher Edward Kaye, MAusIMM
Principal Process Engineer
Mine and Quarry Engineering Services, Inc.