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Escobal Guatemala Project

NI 43-101 Preliminary Economic Assessment

Southeastern Guatemala

REVISION 0
Prepared For:

M3 Engineering & Technology Corporation ● 2051 West Sunset Road, Tucson, AZ 85704 ● 520.293.1488

DATE AND SIGNATURES PAGE

This report is current as of 07 May 2012. The effective date of the Mineral Resource estimate is 23 January, 2012. See Appendix A, PEA Contributors and Professional Qualifications, for certificates of qualified persons. These certificates are considered the date and signature of this report in accordance with Form 43-101F1.

ESCOBAL GUATEMALA PROJECT

FORM 43-101F1 TECHNICAL REPORT

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1 SUMMARY

1.1 PRINCIPAL FINDINGS

Exploration work since 2010 has resulted in significant increase in the mineral resources of the Escobal site, leading to a new Preliminary Economic Assessment ("PEA") to analyze increased mine and plant throughput associated with extraction of the additional resources. The new assessment indicates that throughput increases from 3,500 metric tonnes per day (MTPD) to 4,500 MTPD and/or 5,500 MTPD will improve economic results.

- Indicated mineral resources for the Escobal deposit are 27.1 million tonnes grading 422 g/t Ag, 0.43 g/t Au, 0.71% Pb and 1.28% Zn at a cut-off grade of 150 g/t silver-equivalent, which represents a 50% increase in Indicated silver ounces as compared to the Indicated silver resources reported in November 2010. Indicated silver-equivalent ounces now total 429.7 million.

- Inferred mineral resources for the Escobal deposit are 4.6 million tonnes grading 254 g/t Ag, 0.59 g/t Au, 0.34% Pb and 0.66% Zn at a cut-off grade of 150 g/t silver-equivalent. Inferred silver-equivalent ounces total 44.7 million.

- Although the Escobal mineral resource has increased significantly since 2010, the Escobal deposit has not been fully delineated and remains open along strike and down dip.

- The previous Preliminary Economic Assessment (November 2010) indicated that a 3,500 MTPD underground mine producing lead and zinc concentrates over a production life of 18 years is economically viable. The new Preliminary Economic Assessment demonstrates that increasing the throughput to 4,500 MTPD and/or 5,500 MTPD will enhance the economic results over the previous plan. The contemplated operations would provide direct employment of approximately 650 employees in Guatemala.

- Metallurgical studies to-date continue to confirm that processing of the indicated and inferred resources through differential sulfide flotation will produce marketable lead and zinc concentrates. Process recovery rates are expected to average 87% for silver, 75% for gold, and 83% for lead and zinc.

- Production in years 1 through 10 would average more than 20 million ounces of silver per year at total cash cost of less than $5.00 U.S. (net of by-product credits) at the base case metal prices used in the study ($25.00/oz Ag, $1300/oz Au, $0.95/lb Pb, $.090/lb Zn). In both expansion cases, production life would increase to 19 years, approximately one year longer than the 3,500 MTPD case contemplated in the previous PEA.

- After tax net present value for the 4,500 MTPD case at a 5% discount rate is $2.94 billion at the base case metal prices, with an after tax internal rate of return (IRR) of 68.3% on an initial capital cost of $372.8 million.

- After tax net present value for the 5,500 MTPD case at a 5% discount rate is $2.99 billion at the base case metal prices, with an after tax internal rate of return (IRR) of 68.5% on initial capital costs of $405.4 million.

- Significant exploration upside remains in the Escobal vein trend and in other regional exploration opportunities.

- Based on financial and technical measures, exploration work and project advances to date, M3 recommends that Tahoe complete the detailed engineering and development of the Escobal project and begin taking steps to increase mine and mill capacity to 4,500 MTPD.

- Based on these same factors, positive economic benefits may be realized from expansion above 4,500 MTPD. M3 recommends that Tahoe continue to explore adjacent to the known Escobal mineral resources and advance detailed engineering to further define and optimize potential mine and plant capacity beyond 4,500 MTPD.

This PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. The basis for the PEA is the Indicated mineral resources and Inferred mineral resources as reported herein. No pre-feasibility or feasibility study has been carried out with respect to the Escobal project.

1.2 PROPERTY DESCRIPTION AND OWNERSHIP

Escobal is located in southeast Guatemala, approximately 40 kilometers east-southeast of Guatemala City and three kilometers east of the town of San Rafael Las Flores in the Department of Santa Rosa. San Rafael Las Flores has a population of 3,500 people and is 70 km from Guatemala City by paved road. Access to the area is also possible from the northeast on a paved highway via the town of Mataquescuintla. The Company is not aware of any significant indigenous population residing in the area of the Escobal Project. According to Guatemala's National Institute of Statistics (Census 2002) San Rafael Las Flores' population is 99.6% "Ladino", i.e., of Hispanic origin and non-indigenous.

The local climate consists of two major seasons; a "rainy" season between May and November and a "dry" season between November and May. Annual precipitation averages 1,689 mm. Average temperatures vary between 14°C and 33.1°C.

There is a high voltage electrical line extended to the town of San Rafael Las Flores, which will be upgraded to handle the anticipated load requirements for the Project. Telephone and internet services are currently available at the project site and nearby facilities in San Rafael Las Flores. There are water wells within the Project area that are capable of providing water for operations.

1.3 MINERAL TENURE, SURFACE RIGHTS, AND ROYALTIES

Tahoe Resources Inc. acquired the project from Goldcorp through a transaction completed in June, 2010. The Project comprises three exploration concessions, totaling approximately 129 km^2, called Oasis, Lucero and Andres, which were granted on March 26, 2007, August 21, 2007 and November 15, 2007, respectively to Entre Mares de Guatemala S.A., a wholly owned subsidiary of Goldcorp. The concessions are now controlled by Minera San Rafael S.A., a wholly owned subsidiary of Tahoe. The Oasis concession covers the entire Escobal vein and project area.

The first three-year term of the Oasis concession expired in March 2010; Tahoe renewed this concession in August 2010. Extension requests have been filed for Lucero and Andres with approvals pending. The concessions can be extended for an additional two-year term. There is a reasonable expectation that the extensions will be granted for the concessions.

Applications for additional exploration concessions have been submitted to the Guatemalan government and as of the effective date of the Report are pending approval.

Land surrounding the Project area is privately owned by local farmers and used for growing coffee in the higher elevations and vegetables and other crops in the flatter low lying areas. Tahoe/Minera San Rafael has acquired all project surface rights needed to support the areas required for operations, tailings, waste rock disposal, processing, and ancillary surface facilities as contemplated in this Preliminary Economic Assessment.

The current mineral royalty in Guatemala is 1% paid to the federal government, of which a portion is returned to the local community. In January 2012, the Guatemalan Ministry of Energy and Mines (MEM) and the Guatemalan Mining Association (GREMIAL) agreed to a voluntary royalty of 4% for precious metals and 3% for base metals. This voluntary royalty increase is reflected in the economic analysis included in this Preliminary Economic Assessment. In addition, a profit sharing program in the form of an NSR royalty of 0.5% will be paid to an Association of Land Owners.

1.4 PERMITS

The Escobal Project is currently in the exploration phase and exploration activities are permitted by both MEM and Ministry of Environment and Natural Resources (MARN). All required permits to continue surface and underground exploration are in place. The environmental requirements from MARN are specified in Resolution 4590-2008/ELER/CG dated December 23, 2008 that applies to exploration activities. License of rights was transferred from Entre Mares de Guatemala to Minera San Rafael as specified in Resolution 1918-2010/ECM/GB, dated September 3, 2010. An Environmental Impact Assessment (EIA) that addresses the environmental impacts associated with the ongoing exploration declines was submitted to MARN in November 2010 and an Environmental License submitted in March 2011. An appropriate level of public disclosure and involvement was required and developed at this stage. MARN accepted the work plan for the exploration declines on April 5, 2011, clearing the way for starting the underground exploration program. An Environmental Impact Study (EIS) that addresses the environmental impacts associated with exploitation of the mineral body was approved by MARN on October 21, 2011 by Resolution 3061-2011/DIGARN/ECM/beor. This resolution clears the way for construction of the mine, processing plant, and surface facilities required for exploitation. Minera San Rafael submitted the application to MEM for the Exploitation license, required for producing concentrates from the mine, in November 2011 and expects the license to be granted in the first half of 2012.

All other permits required for construction and operation have been obtained.

Permits to continue surface exploration activities are in place.

1.5 ENVIRONMENT

The mandate from Tahoe is to meet or exceed the standards of sustainability and environmental management based on North American practice and regulation. No impacted waters and materials will be directly discharged from the site. Impacted water will require lined containment and treatment prior to being released to the environment. The environmental management program will include the following:

- Dry stack tailings
- Lined storm water and waste facilities
- A concurrent reclaim program
- Process water recovery and recycling
- Process/contact water treatment systems
- Underground paste backfill

These environmental controls represent the state of the art in sustainable design.

1.6 GEOLOGY AND MINERALIZATION

The Escobal deposit is an intermediate-sulfidation fault-related vein formed within Tertiary sedimentary and volcanic rocks within the Caribbean plate. The Escobal vein system hosts silver, gold, lead and zinc, with an associated epithermal suite of elements, within quartz and quartz-carbonate veins. Quartz veins and stockwork up to 50 m wide, with up to 10% sulfides, form at the core of the Escobal deposit and grade outward through silicification, quartz-sericite, argillic and propylitic alteration zones.

Drilling to date has identified continuous precious and base metal mineralization at Escobal over 2,200 meters laterally and 1,000 meters vertically in four zones; the East, Central, West/Margarito and East Extension zones. The vein system is oriented generally east-west, with variable dips. The East and East Extension zones dip to the south from 60° to 75° to the south with recent drilling showing a change to vertical dip at depth. The majority of the mineralized structure(s) in the Central and Margarito zones dip from 60° to 75° to the north, steepening to near-vertical at depth. The upper eastern portion of the Central Zone dips 60° to 70° to the south as in the East Zone.

1.7 EXPLORATION STATUS

In 2011, exploration drilling at the Escobal project was designed to improve confidence in the established Indicated and Inferred mineral resources as well as to explore extensions of the deposit where it was open laterally and to depth. This program succeeded in expanding the November 2010 Indicated resource by 50% to 367.5 million silver ounces. Inferred resources now total 36.7 million silver ounces.

The Escobal deposit is open along strike in both directions and down dip. Tahoe is aggressively exploring for the continuation of mineralization at Escobal, with five drills currently active on the property. The drilling is designed to identify new areas of mineralization within the Escobal structure through wider and deeper extensional step-outs. This program will be carried out principally from the surface, with underground drilling to commence in mid-2012 once drill platforms are available in the ongoing exploration declines.

The Escobal vein is one of 14 vein showings identified in the district and the only vein system that has been adequately drilled to date. Using the geologic model developed at Escobal, prospective areas will continue to be evaluated throughout the region while a more concentrated effort will be made to upgrade and drill viable targets within Tahoe's currently held concession areas. In late 2011, drilling was expanded to evaluate other regional targets.

1.8 DRILLING

Drilling on the Oasis concession has been conducted by Entre Mares (Goldcorp) and Tahoe from 2007 to the present, using a combination of contracted and company-owned drills. A total of 381 drill holes (136,615 meters) have targeted the Escobal vein system and other veins within the concession. Data from drilling on the Escobal vein through December 2011 have been used for the resource model and estimate reported herein. With minor exception, all drilling at Escobal has been by diamond drill (core) methods, with the majority (66%) of mineralized intercepts drilled using NTW-size or larger drill core. Core recovery averages 96% over the life of the project.

1.9 SAMPLE PREPARATION AND ANALYSIS

BSI Inspectorate has been the primary analytical laboratory for all of the Escobal drill sample preparation and analysis, with only minor exceptions. All samples have been prepared and analyzed using industry-standard practices suitable for the mineralization at Escobal. Entre Mares and Tahoe conducted quality assurance and quality control (QA/QC) programs throughout all of the drill campaigns at Escobal, which included check assaying, duplicate sample assaying at other laboratories, and the use of blind assay standards and assay blanks.

The core sampling procedures, sample analyses, QA/QC procedures, and sample security have provided sample data that are of sufficient quality for use in the resource estimation.

1.10 DATA VERIFICATION

Data verification was supervised by Paul Tietz, CPG, of Mine Development Associates (Reno, Nevada USA). Mr. Tietz conducted site visits to the Escobal property in 2010 and 2012, which included verifying drill locations in the field, reviewing sample handling and data collection procedures, verifying downhole survey data, and independent verification sampling of drill core. MDA also completed a full audit of the project database, analysis of the QA/QC data, and study of core recovery and its relationship to metal grades. The results of this verification program support the estimation of the Escobal resource and the assignment of an Indicated classification to much of the stated resource.

1.11 MINERAL PROCESSING AND METALLURGICAL TESTING

McClelland Laboratories (McClelland) Sparks, Nevada, USA conducted the initial metallurgical tests in 2009 on three drill core samples from the Escobal mineral deposit. It was concluded from the results of the tests that a differential lead/zinc flotation producing a high value lead concentrate containing most of the silver and gold in the mineral resource and a salable lower value zinc concentrate was the optimum processing route.

In June 2010, FLSmidth Dawson Metallurgical Laboratories was contracted to conduct metallurgical testing on drill cores representative of the mineralization from the Escobal Project. The primary objective of the test program was to determine process design criteria for crushing, grinding and flotation for the Escobal sulfide deposit. Results of the differential flotation tests indicate that the Escobal sulfide mineralization will respond to widely used and proven mineral processing techniques. The test programs conducted to date show that good recoveries of gold, silver, lead and zinc and acceptable reagent consumptions can be obtained by using conventional lead zinc differential flotation process. Metallurgical testing of material from newly discovered extensions of the Escobal vein (i.e., East Extension and West/Margarito zones) has been initiated with results pending.

Flotation feed will consist of a primary grind size of 80% passing 105 µm in the rougher flotation circuits and regrind size of 80% passing 37 µm in the cleaner flotation circuits. Expected recoveries from the sulfide mineral processing are, 86.8% for silver, 75.1% for gold, 82.5% for lead, and 82.6% for zinc.

1.12 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

1.12.1 Mineral Resources

The mineral resource estimate for the Escobal deposit contains 367.5 million ounces of silver classified as Indicated resources and 36.7 million ounces of silver classified as Inferred resources, with significant amounts gold, lead, and zinc reported in both resource categories. A summary of the Indicated and Inferred resources, using a cutoff grade of 150 grams per tonne silver equivalent, is provided in the following table:

Table 1-1: Summary of Indicated and Inferred Resources

Resource Classification	Tonnes (M)	Silver (g/t)	Gold (g/t)	Lead (%)	Zinc (%)	Silver (Moz)	Gold (koz)	Lead (kt)	Zinc (kt)
Indicated	27.1	422	0.43	0.71	1.28	367.5	373	192	347
Inferred	4.6	254	0.59	0.34	0.66	36.7	85	15	30

Mineral resources at Escobal were estimated using approximately 22,900 samples obtained from 350 diamond drill core holes, totaling 121,639 meters. Mine Development Associates modeled and estimated the Escobal deposit resources by refining the geologic model, evaluating the drill data statistically, interpreting mineral domains on cross sections and level plans, analyzing the modeled mineralization statistically to establish estimation parameters, and estimating silver, lead, gold, and zinc grades into a three-dimensional block model using inverse distance cubed (ID^3**).**

Silver-equivalent Indicated resources total 429.7 million ounces at an average grade of 493 g/t and silver-equivalent Inferred resources total 44.7 million ounces at an average grade of 309 g/t. Silver-equivalent values for the resources were calculated using metal prices of $25.00/oz Ag, $1300/oz Au, $0.95/lb Pb, and $0.90/lb Zn. The effective date of the mineral resource estimate is 23 January 2012.

1.12.2 Mineral Reserves

There are no mineral reserves reported for the Escobal project.

1.13 MINING

Exploration work since completion of the November 2010 PEA has added significant high quality resources to the Escobal mineral inventory. The additions have prompted the need to evaluate the ability for expansion of the mine and process plant production capacity. With the additional resources the mine clearly has excess annual production capacity beyond that contemplated in the November 2010 PEA. In order to take advantage of that capacity, transverse long-hole stoping will replace longitudinal long-hole stoping in areas where the horizontal dimensions across the strike of the vein are greater than 15 meters. This will allow an increase in the number of active producing workplaces in the mine at any given time.

Approximately 15,000 meters of additional primary development ramps and 1,100 meters of raise will be required to access the new resource. Primary development has been accelerated compared to the previous plan in order to access the new resources and allow increased production. This will be accomplished through the addition of development crews and equipment rather than an increase in productivity. Footwall laterals in waste will be used to access stoping areas in lieu of the individual spiral ramps that were utilized in the earlier study for stope development. Two cases have been analyzed in this study; increasing mine production to 4,500 MTPD and capping it at that rate throughout the mine life, or increasing mine production to 4,500 MTPD, making major modifications to the process plant and then further increasing mine production to 5,500 MTPD for the remainder of the mine life. The mine plan for the two cases only differ in the timing of development and a slightly smaller equipment fleet for the 4,500 MTPD.

The Escobal deposit will be accessed through two main portals. Primary ramps will access the Central Zone and a third primary ramp will be driven into the East Zone from the West Central ramp. The three primary ramps will connect to a system of secondary footwall laterals spaced 25 meters vertically in the Central Zone and spiral access ramps in the East Zone. Stopes will be accessed from the secondary development openings. The primary and secondary development will be excavated at a maximum incline of 15%. The main access ramps are located nominally 75 and 150 meters from the vein and will be driven 5 meters wide by 6 meters high. Internal ventilation raises will be driven between the various ramps, footwall laterals, and accesses.

Filtered tails from the process plant will be combined with cement and water to make a structural fill for use underground. Backfill will be required for all stopes for stability reasons and as a preferred place to store tailings. A paste backfill plant located near the East portal will produce backfill for delivery via a system of steel and HDPE pipe into the mine for placement in the mined out stopes. Stope production will be hauled directly from the stopes to the process plant by truck and development waste will be placed in mined stopes where possible, or trucked to a surface waste dump facility. The mine plan contemplates a network of infrastructure to dewater the mine, supply electrical power, fresh water for operations and dust control, compressed air and communication systems. The mine is expected to deliver a total of 29.8 million tonnes at average grades of 383 g/t silver, 0.38 g/t gold, 0.62% lead, and 1.10% zinc to the mill for processing. This total includes dilution of 4.7 million tonnes at an average grade of 71 g/t silver, 0.12 g/t gold, 0.10% lead, and 0.22% zinc. A mine wide cut-off value of 150 g/t equivalent silver has been determined as optimal for the operation and approximately 95% of the resource above this cut-off is extracted in the mine plan.

1.14 PROCESS FLOWSHEET

Mineralized rock will be transported from the underground mine to a run of mine stockpile from where it will be transported via front end loader or trucks to the processing facility. Mineral concentrates of gold, silver, lead and zinc will be produced by mineral flotation technology. The sulfide concentrator will consist of a three stage crushing circuit followed by one ball mill. This will be followed by a conventional lead zinc differential flotation circuit consisting of tank cells with separate circuits for lead and zinc. Lead and zinc concentrates produced at the concentrator facility will be packaged and loaded onto trucks for shipment to concentrate smelters and metal refineries.

The design basis for the processing facility is 4,500/5,500 dry metric tonnes per day or 1,642,500/2,007,500 dry tonnes per year. Sulfide and mixed oxide-sulfide resources (diluted) are available for approximately 19 years at an average silver grade of 383 g/t, gold grade of 0.38 g/t, lead grade of 0.62% and zinc grade of 1.1%.

1.15 TAILINGS AND WASTE ROCK FACILITY

The above ground disposal of tailings will be "dry stacked". Tailings that are thickened and filtered to 10% to 15% moisture content are commonly termed dry tailings. Benefits include a reduced footprint and the water removed from the tailing is returned to the process stream, providing a direct effect to make-up water. M3 anticipates approximately 45% of the tailings will be dry stacked, with the remainder of the tailings returned underground as paste fill. In addition, dewatering for reclamation and closure is completed concurrent with operations greatly simplifying environmentally sound closure of the facility.

1.16 INFRASTRUCTURE

The project is approximately 2 kilometers from San Rafael Las Flores, a town of 3,500 people, and approximately 70 kilometers by paved highway from Guatemala City. All year access to the area is good via paved highway from Guatemala City.

Electrical power will be provided to the project from the existing Guatemala national grid by means of connecting to the existing San Rafael Las Flores substation at a voltage level of 69 kV, and bringing a new 7 km long 69 kV line to site. Additional power required for the expansion cases as well as for peak use demand periods in the 3,500 MTPD case will require on-site generation. Current plans indicate that the national grid will in the future complete a closed loop to the project rather than the current radial feed. However, power requirements beyond those currently assigned to the property have been envisioned and estimated as on-site generators with diesel being the fuel.

Hydrological studies indicate that sufficient water will be available to supply process and potable requirements for the project.

1.17 TRANSPORTATION AND LOGISTICS

The major process and mining equipment will be procured overseas and shipped to Guatemala. No special handling requirements are foreseen, and normal shipping routes and ships can be utilized. Logistics to date has not proven to be problematic.

Guatemala has ports on both the Pacific and the Caribbean coasts. Access to the mine site from both ports is on paved highway.

Filtered concentrate will be placed in 1,000 to 2,000 pound super-sacks, placed in sea-going containers, and carried on highway tractor trailer units along paved highway to either port for shipment to international smelters.

1.18 RECLAMATION

The entire facility will be designed with closure in mind, to the greatest extent practicable. The facilities will be designed and operated to minimize the footprints and areas of disturbance and to utilize the most advanced planning and reclamation techniques available including dry stack tailings, concurrent reclamation and geomorphic landform grading.

Surface disturbance of this underground mine will be small as all mining activity will be underground. Reclamation will commence as soon as is practical during the development and operations by placing salvaged topsoil on outslopes and encouraging vegetation. Final reclamation of the top surface will occur at final closure at the end of mine life.

1.19 OPERATING COST ESTIMATE

The operating costs for the 4,500 MTPD and the 5,500 MTPD cases were calculated for each year during the life of the mine using the annual production tonnage as a basis. Table 1-2 reflects the approximate production of zinc and lead concentrates and metal contained in each concentrate.

Table 1-2: Approximate Concentrate Production and Content

4,500 MTPD Case	Tonnes (000's)	Zinc (klbs.)	Silver (kozs.)	Gold (kozs.)
Zinc Concentrate	515	596,372	15,807	15
Lead Concentrate	299	336,480	303,275	258
5,500 MTPD Case				
Zinc Concentrate	519	600,758	15,830	15
Lead Concentrate	300	337,734	303,708	258

Table 1-3 shows the unit cost per tonne for the life of the mine for both cases.

Table 1-3: Operating Costs by Area

	4,500 MTPD	5,500 MTPD
Life of Mine	$/tonne	$/tonne
Ore Tonnes	29,826,845	29,924,285
Mining Operations	$29.03	$27.22
Mill Operations		
Crushing & Conveying	$1.95	$1.85
Grinding & Classification	$5.06	$6.29
Flotation and Regrind	$4.20	$4.19
Concentrate Dewatering, Filtration & Dewatering	$1.05	$0.99
Tailing Disposal	$5.26	$4.92
Laboratory	$0.52	$0.50
Ancillary Services	$1.50	$1.42
Subtotal Processing	$19.54	$20.16
Supporting Facilities		
General and Administrative	$6.67	$6.87
Subtotal Supporting Facilities	$6.67	$6.87
Total Operating Cost	$55.24	$54.25

1.20 CAPITAL COST ESTIMATE

All costs for the options presented in this report are in addition to the costs of the 3,500 MTPD plant, which is currently under development. Table 1-4 shows a summary of budgeted initial capital expenses for the original plant.

Table 1-4: Initial Capital Cost Control Budget (3,500 MTPD)

Description	Cost
Direct Costs	
General Site	$14,045,472
Mine, West Portal – By Owner	$0
Mine, East Portal – By Owner	$0
Primary Crushing	$4,098,061
Secondary & Tertiary Crushing	$5,737,647
Fine Ore Storage & Reclaim	$5,775,776
Grinding	$15,074,923
Flotation & Regrind	$20,755,307
Reagents	$3,734,928
Concentrate	$9,613,887
Tailing Dewatering	$16,990,634
Paste Backfill Plant	$6,723,845
Tailing Dry Stack	$2,073,044
Water Systems and Well Field	$10,313,094
Sewage Treatment	$639,615
Main Substation	$6,065,325
Overhead Power Line	$2,402,365
Ancillaries	$20,893,022
Insurance/Capital Spares	$2,000,000
Freight	$10,388,696
Duties	$3,001,033
Subtotal DIRECT COST	**$160,326,674**
Indirect Costs	
CONTINGENCY	$26,646,797
Other Indirects Including EPCM,	$30,040,626
Contractor Power, Vendor Supervision	
and Commissioning	
IVA @ 12% (Eventually Refundable)	$10,697,853
TOTAL EPCM CAPITAL COST	**$227,711,950**
TOTAL MINE CAPITAL COST	**$78,494,050**
OWNER'S COST	**$20,443,000**
TOTAL	**$326,649,000**

As of March 31, 2012, this project is under development. The project has committed $100,467,621 (i.e. 44%) of the Engineering, Procurement, and Construction Management (EPCM) budget of $227,711,950 and has used $6,116,714 (i.e. 23%) of the contingency budget of $26,646,797 allotted for the project. The Owner has committed $68,511,671 (69%) of the Mine Development and Owners budget of $98,937,988. The owner has not yet used any contingency but has allocated all of the $13.9 million of the contingency budgeted to underground development and purchases of mining equipment.

The capital costs for the option to increase production to 4,500 MTPD are as follows:

Table 1-5: Capital Cost Estimate for the 4,500 MTPD Expansion:

Total Costs for the 3,500 MTPD Project	**$326,649,000**
Additional Costs for the Expansion to 4,500 MPTD from 3,500 MTPD	
Mine Expansion Costs	$28,101,501
Direct Costs	$12,710,911
Indirect Costs	$5,323,588
Total Plant Expansion Costs	$18,034,499
Grand Total	**$372,785,000**

The 4,500 MTPD expansion will cost $18,034,499 in plant expansion and $28,101,501 in mine development and equipment in addition to the costs for the 3,500 MTPD project, for a total of $372,785,000.

The capital costs for the option to increase production to 5,500 MTPD are as follows:

Table 1-6: Capital Cost Estimate for the 5,500 MTPD Expansion

Total Costs for the 3,500 MTPD Project	**$326,649,000**
Additional Costs for the Expansion to 5,500 MPTD from 3,500 MTPD	
Mine Expansion Costs	$28,101,501
Direct Costs	$35,147,120
Indirect Costs	$15,015,844
Total	$50,162,964
Grand Total	**$405,413,465**

The 5,500 MTPD expansion will cost $50,162,964 for plant expansion and $28,601,501 in mine development and equipment costs in addition to the costs for the 3,500 MTPD project, for a total of $405,413,465.

1.21 FINANCIAL ANALYSIS

The 4,500 MTPD case economic analysis indicates that the project has an $NPV_{5\%}$ of \$2.94 billion and an Internal Rate of Return (IRR) of 68.3% with a payback period of 1.5 years.

The 5,500 MTPD case economic analysis indicates that the project has an $NPV_{5\%}$ of \$2.99 billion and an Internal Rate of Return (IRR) of 68.5% with a payback period of 1.5 years.

Table 1-8 (4,500 MTPD) and Table 1-9 (5,500 MTPD) compare the base case project financial indicators with the financial indicators for other cases when the sales price, the amount of capital expenditure, and operating cost are varied from the base case values by 10% and 20% while metal recoveries are varied by 1% and 2%. Two additional cases evaluate the sensitivity of the project to metal prices. The prices used in those cases are shown below. By comparing the results of this sensitivity study, it can be seen that the project IRR is most sensitive to metal price and capital cost.

Table 1-7: High/Low Metal Price

The High Metal Price Case was calculated using the following prices:	The Low Metal Price Case was calculated using the following prices:
Ag - \$35	Ag - \$18
Au - \$1800	Au - \$1100
Pb - \$0.95	Pb - \$0.95
Zn - \$0.90	Zn - \$0.90

Table 1-8: 4500 MTPD Case – Sensitivity Analysis

Sensitivities - After Taxes					
Change in Metal Prices	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
20%	$6,339,750	$3,902,357	$2,568,846	83.6%	0.9
10%	$5,576,018	$3,420,385	$2,240,478	76.0%	1.1
0%	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
-10%	$4,048,553	$2,456,439	$1,583,741	59.8%	1.4
-20%	$3,284,821	$1,974,466	$1,255,373	51.2%	1.7
Change in Operating Cost	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
20%	$4,482,749	$2,736,312	$1,777,190	64.5%	1.3
10%	$4,647,517	$2,837,362	$1,844,650	66.3%	1.3
0%	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
-10%	$4,977,054	$3,039,462	$1,979,569	69.8%	1.2
-20%	$5,141,822	$3,140,512	$2,047,029	71.6%	1.1
Change in Initial Capital	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
20%	$4,760,314	$2,887,240	$1,861,664	60.2%	1.4
10%	$4,786,300	$2,912,826	$1,886,887	63.9%	1.3
0%	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
-10%	$4,838,271	$2,963,998	$1,937,332	72.9%	1.1
-20%	$4,864,257	$2,989,584	$1,962,555	78.7%	1.0
Change in Recovery	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
2.0%	$4,950,739	$3,025,832	$1,971,676	69.5%	1.2
1.0%	$4,881,512	$2,982,122	$1,941,893	68.8%	1.2
0.0%	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
-1.0%	$4,743,059	$2,894,702	$1,882,326	67.3%	1.2
-2.0%	$4,673,832	$2,850,992	$1,852,543	66.6%	1.3

Table 1-9: 5500 MTPD Case - Sensitivity Analysis

Sensitivities - After Taxes					
Change in Metal Prices	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
20%	$6,345,786	$3,963,392	$2,627,389	83.7%	1.0
10%	$5,580,592	$3,474,281	$2,292,385	76.1%	1.1
0%	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
-10%	$4,050,205	$2,496,061	$1,622,378	60.2%	1.4
-20%	$3,285,011	$2,006,950	$1,287,374	51.6%	1.7
Change in Operating Cost	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
20%	$4,490,725	$2,784,266	$1,822,839	64.9%	1.3
10%	$4,653,062	$2,884,719	$1,890,110	66.6%	1.3
0%	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
-10%	$4,977,736	$3,085,623	$2,024,653	70.0%	1.2
-20%	$5,140,073	$3,186,075	$2,091,924	71.7%	1.2
Change in Initial Capital	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
20%	$4,763,427	$2,933,999	$1,906,936	60.5%	1.4
10%	$4,789,413	$2,959,585	$1,932,159	64.1%	1.3
0%	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
-10%	$4,841,384	$3,010,757	$1,982,604	73.1%	1.2
-20%	$4,867,370	$3,036,343	$2,007,827	78.8%	1.1
Change in Recovery	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
2.0%	$4,954,100	$3,073,868	$2,018,138	69.7%	1.2
1.0%	$4,884,749	$3,029,520	$1,987,760	69.0%	1.2
0.0%	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
-1.0%	$4,746,048	$2,940,822	$1,927,003	67.6%	1.3
-2.0%	$4,676,698	$2,896,473	$1,896,625	66.9%	1.3

1.22 CONCLUSIONS AND RECOMMENDATIONS

The results of this PEA demonstrate that:

1. An economically viable and environmentally suitable underground mining operation can be designed and constructed at the Escobal project.

2. Based on this analysis, M3 recommends Tahoe continue to advance the project.

3. Additional underground exploration, work to finalize project permitting for exploitation, and detailed engineering and design for feasibility of the expansion cases is recommended.

4. In addition, M3 recommends Tahoe continue to explore adjacent to the known Escobal mineral resources and advance detailed engineering to further define and optimize potential mine and plant capacity beyond 4,500 MTPD.

2 INTRODUCTION

2.1 PURPOSE AND BASIS OF REPORT

M3 Engineering & Technology Corporation ("M3") of Tucson, Arizona in the USA was commissioned by Tahoe Resources Inc. ("Tahoe") to provide an independent Qualified Person's Review and Technical Report (the "Report") for the potential expansion of the Escobal Project in Guatemala (the "Project"). This review is warranted by the substantial increase in resources resulting from continued exploration since the previous Preliminary Economic Assessment (PEA) in November 2010. The Project location is shown in Figure 2-1.

Tahoe is the sole proprietor of the Project through its subsidiary, Minera San Rafael, S.A. ("MSR"). The Project is comprised of three exploration concessions, 129 km^2 (129,000 ha) called Oasis, Lucero and Andres, granted on March 26, 2007, August 21, 2007 and November 15, 2007 respectively. The Oasis concession covers the entire Escobal vein.

This Report uses metric measurements, except where noted. The currency used in the Report is U.S. dollars. The local currency of Guatemala is the Quetzal. At the Report effective date, the exchange rate was US$1 equals 8.00 Quetzals.

2.2 SOURCES OF INFORMATION

Tahoe previously filed a Technical Report on the Escobal project entitled "Escobal Project Guatemala NI 43-101 Technical Report" dated 30 April 2010. This report was prepared by AMEC Americas Limited of Vancouver, Canada under the guidance of Mr. Greg Kulla, P. Geo., a Qualified Person (QP) as defined by NI 43-101.

Tahoe issued a second Technical Report on the Escobal project entitled "Escobal Guatemala Project NI 43-101 Preliminary Economic Assessment" dated 29 November 2010. This report was prepared by M3 under the guidance of Mr. Conrad Huss, P.E., a QP as defined by NI 43-101. The November 2010 PEA reported an increase in the mineral resources of the Project and provided technical and economic analyses of the potential viability of those mineral resources.

Additional information was obtained by M3 or provided by Tahoe, and is contained herein.

2.3 QUALIFIED PERSONS AND SITE VISITS

The Qualified Person and Principal author for this report is Conrad Huss, P.E., of M3 Engineering & Technology Corporation. All M3 personnel for this project are supervised by Conrad Huss. Mr. Huss visited the Project site on 1 December 2010.

The Qualified Person responsible for the review of the civil and environmental controls for the Escobal project is Daniel Roth, PE, of M3 Engineering & Technology Corporation. Mr. Roth visited the Escobal project site on numerous occasions in 2010, 2011, and 2012.

The Qualified Person responsible for the review of the metallurgical testing and flow sheets for the Escobal project is Thomas L. Drielick, PE, of M3 Engineering & Technology Corporation.

Other M3 staff members that have visited the project site include:

Randy Hensley – Construction Manager
Alberto Bennett – Electrical Engineer
Lorena Montano – Environmental

The Qualified Person responsible for the review of the drilling, sampling method, sample preparation and analysis, data verification, and resource estimate for the Escobal project is Paul Tietz, CPG, of Mine Development Associates, an independent mining consulting firm. Mr. Tietz visited the Escobal project site in September 2010 and February 2012.

Figure 2-1: Project Location Map

2.4 EFFECTIVE DATES

The effective date of the Report is 07 May 2012; the effective date of the Escobal resource estimate is 23 January 2012.

Tahoe's exploration drilling program is ongoing as of the effective dates of the Report and resource estimate. Where applicable, results received to date from this recent drilling generally corroborate the updated resource model.

There were no material changes to the information on the Project between the effective date and the signature date of the Report.

2.5 UNITS AND ABBREVIATIONS

The report considers US Dollars ($) only. Unless otherwise noted, all units are metric. However, as noted and as standard for projects of this nature, certain statistics are reported as avoirdupois or English units, grades are described in terms of percent (%), grams per metric tonne (g/tonne or g/t) or troy ounces per short ton (oz/t),. Salable base metals are described in terms of metric tonnes, English pounds. Salable precious metals are described in terms of troy ounces.

The following abbreviations are used in this report.

Table 2-1: Terms and Abbreviations

Abbreviation	Unit or Term
AA	Atomic Adsorption
Ag	Silver
AG	Autogenous Grinding
AT	Assay Ton
Au	Gold
cfm	Cubic feet per minute
CO_3	Carbonate
COG	Cutoff grade
Cu	Copper
CV	Coefficient of Variation (standard deviation/mean)
dba	doing business as
DDH	Diamond Drill Hole
FA	Fire Assay
g/tonne or g/t	grams per metric tonne
GPS	Global Positioning System
HP / hp	Horsepower
ICP	Inductively-Coupled Plasma
IRR	Internal Rate of Return
kg	Kilograms
km	Kilometer
k	Thousands
kPa	Kilopascal
kW-h	Kilowatt-hour
L	Liters
LOM	Life of Mine
Ma	Million years old
MDA	Mine Development Associates
Mn	Manganese
MTPD	Metric Tonnes per Day
MY	Million years old
NPV	Net Present Value

Abbreviation	Unit or Term
NSR	Net Smelter Return
opt	Troy ounces per English ton
oz/t	troy ounce per short ton
Pb	Lead
PSD	Particle Size Distribution
ppm	Part per million
%	Percent by weight
QA/QC	Quality Assurance/Quality Control
RC	Reverse Circulation
tpa	Tonnes per annum
tpy	Tonnes per year
tpd	Tonnes per day
US$ / USD	United States Dollars
XRD	X-Ray Diffraction
XRF	X-Ray Fluorescence
Zn	Zinc
2-D	Two-Dimensional
3-D	Three-Dimensional
4WD	Four-Wheel Drive

3 RELIANCE ON OTHER EXPERTS

The QP as author of this Report states that he is a qualified person for the Report as identified in the “Certificate of Qualified Person” attached to the Report. The author has relied upon and disclaims responsibility for information derived from the following expert reports pertaining to mineral rights, surface rights, and permitting issues.

In cases where the M3 PEA author, Conrad Huss, P.E., Qualified Person, has relied on contributions of the Qualified Persons listed in Appendix A, the conclusions and recommendations are exclusively the Qualified Persons’ own. The results and opinions outlined in this report that are dependent on information provided by Qualified Persons outside the employ of M3 are assumed to be current, accurate and complete as of the date of this report.

Reports received from other experts have been reviewed for factual errors by Tahoe and M3. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statements and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of these reports.

Metallurgical testing done by Tahoe’s consultants depends on the samples’ accuracy representing the Escobal deposit.

The base case metal prices utilized herein were provided by M3.

Mining is a business inherent with risk. The risk must be borne by the Owner. M3 does not assume any liability other than performing this technical study to normal professional standards.

3.1 MINERAL TENURE

M3 has not examined mineral tenure, nor independently verified the legal status or ownership of the Project area or underlying property agreements. M3 has fully relied upon independent legal experts for this information through the following documents:

- Arenales & Skinner-Klee, 2010: unpublished legal opinion letter prepared by Arenales & Skinner-Klee for Entre Mares, S.A., 23 February, 2010.
- Arenales & Skinner-Klee, 2010: unpublished legal opinion letter prepared by Arenales & Skinner-Klee for Entre Mares, S.A., 21 May, 2010.
- AMEC, 2010: Escobal Project Guatemala NI 43-101 Technical Report, prepared by Mr. Greg Kulla, 30 April, 2010

Data and information derived from work done by previous owners of Escobal and more recent work by Tahoe Resources Inc.

3.2 SURFACE RIGHTS, ACCESS, AND PERMITTING

M3 has fully relied on information regarding the status of the current Surface Rights, Road Access and Permits through opinions and data supplied by independent legal experts through the following document:

- Arenales & Skinner-Klee, 2010: unpublished legal opinion letter prepared by Arenales & Skinner-Klee for Entre Mares, S.A., 23 February, 2010.
- Arenales & Skinner-Klee, 2010: unpublished legal opinion letter prepared by Arenales & Skinner-Klee for Entre Mares, S.A., 21 May, 2010.
- AMEC, 2010: Escobal Project Guatemala NI 43-101 Technical Report, prepared by Mr. Greg Kulla, 30 April, 2010

Data and information derived from work done by previous owners of Escobal and more recent work by Tahoe Resources Inc.

3.3 RESOURCE MODELING

The Qualified Person in charge of Resource Modeling is Paul Tietz of Mine Development Associates (MDA).

3.4 MINE TABULATION

The Qualified Person in charge of Mine Tabulation and Costing Review is Conrad Huss of M3 Engineering & Technology Corporation.

3.5 DRILLING, SAMPLE PREPARATION AND SECURITY, DATA VERIFICATION

The Qualified Person in charge of Drilling, Sample Preparation and Security, and Data Verification is Paul Tietz of MDA.

3.6 METALLURGICAL TESTING

The Qualified Person in charge of Metallurgical Testing is Thomas Drielick, P.E. of M3 Engineering and Technology Corporation.

3.7 FLOW SHEETS

The Qualified Person in charge of Flow Sheets is Thomas Drielick, P.E. of M3 Engineering and Technology Corporation.

3.8 CIVIL AND ENVIRONMENTAL CONTROLS

The Qualified Person in charge of Civil and Environmental Controls is Daniel Roth, P.E. of M3 Engineering and Technology Corporation.

3.9 PROCESS PLANT AND COSTING

The Qualified Person in charge of Process Plant and Costing is Conrad Huss, P.E. of M3 Engineering and Technology Corporation.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

Escobal is an advanced stage is an exploration project located in southeast Guatemala; approximately 40 kilometers east-southeast of Guatemala City and two kilometers east of the town of San Rafael Las Flores in the Department of Santa Rosa (Figure 2-1). The Project is centered at UTM coordinate 806,500E 1,601,500N (NAD27, Zone 15).

The Project consists of three exploration concessions, Oasis, Lucero and Andres covering 129 square kilometers that is 100% owned by Tahoe Resources through its wholly-owned subsidiary Minera San Rafael S.A.

4.2 MINERAL TENURE AND AGREEMENT

4.2.1 Mineral Rights

The Project comprises three exploration concessions covering 129 km^2 (129,000 ha) called Oasis, Lucero, and Andres (Figure 4-1), granted on March 26, 2007, August 21, 2007 and November 15, 2007 respectively to Entre Mares de Guatemala S.A. The Oasis concession covers the entire Escobal vein, the Lucero concession is located approximately 20 km east of Escobal and the Andres concession lies roughly 13 km to the northwest. The original concessions covering a total area of 129 km^2 (129,000 ha) were transferred to Mineral San Rafael S.A. through a transaction agreement, dated 3 May, 2010, with two wholly-owned subsidiaries of Goldcorp Inc.

Exploration concessions in Guatemala are granted for an initial period of three years which can be extended for two additional periods for two years each, for a total holding period of seven years. The first three-year term of the Oasis concession expired in March 2010, at which time a renewal application was filed to extend the exploration concession for two more years. As part of the renewal process requirement the Oasis concession was reduced in area from 50 km^2 to 40 km^2 and three new exploration concessions were applied for to fill the 20% gap created by the area reduction; the Melissa, Cipreses and Puente Quebrado concessions cover a total area of 10 km^2 in the northeast and south areas of the original Oasis concession. The renewal application was approved by the Guatemalan Ministry of Energy and Mines (MEM) on 28, April, 2010, prior to its effective transfer to Tahoe at the close of the Initial Public Offering ("IPO") on June 8, 2010.

In June 2010 an application was filed to extend the term of the Lucero license for two years from its July 2010 expiration date. The renewal application provides for a reduction in the area of the license from 52.8 km^2 to 40 km^2. One new license application (Valencia) was filed to fill the approximate 12.8 km^2 gap created by the reduction of the original Lucero license. Similarly, an application was made in October 2010 to extend the Andres license for two years from the original license expiration in December 2010. In this case the Granada license was applied for to fill the gap left through reduction of the Andres license. The renewal of the Lucero and Andres licenses are pending.

According to Guatemala law, a second two-year extension can be applied for all three licenses in 2012, but after 2014, no more extensions are permitted and an exploitation license application must be made. Prior to the application of the exploitation license an economic study, mine plan and environmental impact assessment must be completed as preconditions for granting of an exploitation license.

In addition to the three granted exploration licenses, applications for the Soledad reconnaissance license and the El Olivo and Juan Bosco exploration licenses were submitted to MEM by Entre Mares in 2006, 2007 and 2008, respectively. San Rafael acquired the right to these applications as part of the Escobal Acquisition. San Rafael later filed an application for the Cristina exploration license (October 2010) and the El Silencio reconnaissance and Barrera exploration licenses (November 2010). In 2011 five exploration concessions – Nacimiento, Pajarita, El Durazno, Teresa and Pajal, were applied for over newly identified prospective areas within the El Soledad and El Silencio reconnaissance concessions. The Company was subsequently notified by MEM that these applications could not be registered until the reconnaissance concessions were granted and requested exploration areas were formally excluded. All of these licenses will cover land within the Escobal Project area when issued.

On July 8, 2011 an application was submitted to MEM for the Escobal Exploitation concession, covering 20.0 km^2 of area designated for mine development in the original Oasis exploration concession. Upon filing of the exploitation concession, three new exploration concessions (Oasis I, II, III) were requested to occupy the area liberated through elimination of the original Oasis concession.

According to Guatemalan requirements the concession is “coordinate staked”; filed only referenced to UTM coordinates and nothing is located on the ground. No physical survey of exploration concession boundaries is required.

The following table shows concession type, size and application/grant dates for all San Rafael concessions:

Table 4-1: San Rafael Concessions

Concession	Type	Size (Km2)	Application Date	Grant Date	1st Extension Filed	1st Extension approved
SOLEDAD	Recon	802.5	12/6/2006	NA		
OASIS	Exploration	40.0	10/25/2006	3/15/2007	10/9/2009	4/27/2010
LUCERO	Exploration	45.8	10/25/2006	7/20/2007	6/21/2010	
ANDRES	Exploration	44.0	5/18/2007	12/17/2007	10/6/2010	
EL OLIVO	Exploration	36.0	5/18/2007	NA		
JUAN BOSCO	Exploration	59.9	11/12/2008	NA		
PUENTE QUEBRADO	Exploration	3.0	10/9/2009	NA		
MELISA	Exploration	3.0	10/9/2009	NA		
CIPRESES	Exploration	3.0	10/9/2009	NA		
VALENCIA	Exploration	7.0	8/23/2010	NA		
GRANADA	Exploration	5.0	10/6/2010	NA		
CRISTINA	Exploration	52.5	10/6/2010	NA		
EL SILENCIO	Recon	1098.1	11/4/2010	NA		
BARRERA	Exploration	9.0	11/17/2010	NA		
NACIMIENTO	Exploration	7.6	2/19/2011	NA		
PAJAL	Exploration	66.0	5/4/2011	NA		
ESCOBAL	Exploitation	20.0	7/8/2011	NA		
EL DURAZNO	Exploration	48.9	7/29/2011	NA		
PAJARITA	Exploration	57.0	7/29/2011	NA		
TERESA	Exploration	68.5	8/17/2011	NA		
OASIS I	Exploration	12.8	8/31/2011	NA		
OASIS II	Exploration	7.0	8/31/2011	NA		
OASIS III	Exploration	0.2	8/31/2011	NA		

Yearly payments to the MEM for each 50 km^2 exploration concession includes an approximate Q. 30,000 (~US$ 3,750) concession holding fee and a Q. 750 (~US$ 90) exploration report filing fee. All required payments are current for all concessions through 2010. Similar payments will be required for subsequent extension periods.

There are no defined work requirements to keep an exploration concession valid, although exploration activity (sampling, mapping, etc.) must to be conducted and results filed with the Ministry of Mines (MEM) on an annual basis. Exploration activity reports have been filed with MEM for all exploration concessions each year as required.

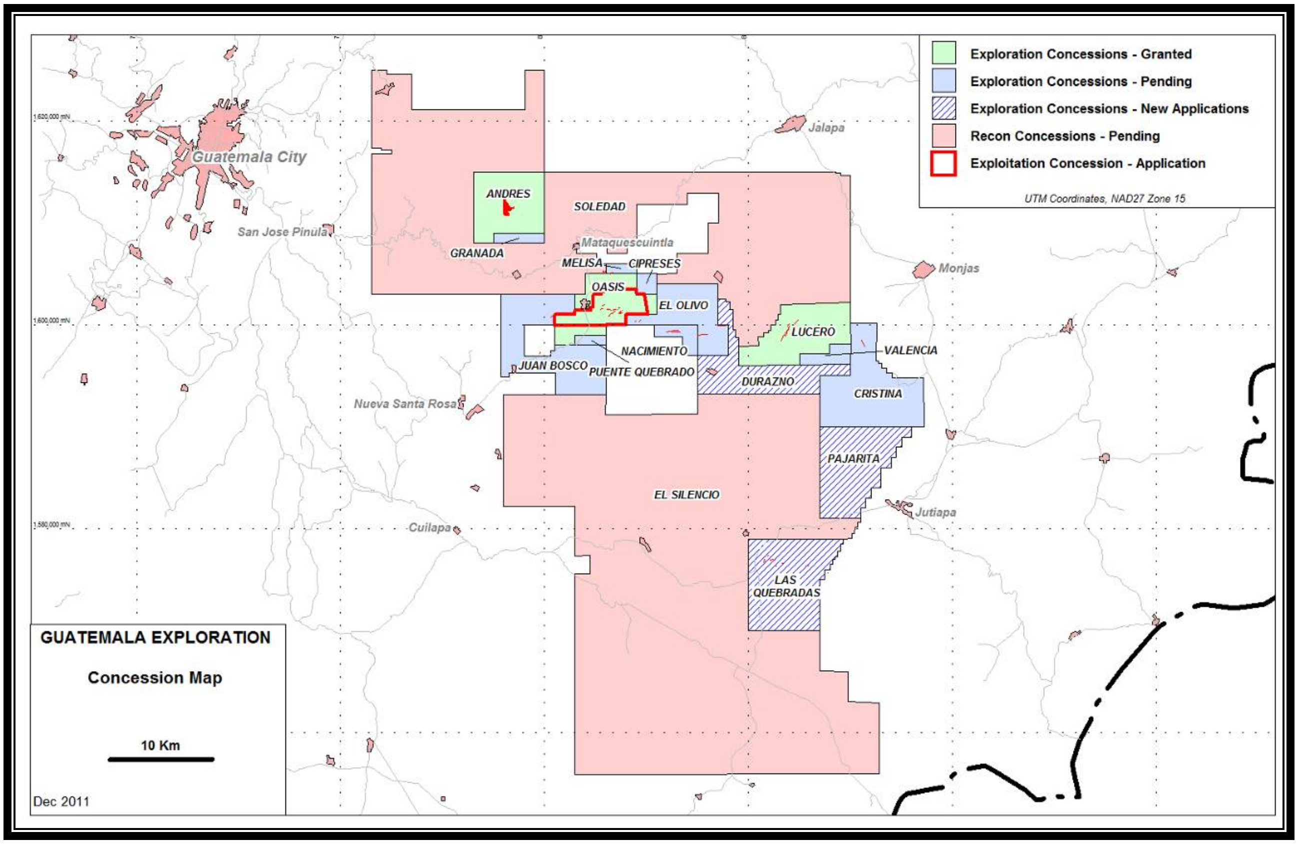


Figure 4-1: Exploration Concession Showing Regional Veins

4.2.2 Surface Rights

In Guatemala, the surface rights are independent of mining rights and must be negotiated separately. There is no allowance for expropriation in Guatemala. The land in the area of the project is privately owned by local farmers and is used for growing coffee in the higher elevations and vegetables and other crops in flat low lying areas.

Approximately 281 ha of surface rights cover the area of the Project. All Project surface rights needed to support the areas required for mining, tailings, waste rock disposal, processing plant and ancillary surface facilities have been acquired.

Based on an internal survey conducted in 2008 and an independent survey conducted in 2009 the average value of land in the area is approximately US$13,500 per hectare.

In areas peripheral to the project where surface rights have not been purchased annual rental fee agreements are in place with a number of land owners that provides for access and site preparation to accommodate exploration activities and drilling. No liabilities currently exist for land usage.

Construction and temporary operations offices have been built in the property, as well as a temporary maintenance shop, warehouse, internal and access roads, internal power lines, a temporary electrical substation and infrastructure to support the underground exploration development at both portals, including air compressors, generators and ventilation fans. General construction of the process plant, offices and ancillary buildings is underway.

4.2.3 Agreements

On 3 May 2010, Tahoe executed an agreement to acquire the Escobal Project with Goldcorp's indirectly wholly owned subsidiaries, Goldcorp Holdings Barbados Ltd. and Guatemala Holdings Ltd., which respectively hold 9.1% and 91.9% of Entre Mares de Guatemala S.A. de C.V. On 8 June 2010 upon successful completion of Tahoe's Initial Public Offering, Tahoe acquired all of the common shares of Entre Mares including the Escobal project and all of the exploration properties discussed in this Report.

4.2.4 Royalties

The current mineral royalty in Guatemala is 1%, shared equally between the local Municipality and Federal government. However, in January 2012 a voluntary royalty was agreed to between the Ministry of Mines (MEM) and the Guatemalan Mining Association (GREMIAL) that would impose a higher royalty on precious and non-precious metal mining. The voluntary rates were established at 4% for precious metals and 3% for base metal mining. The royalty on non-metal mining was maintained at 1%, while Goldcorp agreed to pay a 5% royalty rate at their Marlin gold-silver operation.

Tahoe is developing a profit sharing program which may be considered to be a royalty that will be implemented to provide ex-land owners benefits throughout the life of the Project. The concept is to pay an amount of 0.5% of net smelter returns to an Association of Land Owners and individual land owners. A certain percentage of this money will be deposited in a special fund, administrated by the association board of directors and used for improvements in local communities on behalf of the members of the association. Land purchase agreements include a provision that provides land owners the right to buy their land back from Tahoe at a significantly reduced price at the end of the life of the mine, once all reclamation has been completed.

4.2.5 Permits

All permits to continue exploration activities are in place. The application for the first two year extension of all three exploration license has been submitted. Tree-cutting permits, generally required by the National Institute of Forests (INAB), have not been required for exploration drilling as no road building has been undertaken due to minimal surface disturbance by the use of man-portable drills. Reclamation of drill sites is conducted once each drill hole is completed. INAB permits have been obtained for specific areas of site facilities where tree cutting is required for project development. Land use changes in the project area have also been approved by INAB as required.

This is an early-stage development project with exploration activities permitted by both MEM and MARN. All required permits to continue surface and underground exploration activities are in place. No other parties hold interest in the project.

The environmental requirements for the Escobal Project from MARN are specified in Resolution 4590-2008/ELER/CG, dated December 23, 2008. This resolution applies to surface exploration activities. These requirements were transferred from Entre Mares to Minera San Rafael as specified in Resolution 1918-2010/ECM/GB, dated September 3, 2010.

Development of an underground exploration program including the construction of two declines to gain access for additional drilling of the Escobal deposit is a permitted activity under the terms of the existing exploration license. An EIS addressing the additional activities associated with underground exploration was required prior to commencement of these activities and was filed with the MARN in November 2010 and an Environmental License filed on March 17, 2011. Approval of the Environmental Assessment was required before the underground exploration commenced; MEM notified the company of the reception and acceptance of the work program for the exploration declines on April 5, 2011, clearing the way for the start of the underground exploration program.

An EIA for the exploitation phase of the project was prepared and submitted to MARN for approval in August of 2011. MARN approved the EIA for exploitation by issuing Resolution 3061-2011 in October of 2011. This approval allowed full construction of the mine, process plant and all surface and underground facilities to be conducted. Application for the Exploitation License was submitted to the MEM in November 2010 and is awaiting final approval by the agency. Approval of this license is required before production can commence.

The environmental impact statements require documentation of baseline conditions, a project description, and an analysis of potential impacts and their mitigation measures. Public disclosure and involvement has been required and developed throughout each stage of the project and the permitting.

4.3 ENVIRONMENTAL MANAGEMENT AND STEWARDSHIP

The Escobal Project is a Greenfield project and as such warrants a high level of environmental stewardship. The mandate from Tahoe is to meet or exceed the standards of sustainability and environmental management based on North American practice and regulation. This section summarizes the elements of design and practice relating to environmental management and stewardship at the Escobal Project.

No impacted waters and materials will be directly discharged from the site. Impacted water will require lined containment and treatment prior to being released to the environment. The environmental management program includes:

- Primary Watershed Considerations
- Dry Stack Tailings
- Lined stormwater and waste facilities
- Concurrent Reclamation
- Process water recovery and recycling
- Process/Contact Water Treatment Facility
- Underground Paste Backfill

- Geochemical Characterization
- Environmental Impact Management Program

4.3.1 Primary Watershed

The plant site and Tailings and Waste Rock facilities are designed and located such that the upstream primary natural watershed will not be diverted. Only the portion of the drainage near the operational facilities will be realigned and strengthened. The avoidance of diverting this major watershed will reduce the overall area of disturbance as well as maintaining the historic flow of water through the property.

4.3.2 Dry Stack Tailing

Dry stack tailing management provides significant environmental and operational advantages over traditional wet or slurry tailings disposal methods.

The primary benefit derived from a dry stack tailing system is water balance. As the tails are filtered to 10~15% moisture, the remaining water is returned to the process stream providing a direct offset to make-up water normally obtained from ground water pumping.

Another benefit to a dry stack tailing system is the reduced footprint compared to a wet system.

4.3.3 Lined Stormwater and Waste Facilities

All facilities located on permeable ground that contain or receive impacted waters or acid generating material will be lined for containment.

4.3.4 Concurrent Reclamation

Concurrent reclamation of the Tailings and Waste Rock Storage Facility outslopes will allow for early reclamation with either native seed mix or a return to agricultural crops. Natural landform grading will be incorporated to provide a more stable, sustainable and natural functioning final surface.

4.3.5 Process Water Recovery and Recycling

The process design in both the Tails and Concentrate circuits maximizes process water recovery and reuse. In addition, contact water from the Tailings and Waste Rock Storage Facility as well as contact water from the haul roads and active mill and plant areas will be collected in channels and stormwater ponds for reuse in the process stream. Recycling and utilizing this water for operational uses will reduce the need for make-up process water from fresh water sources and minimize the potential for aquifer impacts in the region.

4.3.6 Process/Contact Water Treatment Facility

Process and contact water not utilized in the process stream or underground operations will be processed at the Process Water Treatment Facility where it will meet North American standards before being released to the environment. The project team is investigating utilizing treated water that cannot be used in the operation as irrigation water in reforestation and reclamation projects rather than direct discharge to surface waters.

4.3.7 Paste Backfill

Up to 60% of the tailings produced will be mixed with cement and water in a batch plant and disposed of underground as paste backfill, providing several environmental advantages.

- Provides stability to the underground workings, increasing safety and reducing the possibility of subsidence expressions reaching the surface.
- Provides an opportunity to encapsulate any potentially acid generating development materials, isolating them from water and oxygen thus preventing any potential metals leaching or acid generation.
- Provides reduction of storage area required on the surface. As little as 40% of the tails produced will be disposed of above ground.

4.3.8 Geochemical Characterization

Geochemical Characterization of the waste rock samples and tailings samples to date indicates a large net neutralizing capacity. Humidity cell tests with representative samples from both waste rock and tails have been in progress for over a year, with average pH values of 7.8 and 7.2 for the waste rock and tailings, respectively. No deleterious metals in the waste rock or tailings effluent exceed regulatory limits. Samples are being collected systematically and regularly as the declines are being excavated and as metallurgical work continues. Testing of these samples is ongoing and delivering consistent and favorable results indicating almost no potential for acid generation. Sampling and characterization of the waste rock and tailings continues.

4.3.9 Environmental Impact Management Program

Potential impacts from the envisioned mining operations will be characterized, monitored and managed by a comprehensive Environmental Impact Management Program developed specifically for the conditions at Escobal. Based on North American standards, the program will function to avoid, minimize, mitigate and remediate, in that order, all potential impacts. The management plan is designed to comply with the requirements of the Exploration Decline EIS, the EIS for Exploitation, other permit and governmental regulations.

4.4 PERMITTING

All activities on the Escobal Project have been permitted by both the MEM and MARN as well as other agencies of the Guatemalan Government. The environmental requirements for the initial exploration program from the MARN are specified in Resolution 4590-2008/ELER/CG dated December 23, 2008. License of rights was transferred from Entre Mares de Guatemala to Minera San Rafael as specified in Resolution 1918-2010/ECM/GB, dated September 3, 2010. An EIA that considers the environmental impacts associated with the underground exploration drifts was been prepared and submitted to MARN. This EIA authorizing excavation of the two declines, temporary facilities for to support the underground effort, the access road and installation of the power line was approved by MARN Resolution 262-2011. An EIA for the exploitation phase of the project was prepared and submitted to MARN for approval August of 2011. MARN approved the EIA for exploitation by issuing Resolution 3061-2011 in October of 2011. This approval allowed full construction of the mine, process plant and all surface and underground facilities to be conducted. Application for the Exploitation License was submitted to the MEM in November 2010 and is awaiting final approval by the agency. Approval of this license is required before production can commence.

The environmental impact statements require documentation of baseline conditions, a project description, and an analysis of potential impacts and their mitigation measures. Public disclosure and involvement has been required and developed throughout each stage of the project and the permitting.

4.4.1 Baseline Studies and Permits

The following is a list of baseline data that is typically required for a mining project proposal and includes the current status of Escobal Project studies:

- Flora and Fauna. First Aquatic and Terrestrial biology survey completed in 2008, two in 2009, and two in 2010, and continued in 2011; surveys are conducted twice per year, once in the rainy season and once in the dry season.
- Archaeological Resources. First inspection completed in June 2009. Second inspection completed in October 2010. An extensive third inspection associated with analysis of full project construction was conducted in 2011 approval of the final report was completed and referenced in the approved EIA for construction of the project.
- Socioeconomic Conditions. Studies conducted in 2010 in association with the Environmental Impact Assessment for Exploration. Additional studies were conducted and completed in 2011 in support of and approval of the Exploitation EIA and application for the Exploitation License.
- Air Quality. Data collection commenced in 2009. More than two years of quarterly samples have been collected to complete a baseline record for the approved EIAs and permits. Air Quality continues to be collected as conditions of the approved permits and in support of the construction activities.
- Ambient Noise Levels. Data collection commenced in 2009 and two years of quarterly samples have been collected to complete a baseline record. Noise data continues to be collected as conditions of the approved permits and in support of construction activities.
- Vibration Monitoring. A vibration study was completed in 2010 and the data was included in the underground exploration EIA completed in Nov. 2010. Monitoring continues in support of the exploitation EIA approval and license as well as a condition of approved permits and in support of construction activities.
- Soil Characteristics. Description and characterization of soils completed in 2010. Ongoing analysis is conducted as needed to support additional permit requirements.

- Climatic Information. A simple rain gage was installed in 2009. Weather Station installed in 2010. Climatic information for previous years was obtained from Weather Stations in Los Esclavos and Portezuelo, the closest stations to the project. Climatic data will continue to be collected throughout the mine life.
- Groundwater Quality. Ground water quality sampling has been designed to monitor the natural springs in the area. Data gathering commenced in 2008 and has continued on a systematic basis through the present. Data collection will continue to meet permit conditions and in support of construction activities.
- Surface Water Quality. Data collection commenced in 2008 and has continued on a regular schedule through the present time. Data collection will continue to meet permit requirements and in support of construction activities.
- Hydrology. Study commenced in 2010 and has continued through the present. Data collection will continue to meet permit requirements and in support of construction activities.
- Hydrogeology. Study commenced in 2010 and has continued through the present. Data collection will continue to meet permit requirements and in support of construction activities.
- Geochemistry. Data collection commenced in 2009 and has continued through the present. Data collection will continue to meet permit requirements and in support of construction activities.
- Geology. Data collection commenced in 2007and continues to date.

The EIA for exploitation was completed, submitted to the agencies, and approved in 2011. This allowed construction of all underground and surface facilities necessary for production to commence. Application for the Exploitation License was made to MEM in November 2011. Tahoe contemplates approval of the license in the first half of 2012. The license is required before production can commence.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Escobal project is in southeast Guatemala, 40 kilometers east-southeast of Guatemala City and two kilometers east of the town of San Rafael Las Flores (pop. 3,500) in the Department of Santa Rosa. The property is centered at UTM coordinate 806,500E 1,601,500N (NAD27, Zone 15).

The principal access route to the project is from Guatemala City by 70 km of paved to the town of San Rafael Las Flores, and then east 3 km by dirt road to the center of the project area. The project is accessible all year; however access to the upper elevations of the deposit is limited to four-wheel drive vehicles on less developed roads. Project offices and facilities are currently located at the project site. Housing and food services are located in the town of San Rafael.

5.2 CLIMATE

The local climate consists of two major seasons; a "rainy" season between May and November and a "dry" season between November and May.

Average annual precipitation amounts to 1,689 mm (66 in) with June and September the rainiest months with 315 mm (12 in) and 335 mm (13 in), respectively. January and December are the driest months with 0.6 mm (0.02 in) and 10.6 mm (0.4 in) of rain, respectively.

Average temperatures vary between April, the hottest month with average lows of 19°C (66°F) and highs of 33.1°C (91°F) and January, the coldest month, with temperatures averaging lows of 14°C (58°F) and highs of 30°C (86°F). Climate measurements are from a combination of sources including the project site in 2010, Los Esclavos, Cuilapa, and Santa Rosa located 30 km southwest of the Project area over several years.

Exploration and development activities are carried out year-round without interruption due to weather. Mining activities are expected to be conducted year-round.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The town of San Rafael Las Flores (pop. ~ 3,500) has basic services such as banks, health center and schools. Mataquescuintla (pop. ~ 8,000), located approximately 7 km from San Rafael, is more developed with more diverse banking, commerce and health services. Although there is some historic mining in the area, there is no local workforce experienced in modern mining, and appropriate training programs for the local workforce has commenced. Several smaller villages surround the project area and contribute to the project labor pool.

5.4 EXISTING INFRASTRUCTURE

There is a 13.2 kV medium voltage line to the town of San Rafael Las Flores; however, this line is not capable of handling the anticipated load requirements for the project. Project power requirements may be met by upgrade of an existing 69 kV line and substation located in Mataquescuintla, approximately seven kilometers north of the project. This will be done by expanding the existing 69 kV bay and adding a capacitor bank to improve voltage regulation. Project power alternatives are currently being assessed.

As the project has a relatively low overall power requirement, self-generation remains a fallback position. Project economics can withstand increased operating costs with relatively little impact on the financial metrics.

All year access to the area is good via paved highways either from Guatemala City (approximately 70 km by road) via Barberena and Nueva Santa Rosa (approximately 75 km by road) to the south or alternatively via paved roads from Mataquescuintla (approximately 5 km by road) and Jalapa (approximately 40 km by road) to the north.

Satellite internet services and telephone are currently available at the project site and in San Rafael Las Flores. A fiber optic communication line is currently available in San Rafael and will be operational when permanent facilities are established at the Project site.

Hydrological studies of the Project are currently underway. There are water wells within the Project area which may provide sources of water for potable and process needs.

Sufficient land has been purchased to host the required tailings, waste, plant, and underground access for a mining operation.

Many general supplies required for a mining operation are available in Guatemala, but major mining-specific supplies are not available in-country and will be imported.

5.5 PHYSIOGRAPHY

The project area lies within mountainous terrain interspersed with rolling hills and valleys. Elevations range from 1,300 m in the valley on the west end of the Escobal vein to 1,800 masl in the drilled east extension. The high mountain range of Montaña Soledad Grande north and east of Escobal rises to an elevation of 2,600 m.

Vegetation is characterized by natural mountain forest species that consist of oak, pine and cypress tree varieties and lower strata scrub-brush species.

Agricultural products in the area include corn and beans for local consumption, and commercial production of onions, tomato and coffee.

6 HISTORY

Guatemala does not have a well-recognized mining history, though the area of Mataquescuintla, approximately 7 km north of the Escobal Project, is the site of copper-silver production from underground mining around the turn of the 20th century.

A small underground operation was developed on an antimony showing at the Loma Pache prospect 600 m north of the Escobal vein in the 1970s. There are no drilling records available from the development of the Loma Pache prospect. Production records for both operations are incomplete; the Mataquescuintla (*a.k.a.* Colis) mine reportedly produced 8,000–10,000 tons of concentrate of unknown grade. Underground grades are reported to be 217 g/t silver (Ag), 0.2 g/t gold (Au), 1.27% copper (Cu) and 24% sulphur (S).

Interest in the Escobal area dates back to 1996 when Entre Mares de Guatemala S.A., the predecessor of Minera San Rafael SA., prospected in the area and identified high-grade gold values associated with surface quartz veins in the western portion of the Escobal vein zone. Size potential of the zone was deemed uneconomic at the time and exploration activities were discontinued later that year. In 2006, Entre Mares reinitiated regional exploration in the area, partially based on verifying geochemical anomalies in the company database. In late 2006, significant silver and gold grades were detected from surface sampling along an extensive alteration zone developed over the Escobal vein. An exploration concession was applied for in October 2006 and was granted in March 2007. Exploration drilling commenced in May 2007 and as of the effective date of this Report is ongoing.

In early 2010 Goldcorp, predecessor to Tahoe in ownership of Escobal, reported a Measured and Indicated mineral resource estimate for Escobal of 6.97 Mt at 0.63 gpt Au and 580.3 gpt Ag and an Inferred mineral resource of 13.15 Mt at 0.53 gpt Au and 443.4 gpt Ag (February 17, 2010 Goldcorp news release). Goldcorp did not release a technical report to support the mineral resource declaration at that time.

In an independent study conducted in April 2010, AMEC Americas Ltd. carried out a resource calculation based on 46,333 m of drilling in 175 holes. This study reported an Indicated Mineral Resource of approximately 100 million ounces of silver contained in 4,570,000 tonnes at a silver grade of 684 g/t and an Inferred Mineral Resource of approximately 176 million ounces of silver contained in 12,800,000 tonnes at a silver grade of 427 g/t. (Source: Mineral Resource NI 43-101 Technical Report – AMEC Americas Ltd. dated April 30, 2010, prepared under the guidance of Mr. Greg Kulla, P. Geo, a Qualified Person.)

In 2010 Tahoe Resources engaged M3 Engineering & Technology Corporation ("M3") to prepare the Escobal Preliminary Assessment Report, dated November 29, 2010 that contained an updated mineral resource estimate based on data from 61,469 meters in 220 diamond drill holes. The Preliminary Assessment reported 245.2 million ounces of silver classified as Indicated Mineral Resources, based on 15.3 million tonnes at an average silver grade of 500 g/t and 71.7 million ounces of silver classified as Inferred Mineral Resources, based on 8.3 million tonnes at an average silver grade of 271 g/t. In addition, both mineral resource categories reported significant amounts of gold, lead, and zinc (Source: Escobal Guatemala Project, NI 43-101

Technical Report Preliminary Economic Assessment – M3 Engineering and Technology Corporation dated November 29, 2010, prepared under the guidance of Mr. Conrad Huss, P.E., a Qualified Person).

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

Guatemala comprises two geologic terrains formed as the result the convergence of a major tectonic plate boundary. The North American plate comprises the northern half of Guatemala, and the Caribbean plate comprises the southern half with three major east-west trending, left-lateral transform faults forming the plate collision boundary. From north to south this boundary is defined by the Polochic, Motagua and Jocotan fault systems (Figure 7-1). The Escobal deposit lies within the southern, Caribbean plate, south of the Motagua fault. The northern side of the Motagua fault system contains Paleozoic metasediments, schist and gneiss, while the south side contains a series of Tertiary mafic volcanic eruptive events composed mostly of dacitic to andesitic tuff, lahar and andesitic to basaltic flows. These eruptive units are separated by thin beds of water-lain sediments consisting mostly of fine to medium grained clastic and tuffaceous sediments. Tertiary volcanics are commonly covered by Quaternary and recent dacitic volcanic eruptive ash units. The Escobal deposit is within the Tertiary mafic eruptive units that trend parallel to the Motagua fault system.

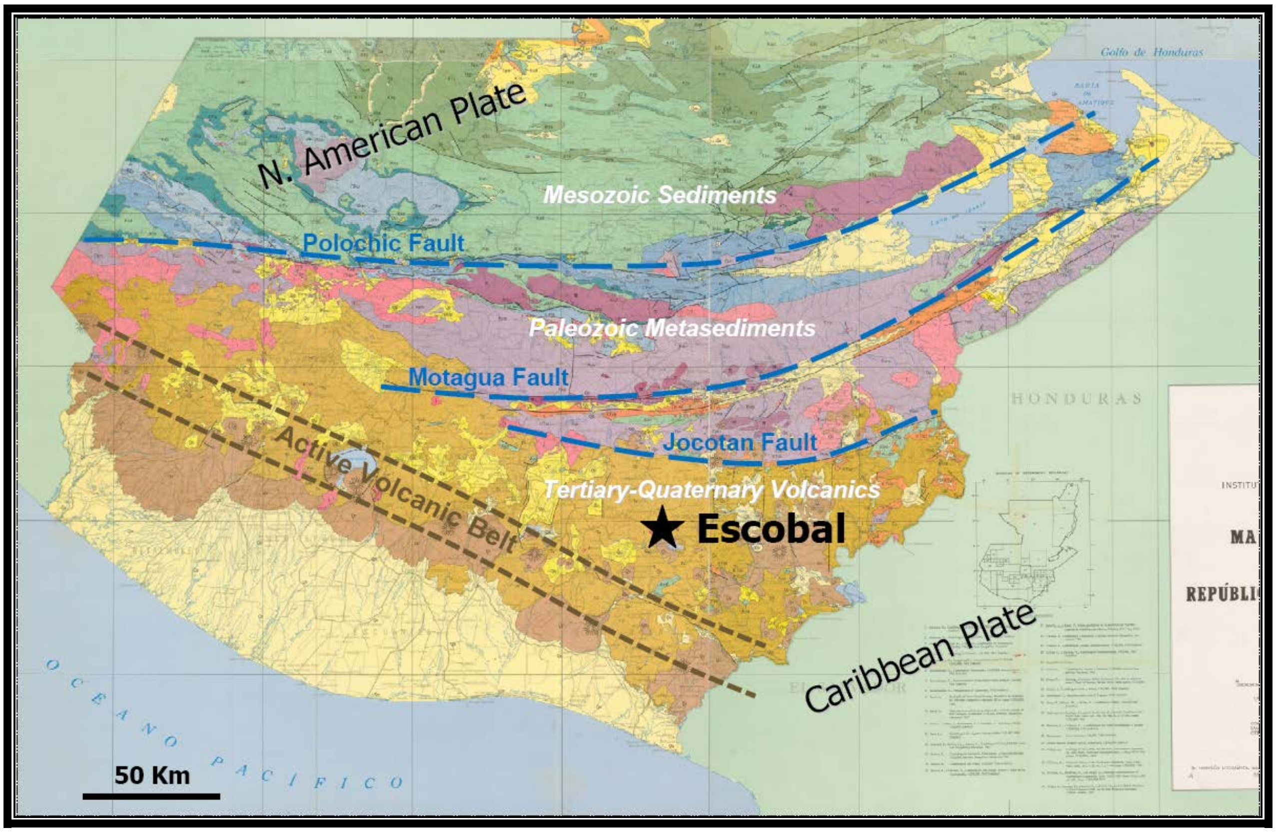


Figure 7-1: Regional Geology

7.2 LOCAL GEOLOGY

Project area surface geology is shown in Figure 7-2. This area is underlain by the Eocene Subinal Formation, a series of interbedded volcaniclastic sediments that include siltstone, fine- and coarse-grained sandstone, tuff, and limestone-clast conglomerate. This formation is unconformably overlain by a package of medium- grained, massive porphyritic andesite and lithic tuff composed of fine- to coarse-grained lapilli. Magnetic andesitic dikes, the youngest lithological units in the Project area, cross-cut all rock units. A thin unit of Quaternary pyroclastic ash irregularly overlies all lithological units over large portions (~ 60%) of the project area.

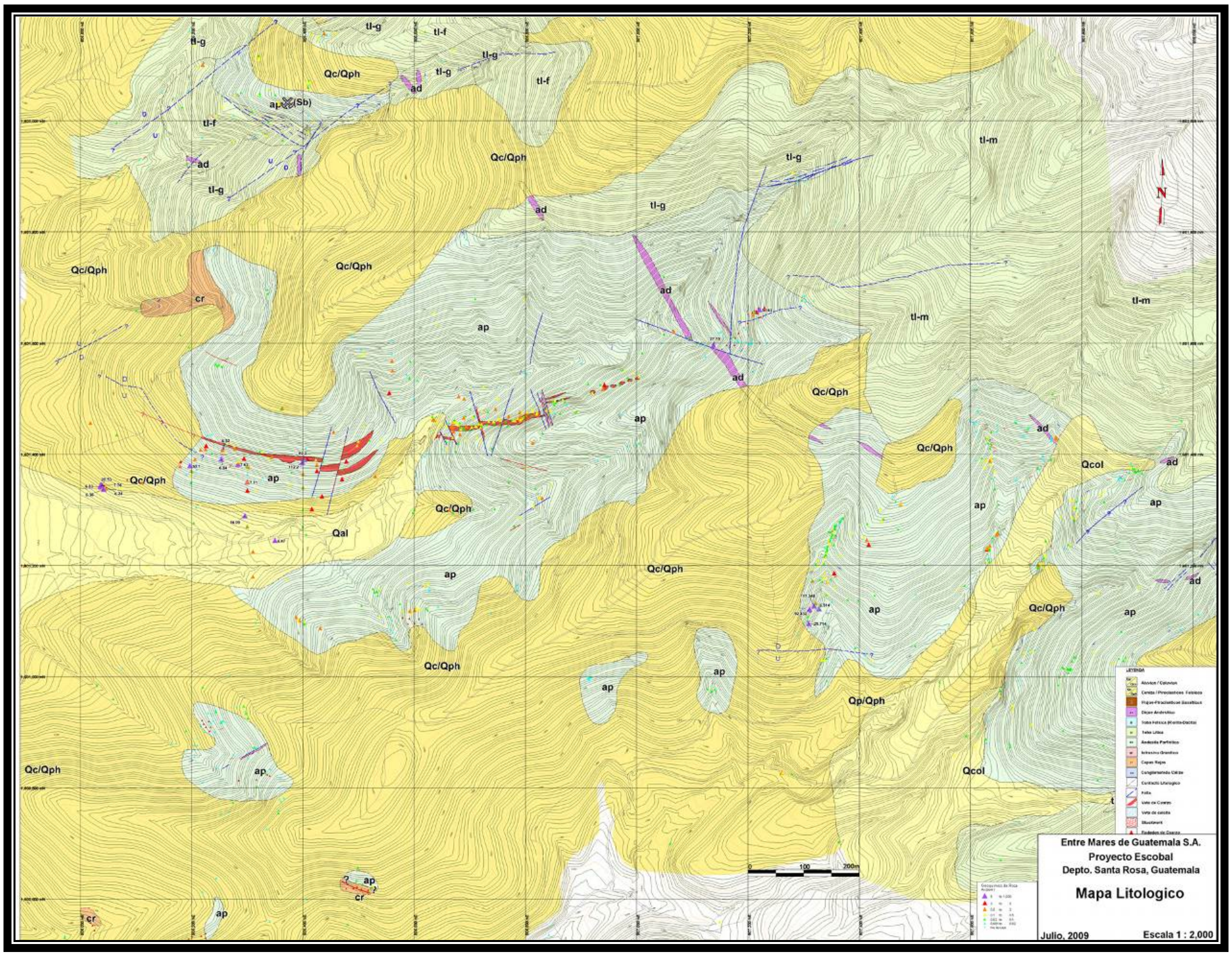


Figure 7-2: Local Geology

7.3 LITHOLOGIES

Specific lithological units from oldest to youngest in age and their corresponding map designations are described below:

Volcaniclastic-Clastic Sequence (Subinal Fm) (cr):

- A volcaniclastic sedimentary sequence related to regional redbeds forms the local basement in the Escobal area. These rocks are believed to correlate with the Subinal Formation, a continental clastic sequence that is distributed throughout central and southeast Guatemala. The volcaniclastic sequence at Escobal contains subunits of lapilli, andesite and crystal tuff intercalated with siltstone, sandstone and conglomerate. Individual beds range from 5 m to 200 m widths. The unit is exposed as irregularly-distributed windows in drainages and has a minimum thickness of 500 m.

- Sedimentary and volcanic subunits prove to be difficult to use as marker beds, due to their irregular distribution and repetitive occurrence. Recent drilling in the West Zone has identified a specific narrow sub-horizontal andesite unit in the extreme west drill sections. Distribution of this andesite bed shows a marked displacement; 150-250 meters down-to the west, suggesting normal basin margin faulting around the 805,900E section.

Porphyritic Andesite (ap):

- A sub-horizontal shaped body of porphyritic andesite unconformably overlies basement sediments throughout the Escobal area. The unit is massive to medium-grained and porphyritic, with feldspar, biotite and quartz phenocrysts in a fine-grained matrix.

- This unit is thought to be hypabyssal or intrusive in origin as it is texturally very consistent and shows no mineralogical zonation. The unit forms rare outcrops in the Escobal area and has been defined in drilling over a thickness of 500 m. Based on regional geological relationships, the porphyry is believed to be Upper Miocene in age.

- Recent drilling in the extreme west portion of the project area encountered a thick unit of andesite breccia that is interpreted as a flow within or proximal to a volcanic vent. The monolithic matrix-supported breccia is composed of sub-angular to sub-rounded porphyritic andesite clasts within a similar composition matrix. Due to limited drilling in the area, the size and distribution of this unit is not currently know, but interpretation will develop with planned deep drilling in 2012.

Lithic Tuff (tl):

- A unit of young post-mineral lithic tuff overlies the andesite porphyry in the northeast and far-west portions of the Escobal area. The unit consists of white, non-welded ashflow tuff with angular to sub-rounded, lapilli to pebble-sized lithic fragments of basalt to rhyolite composition. This unit masks the eastern extension of the East Zone of the Escobal vein with observed thickness of 50 m to 150 m, thickening to the east where less erosion is evident.

Andesite Dikes (ad):

- Andesite dikes cut all three lithological units and are believed to be post mineral of late-Tertiary (post Miocene) age. Dikes occur in the eastern portion of the Escobal Vein where bodies can be followed for 3 km along a N40W regional trend. The dikes are near-vertical tabular bodies that range in width from 20 cm to 10 m, which primarily occupy the footwall of the East Escobal vein. Dikes are composed of euhedral feldspar crystals immersed in a very fine grained matrix. The dikes are generally fresh to weakly altered and contain rare minor quartz veinlets and are thought to be of pre- or syn-mineral age. The dikes are generally magnetic, though magnetism varies in intensity with the degree of alteration/weathering.

Quaternary ash-airfall tuff (Qc/Qph):

- Non-lithified ash and pumice-rich tuff is widespread and covers most ridges and topographic highs in the project area. Thickness is variable, though is commonly several meters thick on hilltops and slopes. Ash is typically eroded from drainages and valleys, though where reworked and transported, can form up to 20 meter-thick deposits.

- The ash unit comprises two layers; a basal very coarse-grained, unconsolidated, heterolithic layer; and an upper layer of medium- to fine-grained unconsolidated ash.

7.4 STRUCTURE

The dominant structural trend in the region parallels the regional Montagua fault along an east-west to N60E trend. At Escobal this structural trend is represented by a series of east-west trending normal faults that exhibit down-to-the south movement, typical of an extensional structural regime. These faults are evidenced by lithologic displacement, shear and gouge zones and placement of the high-angle south dipping veins that define the East Escobal vein and upper and lower limbs of the variably-dipping Central Zone.

Dilation jogs, or tensional shears are commonly observed in extensional fault terrains (pull-apart basins or grabens) where the area between individual normal faults exhibit wide zones of disruption as a response to the structural event. These zones are commonly wider and more gently dipping than the primary steeply dipping structures. The wider moderate north-dipping Central mineralized zone is interpreted as occupying a dilational jog between the normal faults represented by the East Escobal vein zone and the upper and lower limbs of the Central vein zone.

A N40W trending structure dissects the Escobal vein between the East and Central zones. This feature is evidenced by the occurrence of steep dipping (70°SW) andesite dikes and an apparent (~ 100 m) left-lateral shift of mineralization. Based on relative elevations of mineralization and lithologic markers, this is a normal fault with vertical movement on the order of 50 m up-to-the northeast.

In the extreme west margin of the Escobal vein, drilling delineated an andesite marker horizon within the volcaniclastic sequence that suggests a fault at the margin of the San Rafael valley. Relative location of the sub-horizontal andesite suggests normal fault movement on the order of 150-250 meters, down-to-the west.

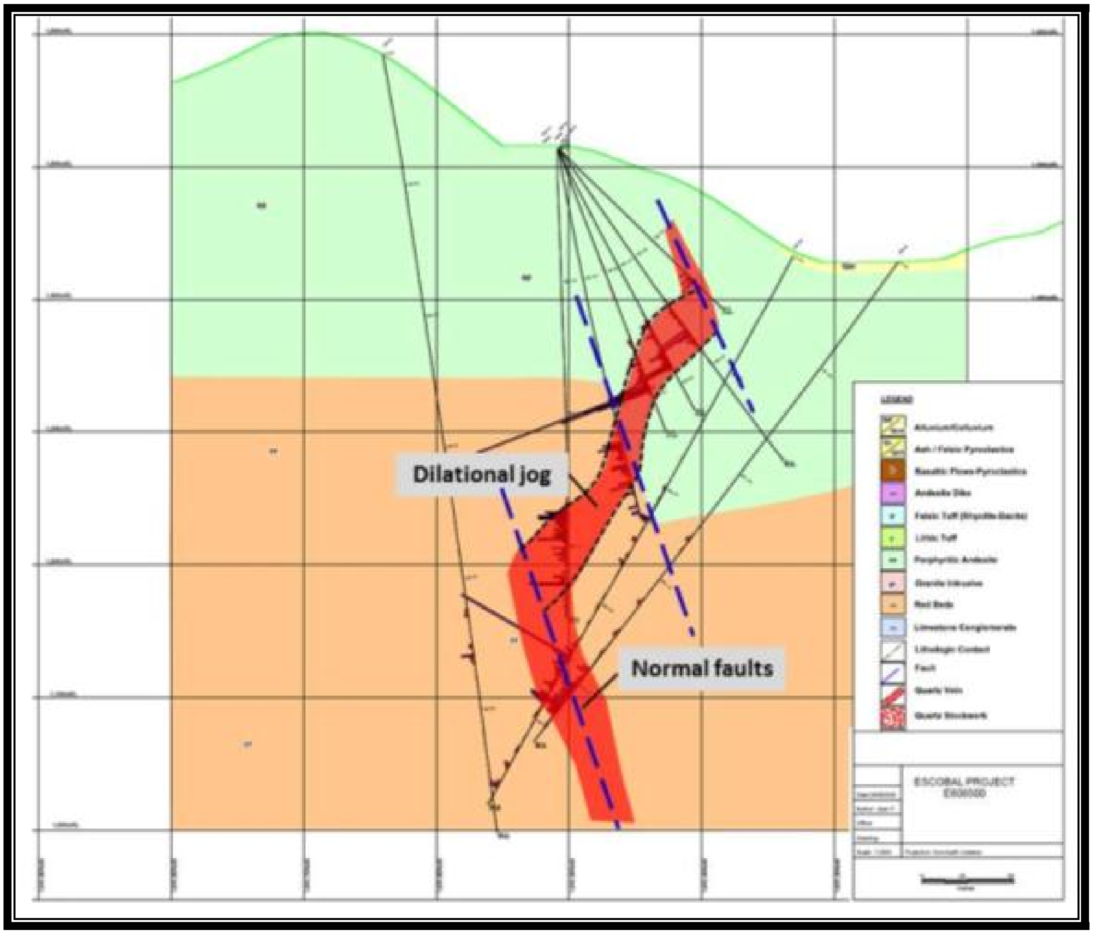


Figure 7-3: Interpretation of Central Escobal Vein along normal faults and dilational jog. (looking east)

Alteration

- Alteration mineralogy is typical of intermediate sulfidation epithermal systems. Quartz veins and stockwork up to 50 m wide, with up to 10% sulfides form at the core of this alteration pattern and grade outward through silicification, quartz-sericite, argillic and propylitic zones. The following descriptions provide additional detail on the alteration types:

Silicification

- Pervasive silicification is intimately related to zones of mineralization and forms as halos on both sides of the principal veins. Silica replacement is common in the matrix and occasionally replaces minor accessory minerals. Where strongly silicified, the rock is totally replaced leaving only casts of replaced minerals. This thoroughly pervasive texture is common where hydrothermal breccia occurs. Disseminated pyrite is commonly associated with silica replacement. Silicification halos surround mineralized veins for thicknesses up to 50 m.

Quartz-Sericite alteration

- Quartz-sericite alteration forms larger zones surrounding veins and silicified zones and generally indicates the zones proximal to mineralization. The alteration is characterized by homogenous zones of mixed quartz and sericite that form up to 100 m thick.

Argillization

- Argillic alteration commonly forms in select fault and shear zones. Commonly clay (kaolinite), sericite and jarosite form within narrow (centimeter to meter wide) zones as the alteration products of feldspar, biotite and rock matrix.

Propylitization

- Propylitic alteration forms as weakly pervasive and stronger fault-controlled zones of chlorite-calcite-pyrite. Propylitic alteration forms furthest from mineralization and commonly borders fresh-unaltered rock.

7.5 MINERALIZATION

Economic mineralization at Escobal comprises silver, gold, lead, and zinc hosted within quartz veins, stockwork zones and hydrothermal breccias. The mineralization is identified by drilling over a 1,700 m strike length and 800 m vertically. Average vein widths vary from 10 m in the East Zone to over 30 m in the Central Zone. The mineralization is open at depth and to the east and west where it is covered by alluvium and post-mineral volcanic rocks.

The deposit predominantly comprises sulfide mineralization. Silver, lead, and zinc sulfide mineralization predominates in the Central and West zones though elevated gold values also occur at depth in the Central Zone. In the East Zone gold-rich mineralization is associated with the upper mixed sulfide-oxide horizon. Silver mineralization in all zones shows a close association with galena and low-iron sphalerite.

A petrographic study of vein samples indicated a fairly simple and consistent paragenesis. Stage I veining consists of banded to massive chalcedony intercalated with quartz and carbonate. This is the volumetrically-dominant vein event and contains the bulk of sulfide minerals. Volumetrically lesser Stage II consists of sulfide-bearing granular chalcedony. Various episodes of post-sulfide quartz, and late barren calcite veining locally cut and/or overprint the main banded vein.

Based on analysis of petrographic characteristics at least five events of quartz veining are interpreted. These include, from oldest to youngest:

1) Dominant banded quartz-chalcedony vein.
2) Silica flooding event (quartz-chalcedony)
3) Narrow chalcedony/quartz veinlets.
4) Narrow euhedral quartz veinlets
5) Late hematite-goethite-chlorite-sericite and calcite replacement veinlets

Gold-silver mineralization occurs exclusively in the first two events. Narrow later-stage quartz, chalcedony veinlets are not considered precious or base metal depositing events.

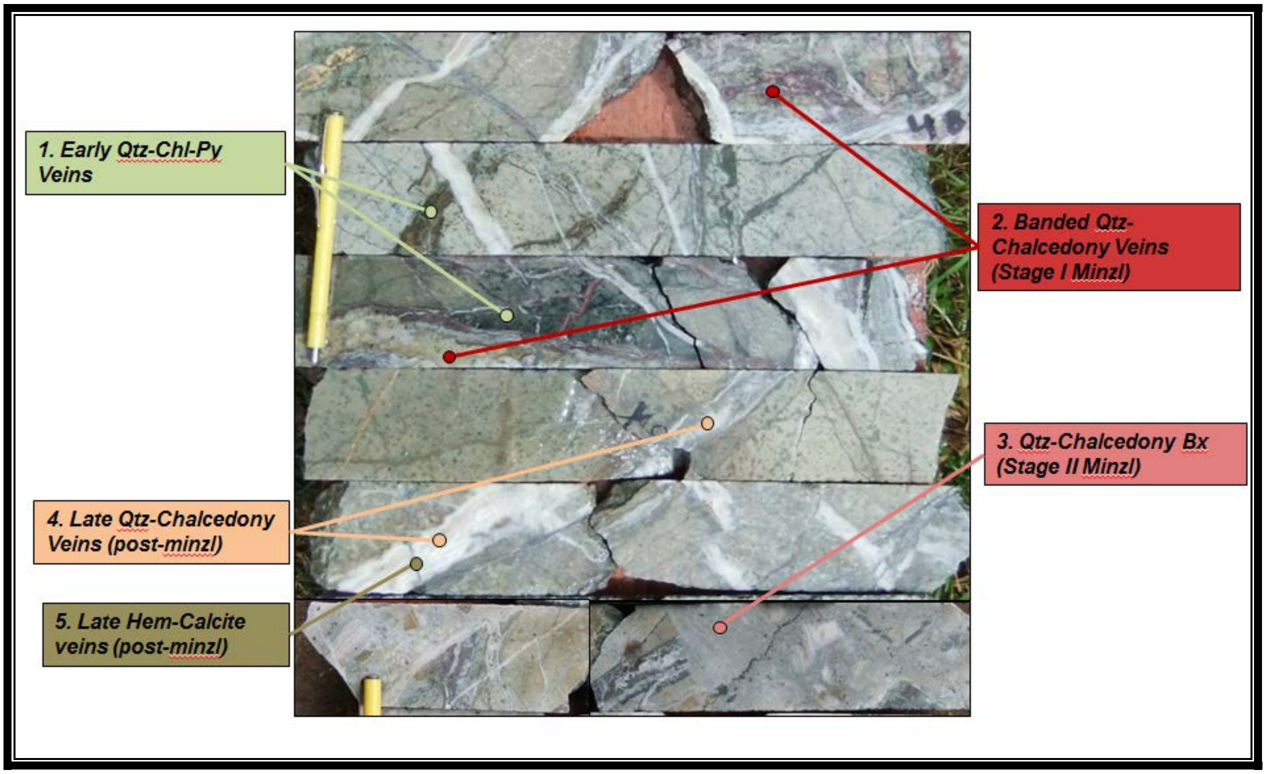


Figure 7-4: Vein episodes and generalized relationships

Silver minerals are dominantly proustite (+/- pyrargyrite), lesser amounts of acanthite and minor native silver. Gold minerals include electrum and native gold. These minerals and other sulfides occur as aggregates of abundant finely disseminated grains most commonly in chalcedony and of interstitial grains to quartz in select bands and as more isolated grains, especially in visible gold sites, throughout the vein in chalcedony/quartz. Aggregates of grains commonly consist of pyrite, acanthite, proustite, visible gold, ± galena, ± sphalerite, and ± chalcopyrite. Acanthite, proustite, and visible gold commonly are found together as disseminated aggregates exhibiting no, or rare, mutual contacts. In places, gold exhibits mutual contacts with acanthite and proustite.

Figure 7-5: Escobal Central Zone. Drillhole E08-110 Breccia with Red Proustite Bands

There is no definitive boundary to the overall vein width or to the up or down dip extensions of the vein. Silicification and stockwork concentrations increase towards massive banded and/or brecciated quartz ± carbonate vein material. Mineralization may start abruptly or may gradually increase through the stockwork. The vein appears to be better constrained in the volcanic host rocks and more diffuse and unconstrained in the sediment host rocks.

Drilling in the far west end of the Escobal vein (E11-312) encountered a wide zone of massive gypsum veins. This is the only gypsum occurrence recognized in the project area. Because it is associated with a similarly rare wide zone of brecciated andesite, it is believed to represent a low-temperature hot springs environment related to the margin of an andesitic volcanic vent.

7.6 ESCOBAL VEIN ZONES

The Escobal vein is divided into three zones:

East Zone

- In the East Zone, the Escobal vein follows an east-west to N80E normal fault that dips variably (60-75°) to the south and can be followed for at least 450 m on strike and over a vertical range of 450 m.

- Recent drilling in the area directly below the initial East Zone (between sections 807,500 and 807,600E) demonstrated that the typical south-dipping vein transitions to a north-dipping dilational jog similar in character, rock type and elevation as found in the Central Zone. This deeper extension in the East Zone remains open and untested down-dip to the north and laterally to the east and west.

- Recent widely-spaced step-out drilling in the deep eastern margin of the East Zone has identified a new zone of mineralization that can be followed for 350 meter along an east-west strike and to a 400 meter depth. The "East Extension" comprises multiple zones of moderate (2-20 meters) width veins and stockwork zones with a near vertical dip. Mineralization extends from 1400 to 1000 meter elevation and remains open to depth. The zone is capped by a 300 meter deep zone of un-mineralized narrow veining that was recognized in earlier drill campaigns.

- Geochemistry in the main East Zone is characterized by a gold-rich sector in the near surface oxide/mixed sulfide-oxide zone that abruptly changes at depth to silver-rich mineralization across the sulfide interface. Lead and zinc concentrations show a strong correlation to silver mineralization in the lower portion of the sulfide zone and increase with depth relative to silver. A gradational zoning pattern is observed with silver giving way to lead and then zinc with depth. The East Extension is characterized by high silver and relatively low grade for lead, zinc and gold.

Central Zone

- The Central portion of the Escobal vein is the thickest part of the vein system. The zone extends 700 meters on strike and covers a nearly 600 meter vertical range, from outcrop at 1500 meter elevation to the deepest drill intercept at 900 meter elevation. The zone strikes east-west with the main portion of the vein dipping moderately (60-70°) to the north. Flexures in mineralization in the upper and lower reaches of the Central Zone are controlled by high-angle faults. The wide moderately north-dipping main portion of the vein represents mineralization along the dilational jog, or tensional shears between two major normal faults.

- There is no near-surface gold zone observed in the Central Zone though gold does increase at depth; increasing at 1200 m elevation and extending to 900 m elevation, where it remains open to depth. This deep gold mineralized zone is unrelated to the near-surface gold zones observed in the East and West zones and may represent a distinct deep-seated gold-rich mineralized zone.

- High-grade silver occurs throughout the Central Zone, the bulk of which forms a wide roughly horizontal zone related to the wide north-dipping dilational structure. The zone narrows towards the east and exhibits greatest vertical extent on its western margin where it abruptly terminates along a barren "gap zone" bounding the western Margarito area. Lead and zinc concentrations correlate extremely well with silver grades in the Central Zone, though silver grades are maintained at depth, contrary to the gradational Ag-Pb-Zn vertical zoning that is evident in the East Zone.

West Zone

- The West mineralized zone has been significantly expanded through recent drilling. The zone is characterized by surficial gold occurrences that give way to a wide zone of silver, gold, and base metal rich vein stockwork at depth. The deeper “Margarito” mineralization is a discrete shoot as it is separated from the Central Zone and the upper gold zone by a 50-100 meter wide barren “gap” in mineralization. The zone as currently modeled extends over a 350 meter strike length and spans +400 meters vertically, raking down to the east. The top of mineralization is entirely preserved with significant grades commencing 250 meters below the surface. The zone is open to depth and down-rake to the east, while the western margin is believed to be down-dropped further west along a normal basin-bounding fault, interpreted through marker-bed offset.

- The West Zone follows a semi-arcuate trace with moderate north dips in the upper reaches of the zone giving way to steep, near vertical inclination at depth. The upper portion of the zone is characterized by high gold values in the mixed-oxide zone. The deeper Margarito shoot exhibits very wide (30-50m) zones of stockwork-veining with moderate silver grades, moderate-high gold grades throughout. Base metal values show a marked increase in the lower portion of the zone.

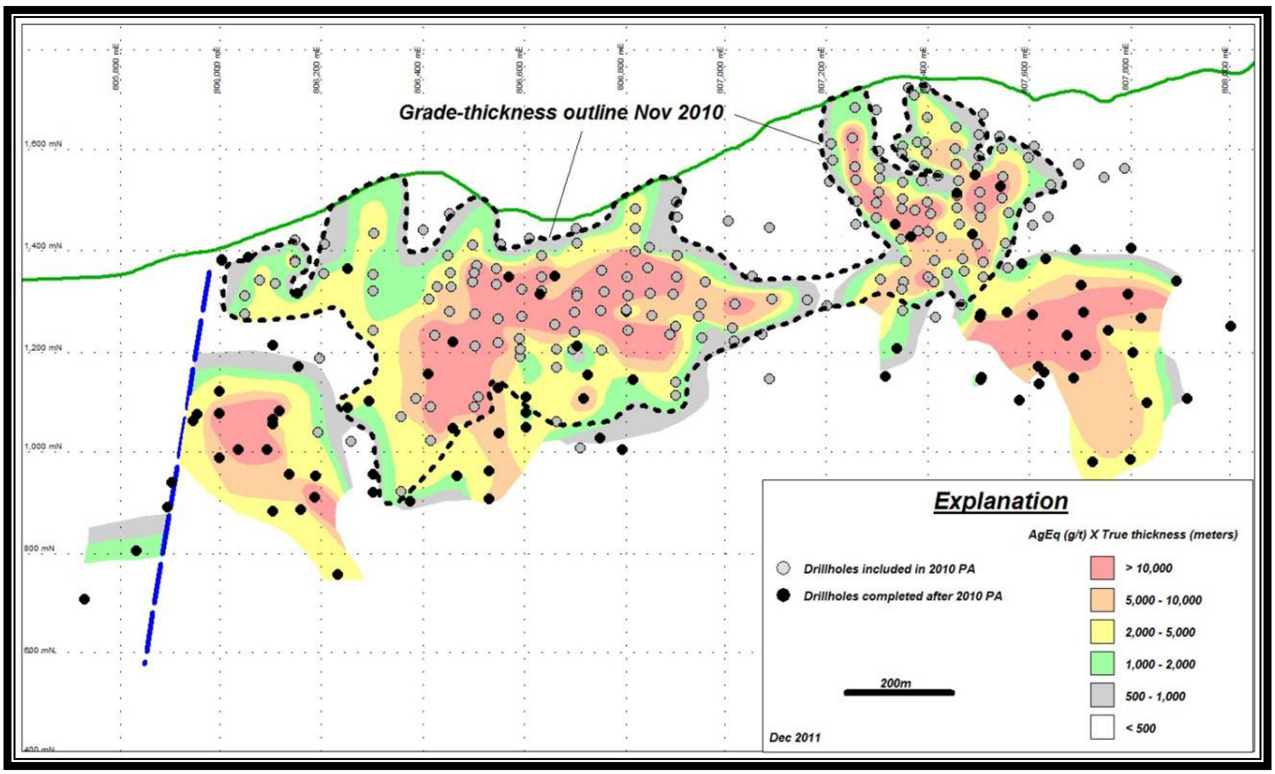


Figure 7-6: Escobal Long Section. Viewed to north.

Oxidation

- The bulk (+75%) of the deposit is un-oxidized. Wall rock oxidation was modeled on 50 meter-spaced north-south drill sections with oxidation to depths up to 200 meters while the vein itself, due to its relative permeability is partially oxidized from the surface to 250 meters depth. Secondary manganese oxide is concentrated near the base of the oxide zone.

- The current mixed oxide-sulfide domain boundary is defined by the last observation of any limonite. Oxidation within vein intervals above the domain boundary vary from moderately to completely oxidized in drill holes. Primary sulfide concentrations increase with depth, from none near surface, to 100% at the domain boundary. There are intensely oxidized vein intersections with high gold grades in upper level of the East Zone which may be amenable to leach processes.

7.7 VEIN MODEL

Vein attributes have been compiled to support metallurgical sample collection and to aid on-going exploration. Physical attributes include estimated true vein width and vein volume percent across the defined zones. Mineralogic attributes include iron oxide/sulfate mineral intensity, manganese oxide mineral intensity, observed proustite and total sulfide intensity. Geochemical attributes include average Ag, Au, Pb, Zn, Cu, As, and Sb contents for each vein intercept, as well as calculated Ag/Au, Ag/Pb, and Ag/Cu ratios. All vein attributes were contoured on a long section in the plane of the vein (vein intercepts were projected horizontally at 90 degrees to a common east-west plane). Key observations include the following:

- Four high-grade (plus 500 g/t Ag) Ag-(Au-Pb-Zn) "ore shoots" are defined by drilling. The East and Central zones are well-defined by drilling while the East Extension and West/Margarito zones remain partially open. All zones contain bonanza-grade (plus 1000 g/t Ag) intervals.

- The East Zone "ore shoot" begins approximately 100 meters beneath the surface, is at least 350 meters in strike length, and spans over 400 meters elevation.

- The Central Zone "ore shoot" begins approximately 50 meters beneath the surface, is at least 700 meters in strike length and spans approximately 600 meters in elevation.

- The East Extension "ore shoot" begins approximately 300 meters beneath the surface. The zone is partially defined by drilling and is open to the west, east and down dip. As currently defined, the zone covers a 350 meter strike length and 400 meter vertical range.

- The West/Margarito Zone begins approximately 250 meters below the surface and as currently defined extends over a 350 meter strike length and 400 meter vertical range. The zone remains open to depth, down-rake to the east and to the west where believed to be offset by faulting.

- Distribution of gold, silver, lead and zinc show a general trend of mineralization with a gentle (~ 20°) rake, down to the west; in effect the East Zone is about 200 meters higher in elevation than the Central Zone, which is in turn is about 200 meters higher in elevation than the Margarito Zone.

- Individual "ore shoots" in the East and Margarito zones show rakes 20-50°± to the east in the plane of the vein. The Central Zone is roughly horizontal with a more extensive vertical plume along its western margin. The East Extension, as currently defined, trends sub-horizontal with an apparent moderate (~ 50°) rake to the east where open to depth.

- Gold and arsenic are erratically anomalous above and peripheral to the higher grade Ag-Pb-Zn-(Au) in partially oxidized vein in the West and East zones. Gold is significantly more prevalent in deep western portion of the Escobal vein system. Deep drilling in sulfide-rich portions of the western Central and Margarito zones show consistently elevated gold values vector down to the west.

8 DEPOSIT TYPES

The Escobal deposit formed in an intermediate sulfidation epithermal quartz vein system of probable Upper Miocene to Lower Pliocene age. These deposits are commonly included in the low-sulfidation epithermal class of deposits. Distinguishing characteristics indicative of the "intermediate sulfidation" environment include mineral assemblages indicating a sulfidation state between those of high and low sulfidation types, relatively high total sulfide content of 5 to 10 percent, low-iron "blond" sphalerite, presence of silver sulfosalts, and association with andesitic to dacitic volcanics. Magmatic-associated fluids are implied.

Epithermal deposits form as high-temperature mineralizing fluids rise along structural pathways and deposit quartz and precious and base metals minerals in open spaces in the response to boiling, which is usually coincident to a release of pressure within the hydrothermal system. This quartz and metal deposition followed by resealing of the system is repeated over the life of the hydrothermal system resulting in crosscutting and overprinted breccia and vein textures. Typically, the largest and highest grade deposits are associated with long hydrothermal systems marked by complex overlapping veins.

These deposits are strongly structurally controlled. Mineralized fluids are directed along structural pathways with high-grade "ore shoots" typically concentrated in open dilatant zones. These dilatant zones commonly form where inflections occur vertically and laterally along the vein.

Metal deposition and zoning in epithermal deposits are related to the level of boiling. Typically precious metals deposit above the boiling level while base metals precipitate below. Boiling may occur at different levels as the hydrothermal system evolves producing an overprint of various episodes.

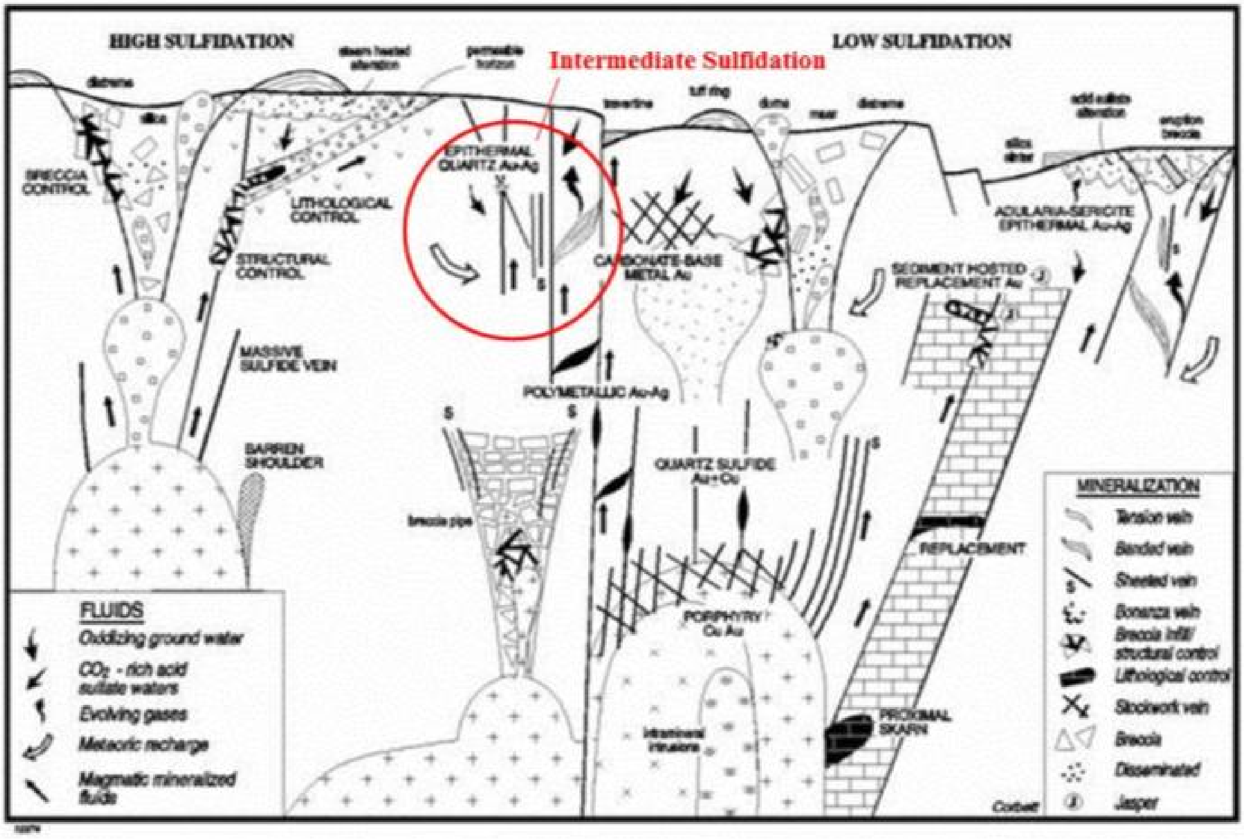


Figure 8-1: Generalized Diagram showing the Spatial Relationship of Intermediate Sulfidation Deposits

(after Corbett 2002, Epithermal Gold for Exploration, AIG News No. 67, 8p)

8.1 ESCOBAL DEPOSIT

The Escobal deposit occurs in a similar geologic setting with host rocks, vein characteristics and mineralogy typical of other intermediate sulfidation systems. Specific definitive features include banded, cockscomb, and drusy vein textures; massive, stockwork and breccia veins; intermediate argillic and quartz-sericite alteration; appreciable base metal and silver-sulfosalt mineralogy and associated arsenic and antimony.

9 EXPLORATION

Exploration at Escobal employs prospecting, mapping and surface geochemical sampling to identify prospective vein zones, followed by drilling. Preliminary work was conducted by Entre Mares de Guatemala as early as 1996 though the project was suspended due to lack of a recognized economic target. Interest in the project was revived in 2006 through more attentive prospecting, recognition of high-grade gold mineralization in the West and East Escobal zones and presence of the extensive silver-base metal-rich Central Escobal zone.

Evolution of the mineralization model at Escobal has been instrumental in identifying the projects resource potential. The recognition of a deep silver-base metal zone below the remnant near-surface gold-bearing cap was paramount in the discovery success. Of equal importance, the understanding of the structural control of the variably-dipping Central Zone vein, and more recently the change in dip of the deep East Zone vein and the discoveries of the West/Margarito and East Extension zones continues to add potential to the project as deeper open portions of the vein zones continue to be defined.

The exploration strategy at Escobal utilizes straightforward exploration techniques that include prospecting for vein and altered outcrops and float, subsequent detailed geologic mapping and surface geochemistry followed by drilling to test the lateral and vertical projections of the surface veins.

Recent step-out drilling has been successful in identifying buried mineralized zones both laterally and to depth from the originally defined Escobal resource areas. These discoveries resulted through systematic drilling along the projections of geochemical anomalies from prior drilling. The new information gained contributes to the understanding of the distribution, zoning and strength of the mineralized system. Based on the results to date, it is believed that there remains significant potential for discovery of still unrecognized mineralization of significance along the Escobal structure.

Several other veins have been identified in the district. The geologic model developed from the Escobal vein will be applied to interpreting these veins and identifying additional targets within the district. At the same time, geologic mapping and prospecting will continue to help identify other styles of mineralization in the district that may include mineralized intrusive and breccia bodies.

Supplementary studies are being undertaken to aid interpretation and discovery of buried veins throughout the region. Currently, spectroscopic (Terraspec®) analysis is being carried out on Escobal and other veins to develop an alteration zoning model that may enhance interpretation and develop other district drill targets. No geophysical studies have yet been employed, though the use of geophysics is being considered in the future to expand exploration in areas of thick alluvial and or post-mineral volcanic cover.

9.1 GEOCHEMISTRY

Gold and silver mineralization in the Escobal vein is typical of intermediate sulfidation deposits with associated epithermal suite of elements including arsenic, antimony, lead and zinc.

Generally, high arsenic, lead and zinc grades correlate with anomalous silver mineralization. Moderate correlations are also observed between silver with antimony and gold with arsenic and lead. Manganese is anomalous throughout the deposit, both as pyrolusite in the oxide portion and possibly as a product of sphalerite in the non-oxide portion of the deposit.

Soil sampling has been completed at 100 m by 25 m spacing over the entire Escobal vein and adjoining areas. Soil and rockchip anomalies confirm trends identified through geological mapping and drilling (Figure 9-1 and Figure 9-2).

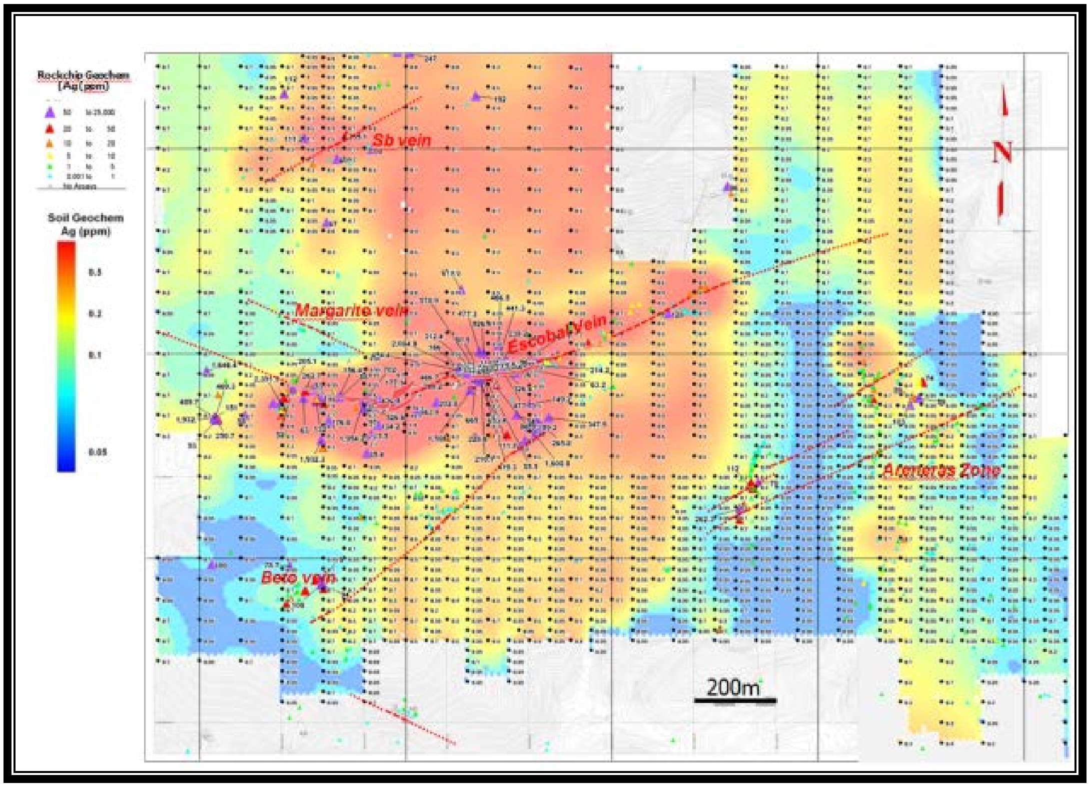


Figure 9-1: Soil and Rockchip Geochemistry – Silver

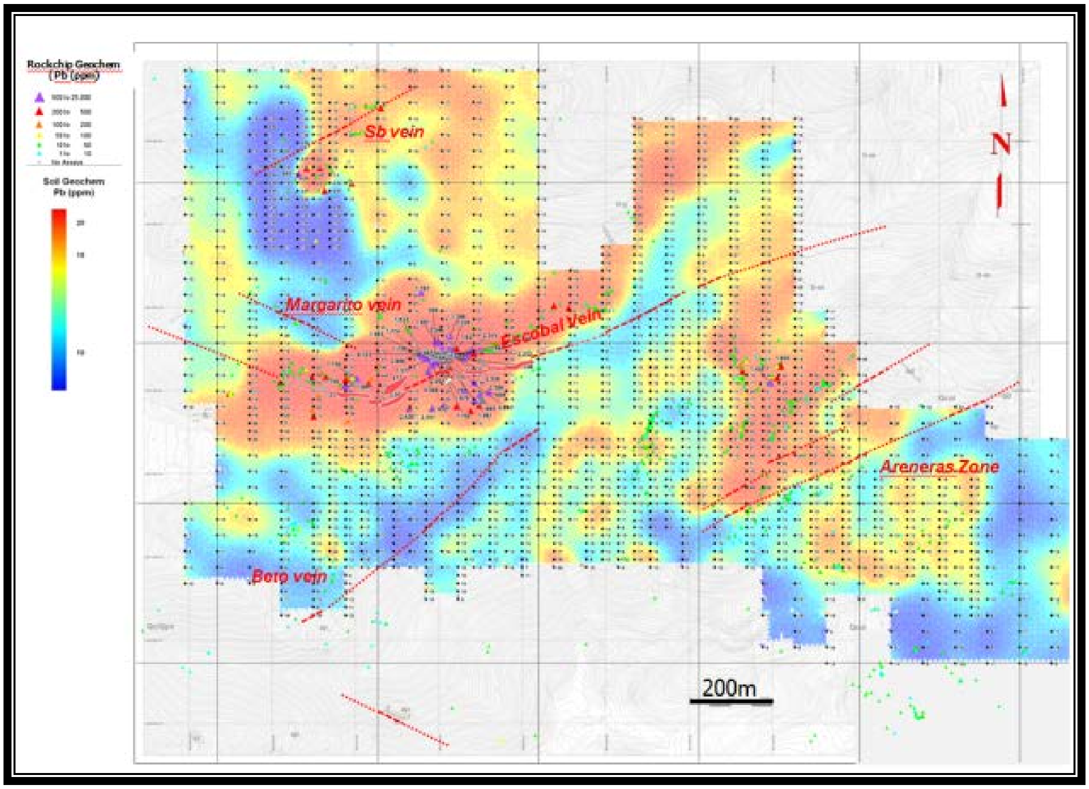


Figure 9-2: Soil and Rockchip Geochemistry – Zinc

The style of mineralization and geochemical zoning patterns vary laterally across the deposit:

East Zone: East Zone geochemistry is characterized by a gold-rich sector in the near surface mixed/oxide zone that abruptly changes with depth to silver-rich mineralization at the oxide/sulfide interface. A small zone of gold mineralization occurs in the deep-east edge of the East Zone, at a similar elevation and possibly related to the deep Central Zone gold domain.

Anomalous lead and zinc concentrations are related to silver mineralization in the sulfide zone. A gradational zoning pattern is observed with silver giving way to lead and then zinc with depth. Absolute lead and zinc values increase with depth relative to silver, suggesting that a pure base metal zone may be imminent below current drilling.

Arsenic is coincident with gold mineralization in the East Zone. Generally arsenic is vertically constrained throughout the deposit, occupying a horizon between 1200 to 1600 meters. In the East Zone anomalous arsenic correlates with the two east-plunging gold "ore shoots" in the mixed/oxide zone, and is irregularly dispersed below gold mineralization in the sulfide zone. Anomalous antimony correlates well with silver and shows a slightly wider dispersion pattern than arsenic.

Central Zone: Geochemistry in the Central Zone is distinctive as no near-surface gold zone is observed. Silver mineralization occurs at a slightly lower elevation than the East Zone and remains open to the east. Anomalous gold grades occur at depth in the central core of the Central Zone with anomalous gold grades at 1350 meter elevation (~ 100-150 meters below surface) extending to a deeper intercept at 1025 meter elevation. This deep gold mineralized core of the Central Zone is believed to represent a distinct zone unrelated to the near-surface gold zones observed in the East and West zones.

Lead and zinc concentrations correlate extremely well with silver in the Central Zone though gradational silver-lead-zinc vertical zoning is not evident as in the East Zone.

Arsenic forms as a large dispersion halo west and above Central Zone mineralization. Antimony correlates well with and shows a minor dispersion around gold-silver mineralization.

West Zone: The West Zone is the most geochemically inconsistent portion of the deposit characterized by surficial gold occurrences, lack of distinct zone of silver mineralization and irregular lead, antimony and arsenic anomalies below the surface gold zone. The West Zone surface gold anomaly occupies a similar elevation range as the upper mixed/oxide East gold zone and is interpreted as the erosional remnant of the same zone. As exploration drilling in the west zone has been largely geared towards definition of the near-surface gold targets, deeper drilling is required to explore zones of deeper silver and gold mineralization and evaluate geochemical signatures in this area.

9.2 DRILLING

The most effective exploration tool at Escobal has been core drilling. Drilling commenced in the spring of 2007 and has continued uninterrupted to the present. As of March 31, 2012 a total of 136,615 meters in 381 holes have been drilled on the Oasis concession. Drilling is discussed in Section 10 of this Report.

Physical attributes recorded through core logging include vein style and intensity, alteration style and intensity, structural style and intensity and sulfide, iron oxide and manganese oxide mineral concentrations. Geochemical attributes include average Ag, Au, Pb, Zn, Cu, As, and Sb contents for most vein intercepts.

The deposit remains open down dip and along strike in both the east and west directions (Figure 9-3). Drilling will continue to define the margins of the identified mineralized zones in the current model and explore for additional zones laterally and at depth. The focus of ongoing exploration is to test deep targets below all mineralized zones as well as the projection of the vein to the west under alluvial valley fill and to the east under post-mineral volcanic cover. The use of larger surface drills and the drilling from underground drill stations are included in the current drill plan.

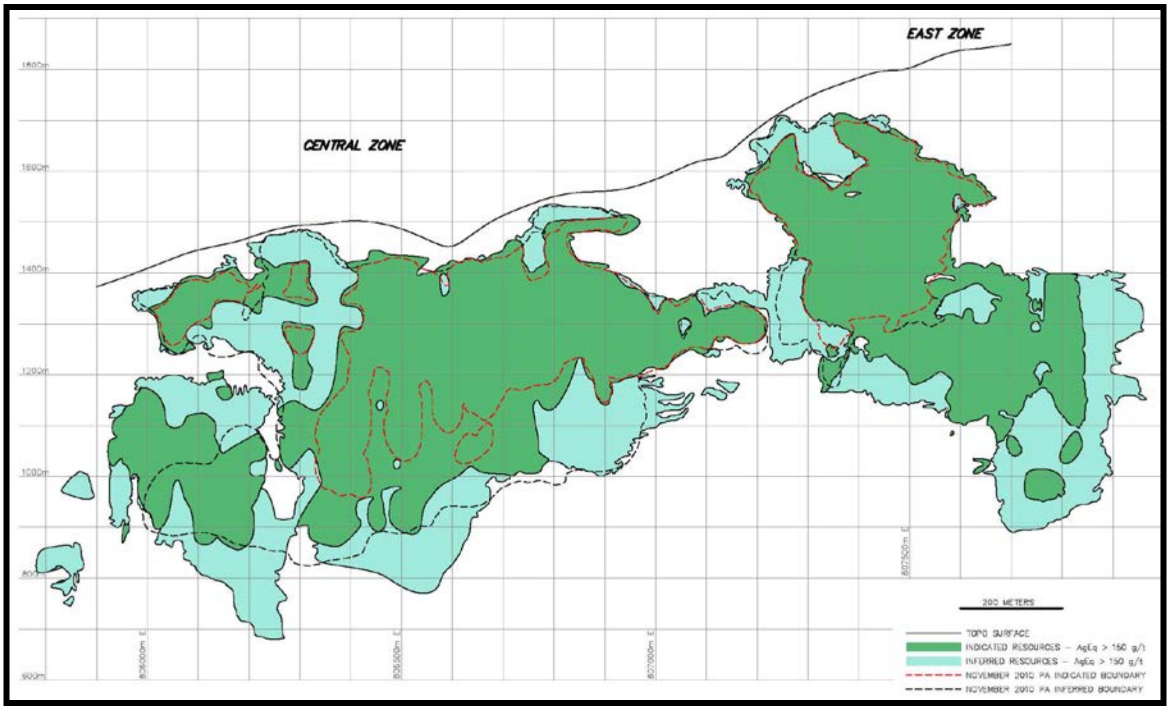


Figure 9-3: Known Mineral Resource and Exploration Potential

9.3 REGIONAL TARGETS

Regional exploration has identified a number of additional targets surrounding the Escobal Project on exploration or reconnaissance concessions which are under application or have been approved. These targets are illustrated in Figure 9-4 and Figure 9-5.

The Neque target parallels the Escobal vein and is exposed in an 800 m diameter window of post-mineral tuff where grades of up to 375 g/t Ag and 16.0 g/t Au were produced through rockchip sampling. Previous work on this property, by Entre Mares includes prospecting in 2006, rock sampling in 2007, soil geochemistry lines in 2007 and geological mapping in 2007. No drilling has been conducted on this property.

The San Nicholas area contains a high-sulphidation alteration target with Au values reported up to 30 g/t within a 1 km^2 alteration area. No drilling has been conducted in the area. The area has been explored by Entre Mares by prospecting in 2000, rock chip sampling in 2007 and 2008, soil sample lines in 2007 and 2008, and geological mapping in 2008. Tahoe carried out some field work, community relations and drill site preparation in 2010 and 2011. Environmental baseline studies commenced in 2011 and a drill plan has been approved by MARN.

The Las Flores target is comprised of a series of east-west trending quartz veins exposed intermittently over a 5 km strike length where grades of up to 150 g/t Ag and 7.0 g/t Au are reported. Previous work on this property by Entre Mares included prospecting in 2006 and 2008, rock sampling from 2006 to 2009 inclusive, soil geochemistry lines in 2008 covering a portion (approximately 5%) of the West Area and preliminary mapping in 2008. This property has not been drill tested.

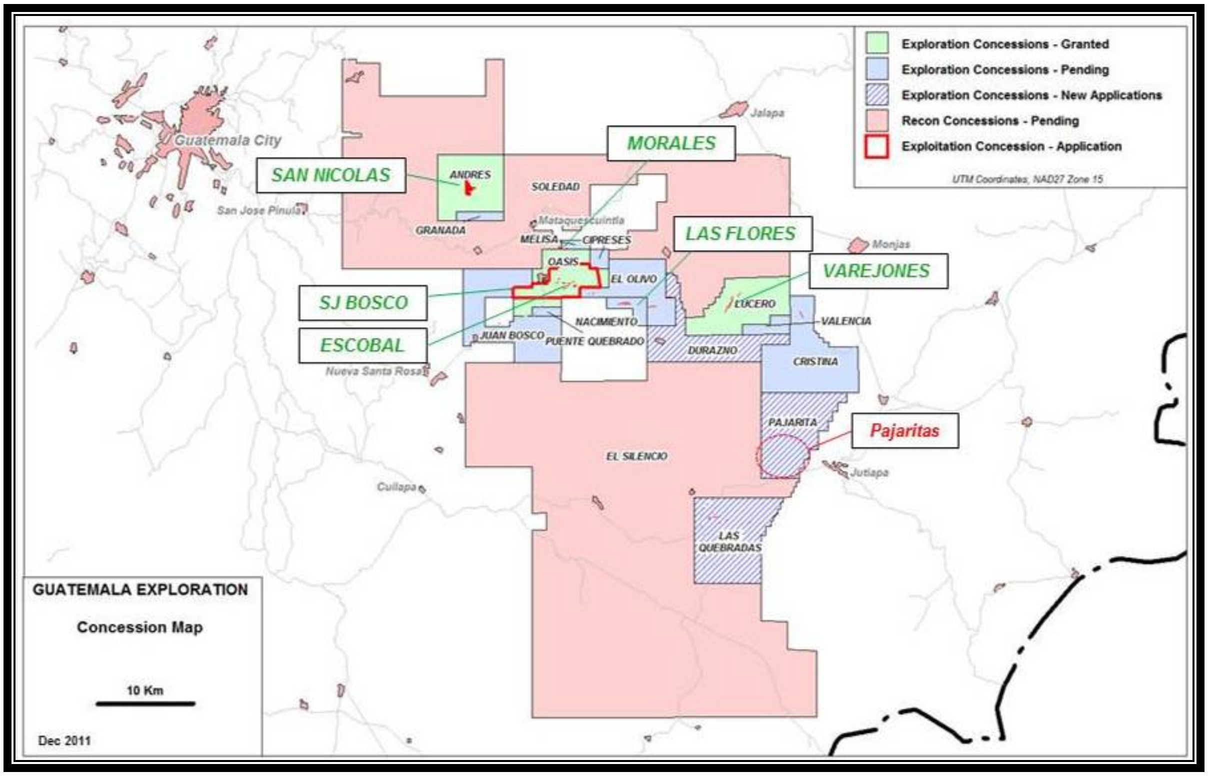


Figure 9-4: Regional Exploration Targets

The San Juan Bosco vein located 6 km west of and parallel to the Escobal vein is a 5 m wide vein that can be traced along a 1 km strike length. A reconnaissance surface sampling program generated results grading up to 15 g/t Ag and 1 g/t Au. Previous work by Entre Mares on this property included prospecting in 2008, rock sampling in 2008, soil geochemistry lines in 2008 and geological mapping in early 2009. Five drill holes were drilled in 2009 testing a portion of the vein where permitted on the Oasis exploration license; JB09-01 contained an 86.64 m interval grading 0.24 g/t Au, which terminated in mineralization. Drill hole JB09-04 contains a single 3 m interval grading 2.37 g/t Au. Additional drilling is warranted to test the higher-grade portion of the vein.

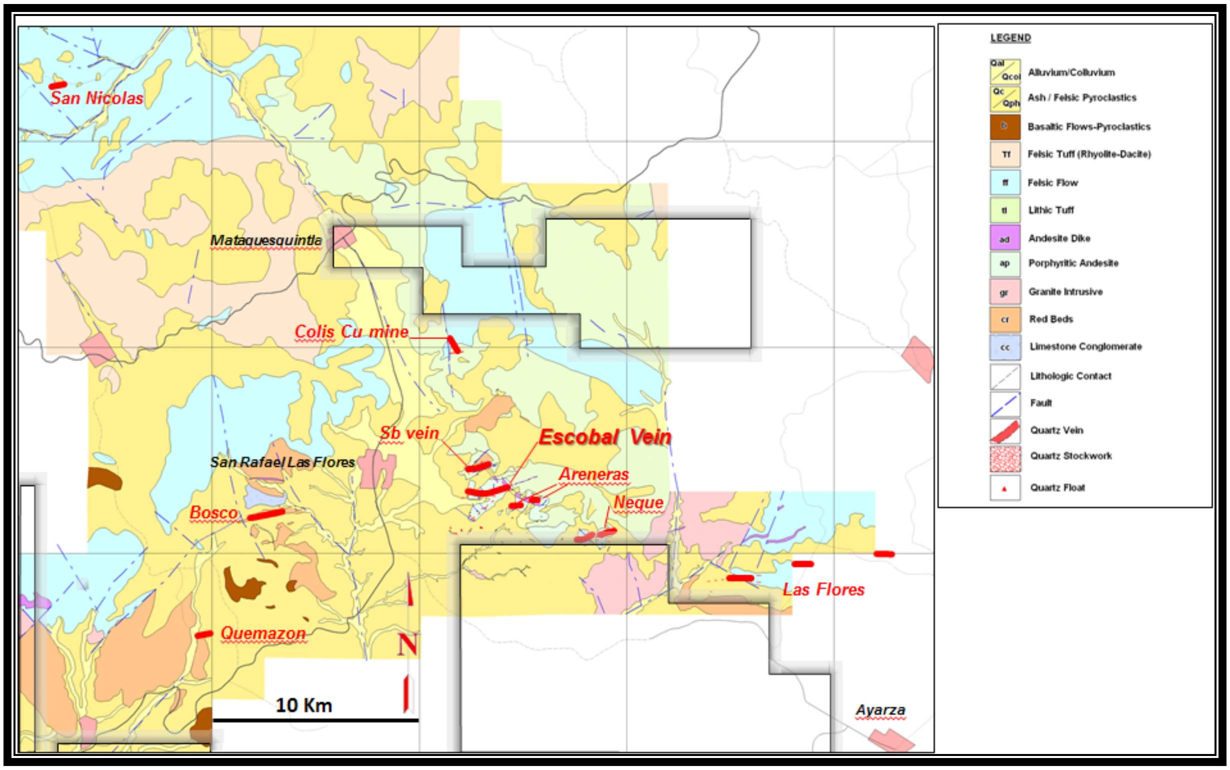


Figure 9-5: Escobal District Exploration Targets

The Varejones target is comprised of numerous red-bed hosted low-sulphidation veins exposed over a 2 km northeast trend where surface results generated values of up to 200 g/t Ag and 4.9 g/t Au. Previous work by Entre Mares on this property includes prospecting in 2001 and 2006, rock sampling in 2007, and soil geochemistry lines in 2007. The target has not yet been drilled.

10 DRILLING

Drilling on the Oasis concession has been conducted by Entre Mares and Tahoe from 2007 to the present. There have been 381 drill holes totaling 136,615 meters completed on the Escobal and surrounding veins through March 31, 2012. Data acquired through December 31, 2011 have been used for the Escobal resource model and estimate reported herein; the dataset used for resource estimation was comprised of 350 drill holes totaling 121,639 meters, including data from 21 holes drilled to obtain metallurgical samples.

Holes drilled in the Oasis concession and not within the current Escobal resource estimate include ten exploration drill holes completed by Entre Mares in 2007 that targeted outlying exploration areas (Areneras, Bosco, and Granadillo veins) and three exploration holes drilled by Tahoe at the Morales exploration area in 2011.

Drilling at Escobal has been by diamond drill (core) methods, using 1.52 m and 3.04 m (5-ft and 10-ft) core barrels. The majority (66%) of mineralized intercepts were drilled using NTW-size or larger drill core, with lesser amounts of NQ2-, BTW-, and BQTK-size drill core. Core recovery averages 96% over the life of the project.

Six diamond drill holes were precollared through unmineralized rock using reverse circulation (RC) drilling; four drill holes precollared by RC have yet to be continued with diamond drilling. In addition, 37 small diameter (AQ-size) Winkie core holes were drilled at Escobal in 2010 and 2011; these drill holes were used as a ‘first pass’ prospecting tool or to gather near-surface geologic data for future plant site construction planning. No RC or Winkie drill samples are included in the drill hole database used for the resource estimate.

Summaries of the drilling completed to date and drill holes used for the resource estimate are presented in Table 10-1 and Table 10-2, respectively. Figure 10-1 is a plan map illustrating the drill hole locations at Escobal.

Table 10-1: Total Oasis Concession Drilling through March 31, 2011

Company	Area	Type	No. Drill Holes	Total Length (m)
Entre Mares	Escobal	Exploration	213	58,156
	Areneras	Exploration	3	601
	Granadillo	Exploration	2	425
	Juan Bosco	Exploration	5	1,294
		Total	**223**	**60,476**
Tahoe Resources	Escobal	Exploration	131	68,810
	Escobal	Metallurgical	21	4,943
	Escobal	Piezometer	3	900
	Morales	Exploration	3	1,487
		Total	158	76,139
		Grand Total	**381**	**136,615**

Table 10-2: Escobal Drilling Included in Resource Estimate (Drilling through Dec 31, 2011)

Area	No. Drill Holes	Total Length (m)
East Zone	132	37,597
East Zone Extension	29	17,735
Central Zone	127	46,688
West Zone / Margarito	62	19,619
Total	**350**	**121,639**

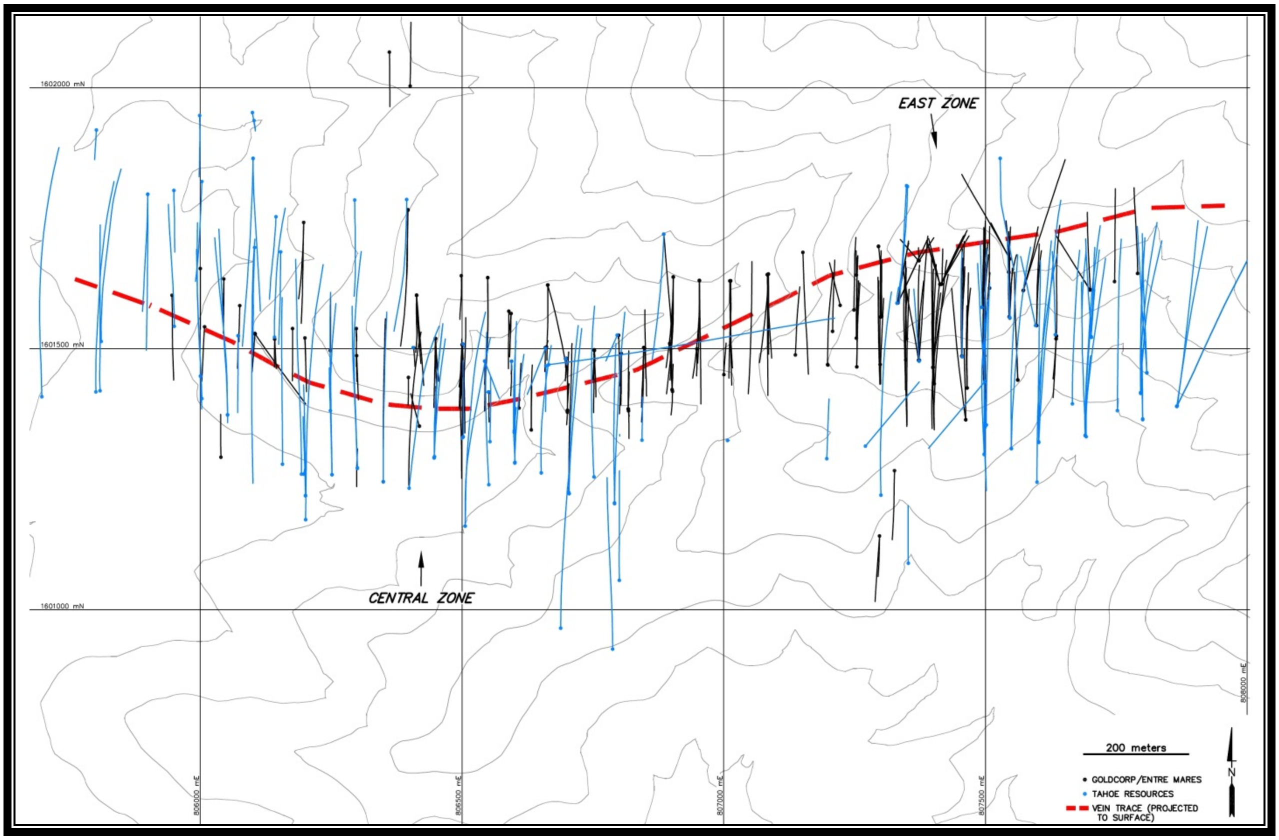


Figure 10-1: Escobal Drill Hole Location Map

10.1 DRILL CAMPAIGNS

Entre Mares

Entre Mares conducted drilling campaigns on the Oasis concession from 2007 to June 2010, during which time they completed 223 diamond drill holes totaling 60,476 meters; all but ten drill holes targeted the Escobal vein system. Entre Mares' drilling was conducted by Kluane Guatemala S.A. (a division of Kluane International), using KD600 and KD1000 drill rigs, and by Entre Mares personnel, using company-owned Hydracore drills.

Tahoe Resources

Upon acquisition of the Escobal property and facilities in June 2010, Tahoe continued the exploration drilling program begun by Entre Mares without interruption, with Kluane Guatemala S.A. as the drill contractor using the same KD600 and KD1000 drill rigs. In addition Tahoe Resources purchased a Hydracore 2000 man-portable drill and an LM-75 drill in early 2011. Both drills were utilized throughout the 2011 exploration drill program.

From June 2010 through March 2012, Tahoe completed 158 drill holes totaling 76,139 meters on the Oasis concession; all but three of these drill holes were targeted at the Escobal vein system.

Tahoe completed 113 exploration drill holes totaling 57,640 meters prior to the December 31, 2011 cutoff date for inclusion of drill data into the resource model.

In mid-August 2010 through to early 2011, Tahoe initiated a diamond drilling program to acquire core samples specifically for metallurgical and physical property testing. Rodio-Swissboring Guatemala S.A. was contracted to drill five large diameter core holes (PQ-size) to acquire samples for comminution testing. No assay data was obtained from this drilling. An additional 16 core holes were drilled to obtain samples for metallurgical variability tests. These holes were drilled by Kluane Guatemala S.A. and Tahoe (HQ- and NTW-size drill core). Assays from the variability test samples are included in the resource estimate.

In addition, data from three piezometer wells are included in the resource estimate.

10.2 DATA COLLECTION

As the project manager and majority of on-site geologic personnel remain unchanged from the transfer of the property from Entre Mares to Tahoe, data collection procedures are generally consistent between the two companies. Hence, the following descriptions are applicable to both the Entre Mares and Tahoe drilling programs, except as noted.

Drill Core Handling

As the core barrel is retrieved from the drill hole, the core is removed and placed in wooden core boxes along with markers labeled with the downhole distance. The core boxes are labeled and transported by pickup truck to the core logging facility at the project site, where company geologists and technicians wash and photograph the core, record the geologic and geotechnical characteristics, and mark drill core intervals for sampling. The core is sampled by sawing the core in half longitudinally. After logging and sampling are complete, the core boxes are transported either to a secured storage facility in San Rafael Las Flores or to storage facilities at the project site, where it is stored on racks inside covered buildings.

Drill Collar Surveys

At the completion of each drill hole, the collar locations are marked in the field with a four-inch plastic (PVC) pipe cemented into the top of the drill hole. The drill hole identification number is indicated by permanent marker on the PVC pipe and etched in the cement at the collar. All drill collar locations were surveyed by Sergio Diaz (2007-2010) or Geotecnología S.A. (2009-2011), both independent professional surveyors based in Guatemala City. Collar locations were determined using non-differential global positioning system (GPS) instruments and post processed. In some cases where topography or heavy vegetation prevented collection of accurate measurements by GPS, the surveyor employed a Total Station instrument using established drill collar locations or project control points with known coordinates as bench marks. All drill hole collar coordinates are reported in UTMm coordinates, NAD 27, Zone 15 and converted to the Escobal site Cartesian coordinate system. The original reports received from the surveyor are archived at the Escobal project office and the reported collar coordinates stored in the Escobal project digital database.

Downhole Surveys

Downhole survey measurements are taken by the drill contractor or company personnel at approximate 50 meter intervals and at the final depth of each drill hole. Entre Mares used a Tropari down-hole survey instrument through to mid-2009, after which they used a Reflex EZ-shot digital down-hole survey tool. All holes drilled by Tahoe were surveyed using the Reflex EZ-shot tool. A 3° west magnetic declination correction was routinely applied to raw azimuth readings for all drill holes. The survey readings are entered into the Escobal project digital database with the original survey datasheets archived at the Escobal project office.

Geological Logging

Geologic data from drill core is originally recorded on paper logging forms and then entered into the digital project database. Data documented from the drill core includes lithology; primary and secondary rock textures, vein lithology; mineralization and alteration; estimated sulfide content; structural features, including the angle of structure to the core axis; and degree of iron oxidation.

Geotechnical Logging

All drill core has been logged for geotechnical data. Geotechnical data collected from the drill core includes core recovery, hardness, rock quality designation (RQD), joint number (Jn), joint roughness (Jr), joint alteration (Ja), joint water reduction factor (Jw), and the stress reduction factor (SRF); all of which is entered into the project database. From this data, geomechanical classifications – tunneling quality index (Q rating) and rock mass rating (RMR) – are calculated to identify the ground control measures appropriate for the rock quality anticipated during underground excavation, determine appropriate stope dimensions (span height and width), and estimate mining dilution. A summary of the geotechnical data and subsequent analysis is detailed in the *Geotechnical Characterization* section.

10.3 DRILLING SUMMARY AND RESULTS

Both the Entre Mares and Tahoe drilling programs at Escobal have targeted the vein system with diamond drill core holes oriented perpendicular to the general east-west strike direction of the deposit, as illustrated in Figure 10-1, and at varying inclinations to explore the deposit along dip.

To date, drilling has intersected mineralization over approximately 2,200 meters of strike length and 1,000 meters of total vertical extent; elevations range from 980 to 1710 meters above sea level (masl) in the East Zone, from 910 to 1540 masl in the Central Zone and from 700 to 1450 masl in the West Zone. In general, the Escobal deposit has been drill-delineated in the east-west direction (*i.e.*, along strike) on nominal 50-meter spaced intervals, with numerous holes drilled between the 50-meter intervals, particularly in the East Zone. The drill sample spacing is sufficient for the geologic modeling and resource estimation of the Escobal vein system.

The East Zone and East Zone Extension strike approximately azimuth 80° and dip to the south from 60° to 80°, with an average dip of approximately 70° south. As such, the majority of drill holes are oriented to the north, with a few holes oriented south to explore for north-dipping secondary veins. Drill hole lengths range from 9.9 meters to 867.5 meters, with an average drill hole length of 343.7 meters. Average drill spacing in the East Zone is approximately 40 meters, with drill spacing in the 'core' of the East Zone at about 34 meters. Average drill spacing in the East Zone Extension is 64 meters.

The Central Zone strikes approximately east-west with variable dips with the western portion (West/Margarito Zone) trending slightly to the north of west. The mineralized structure generally dips from 60° to 70° to the north from the surface down to around 1200 meters elevation and steepens to near-vertical at depth. The upper portion of the deposit in eastern half of the Central Zone dips 60° to 70° to the south (East Zone orientation). Accordingly, drill holes were oriented northerly to explore the south-dipping portion of the mineralization and southerly to explore the north-dipping portion of the mineralization. Drill hole lengths range from 8.2 meters to 995.2 meters, averaging 349.8 meters. Average drill spacing in the Central Zone is approximately 52 meters; drill spacing in the 'core' of the Central Zone is 41 meters. Drill spacing in the West/Margarito area is about 65 meters.

A summary of the drill hole information by zone is presented in Table 10-3. Figure 10-2 and Figure 10-3 are cross sections through the East and Central zones, respectively, illustrating the relationship between drill hole orientation and the geometry of the Escobal vein. As shown, the relationship between drilled length and the true width of mineralization is variable, dependent on the inclination of the drill hole relative to the vein dip.

Significant intercepts for the Escobal deposit, including drilled widths and estimated true widths, are summarized in Appendix B.

Table 10-3: Drill Hole Summary by Zone

East Zone		North Oriented Drill Holes	South Oriented Drill Holes	Vertical (≥ -85°) Drill Holes
Drill Holes	Number	120	8	4
	Min Length (m)	9.9	100.0	83.5
	Max Length (m)	577.3	803.1	467.9
	Avg Length (m)	281.7	334.0	279.3
Azimuth	Range	320° to 040°	174° to 220°	n/a
	Avg	0°	185°	n/a
Inclination	Range	-45° to -80°	-45° to -84°	-85° to -90°
	Avg	-58°	-65°	-87°

East Zone Extension		North Oriented Drill Holes	South Oriented Drill Holes	Vertical (≥ -85°) Drill Holes
Drill Holes	Number	28	1	0
	Min Length (m)	237.9	-	-
	Max Length (m)	867.5	-	-
	Avg Length (m)	611.4	615.7	-
Azimuth	Range	356° to 026°	-	-
	Avg	001°	181°	-
Inclination	Range	-47° to -71°	-	-
	Avg	-58°	-69°	-

Central Zone(1)		North Oriented Drill Holes	South Oriented Drill Holes	Vertical (≥ -85°) Drill Holes
Drill Holes	Number	42	68	16
	Min Length (m)	8.2	46.3	59.4
	Max Length (m)	888.0	831.6	511.1
	Avg Length (m)	459.7	318.3	315.1
Azimuth	Range	357° to 010°	158° to 203°	n/a
	Avg	0°	180°	n/a
Inclination	Range	-45° to -72°	-45° to -84°	-85° to -90°
	Avg	-56°	-67°	-88°

West/Margarito Zone		North Oriented Drill Holes	South Oriented Drill Holes	Vertical (≥ -85°) Drill Holes
Drill Holes	Number	18	41	3
	Min Length (m)	55.2	25.9	45.7
	Max Length (m)	995.2	693.4	146.3
	Avg Length (m)	485.0	258.2	99.9
Azimuth	Range	344° to 000°	145° to 184°	n/a
	Avg	358°	179°	n/a
Inclination	Range	-45° to -69°	-44° to -78°	-89°
	Avg	-57°	-59°	-89°

(1) One hole in Central Zone (E11-342) oriented East at 83° azimuth and -33° inclination excluded from summary

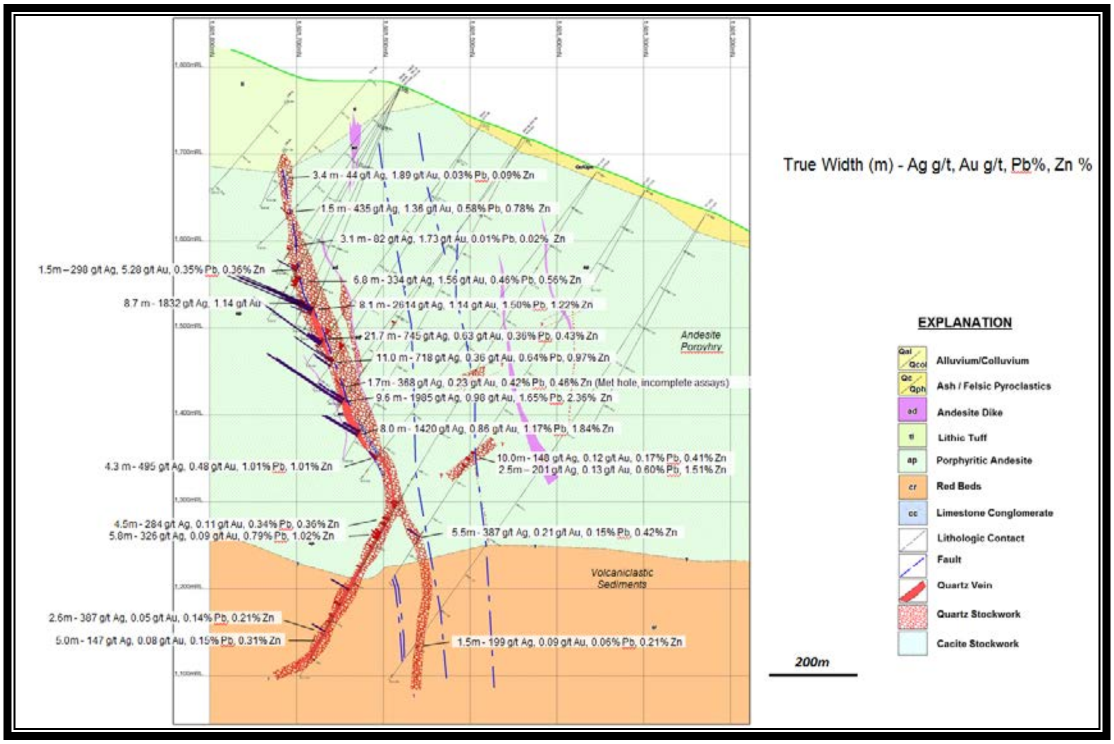


Figure 10-2: Escobal East Zone – Cross Section 807500E (looking East)

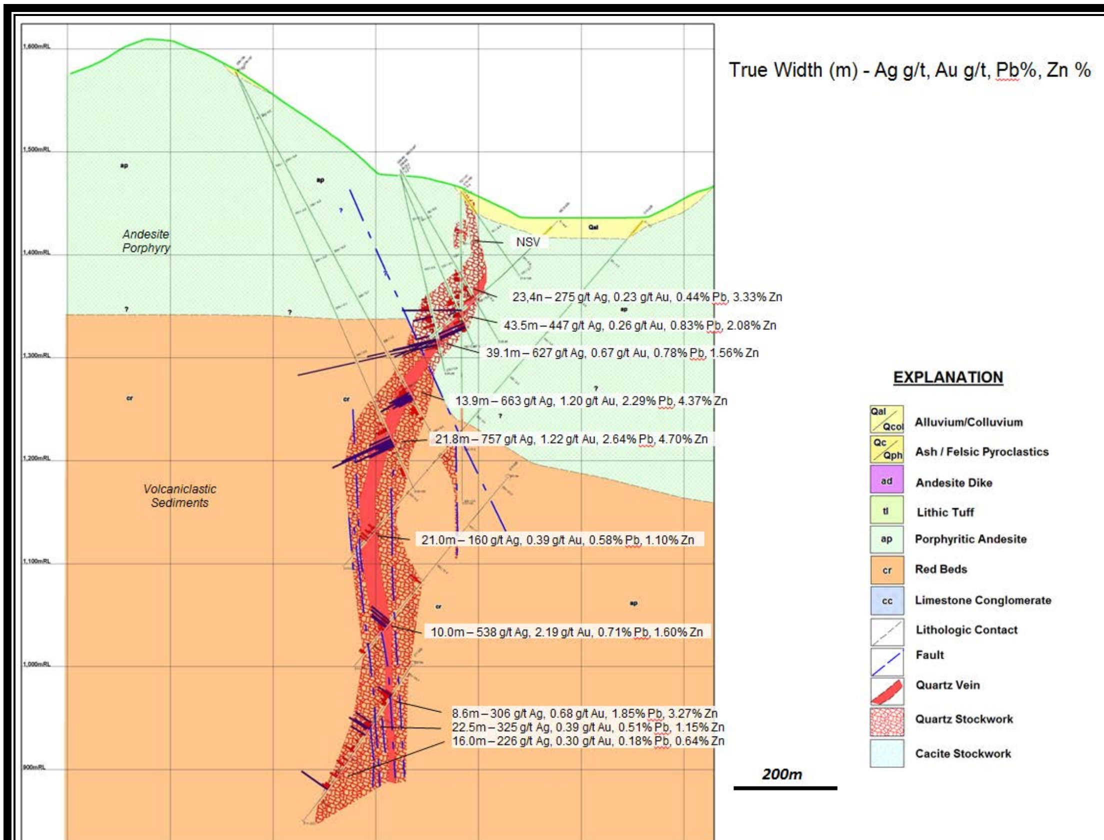


Figure 10-3: Escobal Central Zone – Cross Section 806500E (looking East)

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Sample preparation, analyses, and security procedures generally remained consistent between Entre Mares and Tahoe and the following descriptions are applicable to the practices of both Entre Mares and Tahoe, except as noted. With few exceptions, Entre Mares used BSI Inspectorate as their primary analytical laboratory and Tahoe continues to use BSI Inspectorate (now named Inspectorate, a division of Bureau Veritas) as their primary laboratory. Inspectorate holds current ISO 17025 and ISO 9001:2000 certifications.

11.1 SAMPLE METHOD AND APPROACH

The sampling methodology remained consistent from Entre Mares to Tahoe; whereas the Escobal exploration manager and many of the on-site geologic personnel responsible for the drill core sampling remained in place following the transfer of the property from Entre Mares to Tahoe Resources in 2010.

Once the drill core has been logged for geologic and geotechnical properties (as described in Section 10.2), geologists determined sample intervals based on geologic and/or mineralogic changes. Sample intervals generally varied from less than one meter to one-and-one-half meters in zones of discreet mineralization, and from three meters to locally six meters in weakly mineralized or altered areas. Tahoe Resources' geologists sampled drill core that is visually identified as having significant mineralization on one-and-one-half meter intervals; weakly mineralized core is sampled at three meter intervals. These sample lengths are appropriate for the differing styles and distribution of mineralization at Escobal, though it is recommended that sample intervals do not extend across obvious mineralogic contacts. Intervals of 'fresh' unaltered rock are normally excluded from sampling. Once the sample intervals are determined, the core is marked and sample tags are stapled to the core box dividers.

Core samples selected for analysis are cut lengthwise using mechanized diamond saws. One-half of the core is placed in a plastic sample bag with a sample tag. The remaining half core is replaced in the core box for future reference. The mineralized zones at Escobal are often quite wide (up to 50 meters) and complex (multiple cross-cutting vein events). The practice of submitting one-half of the core provides a reasonable representation of the mineralization for analysis.

11.2 SAMPLE SECURITY

After the drill core is logged and sampled at the project site, the samples are taken to San Rafael Las Flores where they are stored in the secured office/warehouse facility until delivered to Inspectorate's sample preparation laboratory in Guatemala City. From 2007 to 2008, all samples were picked up by Inspectorate at the San Rafael office. Since 2008, the samples have been delivered to the Inspectorate prep lab using Tahoe's drivers and vehicles. BSI holds duplicate sample pulps in secured storage in Guatemala City and returns them on a routine basis to a secured Tahoe facility in San Rafael Las Flores.

11.3 LABORATORY SAMPLE PREPARATION

Since the initiation of drilling at Escobal in 2007, all drill core samples have been prepared by BSI Inspectorate at their preparation facility in Guatemala City. After drying, core samples were crushed to >80 percent passing 2 mm (10 mesh) using a jaw crusher and roll mill. The crushed samples were then passed through a Jones riffle splitter to obtain a nominal 300 gram sample for pulverization. The 300-gram subsample was pulverized to >90 percent passing 150 mesh and split into two sample pulps for primary and check analyses. Barren sand is used to clean the pulverizer after every sample; one sample of the barren sand is inserted into the sample stream per batch where it is reported as an internal laboratory blank. BSI Inspectorate packaged and air-freighted one set of pulps to the BSI Inspectorate laboratory in Reno, Nevada for analysis and delivered the second set of pulps to Entre Mares and Tahoe at site.

11.4 LABORATORY ANALYSES

BSI Inspectorate in Reno, Nevada is the primary laboratory for nearly all of the assaying of drill core at Escobal, with the exception of 79 metallurgical samples from Entre Mares' 2008 drilling campaign that were assayed ALS Chemex for Au and Ag and 315 samples from Tahoe Resources' 2010 metallurgical program that were assayed at Cardwell Analytical (wet assays) and ALS Chemex (ICP).

BSI Inspectorate determined silver grades using aqua regia digestion followed by atomic absorption spectrometry (AAS) and, to a lesser extent, induced-coupled polarization (ICP). The use of ICP for silver grade determination was discontinued in late 2007. For initial silver results exceeding 200 g/t, Entre Mares instructed BSI Inspectorate to automatically re-analyze the sample using fire assay with gravimetric finish. Tahoe continues this practice, but uses a lower grade threshold of 100 g/t.

Gold analyses were done by fire assay (one assay-ton) followed by AAS. Samples returning more than 3 g/t were re-assayed with a gravimetric finish. Assayed sample mass varied depending upon level of sulfide in the sample to limit losses caused by boil-over in the assaying process.

Sample pulps were also analyzed for a multi-element geochemical suite using aqua regia digestion followed by ICP. Lead, zinc, and copper values exceeding 1% were re-analyzed by aqua regia/AAS, which has a higher grade determination threshold than ICP. Base metal samples exceeding the threshold of AAS were assayed using titration methods.

11.5 QUALITY ASSURANCE/QUALITY CONTROL PROCEDURES

Quality assurance/quality control (QA/QC) procedures for drill core sample analyses include the use of standard reference materials, sample blanks, check assays, and duplicate samples, as described herein. Results and analysis of the QA/QC data is presented in 12.0, *Data Verification.*

11.5.1 Standard Reference Materials

Tahoe incorporates blind standard reference material (assay standards) into the sample stream prior to submission of the samples to BSI Inspectorate at a rate of one assay standard per 20 drill samples (5%). Four assay standards of varying silver, gold, lead, and zinc grades are used. Assay standards are prepared and certified by CDN Resource Laboratories Ltd. of Langley, BC. Entre Mares did not use assay standards in their QA/QC program at Escobal.

In addition, BSI Inspectorate also includes reference materials (both in-house and certified reference materials) in its QA/QC program.

11.5.2 Blanks

Entre Mares and Tahoe inserted sample blanks into the sample stream at irregular intervals to check for contamination during the laboratory sample preparation stage. Samples assaying less than five parts per billion gold and 0.1 parts per million silver were collected from local outcrops at Escobal for use as sample blanks. These samples are valid as gold and silver blanks, but do contain trace amounts of lead and zinc. The project database includes 1,554 assay blanks submitted by Entre Mares and Tahoe.

BSI Inspectorate also monitored pulverizer contamination by collecting and analyzing the barren sands used to clean the pulverizer after each sample.

11.5.3 Check Assays

Check assay programs have been in effect at Escobal since the initiation of the exploration drilling campaigns. A total of 5,108 sample pulps, representing 23% of the samples in the Escobal database, were re-assayed by laboratories other than BSI Inspectorate including samples submitted to more than one outside laboratory for redundant check assaying.

Entre Mares' submitted the second pulp split prepared by BSI Inspectorate to a second laboratory for check assaying for nearly all mineralized drill intercepts. Entre Mares used the laboratory at the Marlin Mine in Guatemala for conducting silver and gold check assays beginning with their first drill hole at Escobal in May 2007 through May 2008, and again from July 2009 through the end of their involvement with the property in June 2010. From June 2008 through July 2009, Entre Mares used CAS Honduras for the silver and gold check assaying. A small percentage of samples were also checked by ALS Chemex in Vancouver.

SGS and CAS Honduras both analyzed for silver and gold using fire assay with AAS finish, with high grade results reassayed by fire assay with gravimetric finish. ALS Chemex analyzed for silver and gold using fire assay with gravimetric finish and analyzed for lead and zinc using four-acid digestion with AAS. High grade 'overlimit' lead and zinc results were reanalyzed by volumetric methods (titration).

For Tahoe's check assay program, BSI Inspectorate shipped 5% of the assay pulp splits to ALS Chemex (Vancouver, BC or Reno, Nevada) for reanalysis in 2010. In 2011, BSI Inspectorate shipped 25% of mineralized sample interval pulps to ALS Chemex for check analyses. As before, ALS Chemex analyzes for silver and gold by fire assay and gravimetric finish and analyzes for lead and zinc using four-acid digestion with AAS. High grade 'overlimit' lead and zinc results are reanalyzed by titration.

11.5.4 Duplicates

Duplicate samples are collected after the first stage of crushing (coarse rejects) as opposed to check-assay samples, which are sample pulps. There are a total of 961 duplicate sample analyses in the Escobal database, representing approximately 4% of total samples collected.

From May 2007 through July 2009, Entre Mares submitted coarse reject duplicates generally at the rate of one in 15 samples to CAS Honduras for reanalysis of gold and silver by fire assay/AAS. From July 2009 through May 2010, Entre Mares sent coarse-reject splits to ALS Chemex in Vancouver, though on a much more irregular schedule. ALS Chemex completed the sample preparation process and analyzed the new sample pulps for silver and gold using fire assay with gravimetric finish and for lead and zinc using four-acid digestion with AAS. High grade 'overlimit' lead and zinc results were reanalyzed by titration. Tahoe has discontinued the use of coarse reject duplicate analyses.

11.6 CONCLUSIONS

MDA believes that the core sampling procedures, sample analyses, QA/QC procedures, and sample security have provided samples that are of sufficient quality for use in the resource estimation discussed in Section 14.

12 DATA VERIFICATION

Mine Development Associates ("MDA") verified the Escobal database on two occasions; first in 2010 for the November 2010 mineral resource estimate and again in 2012 for the current resource estimate. The verification work consisted of: 1) completing an audit of the full assay database; 2) checking a significant percentage of the drill location and survey data; 3) conducting two site visits, which included verification sampling and a review of sample handling and logging procedures; 4) reviewing the QA/QC data; and 5) analyzing the core recovery data and its relationship to metal grades. The results of this verification program support the estimation of the Escobal resource and the assignment of an Indicated classification to much of the stated resource.

12.1 DATABASE AUDIT

The discussion that follows includes the information that appeared in the 2010 technical report, with additions describing MDA's 2012 data verification procedures and results.

12.1.1 2010 Assay Audit and Database Reconstruction

The Escobal assay database includes results of the primary analyses for silver, gold, lead, and zinc, plus a 32-element geochemical suite, completed by BSI Inspectorate ("Inspectorate"). The database also includes the secondary check and duplicate analyses completed by ALS Chemex ("Chemex"). The metal values were listed by sample ID number and corresponding drill-hole "from-to" down-hole sample interval. The assay database does not include any of the check assay data from the Marlin Mine lab or the CAS Honduras lab due to the lack of back-up data and the inability to verify any of these analyses.

Each sample interval's final "accepted" metal value was an average of multiple analyses if duplicate and/or check assay values were present. A number of different analytical techniques was employed by the labs which resulted in some uncertainty as to what values should be used in determining the accepted database metal value. As an example, silver techniques ranged from induced-coupled polarization ("ICP") to aqua regia digestion followed by atomic absorption spectrometry ("AAS") to fire assay with gravimetric finish. (See discussion of all analytical techniques in Section 11.0) After discussions with Tahoe, a hierarchy of assay techniques was established for each metal, and it was decided that the "accepted value" would be calculated using only the value(s) for the highest-ranked technique. Using the silver example from above, for intervals with both ICP and AAS techniques, only the AAS value(s) would be considered.

In reviewing the assay data, it was recognized that there was a significant number of analyses in which the analytical technique was not known, so creating an assay hierarchy was not possible. To help rectify this problem, and to also serve as a thorough audit of the assay data, MDA downloaded all of the assay data, including information on assay techniques, directly from the labs and 1) compared these data with the database values, and 2) sorted all data into their proper technique. These imported data were then used to reconstruct the database and determine the final accepted values to be used in the resource estimate.

The sample data sent to the labs uses a sample ID code that is blind to the lab as to the drill hole number and from-to location. MDA conducted an audit of the sample ID/drill hole from-to correlation by comparing the hard-copy sample selection data against the database. Approximately 15 percent of the database was checked and no errors were noted, though a result of this effort was the realization that the current database was missing sample interval data from recent holes E10-221, 223, 224, and 225. All of the missing intervals were in weakly mineralized zones outside of the current mineral model, so their exclusion from the resource estimate is not considered material.

The drill sample down-hole locations were also checked while on site by comparing the database from-to values directly against the sample intervals marked within the core boxes for 12 drill holes. No errors were noted.

12.1.2 2012 Assay Audit

The post-2010 assay analyses were completed by BSI Inspectorate ("Inspectorate") with secondary check and duplicate analyses completed by ALS Chemex ("Chemex") Both labs used similar analytical techniques as in 2010. As in 2010, the current drill samples were submitted to the lab using a sample ID code that is blind to the lab as to the drill hole number and from-to location. The 2012 database was standardized to use only the primary assay values from Inspectorate and no further reconstruction of the assay data was required.

MDA conducted a thorough audit of the post-2010 assay data using the same techniques and procedures as described above for the 2010 resource estimate. All assay data was downloaded directly from the lab and compared with the database values. Two clerical errors concerning a sequence of lead and zinc values were noted and corrected in the database. No other material errors were found.

The drill sample down-hole locations were also checked while on site by comparing the database from-to values directly against the sample intervals marked within the core boxes for six drill holes. No errors were noted.

12.1.3 2010 Drill Sample Locations and Down-Hole Surveys

The drill sample collar locations for the pre-Tahoe drill holes were audited against the original spreadsheet data from the third-party surveyor, and no errors were found. MDA updated the database with the survey data for the recent Tahoe drilling and then checked all locations by plotting the hole collars on cross-sections and comparing the locations with the digital topography. A number of drill holes in areas of steep topography had a ±3m elevation difference with the topography and/or adjoining drill holes. After discussion with Tahoe, MDA adjusted the elevations on 21 drill holes to better match the existing data. The uncertainty in some of the collar elevations is not considered significant for the resource estimation to be classified at a level of Indicated.

The down-hole survey data for 36 holes (approximately 17 percent of the database) were audited. For 17 of the holes audited, MDA compared the database values against a visual inspection of the original Sperry Sun camera discs produced by the Tropari survey instrument; only occasional minor discrepancies (<2 degrees) were noted between MDA's reading of the discs and the current database. For the remainder of the drilling, MDA compared the database survey data against the original survey coupons created on-site by the survey crew. One significant error was noted (a difference of 10 degrees in the dip angle), and the database was corrected to reflect the new data.

12.1.4 2012 Drill Sample Locations and Down-Hole Surveys

The 2012 project drill location database submitted to MDA was in the Escobal site Cartesian coordinate system. Using the X,Y conversion formulas provided by Tahoe, MDA checked for consistency all of the pre-2010 drill collar coordinates against the current database. MDA then audited the post-2010 drill holes against the original spreadsheet data from the third-party surveyor. Several discrepancies in the most recent hole locations were found in the initial audit. After going through the survey records with project personnel, it was determined that the database contained preliminary survey data which had not been replaced with the final coordinates. The database was revised to include all final data. All hole locations were also checked by plotting the hole collars on cross-sections and comparing the locations with the digital topography.

The down-hole survey data for 11 holes (approximately 12 percent of the post-2010 database) were audited. MDA compared the database survey data against the original survey coupons created on-site by the survey crew. No errors were noted.

12.2 SITE VISITS

Paul Tietz of MDA visited the project site on September 7th through the 10th, 2010 and again on February 6^{th} through the 9^{th}, 2012. The purpose of the visits was to review the Escobal deposit drilling and sampling procedures, results, and geology in preparation for the resource modeling, and to complete the remaining data audit tasks required for the 2010 and 2012 resource estimates (as discussed in Section 12.1). Specific data verification items included database construction and recordation, drill-hole location and down-hole survey validation, and QA/QC methods, the results of which are discussed in Sections 12.1.1, 12.1.3, and 12.4, respectively. A limited amount of core was evaluated, and verification samples were collected in 2010 from four core holes. During each site visit, time was spent in the field verifying and discussing drilling and sampling procedures and geologic concepts with project personnel.

12.2.1 Drilling Operations

Core rigs were operating on the property during both of MDA's site visits. A detailed description of the core drilling campaigns and procedures is provided in Section 10. The drilling procedures observed while on-site were consistent with industry standards, and no drilling issues were observed or discussed with project personnel which would negatively impact the resource estimate.

12.2.2 Sampling and Logging Procedures

The current logging and sampling procedures meet industry standards/practices and are sufficient to allow for confidence in the upcoming resource estimate. The one improvement suggested by MDA is the use of more selective sampling for both the thin mineralized veins occurring peripheral to the main mineralized structural zones and also for the discrete barren veins within the mineralization. The existing sampling procedures consist of fairly continuous 1m and/or 1.5m sample-widths within the mineralized horizons and 3m sampling widths within the weakly mineralized wallrock. This sampling style does not adequately characterize the often high-grade nature of the thin (<0.3m-wide) mineralized veins that occur within both the structural zone's weakly mineralized hanging wall and footwall.

12.3 2010 VERIFICATION SAMPLING

MDA collected eight quarter-core verification samples from typical moderate- to high-grade sample intervals within four core holes. Six of the quarter-core samples were collected from three recent Tahoe core holes drilled since the completion of the AMEC resource estimate in early 2010. The MDA samples were delivered to the Inspectorate prep lab in Guatemala City and analyzed using the same analytical techniques as employed for the Escobal drilling.

Table 12-1 and Table 12-2 show the assay results for the MDA samples (columns "XX_MDA") with the original half-core assay values (columns "XX_orig.) for comparison. A third column ("XX_diff") shows the percentage difference between the two assays, with a positive value indicating an increase in grade with the MDA sample. The tables also include the means for the two sample types and also the mean of the percentage difference values.

The verification sample results show similar mean values for silver, lead, and zinc, though individual intervals have differences of up to 80 percent. The MDA gold values are predominantly lower than the original samples, which could be a result of sampling bias or inherently more erratic gold mineralization.

Table 12-1: MDA Verification Samples – Silver and Gold Results

MDA Sample ID	Hole_ID	From	To	Ag_orig. ppm	Ag_MDA ppm	Ag_diff %	Au_orig. ppb	Au_MDA ppb	Au_diff %
MDA-ES1	E08-52	176.5	177.5	481.7	500.1	4%	18.514	12.446	-33%
MDA-ES2	E08-52	177.5	178.5	8628.8	7053.5	-18%	2.234	0.709	-68%
MDA-ES3	E10-210	277.5	279	3171.9	1778.4	-44%	0.146	0.059	-60%
MDA-ES4	E10-210	279	280.5	5961.9	2691.3	-55%	0.589	0.192	-67%
MDA-ES5	E10-211	348	349.5	1439.9	2442.6	70%	0.232	0.331	43%
MDA-ES6	E10-211	351	352.5	3562.7	3213.4	-10%	0.453	0.161	-64%
MDA-ES7	E10-221	574.5	576	311.9	359.5	15%	0.81	0.736	-9%
MDA-ES8	E10-221	577.5	579	291.5	394.9	35%	1.29	1.25	-3%
		mean	value	2981.3	2304.2	-0.3%	3.034	1.986	-32.8%

Table 12-2: MDA Verification Samples – Lead and Zinc Results

MDA Sample ID	Hole_ID	From	To	Pb_orig. ppm	Pb_MDA ppm	Pb_diff %	Zn_orig. ppm	Zn_MDA ppm	Zn_diff %
MDA-ES1	E08-52	176.5	177.5	640	667	4%	730	1153	58%
MDA-ES2	E08-52	177.5	178.5	21500	23400	9%	4093	4339	6%
MDA-ES3	E10-210	277.5	279	5406	4313	-20%	5944	6338	7%
MDA-ES4	E10-210	279	280.5	5765	3025	-48%	4774	3267	-32%
MDA-ES5	E10-211	348	349.5	7019	12700	81%	11000	18100	65%
MDA-ES6	E10-211	351	352.5	22800	29100	28%	32100	32900	2%
MDA-ES7	E10-221	574.5	576	2726	2854	5%	8730	6372	-27%
MDA-ES8	E10-221	577.5	579	2730	2408	-12%	7434	5694	-23%
		mean	value	8573	9808	5.8%	9351	9770	7.0%

The MDA sample results discussed above are not considered to be statistically meaningful due to the limited sampling but serve primarily as a general verification of the Escobal metal grades.

No further verification samples were collected during the 2012 site visit.

12.4 QUALITY ASSURANCE AND QUALITY CONTROL "QA/QC"

MDA evaluated the QA/QC data for the Escobal project on two occasions. In 2010 MDA evaluated the QA/QC data for holes up to and including number E10-225. A summary of that evaluation appeared in the Technical Report of November, 2010. In 2012, MDA evaluated the QA/QC data for holes drilled since E10-225, up to and including E11-348 and PZ11-02. The discussion that follows is the one that appeared in the 2010 technical report, with additions describing MDA's evaluation of the data for the later holes. For clarity as to which generation of data is being discussed, section titles include the words "to E10-225" or "post E10-225", as appropriate.

For holes up to and including E10-225, the quality assurance/quality control (QA/QC) procedures for drill core sample analyses included the use of standard reference materials, sample blanks, and duplicate samples. The duplicate samples consist of pairs of analyses, done on the same samples, at two or more different labs.

Tahoe provided the following description of protocols used from 2010 to the present:

Assay Standards: To be inserted into sample stream prior to submission of the samples to BSI Inspectorate at a rate of one assay standard per 20 drill samples (5%). Assay standards are selected based on which standard's grade more closely matches the geologist's estimate of grade for the proximal drill samples.

Assay Blanks: To be inserted into the sample stream at irregular intervals; particularly internal or immediately following high-grade sample intervals. Samples assaying less than five parts per billion gold and 0.1 parts per million silver were collected from local outcrops at Escobal for use as sample blanks. BSI

Inspectorate also monitored pulverizer contamination by collecting and analyzing the barren sands used to clean the pulverizer after each sample.

Check Assays: In 2010, BSI Inspectorate shipped 5% of the assay pulp splits to ALS Chemex (Vancouver, BC or Reno, Nevada) for reanalysis. Due to a miscommunication, in 2011 Inspectorate shipped approximately 25% of mineralized sample interval pulp splits to ALS Chemex for check analyses. Tahoe used ALS's Vancouver facility exclusively in 2011.

12.4.1 Duplicate Samples to E10-225

Several combinations of pulp "check assay" duplicate samples and coarse reject "preparation" duplicate samples are available, in different pairings of analyses done at Inspectorate, Chemex, CAS in Honduras and at the Marlin Mine lab. Duplicates are available for gold, silver, lead, and zinc. For gold and silver, duplicates are available for wet geochemical and for gravimetric analyses. The available pairings are listed in Table 12-3.

Each duplicate pair in Table 12-3 was evaluated using basic statistical parameters, scatterplots, and relative difference plots. Analysis of the thirty-two duplicate sets in Table 12-3 involved more than 96 charts. Only one set of examples of the charts used is presented in this report.

Table 12-3: Comparison of Analyses of Duplicate Pairs

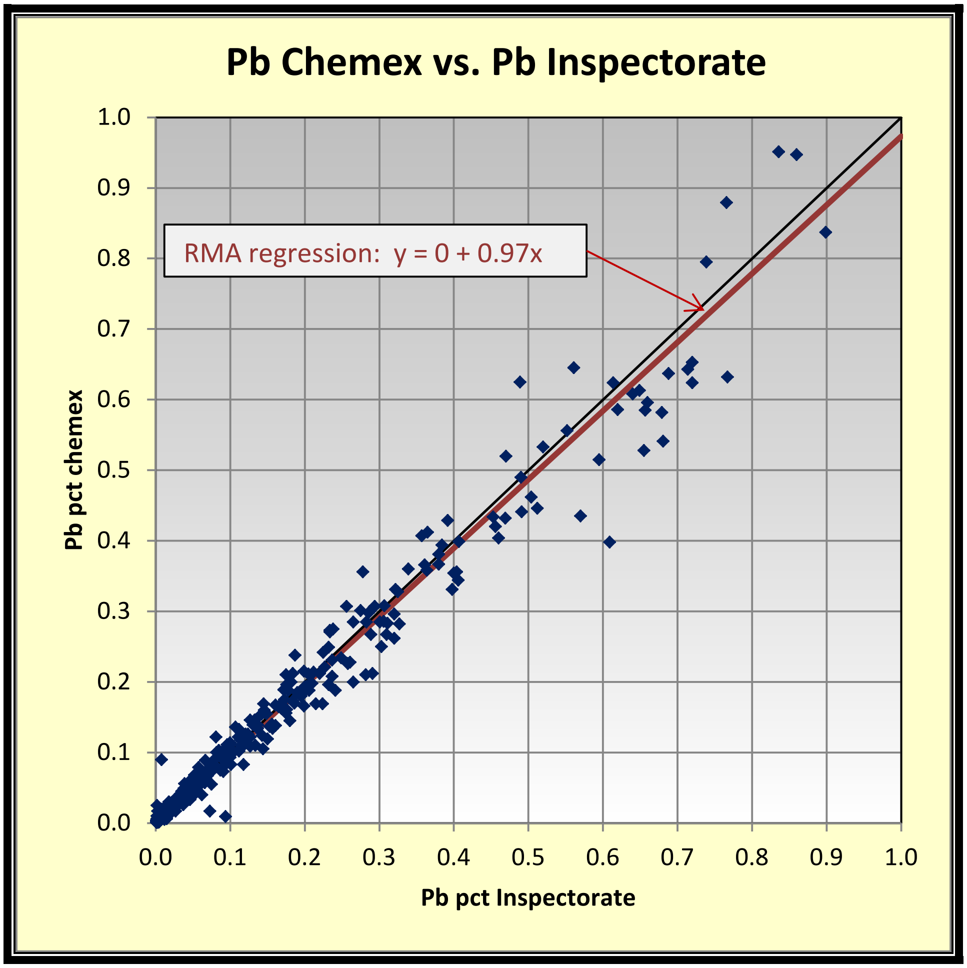
Original (X)	Check (Y)	Material	Count	Erratics Rejected	Pct Diff of Means	Relative Pct Diff	Abs Rel Pct Diff
Insp. Au FA ppb	CAS Au FA30+AA g/tonne	pulp	1368	2	2.6	7.4	29.9
Insp. Au FA ppb	CAS Au FA30+AA g/tonne	coarse reject	43	none	5.7	4.4	25.9
Mine Au g/t	CAS Au FA30+AA g/tonne	pulp	24	5	-2.1	-16.5	53.8
Mine Au g/t	CAS Au FA30+AA g/tonne	coarse reject	27	1	9.7	-9.0	44.4
Insp. Au FA ppb	Mine Au g/t	pulp	2173	12	-5.7	46.7	91.1
Mine Au g/t	Chemex 983 Au ppm	pulp	261	none	10.0	25.5	46.2
Mine Au g/t	Chemex 983 Au ppm	coarse reject	217	none	30.3	62.5	80.0
Insp. Au FA ppb	Chemex 983 Au ppm	pulp	321	none	3.5	41.7	58.8
Insp. Au Grav ppb	Mine Au Grav	pulp	49	1	-3.8	-5.0	11.9
Insp. Au Grav ppb	CAS Au FA30+Grav g/tonne	pulp	19	1	-4.6	-7.4	15.6
Insp. Ag AQR ppm	CAS Ag Wet ppm	pulp	1325	none	-7.4	58.5	134.0
Insp. Ag AQR ppm	CAS Ag Wet ppm	coarse reject	125	2	-7.3	122.0	154.8
Insp. Ag AQR ppm	Mine Ag g/t	pulp	2050	none	-13.9	-6.9	32.8
Mine Ag g/t	Chemex 8106 Ag ppm	pulp	229	3	8.5	4.7	20.4
Mine Ag g/t	Chemex 8106 Ag ppm	coarse reject	230	none	20.8	25.7	43.4
Mine Ag g/t	CAS Ag Wet ppm	pulp	24	none	12.0	30.4	34.8
Mine Ag g/t	CAS Ag Wet ppm	coarse reject	25	none	-24.2	-274.0	290.6
Insp. Ag AQR ppm	Chemex 8106 Ag ppm	pulp	313	1	-4.4	-6.5	21.6
Insp. Ag AQR ppm	Chemex 8106 Ag ppm	coarse	289	2	-1.8	2.1	31.6

Original (X)	Check (Y)	Material	Count	Erratics Rejected	Pct Diff of Means	Relative Pct Diff	Abs Rel Pct Diff
		reject					
Insp. Ag Grav ppm	CAS Ag FA30+Grav g/tonne	pulp	499	none	-3.8	-4.6	7.4
Insp. Ag Grav ppm	CAS Ag FA30+Grav g/tonne	coarse reject	8	none	3.8	5.4	9.1
Insp. Ag Grav ppm	Mine Ag Grav	pulp	585	1	-6.4	-12.0	15.3
Mine Ag Grav	Chemex Ag ME-GRA21 ppb	pulp	74	1	9.9	8.9	12.9
Insp. Ag Grav ppm	Chemex Ag ME-GRA21 ppb	pulp	58	none	1.0	0.2	8.2
Insp. ICP Lead %	Chemex Pb-AA62 %	pulp	152	none	-17.3	-14.5	15.2
Insp. ICP Lead %	Chemex Pb-AA62 %	coarse reject	25	1	-10.1	-16.7	19.6
Insp. ICP Lead Pb ppm	Chemex Pb-AA62 %	pulp	644	none	-2.5	13.1	27.5
Insp. ICP Lead Pb ppm	Chemex Pb-AA62 %	coarse reject	471	2	-2.0	10.4	36.3
Insp. ICP Zinc %	Chemex Zn-AA62 %	pulp	207	2	-6.5	-10.3	12.8
Insp. ICP Zinc %	Chemex Zn-AA62 %	coarse reject	39	2	-9.5	-14.9	16.6
Insp. ICP Lead Zn ppm	Chemex Zn-AA62 %	pulp	619	none	-1.0	8.5	18.5
Insp. ICP Lead Zn ppm	Chemex Zn-AA62 %	coarse reject	477	1	-1.7	8.6	24.6

12.4.1.1 Example of Charts; Lead at Chemex and Inspectorate

Inspectorate, the “primary” lab, did most of the analyses used in the resource estimate. Inspectorate did an “assay” for lead, reported in percent, and an ICP analysis, reported in ppm. The comparison shown is Inspectorate’s ICP analysis with Chemex’s analysis by atomic absorption, comparing “coarse reject” also called “preparation” duplicates. This compares the results of the process of sample size reduction, sample grain size reduction, and chemical analyses at the two labs.

For the comparison illustrated by the charts, MDA began with 533 pairs analyzed at both labs. Sixty samples were eliminated from the comparison as their analyses fell above or below the detection range stated by one or both of the labs. Two samples were eliminated for having extreme differences suggesting either an analytical error or a record-keeping error. MDA has no way to differentiate these two types of errors, although where extreme differences occur, a record-keeping error is suspected.



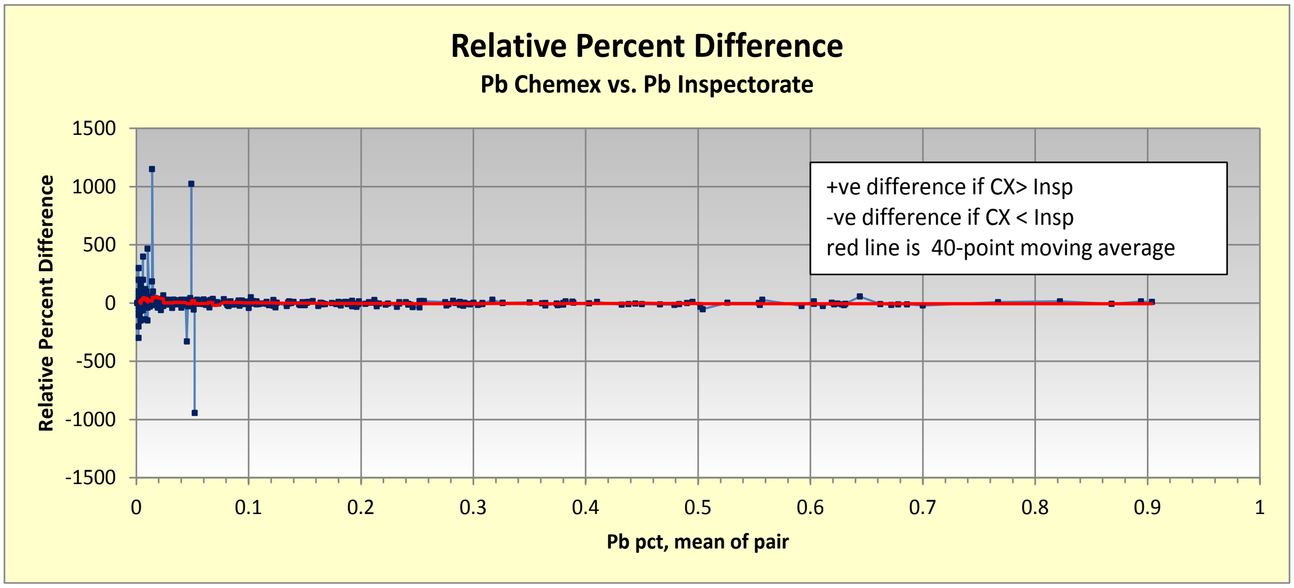
Note: An RMA regression (red line) assumes two independent variables.

Figure 12-1: Scatterplot for Lead, Chemex vs. Inspectorate

Figure 12-1 and Table 12-4 suggest a good correlation, with Chemex tending to be slightly lower than Inspectorate at grades above about half a percent.

Table 12-4: Simple Statistics, Lead

	Std dev	Mean % Pb	Median % Pb	Count
Pb Insp.	0.176	0.119	0.042	471
Pb Cmx	0.171	0.117	0.041	471
% Diff		-2.0		



Note: The relative percent difference is calculated as:
100 x (Chemex-Inspectorate) / lesser of (Chemex, Inspectorate). This calculation gives a worst-case number.

Figure 12-2: Relative Percent Difference for Lead, Chemex vs. Inspectorate

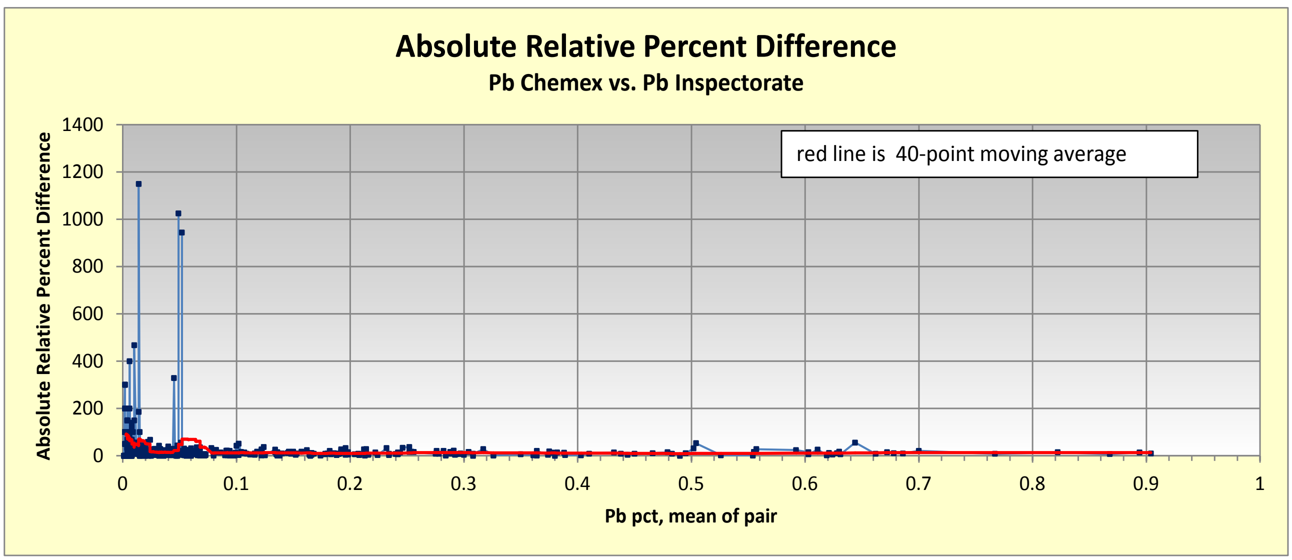


Figure 12-3: Absolute Value of Relative Percent Difference for Lead, Chemex vs. Inspectorate

Table 12-5: Simple Statistics for Percent Differences, Lead

	Std dev	Mean	Count
Rel pct Diff	102.02	10.39	471
Abs Rel pct Diff	95.89	36.31	471

Figure 12-2 and Figure 12-3 suggest that the two labs' analyses for lead compare very well, with some large percentage differences at grades under 0.1% Pb.

12.4.1.2 Summary of the Duplicate Analysis Results

As it is not practical to include large numbers of charts in the present report, the results of the duplicate analyses checks are summarized in Table 12-3.

In Table 12-3, the "Original (X)" analysis is always taken from either the Marlin Mine lab or Inspectorate. Analyses from all other labs are considered to be "Check (Y)" samples. Where Inspectorate is compared to the Marlin Mine lab, Inspectorate's analysis is considered to be the "Original (X)". "X" and "Y" refer to the axes the analyses appear on in charts like Figure 12-1. All differences are calculated as "Check" - "Original".

"Count" is the number of samples left after all those out of the detection range of the analytical method were eliminated. "Erratics Rejected" is the number of samples eliminated because of large differences or other problems. Thus a comparison of "Erratics Rejected" to "Count" gives one estimate of the proportion of suspect data. The total number of "erratic" or suspect pairs in all of the duplicate pairs is 40, out of 12,970 comparisons, or about 0.3 %.

Data may be suspect due to analytical or record-keeping errors, and MDA does not have any means to differentiate these two possibilities. The decision to consider a pair suspect is to some degree subjective and is made in the context of each comparison set.

For each comparison, the lab that yielded the lower average value is shaded blue in Table 12-3. In almost all comparisons involving the Marlin Mine lab, the Mine analyses come out low. Inspectorate appears to be biased high relative to Chemex and the Marlin Mine lab but may be either high or low relative to CAS, depending on the metal and method being evaluated.

For the most part, in the Escobal data set the contrast between a comparison of pulp duplicates and the same comparison using coarse reject duplicates is not large. The contrast does tend to be larger if one of the labs in the comparison is the Marlin Mine lab. It would be useful to have a comparison of coarse reject duplicates in which both the original and duplicate were prepared and analyzed in the Marlin Mine lab.

12.4.2 Duplicates and Check Samples post E10-225

There are two types of duplicate and check assays to be considered in the post E10-225 assay data set. Small numbers of same-lab duplicates run at Inspectorate are the first type, and the second type is a much larger set of pairs consisting of original assays at Inspectorate and check assays done at ALS Chemex.

In the case of the post E10-225 same-lab duplicates, three types of analyses are involved:

- Gold analyzed by fire assay prep with AA finish. MDA notes that the duplicate analysis had a higher upper detection limit than the original. There are 14 such duplicate pairs, but two have values exceeding the upper limit for the first analysis, leaving 12 effective comparisons.
- Silver analyzed by atomic absorption. There are 23 such duplicate pairs, but nine have values exceeding the upper limit for the method, leaving 14 effective comparisons.
- Silver analyzed by fire assay with a gravimetric finish. There are 8 such duplicate pairs.

MDA did not chart the same-lab duplicates described above. A summary comparison appears in Table 12-6, below.

Table 12-6: Summary Comparison of Duplicate Pairs at Primary Lab

Element	Method, units	Count of Pairs	Mean of Original	Mean of Duplicate	Bias percent
Au	FAA, ppb	12	4,787	4,846	1.2
Ag	AA, ppm	14	41.0	43.0	4.9
Ag	FA grav, ppm	8	2,806.2	2,879.7	2.6

Three common mathematical two-sample comparison tests, the F-test for variances, the t-Test for means and the Mann-Whitney test for medians, suggest that at the 95% confidence level each set of duplicates is not statistically distinguishable from the corresponding set of originals (Test done using SigmaXL software).

The post E10-225 data set includes 339 pairs of analyses for each of silver, gold, lead and zinc, consisting of the original analyses done by Inspectorate and check analyses done by ALS Chemex. MDA assessed these check analyses using charts similar to the examples that appear in Section 12.4.1.1. The results of MDA's assessment are summarized in Table 12-7, below.

Table 12-7: Summary Comparison of Original and Check Analyses

			Mean Values of Parameters				
Metal	Count	Erratics Rejected	Original ppm	ALS ppm	Pct Diff of Means	Relative Pct Diff	Abs Rel Pct Diff
Gold	194	4	0.382	0.391	2.4	5.7	14.5
Silver	204	7	308.7	311.4	0.9	1.6	8.6
Lead	207	none	0.85	0.876	3.1	14.1	17.3
Zinc	230	none	1.063	1.115	4.9	11.1	15.5

Note that while there are 339 sample pairs, the counts in Table 12-7 are all substantially less than that. For each metal, analyses below the detection limits and very low-grade analyses were eliminated from the calculations of the statistics that appear in Table 12-7, because small real differences can, at low grades, create large but not very meaningful percentage differences.

Three common mathematical two-sample comparison tests, the F-test for variances, the t-Test for means and the Mann-Whitney test for medians, suggest that for each of the four metals, at the 95% confidence level the set of check analyses is not statistically distinguishable from the set of original analyses.

12.4.3 Blanks to E10-225

The analytical database that MDA received in 2010 for Escobal includes the analyses of blanks listed in Table 12-8.

Table 12-8: Blanks in Escobal Data Set

Lab and Analysis	Number of Blanks
Insp. Ag AQR ppm	1,014
Mine Ag g/t	226
Chemex 8106 Ag ppm	35
CAS Ag Wet ppm	110
CAS Au FA30+AA g/tonne	115
Insp. Au FA ppb	1,035
Mine Au g/t	226
Chemex 983 Au ppm	35
Total	2,796

The material used as blanks was collected from local outcrops where the sample assayed less than five parts per billion gold and 0.1 parts per million silver. The material was not subjected to a rigorous round-robin analysis, so MDA cannot be assured that the material is truly “blank.” MDA evaluated the results of the analyses of blank material making use of charts like the example in Figure 12-4.

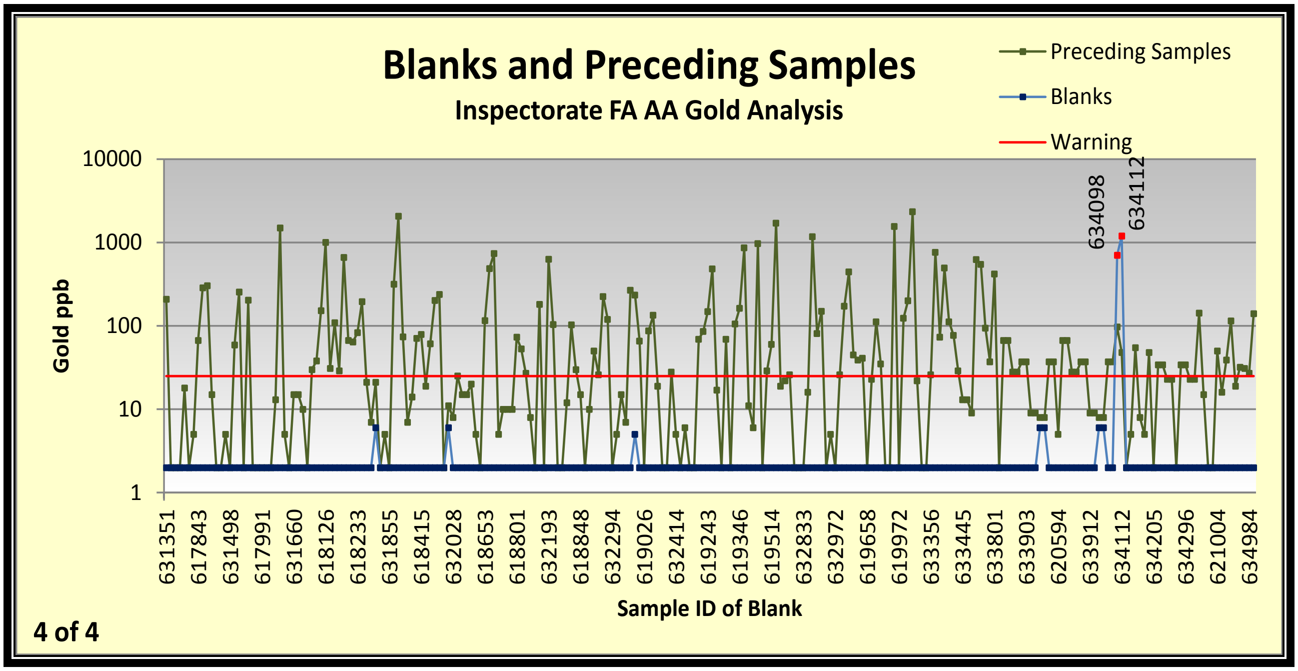


Figure 12-4: Blanks in Inspectorate Gold Analyses

Figure 12-4 shows a plot of the results of the analyses of blanks, with a superimposed plot of the analyses of the samples numerically-preceding each blank. The purpose of superimposing the analyses of the preceding samples is to gain a visual impression as to whether a high grade in a preceding sample tends to produce a higher grade in the immediately-following blank. In the example shown, there is remarkably little evidence of such contamination. In the data set as a whole, there are some suggestions of such contamination, but they are not systematic and they are too few to be of concern.

The red "warning" line on Figure 12-4 is arbitrarily set at five times the lab's lower detection limit. Three to five times the detection limits are typical industry rules of thumb, when no more rigorously-determined rule is available. Two failures are evident in Figure 12-4. The grades in the two failures are so high that MDA suspects they are due to record-keeping errors rather than analytical failures.

In the 2,796 analyses reported to be of blank material, 50 count as failures using five times the lower detection limit as a rule of thumb. This is a rate of 1.8%. MDA cannot determine which of the 50 are record-keeping errors and which are analytical failures.

12.4.4 Blanks Post E10-225

Standards have just recently been introduced by Tahoe. The limited data (<10 analyses per standard) preclude any meaningful analyses or conclusions.

The post E10-225 analytical data set includes 536 analyses of material identified as blanks. Tahoe describes the blank material as "collected from local outcrops of unaltered andesite, which is the one of two primary host rocks for the deposit. The samples were combined and twelve splits were sent to the lab for Au and Ag analysis only; all samples came back <5 ppb Au and <0.1 ppm Ag. However, there were no base metal analyses performed." MDA has reviewed the data from the blanks for gold, silver, lead and zinc, but concludes that the material is not suitable as a blank for lead and zinc.

Continuing to use five times the detection limit as the failure criterion for gold and silver, MDA identified the failures listed in Table 12-9, below.

Table 12-9: Failure List for Blanks Post E10-225

Sample	Analysis	Report Number	Report Date
Gold, ppm			
646766	0.035	11-338-07549-01	18-Oct-11
Silver, ppm			
622327	*0.5*	*10-338-02670-01*	*16-Sep-10*
624977	3	11-338-03457-01	31-May-11
646766	1.1	11-338-07549-01	18-Oct-11
660651	0.6	11-338-08163-01	24-Oct-11
68162	0.8	11-338-08714-01	8-Nov-11
644088	0.5	11-338-08927-01	9-Nov-11
661605	9.1	11-338-10395-01	3-Jan-12
625138	3.8	11-338-03582-01	

The failure rate for gold in the blanks is negligible. The failure rate for silver is 1.5 though the magnitude in assay grade of the failures is still low and would not have a material effect on the resource estimate.

12.4.5 Standards to E10-225

In 2010, standards had just recently been introduced by Tahoe. The limited data (<10 analyses per standard) precluded any meaningful analyses or conclusions in the Technical Report of November, 2010.

12.4.6 Standards post E10-225

Four different standards were used by Tahoe from 2010 onwards. All of the standards were obtained from CDN Resource Laboratories Ltd. ("CDN") of Langley, British Columbia, Canada. A total of 74 analyses of standards are in the data set.

For each metal certified in each standard, CDN's certificates include a "Recommended Value" and a "Between Lab Two Standard Deviations".

For the evaluation described herein, MDA used warning limits set at:

$$Recommended\ Value\ \pm\ 2 * Standard\ Deviation$$

Control limits or failure limits were set at:

$$Recommended\ Value\ \pm\ 3 * Standard\ Deviation$$

MDA obtained the standard deviation by taking half of CDN's "Two Standard Deviations". Information provided by Tahoe indicates that Tahoe used substantially the same failure criteria as MDA.

For each standard, and for each of gold, silver, lead and zinc, MDA plotted a control chart similar to the conventional Shewhart charts. An example is shown in Figure 12-5.

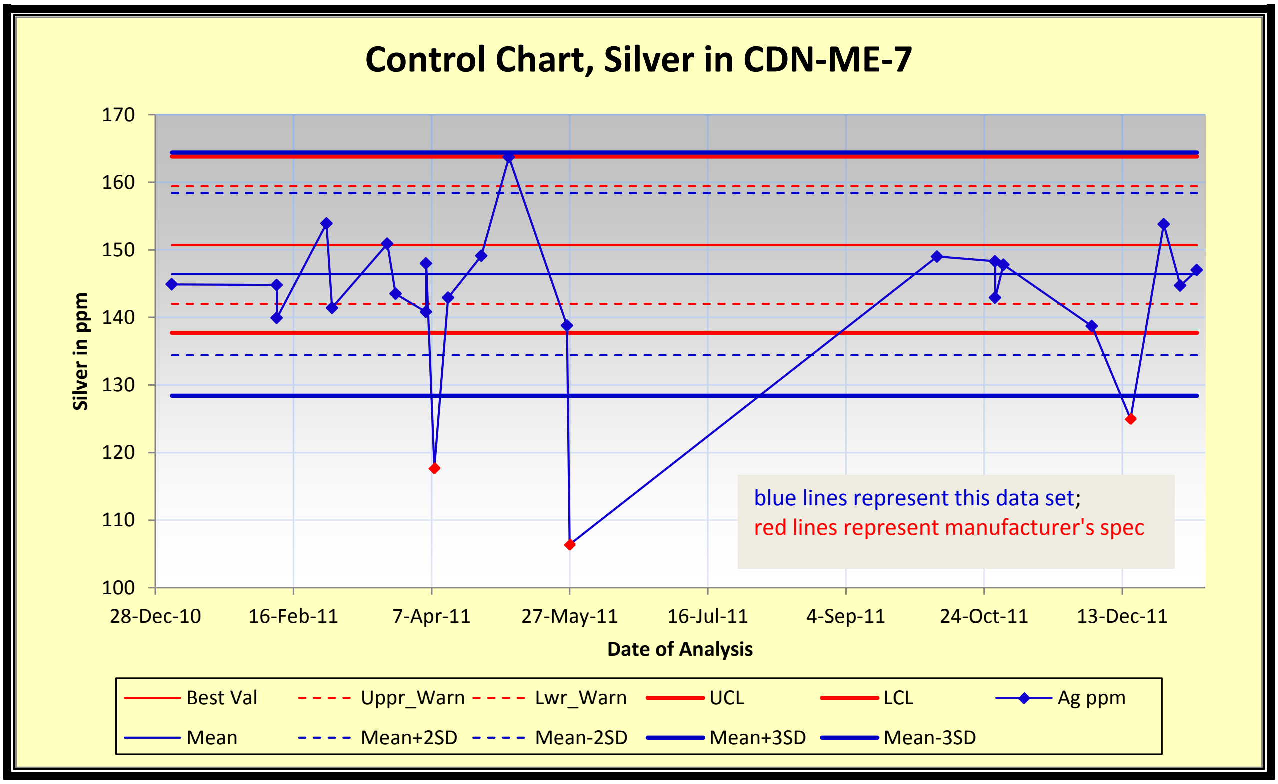


Figure 12-5: Control Chart for Silver in CDN-ME-7

Notes: The red upper and lower warning lines are the $Best\ Value \pm 2 * Std.Dev.$, using statistics calculated by CDN.

The red upper and lower control lines, UCL and LCL, are the $Best\ Value \pm 3 * Std.Dev.$, using statistics calculated by CDN.

The three low-side failures noted were excluded from the calculation of the mean and standard deviation for this data set, shown by blue lines.

Sixteen charts similar to Figure 12-5 were generated in doing the evaluation of the analytical results obtained for the standards. Rather than show all of them in this report, MDA has summarized the results in Table 12-10.

Table 12-10: Summary of Results for Standards

Std. ID	Count	Best Value	Average	Bias pct	Minimum	Maximum	Low Failures	High Failures
CDN-ME-5								
Gold ppm	15	1.07	1.100	+2.8	0.685	1.398	1	3
Silver ppm	15	206.1	208.6	+1.2	197.8	215.3	none	none
Lead ppm	15	21,300	21,633	+1.6	18,200	23,300	1	2
Zinc ppm	15	5,790	5,717	-1.3	5,315	6,452	5	2
CDN-ME-6								
Gold ppm	29	0.270	0.278	+3	0.220	0.315	1	1
Silver ppm	29	101	99.7	-1.3	90.9	110.6	none	none
Lead ppm	29	10,200	9,912	-2.8	8,760	10,900	2	none
Zinc ppm	29	5,170	5,221	+1	4,729	5,702	none	none
CDN-ME-7								
Gold ppm	24	0.219	0.220	+0.5	0.203	0.264	none	1
Silver ppm	24	150.7	142.7	-5.3	106.4	163.7	3	none
Lead ppm	24	49,500	49,271	-0.5	41,300	59,700	3	4
Zinc ppm	24	48,400	47,475	-1.9	44,500	53,200	5	2
CDN-ME-11								
Gold ppm	6	1.380	1.418	+2.8	1.357	1.483	none	none
Silver ppm	6	79.3	76.1	-4	73.8	79.2	none	none
Lead ppm	6	8,600	7,889	-8.3	6,502*	8,289	1	none
Zinc ppm	6	9,600	9,242	-3.7	8,588	10,100	1	none

Notes: "Count" is the number of times that this standard was inserted into the sample stream.

"Best Value" is the value assigned to the standard by the manufacturer, for the metal indicated. In the certificates provided by CDN this is referred to as the "Recommended Value".

"Average" is the average of the values obtained for this standard by Inspectorate when analyzing Tahoe's drill samples. Analyses deemed to be failures were not excluded in calculating this average.

"Bias pct" is calculated using the formula $100 * \frac{(Average - Best\ Value)}{Best\ Value}$

"Minimum" is the minimum of the values obtained for this standard by the lab analyzing Tahoe's drill samples.

"Maximum" is the maximum of the values obtained for this standard by the lab analyzing Tahoe's drill samples.

"Low Failures" is a count of the number of instances in which an analysis of the standard fell at or below the lower control limit.

"High Failures" is a count of the number of instances in which an analysis of the standard fell at or above the upper control limit.

*The low bias for lead in CDN-ME-11 is strongly influenced by this single low value in sample 68137.

Where Tahoe identify analytical failures, their policy is to re-run the affected sample batch, for the element concerned. The final data set made available to MDA incorporates any such re-runs, so the failures identified by Tahoe in the batches that were re-run are not "visible" to MDA and are not counted in the failure columns of Table 12-10. Tahoe has advised MDA that they were most rigorous about identifying failures and having batches re-run for silver, somewhat less rigorous in the case of gold, and least rigorous in the cases of lead and zinc. Silver carries in excess of 85% of the value of the project, and silver also has the fewest failures in Table 12-10, only three and those all on the low side.

Any set of analyses by a single lab will in all probability have some biases relative to the accepted values of standards. The biases listed in Table 12-10 have magnitudes that are, for the most part, typical of those that MDA encounters in assay data sets. There are two biases whose magnitudes exceed 5%, silver in CDN-ME-7 and lead in CDN-ME-11. In both cases the biases are on the low side; in other words the lab produced results that are biased low relative to the accepted values of the standards.

12.4.7 Conclusion and Recommendations

The large quantity of pulp duplicate data up to E10-225, with some coarse reject duplicate data, show less than 1% of erratically-large differences and do not reveal significant biases in the critical components of the data set, those being analyses from Inspectorate and Chemex.

The post E10-225 check analyses contained higher percentages of erratically-large differences, about 2.1% for gold and about 3.4% for silver. On average, ALS Chemex' analyses are marginally higher than those of the primary lab, Inspectorate.

The analyses of blanks yielded a higher-than-expected "failure" rate, notably for silver in the post E10-225 data set. The observed failure rate is not so high as to disqualify the data set for use in the resource estimate described in this report.

12.5 2010 CORE RECOVERY – METAL GRADE ANALYSES

Tahoe provided MDA with the core recovery data for all 218 core holes used in the 2010 resource estimate. MDA checked the recovery data calculations and spot-checked the measurements against the core photos. Only one measurement error was found in the calculations, and the recovery data showed no apparent discrepancies with the visual check of the core photos.

The core recovery data are dominated by measurements of >95 percent recovery with isolated zones of lower recovery. The average core recovery for all readings is approximately 96 percent. The average core recovery for the mineralized intervals used in the resource estimate is approximately 95 percent. Approximately 65 percent of the core recovery measurements have values of 100 percent recovery. The prevalence of exact 100 percent core recovery values is indicative of the massive, weakly fractured nature of the country rock but also suggests possibly less rigorous measurement techniques.

MDA analyzed the relationship between metal grades and core recovery. All four metals (silver, gold, lead, and zinc) were reviewed independently. Figure 12-6 shows the relationship between silver grades and core recovery. The silver grade and number ("Count) of core recovery intervals are presented in the left-hand and right-hand y-axis, respectively. These values are sorted into core recovery "bins" of regular 10 percent intervals as noted along the x-axis. (Each bin represents all intervals within each 10 percent interval; for example, recovery column "80" shows the average silver value and number of sample intervals for all intervals with core recovery values between 80 and 89 percent.) The data shown in Figure 12-6 were filtered for only those sample intervals with silver grades greater than 10g Ag/t to better represent the core recovery effect on significant silver grades.

The data in Figure 12-6 show a noticeable decrease in average silver grade when core recovery drops below 80 percent. Only a small fraction (<10 percent) of all sample intervals have recovery values below 80 percent, so the observed change in silver grade is not believed to result in a material error in the current resource estimate. Since the assay values have been potentially down-graded due to the core recovery loss, the current resource estimate is on the conservative side, indicating a small upside to the resource estimate.

MDA analyzed the gold grade versus core recovery data, and a similar pattern as seen in the silver data was observed.

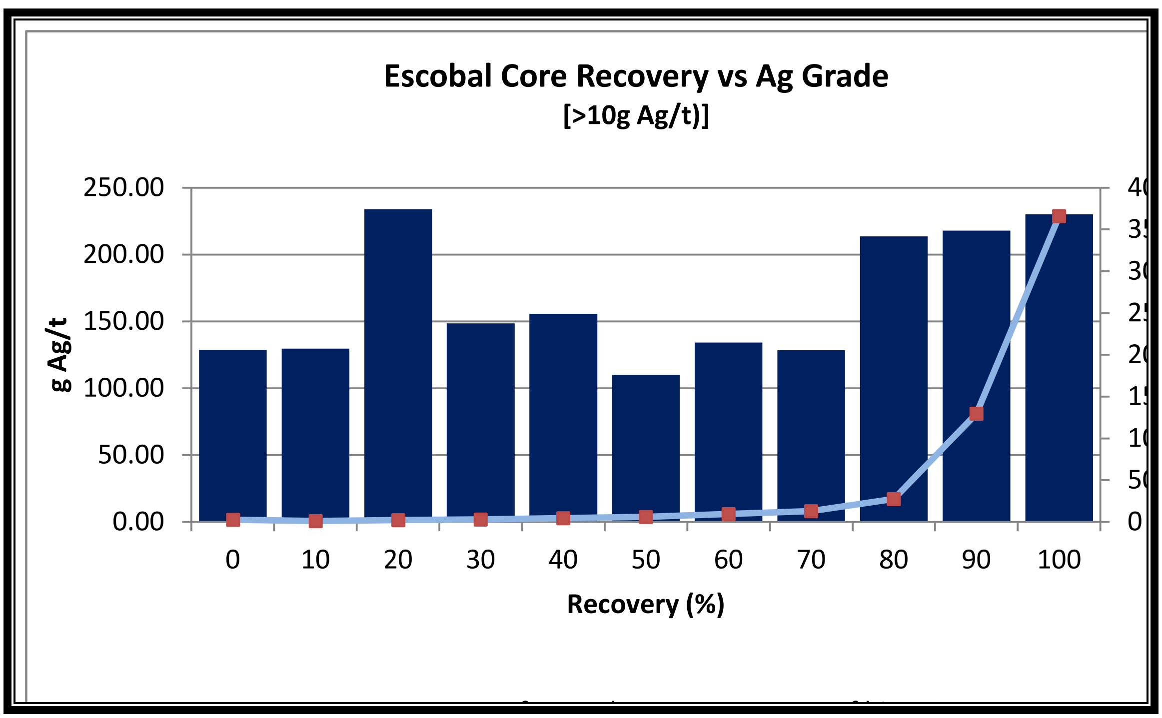


Figure 12-6: Core Recovery – Silver Grade Comparison

Figure 12-7 shows the relationship between lead grades and core recovery. The lead grade and number (“Count) of core recovery intervals are presented in the left-hand and right-hand y-axis, respectively. These data are sorted into the same core recovery 10-percent bins as in Figure 12-6. The data shown in Figure 12-7 include only those sample intervals with lead grades greater than 100 ppm Pb.

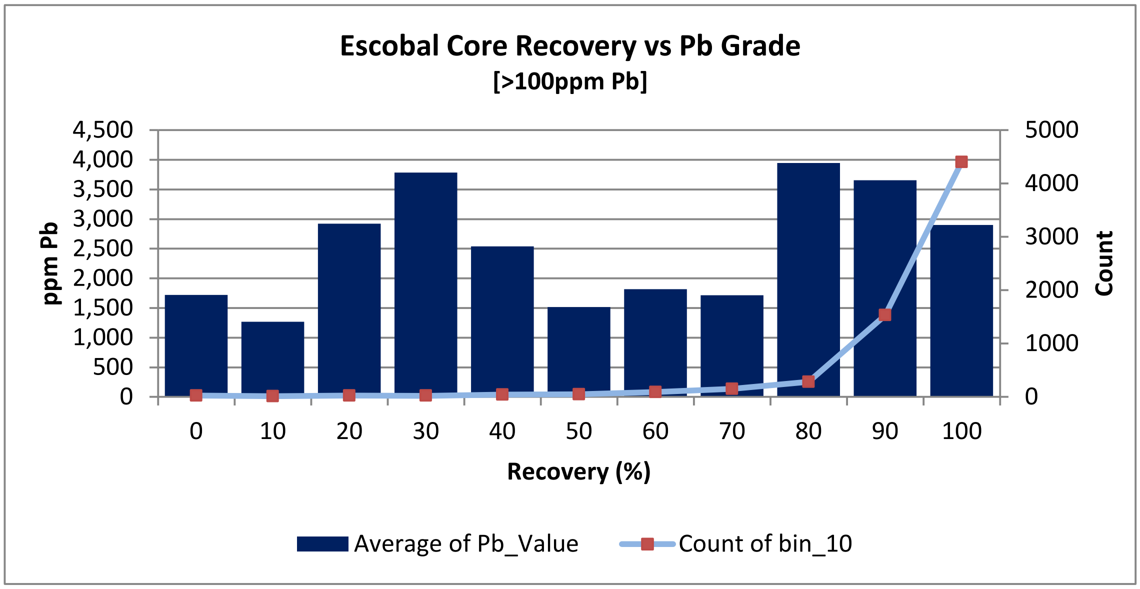


Figure 12-7: Core Recovery – Lead Grade Comparison

The lead versus core recovery data in Figure 12-7 indicate that there is an approximate 25 percent increase in lead grade when core recovery decreases from 100 percent into the 80 percent recovery category. Below 80 percent, the lead data have a similar, though somewhat more erratic, decrease in values as was seen above in the silver data. The latter observed decrease in lead grade is not considered significant due to the small fraction (<10 percent) of all samples with recoveries less than 80 percent. An analysis of the zinc data shows the same relationship with core recovery as seen in the Figure 12-7 lead data.

The increase in lead and zinc grades associated with the 80 percent and 90 percent core recovery intervals could be directly related to a selective increase in grade from core loss. It also could be a natural function of the geology in which the higher-grade, base-metal mineralization occurs preferentially within highly fractured structural intervals. The observed lower core recoveries within these intervals would then have a purely spatial correlation, not genetic, with the increased grades. Further analyses of the data would be needed for a more precise determination of the relationship between core recovery and base-metal grades.

No further analyses of the core recovery data was conducted in 2012. Overall, the data indicate good to excellent core recovery within the mineralized horizons and support the estimation of the Escobal resource to an Indicated level.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Independent and well-respected international testing facilities that have performed metallurgical test work on the Project include:

- FLSmith Dawson Metallurgical Laboratories; Salt Lake City, Utah, USA.
- McClelland Laboratories (McClelland); Nevada, USA
- SGS Lakefield; Canada
- Economic Geology Consulting (EGC); Nevada, USA
- Phillips Enterprises, LLC (PE); Colorado, USA.
- Silver Valley Laboratories (SVL); Idaho, USA.
- Kappes Cassiday Associates, Reno, Nevada, USA.

The laboratories do not hold ISO certification for metallurgical testing activities; this is typical for metallurgical test work facilities. Test work was performed on behalf of Entre Mares during 2008–2009.

Previous metallurgical test work conducted by McClelland Laboratories (McClelland), Utah, USA and Kappes Cassiday Associates, Reno, Nevada, USA (KCA) concluded that differential lead/zinc flotation producing a high value lead concentrate containing most of the silver and gold in the mill feed and a saleable lower value zinc concentrate was the optimum processing route.

The following conclusions were drawn from the test work conducted by McClelland Laboratories in May 2009:

- The Escobal sulfide and mixed oxide/sulfide composites did not respond particularly well to gravity concentration treatment, at an 80%-106µm feed size.
- The Escobal sulfide composites responded well to conventional bulk sulfide flotation treatment for recovery of gold and silver, at an 80%-75µm feed size
- The Escobal sulfide composites showed good potential for selective flotation of contained lead and zinc.
- The Escobal mixed oxide/sulfide composite did not respond as well to conventional bulk sulfide flotation treatment.
- The Escobal composites were moderately amenable to whole ore milling/cyanidation treatment, at an 80%-75µm feed size.
- The EC08-127 composite may have displayed a moderate preg-robbing tendency during whole ore cyanidation.
- Adding activated carbon during whole ore cyanidation (CIL) leaching generally was effective in significantly improving gold and silver recoveries.
- Cyanidation of flotation products, including regrind/intensive cyanidation of flotation rougher concentrates, was not particularly effective in increasing overall leach recoveries, when compared to whole ore CIL/cyanidation leaching.

FLSmidth Dawson Metallurgical Laboratories was selected in June 2010 to conduct a comprehensive metallurgical testing on a composite sample representative of the Escobal deposit. The objective of the Dawson Metallurgical testwork is to advance the design of the differential flotation circuit to process the mineralized material from the Escobal deposit. The sequence of flotation testwork being conducted by Dawson Metallurgical includes the following:

- Grind time determination
- Reagent screening for lead rougher flotation
- Lead rougher flotation
- Reagent screening for zinc rougher flotation
- Zinc rougher flotation
- Preliminary grind size optimization
- Locked cycle testing
- Tailing and concentrate physical property characterization

Hazen Research, Inc. located in Golden Colorado was selected in June 2010 to conduct a comprehensive comminution test program using PQ core samples drilled specifically for these tests. The tests include: JK drop weight tests, Bond rod mill tests, Bond ball mill tests, Bond crushing tests and abrasion testing.

13.1 SAMPLING

A total of 46 buckets of drill core samples were received for sample preparation and assay at Phillips Enterprises and stage crushed to minus 10-mesh prior to flotation testing.

Head assays for the master composite were conducted to determine gold (Au), silver (Ag), lead (Pb), zinc (Zn), copper (Cu), iron (Fe), and antimony (Sb). Also carbon (total/organic), non-sulfide lead and non-sulfide zinc were determined. The results of the head assay analysis for the master composite are presented in a table below.

Table 13-1: Master Composite Head Assay Results

Element	Au, ppm	Ag, ppm	%Pb	%Zn	%Cu	%Fe	%Sb	$\%C_{tot}$	$\%C_{org}$	Pb_{ns}	Zn_{ns}
Assay	0.412	569	0.984	1.56	0.041	2.48	0.044	1.35	0.05	0.10	0.097

13.2 GRINDING TESTS

Two kilogram samples were ground in a mill at 20, 30, 40, 50, and 60 minute intervals to establish the relationship between the grind sizes (P_{80}) and grind times. The time required to achieve various grinds were obtained and tests were run at different grind sizes to ascertain the relationship between P_{80} versus metal recovery.

The test results shown in the tables below indicate that grinding beyond 105 microns did not result in any significant increase in metal recoveries. It was therefore decided that 105 microns was the optimum grind size for rougher flotation. This grind is being used for further testwork as well as the design of the process plant.

Table 13-2: P80 Versus Metal Recovery to the Lead Rougher Concentrate

Test No	Grind P80	Metal Recovery to Pb Rougher Concentrate with SIPX as Collector						
	Microns	%Au	%Ag	%Pb	%Zn	%Cu	%Fe	%Sb
9	231	55.3%	69.5%	83.6%	22.2%	57.7%	16.6%	28.0%
10	144	61.6%	75.5%	88.2%	20.8%	64.3%	18.3%	30.3%
11	105	67.0%	78.7%	88.7%	19.1%	64.0%	18.9%	30.8%
4	74	64.9%	82.9%	89.4%	17.9%	64.8%	22.8%	30.8%
12	46	67.9%	79.6%	82.6%	14.5%	64.6%	13.1%	30.0%
13	37	68.5%	76.5%	67.5%	11.4%	56.4%	10.3%	28.0%

Test No.	Grind Time ($P_{80,}$ microns)	Metal Recovery to Pb Rougher Concentrate With PAX as Collector						
		%Au	%Ag	%Pb	%Zn	%Cu	%Fe	%Sb
22	35(150)	67.6%	81.1%	88.9%	31.9%	71.7%	34.7%	31.4%
23	45(105)	71.8%	81.3%	89.2%	25.7%	70.6%	37.2%	30.8%
24	60(75)	73.9%	81.2%	89.4%	24.1%	71.8%	36.4%	32.3%
25	85(53)	73.9%	80.0%	80.1%	15.1%	64.9%	21.2%	37.9%

13.3 GRINDABILITY TESTS

Phillips Enterprises, LLC conducted three ball mill grindability tests on Escobal Project samples as part of the McClelland Laboratories test work. The resulting ball mill work indices (Wi) from ball mill grindability tests conducted at closing screen of 100 mesh (150 micron) are shown below.

Table 13-3: Resulting Ball Mill Work Indices from Ball Mill Grindability Tests

Sample	Wi (kW-hr/st)	Wi (kW-hr/mt)
EC08 - 122	14.07	15.55
EC08 - 125	17.22	18.99
EC08 - 127	16.44	18.13
Average	15.91	17.56

A circuit consisting of one 5m by 8.5m (16.5 ft x 28 ft) ball mill in closed circuit with a hydrocyclone classifier was selected as a circuit that would likely meet the design tonnage. This circuit was based on the results from the comminution tests to produce a primary grind size of 80% passing 105 µm.

13.4 REAGENT SCREENING TESTS

Initial lead rougher flotation tests were conducted with Sodium Isopropyl Xanthate and Sodium Ethyl Xanthate and 3418A as the main collectors. It was observed that each collector tried worked well with the flotation being very fast and essentially being completed after 4 minutes.

Microscopic examination of the concentrates indicated that concentrate contained galena as the main product with pyrite and gangue as the main contaminants with lesser but significant amounts of sphalerite as the third most common contaminant. Examination of the tails showed that the predominant sulfide minerals were pyrite and sphalerite with pyrite being in the majority. The minerals were very liberated with the only locking seen being small blebs of pyrite attached to gangue. It was also found that Sodium Isopropyl Xanthate (SIPX) performed better than Sodium Ethyl Xanthate. More reagent screening tests were conducted with the stronger xanthate, Potassium Amyl Xanthate (PAX), which improved the recovery of precious metals when compared with Sodium Isopropyl Xanthate as shown in the table below.

Table 13-4: Typical Metal Recovery to Lead Rougher Concentrate

Collector Type	%Au	%Ag	%PB	%Zn	%Fe
SPIX	74.5	79.7	90.9	26.0	30.1
PAX	77.3	83.4	91.8	30.4	37.5
Difference	2.73	3.68	0.91	4.36	7.40

With the objective to maximize the precious metals recovery to the lead concentrate, Potassium Amyl Xanthate (PAX) was used in subsequent tests as the main lead rougher collector.

The tests showed that galena floated well with xanthates as the main collectors. Pyrite and sphalerite floated with galena but not in unusual quantities considering the use of very strong collectors to maximize precious metals recoveries and the fact that pyrite is the most abundant sulfide in the material tested. The high degree of liberation indicates that a grind coarser than 75-microns is possible and that the rougher concentrate will clean well.

Initial zinc reagent screening tests were conducted with the tailings from the lead rougher flotation. Lime ($Ca(OH)_2$), copper sulfate($CuSO_4.5H_2O$) and Potassium Amyl Xanthate (PAX) were added to the lead rougher tails slurry, conditioned for 5 minutes and a zinc rougher flotation step was completed after adding X-133 frother. The pH of the lead rougher tails slurry was raised to 9.5 with lime to depress pyrite and the copper sulfate was used to activate sphalerite.

Sphalerite floated well with PAX as the main collector at pH 9.5 and with about 30 g/t copper sulfate dosage for sphalerite activation. The results shown in the table below indicates that good recoveries of both lead and zinc are achievable in the rougher concentrates.

Table 13-5: Typical Metal Distribution in Rougher Flotation Products

Product Identification	%Au	%Ag	%Pb	%Zn	%Fe
Lead Ro Con	76.72	82.29	91.10	31.7	29.83
Zinc Ro Con	5.56	4.91	2.94	61.89	13.39
Pb +Zn Con	82.28	87.20	94.03	93.64	43.22
Zinc Ro. Tail	17.72	12.80	5.97	6.36	56.78

More tests were conducted in September 2010 to address the following:

- Recycle water. The Escobal Project design will strive to recycle as much water as possible minimizing treatment and discharge. Calcium in the lime used to raise the pH in zinc flotation depresses galena and precious metals and the copper sulfate used to activate sphalerite will increase the amount of sphalerite and pyrite that float to the lead rougher flotation concentrate if process water from the zinc flotation is recycled.

- Flotation Objectives. The best economic benefit for the project is to maximize the recovery of precious metals to the lead rougher circuit where the highest value can be achieved. Flow sheet design will therefore focus on recovery of precious metals to lead concentrate and achieving a marketable lead concentrate grade at the expense of some zinc recovery.

The flotation test program in progress to address the above objectives includes:

- The use of co-collectors to improve the recovery of precious metals to the lead rougher concentrate

- The use of rougher concentrate regrind to improve both the lead and zinc concentrate cleaner flotation response

- The reduction or elimination of lime and or copper sulfate in the zinc rougher flotation circuit

- The investigation of process water treatment options to remove lime and copper sulfate from recycle water.

Thirteen tests were run with co-collectors, 3418A, AF31, Aero 3477, AF 208, Flomin C-4920, Flomin C-4132, Flomin C-4150, Flomin C-7436, Flomin C-4930, Flomin C-7931. The co-collectors were either used with PAX in the lead rougher flotation or used in place of PAX in the zinc rougher flotation. The -10 mesh samples used for the tests were all ground to a P_{80} of 105 microns and X-133 was used as the frother for all the tests. Lime was not added to the zinc flotation circuit.

Table 13-6: Typical Metal Distribution in Rougher Flotation Products

Test #	Product Identification	Co-Collectors Used	%Au	%Ag	%Pb	%Zn	%Fe
42/43	Lead Ro Con	AF31/C-4920	74.39	89.11	94.10	38.21	39.36
42/43	Zinc Ro Con	3418A	15.39	2.85	1.79	56.89	1.88
44/45	Lead Ro Con	3477	78.28	88.00	93.24	34.82	37.20
44/45	Zinc Ro Con	C-7436	10.42	3.90	2.33	60.49	6.86
47	Lead Ro Con	A-208	81.80	88.76	93.51	36.55	38.27
47	Zinc Ro Con	SEX/ C-4132	4.74	2.75	1.65	56.96	4.84
48/49	Lead Ro Con	C-4150	81.44	87.88	93.27	38.10	39.48
48/49	Zinc Ro Con	SIPX/C-4132	6.69	2.83	1.80	56.40	3.59
50/51	Lead Ro Con	C-7436	79.99	86.86	92.68	33.80	38.99
50/51	Zinc Ro Con	SEX/C4150	5.71	3.86	2.35	60.28	4.82
52	Lead Ro Con	C4930	80.98	87.17	93.20	37.04	39.96
52	Zinc Ro Con	PAX/H_2SO_4	4.34	4.73	3.01	60.35	5.97
53/54	Lead Ro Con	C-7931	84.35	87.17	93.89	37.14	36.64
53/54	Zinc Ro Con	C-7931	2.42	3.26	1.94	57.22	5.30

The results of the tests showed that additional silver (and gold) could be recovered to the lead rougher concentrate by using co-collectors. The best results for silver recovery were obtained with a combination of AF31/Flomin C-4920 and Aerofloat 208 as co-collectors. The tests gave silver recoveries of 89.11% and 88.76% to the lead rougher concentrates. A closer examination shows that Aerofloat 208 may be a better co-collector since it floated more gold (81.80% vs 74.39%) and less zinc (36.55% vs 38.21%) and iron (38.27% vs 39.36). than the AF31/Flomin C-4920 combination. Unfortunately the co-collectors also improved the flotation of the main contaminants sphalerite and pyrite to the lead rougher concentrate. More tests and mineralogical studies need to be conducted to ascertain whether there is some association of silver with sphalerite and pyrite. Testing currently in progress is analyzing the alternative of regrinding the rougher concentrate to improve mineral liberation, the use of sphalerite, and pyrite depressants to improve both the recovery of precious metals to and the grade of the final lead concentrate.

The results of the tests using very selective zinc collectors and co-collectors produced good results with less than 6% zinc left in the zinc (final) rougher tails in all the tests. The best result was achieved with SEX/Flomin C-4150 co-collector combination where 60.28% of the zinc in the ore reported to the zinc rougher concentrate with only 2.35% lead and 4.82% iron reporting in the zinc rougher concentrate. The 3418A co-collector had the lowest amount of contaminants of 1.79% lead and 1.88% iron but had only 56.89% of zinc reporting to the zinc rougher concentrate.

The following conclusions are drawn from the test work conducted so far by the Dawson's Metallurgical:

- The Escobal sulfide ore is amenable to selective flotation producing a lead concentrate with most of the silver and gold in the lead concentrate and a clean zinc concentrate with some precious metals content.
- Grinding the ore to 80 percent passing 105 microns produced mineral liberation suitable for the flotation process.
- Ore floated well with normal flotation reagents such as; potassium amyl xanthate (PAX), sodium isopropyl xanthate (SIPX), copper sulfate ($CuSO_4$), zinc sulfate, Aerofloat 208 and Aerofroth X-133.
- Very selective collectors and co-collectors can be used in the zinc circuit at lower pH to eliminate the use of lime.

13.5 DETERMINATION OF RECOVERIES AND REAGENT CONSUMPTIONS

The following recoveries and reagent consumptions will be used based on test work and determinations described in the preceding sections:

Table 13-7: Escobal Concentrator Operational Parameters

Parameter	Value	Units
Silver Recovery	86.7	Percent
Gold Recovery	75.1	Percent
Lead Recovery	82.5	Percent
Zinc Recovery	82.6	percent
Bond Work Index	17.56	kW-hr/Mt
Primary Grind Size (P_{80})	105	Microns

Table 13-8: Reagent Consumptions

Parameter	Value	Units
Collectors	60	g/t
Activators	30	g/t
Depressants	80	g/t
Frothers	30	g/t
Flocculant	60	g/t

13.6 ESTIMATED METALLURGICAL RECOVERIES

13.6.1 Design Throughput

The design basis for the Escobal project processing facility is 4,500/5,500 dry metric tons per day (mtpd) or 1,642,500/2,007,500 dry metric tons per year (mtpy). Sufficient resources are available for 19 years of milling at this rate.

13.6.2 Metallurgy

Process development to determine concentrator unit operations and to set the design criteria for the unit operations has been done by McClelland Laboratories Inc. of Reno, Nevada and FLSmidth Dawson Metallurgical Laboratories of Salt lake City, Utah. M3 has reviewed the data supplied by McClelland Laboratories Inc. and Dawson Metallurgical Laboratories and has relied on it to develop the process design criteria to be used for the design of the process facilities. The metallurgical testing program has followed industry accepted practices and is believed to be technically sound and representative for the deposit, although there can be no guarantee that all mineralogical assemblages have been tested. In addition, results obtained by testing vein samples may not always be representative of results obtained from production scale processing of the whole deposit. M3 has extrapolated the design criteria included in this document from test results. These preliminary design criteria may change as more computer simulation, laboratory, or pilot plant performance testing becomes available.

McClelland's and Dawson Metallurgical froth flotation test data from samples of the Escobal sulfide resources has shown 75.1 percent gold recovery, 86.7 silver recovery, 82.5% lead recovery and 82.6% zinc from feed grades averaging 415 g/t silver, 0.47 g/t gold, 0.72% lead and 1.23% zinc.

McClelland's tests also show that flotation recoveries of 66% and 84% for gold and silver were achievable from a mixed oxide/sulfide sample that had only about half the amount of sulfide sulfur contained in the sulfide samples. Based on experience at comparable deposits it is reasonable to assume that similar recoveries can be achieved for oxide material blended with sulfide material, though further testwork is required.

Metallurgical testing of material from the East Extension and West/Margarito Zone is currently in progress. Samples from each of these areas appear to be same mineralogically as samples tested from the Central and East zones and there is no reason to believe samples from the East Extension and West/Margarito will perform differently metallurgically. Results from this test work are expected in the third quarter of 2012.

14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

The mineral resource estimate described in this technical report is an update of a previous mineral resource estimate completed by MDA and publically reported on November 29, 2010. Mineral resource estimation for the Escobal project follows the guidelines of Canadian National Instrument 43-101 ("NI 43-101"). The modeling and estimation of silver, gold, lead, and zinc resources were done under the supervision of Paul G. Tietz, a qualified person under NI 43-101 with respect to mineral resource estimation. Mr. Tietz is independent of Tahoe by the definitions and criteria set forth in NI 43-101; there is no affiliation between Mr. Tietz and Tahoe except that of an independent consultant/client relationship. There are no mineral reserves estimated for the Escobal project.

The mineral resource described in this Report has an effective date for data input of January 23, 2012 and includes the data and analyses resulting from Tahoe Resources' 2011 work program up to and including drill hole E11-348. For the current resource estimate, MDA audited the data derived from drilling through December 2011, analyzed QA/QC data, conducted a site visit, and collected samples of drill core for verification purposes. All of these subjects are discussed in Section 14 of the technical report.

The Escobal deposit was modeled and estimated by evaluating the drill data statistically, interpreting mineral domains on cross sections and then level plans, analyzing the modeled mineralization statistically to establish estimation parameters, and estimating silver, lead, gold, and zinc grades into a three-dimensional block model. All modeling of the Escobal resources was performed using Gemcom Surpac® software.

All of the procedures and methods used to model and estimate the Escobal deposit are similar to those used by MDA in the previous 2010 resource estimate. Specific data and results have been updated to reflect the current work.

Although MDA is not an expert with respect to any of the following factors, MDA is not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the Escobal mineral resources as of the date of this report.

14.2 DATA

The Escobal deposit mineral resource reported in this technical report is based on project drill database consisting of 355 drill holes totaling 122,665 m. The project database includes the three Areneras and two Granadillo drill holes listed in drill total table in Section 10.1. The large majority of the drilling has been by diamond core drilling methods with the database containing 345 diamond core holes for 119,492 m. The remaining drilling was by reverse circulation "RC" methods (four drill holes for 751 m) or a combination of RC and diamond core (six drill holes for 2,422 m). The Escobal drill-hole assay database contains 22,937 silver assays, 22,936 gold assays, 22,844 lead assays, and 22,844 zinc assays. The geology database includes drill-hole lithology, alteration, vein type and percentages, sulfide content, and oxidation state data. Digital topography at 2 m-contours was supplied by Tahoe.

14.3 DEPOSIT GEOLOGY PERTINENT TO RESOURCE MODELING

Mineralization within the Escobal deposit is associated with two generally east-west trending structures (Central and East) that are characterized by multi-phase brecciation and quartz±sulfide veining. The mineralized structures can be up to 50m wide, though widths of 10-30m are more common. The structure/wallrock boundaries are often not distinct sharp contacts, but consist of a gradual decrease in brecciation/veining over a 5-10m distance. This gradation is especially common within the hanging wall andesite wallrock in the Central and East zones. Conversely, the footwall boundary is more clearly defined within the sedimentary rocks in the deeper sections of the Central Zone. Peripheral to the main mineralized structures, mineralization occurs within thin (<0.5m) sulfide-bearing quartz veins and breccias.

The Central Zone structure extends 1,200 m along a general east-west strike. [The western portion of the Central Zone is often referred to as the West Zone in various project reports and by project personnel. For the purposes of this technical report, the Central Zone refers to the combined West and Central zones.] The upper portion of the Central Zone primarily dips steeply to the north at approximately 70-75 degrees while at depth it becomes near-vertical. Its vertical extent is approximately 950m, and though the mineralization appears to be weakening, the Central Zone is still considered open at depth. Within its eastern half, the upper portion of the Central structure is truncated against a weakly to moderately mineralized, east-dipping structure whose orientation is sub-parallel to the main East Zone structure. Structural interpretations suggest that this Central Zone east-dipping structure is the structurally -offset western extension of the East Zone structure.

A late, generally weakly mineralized, quartz-calcite vein event occurs within the middle and eastern portions of the Central Zone mineralized structure. The late veining occurs as distinct, predominantly post-mineralization veining, often less than 1m thick, that cuts through intervals of higher-grade mineralization. The late veining is prevalent in almost all intercepts to some degree, with the percentage of late vein directly affecting the assay grade of the individual sample intervals. Where the late quartz-calcite vein attains an appreciable thickness (up to 10m), a “metal void” is created in the otherwise generally continuous high-grade mineralization. Isolated instances of higher-grade mineralization within the vein often are associated with the presence of sulfide-rich, clasts or remnant slivers of the mineralized wallrock.

The East Zone structure, which lies a few hundred meters east- northeast of the larger Central Zone structure, extends 850m along a general N80E strike and dips to the south at approximately 60-70 degrees. The East Zone has a vertical extent of approximately 800m, and is still considered open at depth. Drilling in 2011 has extended the East Zone mineralization both along strike to the east and down-dip. A near-vertical, east-trending mineralized structure, similar in orientation to the Central Zone mineralized structure, has been encountered at depth within the East Zone. The east-southeast plunging intersection of this structure with the primary south-dipping East Zone structure is a structurally complex area hosting significant mineralization. Extensions of the near-vertical structure up through the East Zone are marked by narrow, though often high-grade, quartz-sulfide veins.

The tuff lithology which overlies the andesite in the East Zone is a post-mineralization unit, and the East Zone mineralization truncates against the base of the tuff.

The Central and East zones contain predominantly sulfide mineralization, with minor oxide and mixed oxide/sulfide material within the upper portions of both zones. Silver, lead, and zinc occur throughout the Central and East zones, with better grades generally occurring within the sulfide mineralization. Gold also occurs throughout the deposit, though the richest gold mineralization is within the oxide and mixed material within the upper levels of the East Zone. Increased gold mineralization was also encountered in the 2011 drilling within the deeper portions of the Central Zone.

14.4 GEOLOGIC MODEL

A cross-sectional geologic model of the Escobal deposit was created by Tahoe and MDA. The cross-sections looked due east and were evenly spaced on 50m intervals. The cross-sections are numbered using the project's UTM Easting coordinates with the westernmost section being section 805700E and the eastern section is 808000E.

Drill-hole information, including rock type, oxidation, and type and percentage of veining, along with the topographic surface, were plotted on the cross sections. To augment the plotted drill information, the core photos of almost all of the drill holes were analyzed for structural information, especially the angle to core axis orientation of the mineralized veins and breccias. The core photo review also allowed for greater definition on the vein types and structural zone contacts.

The geologic model constructed from this data included 1) the wallrock lithologies, with all apparent structural offsets, 2) the oxidation boundaries showing oxide, mixed oxide and sulfide, and sulfide material, and 3) the mineralized structures within the Central and East zones. The latter structural zone model was used as a template to guide the mineral-domain modeling (discussed below).

Included in the Central Zone structural geology was the modeling of the distinct through-going, late quartz-calcite vein. The location and thickness of the late vein was determined by those drill intercepts which are dominantly composed of massive, late quartz-calcite veining. As discussed in the previous section, the late veining can occur throughout the full width of the mineralized structure zone as less than 1m-thick veins. With the current drill spacing, it is not practical to model the thin veins at the scale of the Escobal deposit. As such, the current resource model, with the one distinct late quartz-calcite vein trending through the heart of the Central mineralization, is simplistic in its representation of this weakly mineralized veining.

The inability to accurately estimate the weakly mineralized, late-stage veining, and its effect on the metal-grade distribution, is an uncertainty in the current resource model.

The lithology and oxidation models were converted into 3-dimensional solids which were used to code the block model. The lithology and oxidation codes were used to assign density values to the block model (see Section 14.6 for details on the block model density), while the oxidation coding was also used for resource classification.

14.5 MINERAL-DOMAIN GRADE MODELS

Cross-sectional mineral-domain models for each of the four metals were created for the Central and East zones. Distribution plots of silver, gold, lead, and zinc grades were made to help define the natural populations of metal grades to be modeled on the cross sections. The natural populations from the distribution plots were checked against the drill data and geologic model to determine if the populations represented realistic, continuous mineral types. Low-grade, moderate-grade, and high-grade mineral domains (domain codes 100, 200, and 300, respectively, in the block model) were determined for all four metals.

The resulting grade populations used to create the mineral domains are shown in Table 14-1.

Table 14-1: Mineral Domain Grade Populations

Metal	Zone	Low-Grade Domain (domain 100)	Mid-Grade Domain (domain 200)	High-Grade Domain (domain 300)
Silver	Central	~ 5 – 130g Ag/t	~ 130 – 1300g Ag/t	~ >1300g Ag/t
	East	~ 5 – 145g Ag/t	~ 145 – 800g Ag/t	~ >800g Ag/t
Gold	Central	~ 0.06 – 0.5g Au/t	~ 0.5 – 2.0g Au/t	~ >2.0g Au/t
	East	~ 0.13 – 0.37g Au/t	~ 0.37 – 1.0g Au/t	~ >1.0g Au/t
Lead	Central	~ 0.0025 – 0.05% Pb	~ 0.05 – 1.5% Pb	~ >1.5% Pb
	East	~ 0.0025 – 0.08% Pb	~ 0.08 – 0.6% Pb	~ >0.6% Pb
Zinc	Central	~ 0.0075 – 0.16% Zn	~ 0.16 – 1.0% Zn	~ >1.0% Zn
	East	~ 0.0075 – 0.06% Zn	~ 0.15 – 0.85% Zn	~ >0.85% Zn

Distinct mineral domains, which were used to control estimation, were created based on the analytical population breaks indicated by the distribution plots and the geological interpretation. The mineral domains as modeled and drawn on the cross sections are not strict "grade shells" but are created using geologic information for defining orientation, geometry, continuity, and contacts in conjunction with the grades. Each of these domains represents a distinct style of mineralization. While all metals are generally spatially related, they are not always exactly coincident, thereby requiring separate domain models for each metal.

The unique metal-grade and geologic characteristics of the late, quartz-calcite vein required the creation of a unique mineral domain for it within each of the four metals (domain code 110). The late vein mineral domain included all late vein assay values and restricted estimation to within the late vein.

At the start of the mineral-domain modeling, it was realized that the low-grade gold domain was smaller in cross-sectional area than the low-grade silver domain. To assure that some level of gold mineralization would be estimated into all blocks containing silver, a dilutional domain (domain code 10) was added to the gold mineral-domain model.

The mineral domain models were constructed by MDA and Tahoe personnel, though the final edits and review were done by MDA. Each metal was modeled independently, though the completed silver sectional model was used to help guide the general trends of the gold, lead, and zinc domain models. For the current resource estimate, a spatial and statistical analysis indicated a close relationship between the lead and zinc low- and mid-grade sectional domains. Accordingly, the low- and mid-grade lead sectional domains were used as a proxy for the corresponding zinc sectional domains. A unique high-grade zinc sectional domain was created due to the increased variation from the lead high-grade domain.

The mineral domain cross sections were three-dimensionally rectified to 5 m level plans, which coincide with the center of the block-model's vertical block size. The rectified levels were used to code domain percentages into the block model.

Typical cross sections through the Central Zone silver domain model are shown in Figure 14-1 and Figure 14-2, while the East Zone silver domain model is shown in Figure 14-3. Also included on the cross-section figures are estimation areas used to both restrict the samples used for estimation, and to define orientations for estimation search ellipsoids. See Sections 14.7 and 14.8 for further details on the estimation areas.

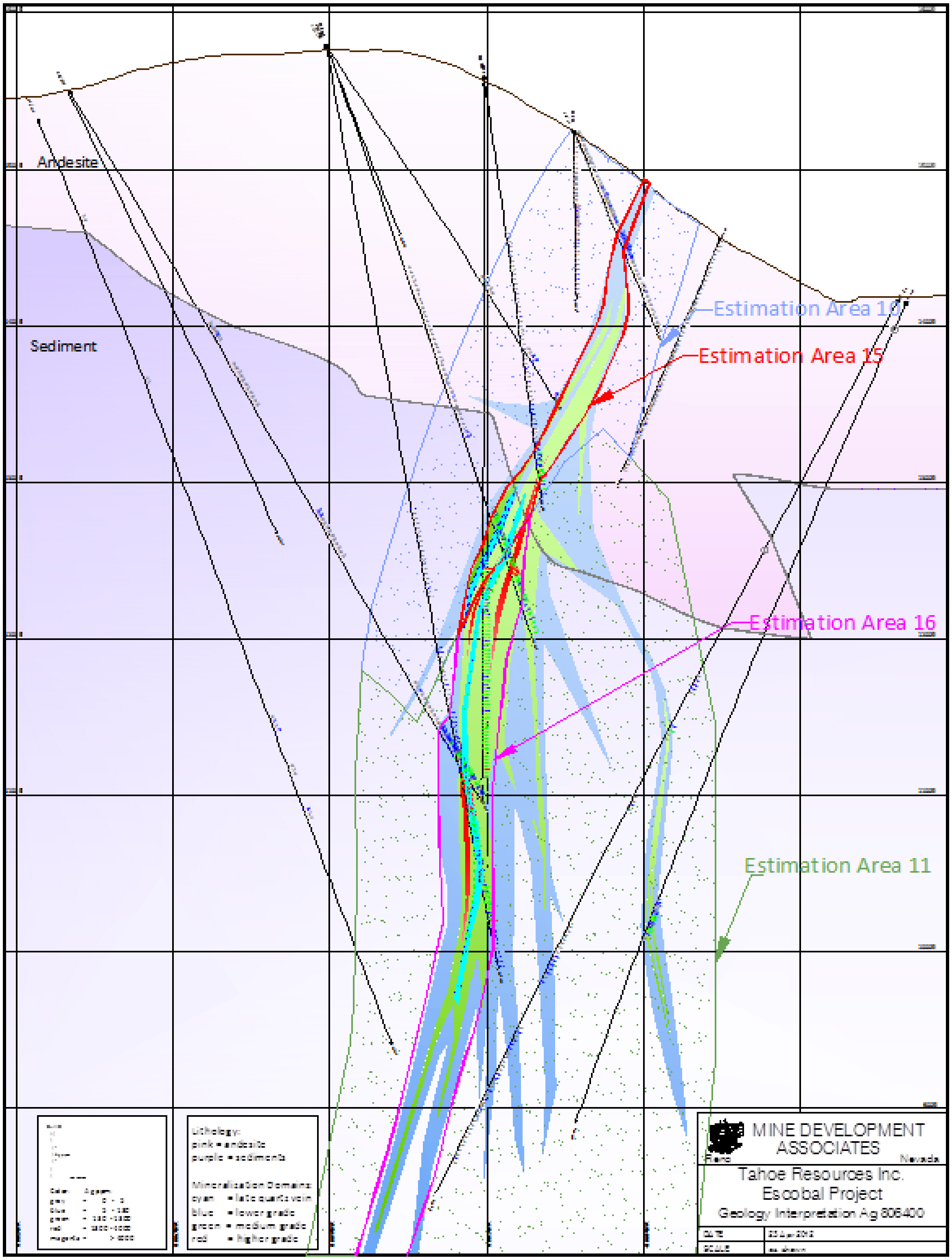


Figure 14-1: Section 806400 – Escobal Central Zone Silver Geologic Model

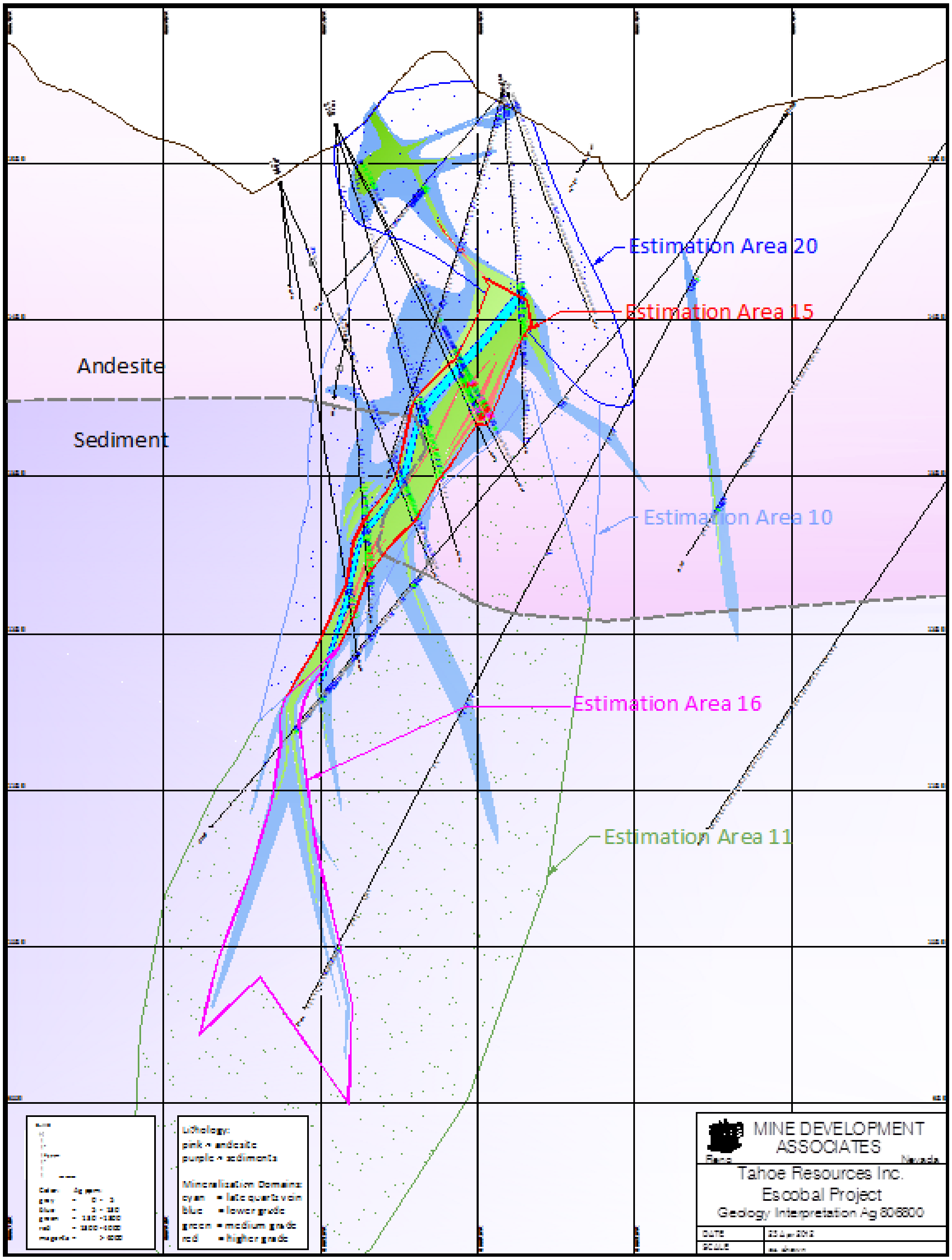


Figure 14-2: Section 806800 – Escobal Central Zone Silver Geologic Model

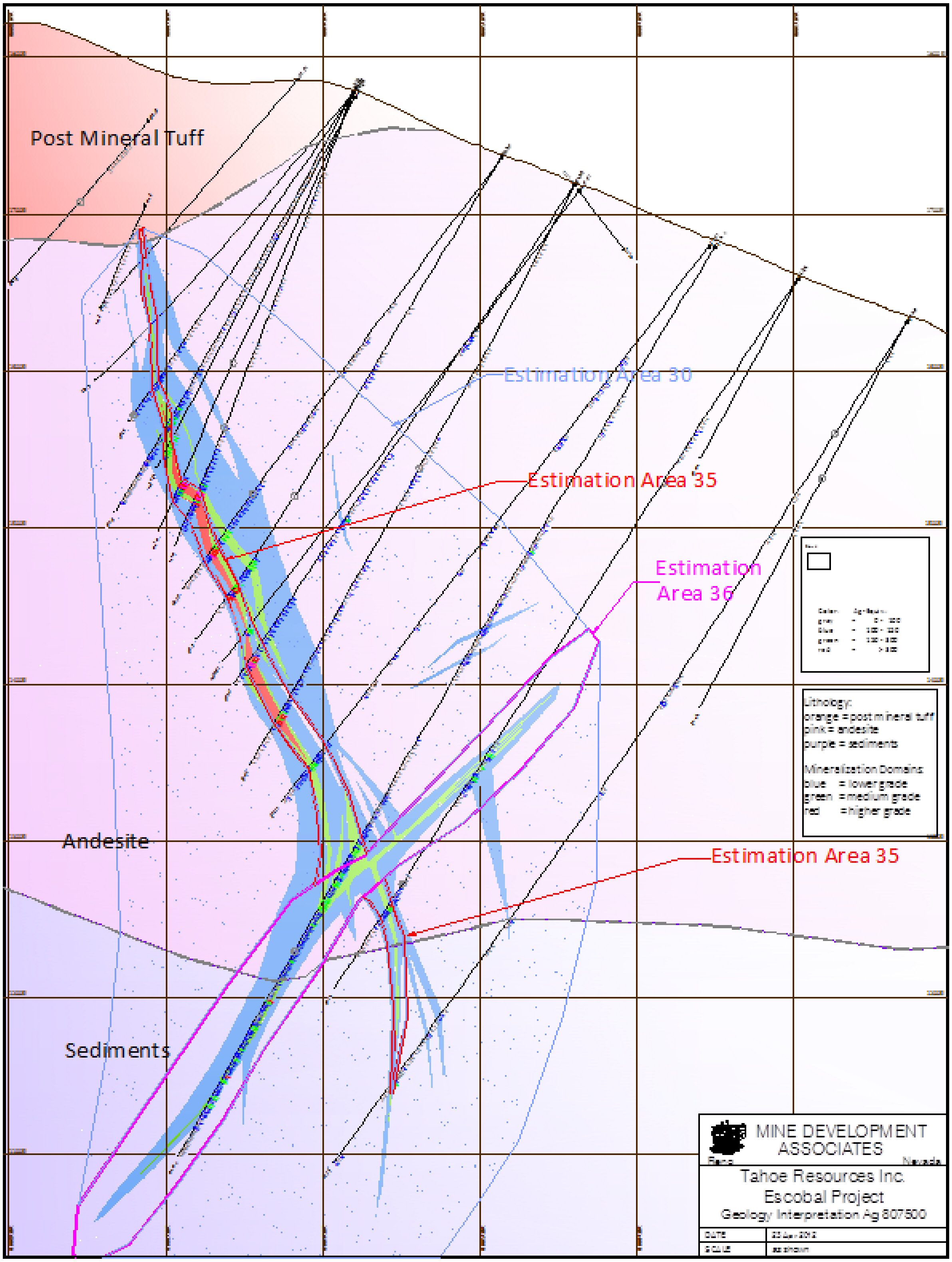


Figure 14-3: Section 807450 – Escobal East Zone Silver Geologic Model

14.6 DENSITY

The following section on Escobal density is taken from the November 2010 technical report. No further analysis of the data has been completed.

The density values used in the current resource estimate are based on 3,397 density measurements collected by Tahoe from diamond drill core in the Escobal resource area. The samples were grouped according to lithology, oxidation state, and lead mineral domains, as discussed in Sections 14.4 and 14.5. The oxidation state (oxidized, mixed oxidized and sulfidic, and sulfidic) and lead domains (100, 110, 200, and 300) were used as a spatial control on model density due to the correlation with sulfide content, which is the dominant factor in density variation within the mineral model. After completing a statistical review of the density data, MDA eliminated two samples as being outliers or improbable and capped two measurements. Due to potential sample collection bias (the use of whole solid core versus fractured, possibly less-dense core), MDA lowered the values of each group by about 1% for use in the current resource estimate. The lithology, oxidation state, and mineral domains were used to code the block model with the assigned density values.

The density values used in the estimate are shown in the “Model” columns in Table 14-2 and Table 14-3. Table 14-2 lists the density values used for the wallrock lithologies. There are no density data for the overlying Tertiary tuff lithology, and a density value of 2.54g/cm^3 was assigned to this lithology. The lack of density data for the tuff is not considered significant to the current resource estimation because no mineralization has been modeled into this post-mineralization rock unit.

The density values used for the mineralized material, which have the most impact on the resource estimate, are represented in Table 14-3. Within all mineral domains, there was a natural grouping of density values by oxidation state; lower density values for the oxidized and mixed material as compared to the higher density values within the sulfide material. The mineral domain densities vary from 2.52g/cm^3 within the oxidized and mixed low-grade (100 domain) material, to 2.82g/cm^3 within the sulfidic high-grade (300 domain) material within the Central Zone. The increased presence of massive sulfide in the Central Zone as compared to the East Zone is demonstrated in the density values; as a result, unique values for the 300 lead domain mineralization are assigned to each zone. This difference in density between East and West zones for the lower grade domains is not observed, so one density value is used for each of the 100 and 200 mineral domains in both zones. The limited sampling in the oxidized/mixed 300 domain resulted in MDA assigning the same values as those for the 200 domain for this rock type. Additional density measurements for the oxidized/mixed high-grade mineralization are recommended.

The block’s density value is the volume-weighted average of the unmineralized lithologic density, combined with the volumes of lead mineralization domains.

Table 14-2: Lithology Density Values Used in Model

Lithology	#	Mean	Median	Min.	Max.	Std.Dev.	Model
andesite	291	2.63	2.65	2.17	2.87	0.09	**2.61**
sediment	246	2.65	2.67	2.14	2.99	0.12	**2.64**
tuff	No data						**2.54**

Table 14-3: Mineral Domain Density Values Used in Model

Pb Domain	Ox. State	#	Mean	Median	Min.	Max.	Std.Dev.	Model
100	Ox-Mix	186	2.55	2.55	2.16	2.79	0.09	**2.52**
200	Ox-Mix	114	2.57	2.57	2.26	2.84	0.09	**2.54**
300	Ox-Mix	6	2.51	2.52	2.30	2.67	0.15	**2.54**
110	all	126	2.64	2.63	2.44	3.64	0.11	**2.59**
100	Sulfide	472	2.65	2.67	2.17	2.99	0.09	**2.63**
200	Sulfide	1835	2.68	2.68	1.81	3.39	0.10	**2.65**
300 (East)	Sulfide	162	2.77	2.74	2.56	3.31	0.14	**2.72**
300 (Central)	Sulfide	418	2.90	2.80	2.15	4.30	0.31	**2.82**

14.7 SAMPLE CODING AND COMPOSITING

Drill-hole assays were coded by the sectional mineral-domain polygons. The coded drill samples were then sub-divided into four groups using 3-dimensional solids created from the estimation areas shown in Figure 14-1, Figure 14-2, and Figure 14-3. The four assay groups are:

1) Outside the Central structural zone (estimation areas 10, 11, and 20),
2) Inside the Central structural zone (estimation areas 15 and 16),
3) Outside the East structural zone (estimation area 30), and
4) Inside the East structural zone (estimation area 35).

The drill samples, and subsequent composite samples, were sub-divided in this manner to restrict estimation across the boundaries of the mineralized structural zones. This was done to restrict the higher-grade structural zone assays to only influence blocks within the structural zone. Allowing the structural zone assays to estimate outside of the primary structure would have resulted in a clear overestimation of grade within the generally weakly altered wallrock.

All mineralization domains were evaluated in these groups, both statistically and spatially, within this geologic context. After these analyses, MDA capped a total of 144 individual metal analyses, for all domains and areas within all metals, which were statistically and spatially deemed beyond a given domain's natural population of samples. This number of samples capped represents approximately 0.3% of the total assay values within the database. The capped analyses occur within all grade ranges and all estimation areas. Descriptive statistics of the uncapped and capped sample grades by estimation area and domain are given in Appendix C.

Compositing was made at 3 m down-hole lengths, honoring all mineral domain and estimation area boundaries. Composite descriptive statistics for the estimation area groups and respective metal domains are presented in Table 14-4.

Table 14-4: Escobal Mineral Domain Composite Statistics

Silver Composites **Areas 10 - 11- 20**

Domain	Valid N	Total Length (m)	Mean (g/t)	Median (g/t)	Std.Dev.	CV	Minimum (g/t)	Maximum (g/t)
100	1730	4960	26.1	13.9	33.9	1.30	0.0	316.6
200	221	583	257.0	188.0	218.4	0.85	10.8	1400.0
300	2	3	1720.5	1775.2	67.9	0.04	1663.9	1775.2

Silver Composites **Areas 15 - 16**

Domain	Valid N 0	Total Length (m)	Mean (g/t)	Median (g/t)	Std.Dev. 0.0	CV 0	Minimum (g/t)	Maximum (g/t)
110	143	360	68.5	37.2	91.2	1.33	0.0	610.0
100	908	2530	40.2	28.7	35.4	0.88	0.0	202.7
200	856	2321	356.3	277.4	254.2	0.71	9.0	1900.0
300	108	247	2617.1	2230.0	1719.7	0.66	151.5	11100.0

Silver Composites **Area 30**

Domain	Valid N 0	Total Length (m)	Mean (g/t)	Median (g/t)	Std.Dev. 0.0	CV 0	Minimum (g/t)	Maximum (g/t)
100	1121	3190	24.4	14.2	26.5	1.08	0.0	200.0
200	102	227	283.2	213.2	188.7	0.67	33.0	995.4
300	4	8	988.6	941.6	287.3	0.29	731.7	1500.0

Silver Composites **Area 35**

Domain	Valid N 0	Total Length (m)	Mean (g/t)	Median (g/t)	Std.Dev. 0.0	CV 0	Minimum (g/t)	Maximum (g/t)
100	622	1707	40.7	28.1	43.4	1.07	0.0	477.4
200	301	724	294.1	248.2	181.7	0.62	10.0	1291.7
300	98	223	2096.0	1535.2	1492.6	0.71	157.5	6614.9

Gold Composites **Areas 10 - 11- 20**

Domain	Valid N	Total Length (m)	Mean (g/t)	Median (g/t)	Std.Dev.	CV	Minimum (g/t)	Maximum (g/t)
10	1250	3546	0.020	0.015	0.022	1.13	0.000	0.200
100	729	2038	0.128	0.098	0.099	0.77	0.000	0.742
200	39	87	0.944	0.791	0.483	0.51	0.330	2.318
300	2	3	3.241	3.241	0.846	0.26	2.643	3.840

Gold Composites **Areas 15 - 16**

Domain	Valid N	Total Length (m)	Mean (g/t)	Median (g/t)	Std.Dev.	CV	Minimum (g/t)	Maximum (g/t)
110	146	367	0.253	0.100	0.512	2.03	0.000	4.834
10	430	1156	0.025	0.022	0.021	0.83	0.000	0.134
100	1042	2844	0.162	0.139	0.106	0.65	0.000	0.860
200	377	924	0.797	0.685	0.436	0.55	0.052	3.975
300	76	177	3.608	3.098	2.134	0.59	0.847	12.000

Gold Composites **Area 30**

Domain	Valid N	Total Length (m)	Mean (g/t)	Median (g/t)	Std.Dev.	CV	Minimum (g/t)	Maximum (g/t)
10	1123	3174	0.020	0.013	0.02	1.21	0.000	0.170
100	119	288	0.147	0.135	0.07	0.50	0.026	0.366
200	13	24	0.597	0.454	0.31	0.53	0.295	1.152
300	5	10	5.553	1.245	10.79	1.94	1.126	29.918

Gold Composites **Area 35**

Domain	Valid N	Total Length (m)	Mean (g/t)	Median (g/t)	Std.Dev.	CV	Minimum (g/t)	Maximum (g/t)
10	584	1607	0.034	0.026	0.03	0.88	0.000	0.180
100	272	647	0.169	0.158	0.08	0.45	0.000	0.526
200	115	231	0.575	0.555	0.24	0.42	0.040	1.750
300	85	175	4.441	2.269	5.49	1.24	0.802	40.000

Table 14-4: Escobal Mineral Domain Composite Statistics (Continued)

Lead Composites — **Areas 10 - 11- 20**

Domain	Valid N	Total Length (m)	Mean (%)	Median (%)	Std.Dev.	CV	Minimum (%)	Maximum (%)
100	2205	6381	0.020	0.010	0.029	1.457	0.000	0.328
200	487	1358	0.226	0.138	0.234	1.038	0.001	2.000
300	4	9	2.350	2.305	0.417	0.177	1.630	2.805

Lead Composites — **Areas 15 - 16**

Domain	Valid N	Total Length (m)	Mean (%)	Median (%)	Std.Dev.	CV	Minimum (%)	Maximum (%)
110	146	367	0.071	0.042	0.087	1.236	0.000	0.508
100	625	1759	0.027	0.017	0.028	1.065	0.000	0.187
200	1068	2990	0.359	0.273	0.293	0.816	0.006	1.900
300	285	740	3.214	2.235	2.582	0.803	0.163	15.000

Lead Composites — **Area 30**

Domain	Valid N	Total Length (m)	Mean (%)	Median (%)	Std.Dev.	CV	Minimum (%)	Maximum (%)
100	1583	4551	0.021	0.015	0.020	0.960	0.000	0.149
200	286	755	0.165	0.128	0.119	0.720	0.008	1.000
300	8	16	1.471	1.371	0.378	0.257	0.763	2.090

Lead Composites — **Area 35**

Domain	Valid N	Total Length (m)	Mean (%)	Median (%)	Std.Dev.	CV	Minimum (%)	Maximum (%)
100	443	1207	0.025	0.020	0.021	0.832	0.000	0.120
200	524	1380	0.190	0.158	0.130	0.681	0.011	0.917
300	114	253	1.402	1.102	0.908	0.648	0.134	4.410

Zinc Composites — **Areas 10 - 11- 20**

Domain	Valid N	Total Length (m)	Mean (%)	Median (%)	Std.Dev.	CV	Minimum (%)	Maximum (%)
100	2205	6381	0.052	0.030	0.065	1.253	0.000	0.712
200	457	1266	0.366	0.268	0.313	0.855	0.006	1.990
300	45	111	2.045	1.670	1.061	0.519	0.690	5.000

Zinc Composites — **Areas 15 - 16**

Domain	Valid N	Total Length (m)	Mean (%)	Median (%)	Std.Dev.	CV	Minimum (%)	Maximum (%)
110	146	367	0.131	0.093	0.126	0.960	0.001	0.567
100	625	1759	0.074	0.049	0.079	1.067	0.002	0.702
200	828	2237	0.455	0.406	0.281	0.617	0.014	2.300
300	554	1484	3.643	2.105	4.324	1.187	0.197	33.355

Zinc Composites — **Area 30**

Domain	Valid N	Total Length (m)	Mean (%)	Median (%)	Std.Dev.	CV	Minimum (%)	Maximum (%)
100	1583	4551	0.060	0.043	0.055	0.922	0.001	0.355
200	276	728	0.341	0.284	0.220	0.644	0.020	1.640
300	20	41	2.130	1.507	1.250	0.587	0.919	4.850

Zinc Composites — **Area 35**

Domain	Valid N	Total Length (m)	Mean (%)	Median (%)	Std.Dev.	CV	Minimum (%)	Maximum (%)
100	443	1207	0.071	0.053	0.063	0.884	0.001	0.500
200	499	1309	0.349	0.309	0.193	0.553	0.018	1.271
300	143	326	1.727	1.430	0.979	0.567	0.250	5.970

14.8 RESOURCE MODEL AND ESTIMATION

The Escobal resource block model replicates the relatively complex metal distributions and geometries observed in the geologic and mineral-domain cross-sectional models. Because of the rather contorted geometries and the unique composite grouping needed to control the estimation, eight separate estimation areas were created at Escobal; five in the Central Zone (areas 10, 11, 15, 16, and 20) and three in the East Zone (areas 30, 35 and 36). The locations of these areas relative to the mineral domains are shown in Figure 14-1, Figure 14-2, and Figure 14-3. The estimation areas were modeled with solids, which were used to code the block model.

The portion of each 5 m by 5 m by 2.5 m block inside each mineral domain was estimated using only composites from inside its respective domain and estimation area group. Grade interpolation utilized Inverse Distance Cubed (ID3), with nearest neighbor and ordinary kriging estimates also being made for checking estimation results and sensitivities. All estimations used three search passes, and successive passes did not overwrite previous estimation passes. The final pass filled the modeled domains. Strict (5m) search restrictions (pullbacks) were employed for the higher-grade values for all four metals within the late-stage quartz calcite vein (domain 110). Less restrictive pullbacks were also employed for specific zinc and gold domains to control the influence of the extreme high-grade samples. The Escobal estimation parameters are given in Table 14-5.

Variography and geostatistical evaluations were made to determine distances for search and classification criteria. A grade relationship for silver of up to 40m outside of the East and Central structural zones and 60 m inside the structural zones was observed in the statistics, and this value was used in the criteria for classifying Indicated resources.

Table 14-5: Escobal Estimation Parameters for Mineral Resources

Description	Parameter
SEARCH PARAMETERS: All Estimation Areas	
Samples: minimum/maximum/maximum per hole (1st pass search)	2 / 9 / 3
Samples: minimum/maximum/maximum per hole (2nd and 3rd pass searches)	1 / 9 / 3
First Pass Search (m): major/semimajor/minor	75 / 75 / 37.5
Second Pass Search (m): major/semimajor/minor	150 / 150 / 75
Third Pass Search (m): major/semimajor/minor	Fill all domains
SEARCH ELLIPSOID ORIENTATIONS	
Search Bearing/Plunge/Tilt : Estimation areas 10 and 15	270° / 0° / -62.5°
Search Bearing/Plunge/Tilt : Estimation areas 11 and 16	270° / 0° / 90°
Search Bearing/Plunge/Tilt : Estimation area 20	260° / 0° / 60°
Search Bearing/Plunge/Tilt : Estimation areas 30 and 35	260° / 0° / 65°
Search Bearing/Plunge/Tilt : Estimation areas 36	260° / 0° / -70°

SEARCH RESTRICTIONS

Domain	Areas	Grade Threshold	Search Restriction (m)	Estimation Pass
Ag 110	all	>250 g/t	5	all
Pb 110	all	>0.25 %	5	all
Zn 110	all	>0.4 %	5	all
Au 110	all	>1.0 g/t	5	all
Zn 300	15, 16	>1.0 %	75	all
Au 300	30	>25 g/t	75	all
Au 300	35	>6 g/t	75	all

14.9 RESOURCE CLASSIFICATION

MDA classified the Escobal resources in order of increasing geological and quantitative confidence into Inferred and Indicated categories defined by the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" in 2005, in compliance with Canadian National Instrument 43-101. CIM mineral resource definitions are given below:

Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

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

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

Inferred Mineral Resource

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

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Indicated Mineral Resource

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

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

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

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

14.10 MINERAL RESOURCES

MDA classified the Escobal resources by a combination of distance to the nearest sample and the number of samples, while at the same time taking into account reliability of underlying data and understanding and use of the geology. All estimated mineralization was assigned to be at least Inferred. There are no Measured resources within the Escobal deposit at this time, primarily due to limited QA/QC data and some spatial uncertainty within parts of the model. To be classified as Indicated, the blocks outside of the East and Central structural zones must be within an average distance of 40m to two silver composites, coming from two different drill holes, within an 80m isotropic search. The isotropic search is limited to those composites outside of the structural zones. This effectively creates a requirement of two drill holes having the closest sample within 40m. Within the East and Central structural zones, Indicated blocks must be within an average distance of 60m to two silver composites, coming from two different drill holes, within a 120m isotropic search. The 120m isotropic search is limited to those composites inside of the structural zones. There are no Indicated resources within the oxide portions of the deposit or in the gold-dominant oxide and mixed material within the upper levels of the East zone, due to the reasons noted above and also due to a lack of metallurgical data and some uncertainty in the density data within the oxide material. None of these issues detract from the overall confidence in the global project resource estimate, but they do detract from confidence in some of the accuracy which MDA believes is required for Measured and Indicated in these specific areas. The resource classifications will likely rise when those issues listed above are resolved.

Because of the requirement that the resource exists "in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction," MDA is reporting the resources at an approximate economic cutoff grade that is reasonable for deposits of this nature that will likely be mined by underground methods. As such, some economic considerations were used to determine cutoff grades at which the resource is presented. MDA considered reasonable metal prices and extraction costs and recoveries, albeit in a general sense.

The Escobal reported resource is summarized in Table 14-6, while the Escobal estimation results are tabulated by classification and oxidation state in Table 14-7; the latter table provides the resource numbers at various AgEq cutoff grades to better assess grade-tonnage fluctuation. The stated resource comes from the block-diluted grade within the entire 5m by 5m by 2.5m blocks and is tabulated on a silver-equivalent ("AgEq") cutoff grade of 150g AgEq/t. All material, regardless of which metal is present and which is absent, is tabulated. Because multiple metals exist, but on a local scale do not necessarily co-exist, the AgEq grade is used for tabulation. Using the individual metal grades of each block, the AgEq grade is calculated using the following formula:

$$\text{g AgEq/t} = \text{g Ag/t} + (0.0026 * \text{Pb ppm}) + (52 * \text{g Au/t}) + (0.024 * \text{Zn ppm})$$

This formula is based on prices of US$25.00 per ounce silver, US$0.90 per pound zinc, US$0.95per pound lead, and US$1300.00 per ounce gold. No metal recoveries are applied, as this is the *in situ* resource, though expected recoveries are similar across all metals resulting in no change to the stated equivalency formula. Typical cross sections through the Escobal block model showing AgEq block grades for the Central and East zones are given in Figure 14-4, Figure 14-5, and Figure 14-6. These are the same cross-section locations used to depict the mineral-domain models in Section 14.5.

Silver is the dominant metal throughout much of the Escobal deposit accounting for, on average, approximately 85 percent of the *in situ* block values. The one exception to the silver-dominant mineralization style is within the upper levels of the East Zone where the mineralization is primarily gold with decreased silver, lead, and zinc. The highest estimated gold grades (up to 40g Au/t) at Escobal occur within this area, and much of the AgEq stated resource is driven by the gold content in that area. Drilling within this localized area is more widely-spaced than in much of the East Zone, and the spatial location of the high-grade gold is not fully understood.

MDA recommends that additional infill drilling must be completed to better characterize the gold mineralization.

Table 14-6: Escobal Deposit Reported Resource

Escobal Reported Resource (150g AgEq/t cut-off grade)

Class	Mineral Type	Tonnes	Silver (g Ag/t)	Lead (% Pb)	Gold (g Au/t)	Zinc (% Zn)	Silver (oz)	Lead (lbs)	Gold (oz)	Zinc (lbs)
Indicated	Mixed	1,200,000	274.8	0.234	0.406	0.418	10,420,000	6,090,000	15,000	10,870,000
Indicated	Sulfide	25,900,000	428.4	0.731	0.430	1.318	357,050,000	418,020,000	358,000	753,140,000
Indicated	**Total**	**27,100,000**	**421.7**	**0.710**	**0.429**	**1.279**	**367,470,000**	**424,110,000**	**373,000**	**764,010,000**
Inferred	Oxide	800,000	292.2	0.185	0.803	0.315	7,190,000	3,120,000	20,000	5,310,000
Inferred	Mixed	400,000	202.2	0.158	2.496	0.241	2,360,000	1,260,000	29,000	1,930,000
Inferred	Sulfide	3,400,000	250.8	0.400	0.332	0.788	27,180,000	29,730,000	36,000	58,550,000
Inferred	**Total**	**4,600,000**	**253.9**	**0.344**	**0.6**	**0.663**	**36,730,000**	**34,110,000**	**85,000**	**65,790,000**

Table 14-7: Escobal Deposit AgEq Resource Tabulation

Indicated mixed material:

Cutoff g AgEq/t	Tonnes	Silver (g Ag/t)	Lead (% Pb)	Gold (g Au/t)	Zinc (% Zn)	Silver (oz)	Lead (lbs)	Gold (oz)	Zinc (lbs)	AgEq (g/t)
75	1,800,000	209.2	0.183	0.354	0.333	12,210,000	7,340,000	21,000	13,340,000	240.4
100	1,500,000	234.4	0.204	0.372	0.368	11,600,000	6,930,000	18,000	12,470,000	267.9
120	1,400,000	251.1	0.217	0.384	0.389	11,150,000	6,610,000	17,000	11,860,000	286
140	1,200,000	267.5	0.229	0.400	0.410	10,650,000	6,260,000	16,000	11,180,000	304
150	1,200,000	274.8	0.234	0.406	0.418	10,420,000	6,090,000	15,000	10,870,000	312
160	1,100,000	283.2	0.239	0.414	0.426	10,150,000	5,880,000	15,000	10,470,000	321.2
170	1,100,000	291.4	0.243	0.421	0.435	9,880,000	5,660,000	14,000	10,120,000	330
180	1,000,000	299.3	0.247	0.427	0.443	9,620,000	5,450,000	14,000	9,770,000	338.5
190	900,000	307.4	0.252	0.432	0.452	9,350,000	5,260,000	13,000	9,420,000	347.2
200	900,000	314.6	0.257	0.439	0.459	9,100,000	5,110,000	13,000	9,110,000	355.1
210	900,000	322.3	0.264	0.448	0.470	8,820,000	4,950,000	12,000	8,810,000	363.7
220	800,000	329.1	0.270	0.454	0.478	8,570,000	4,820,000	12,000	8,540,000	371.2
230	800,000	336.9	0.277	0.462	0.487	8,290,000	4,680,000	11,000	8,210,000	379.8
240	700,000	344.4	0.284	0.468	0.496	8,020,000	4,530,000	11,000	7,910,000	388
250	700,000	351.6	0.291	0.476	0.505	7,750,000	4,400,000	10,000	7,640,000	396.1
300	500,000	401.6	0.343	0.530	0.569	6,080,000	3,570,000	8,000	5,900,000	451.7
350	300,000	460.5	0.410	0.604	0.663	4,590,000	2,800,000	6,000	4,540,000	518.5
400	200,000	523.4	0.468	0.660	0.704	3,540,000	2,170,000	4,000	3,270,000	586.8
450	100,000	604.6	0.567	0.755	0.756	2,640,000	1,700,000	3,000	2,270,000	676.7
500	100,000	700.9	0.713	0.849	0.892	2,000,000	1,400,000	2,000	1,750,000	785
1000	20,000	1194.5	1.372	0.821	1.293	770,000	610,000	1,000	570,000	1303.9

Indicated sulfide material:

Cutoff g AgEq/t	Tonnes	Silver (g Ag/t)	Lead (% Pb)	Gold (g Au/t)	Zinc (% Zn)	Silver (oz)	Lead (lbs)	Gold (oz)	Zinc (lbs)	AgEq (g/t)
75	38,700,000	314.2	0.563	0.331	1.020	390,760,000	479,880,000	412,000	870,080,000	370.6
100	32,500,000	361.5	0.633	0.372	1.147	377,510,000	453,220,000	389,000	821,620,000	424.8
120	29,400,000	390.9	0.675	0.397	1.222	368,940,000	436,970,000	375,000	790,540,000	458.5
140	26,900,000	416.6	0.714	0.420	1.287	360,900,000	423,790,000	364,000	764,590,000	487.9
150	25,900,000	428.4	0.731	0.430	1.318	357,050,000	418,020,000	358,000	753,140,000	501.4
160	25,000,000	439.8	0.749	0.440	1.348	353,170,000	412,580,000	353,000	742,210,000	514.6
170	24,100,000	451.0	0.767	0.450	1.378	349,290,000	407,330,000	348,000	731,580,000	527.4
180	23,200,000	462.3	0.785	0.460	1.408	345,330,000	402,060,000	343,000	721,190,000	540.4
190	22,400,000	473.5	0.803	0.469	1.438	341,280,000	396,920,000	338,000	710,700,000	553.3
200	21,600,000	484.9	0.822	0.479	1.468	337,160,000	391,680,000	333,000	700,080,000	566.4
210	20,900,000	496.3	0.840	0.489	1.498	333,010,000	386,540,000	328,000	689,430,000	579.5
220	20,100,000	507.8	0.859	0.499	1.528	328,800,000	381,370,000	323,000	678,600,000	592.8
230	19,400,000	519.8	0.879	0.510	1.560	324,370,000	376,030,000	318,000	667,510,000	606.6
240	18,700,000	532.1	0.900	0.521	1.593	319,810,000	370,780,000	313,000	656,560,000	620.8
250	18,000,000	545.0	0.921	0.533	1.627	315,090,000	365,170,000	308,000	645,050,000	635.7
300	14,900,000	610.2	1.030	0.591	1.797	292,130,000	338,050,000	283,000	590,060,000	710.8
350	12,500,000	675.7	1.134	0.646	1.959	270,930,000	311,650,000	259,000	538,650,000	785.8
400	10,500,000	743.1	1.233	0.698	2.108	251,380,000	285,890,000	236,000	489,020,000	862.1
450	8,800,000	820.3	1.334	0.753	2.256	231,820,000	258,480,000	213,000	437,110,000	948.3
500	7,300,000	905.7	1.430	0.807	2.388	213,670,000	231,280,000	190,000	386,370,000	1042.1
1000	2,300,000	1697.1	2.115	1.114	2.995	126,290,000	107,900,000	83,000	152,850,000	1881.9

Table 14-8: Escobal Deposit AgEq Resource Tabulation (continued)

Inferred oxide material:

Cutoff g AgEq/t	Tonnes	Silver (g Ag/t)	Lead (% Pb)	Gold (g Au/t)	Zinc (% Zn)	Silver (oz)	Lead (lbs)	Gold (oz)	Zinc (lbs)	AgEq (g/t)
75	1,000,000	235.9	0.152	0.711	0.268	7,880,000	3,470,000	24,000	6,140,000	283.2
100	900,000	258.3	0.165	0.755	0.288	7,640,000	3,340,000	22,000	5,850,000	308.7
120	900,000	272.5	0.173	0.779	0.300	7,460,000	3,250,000	21,000	5,630,000	324.7
140	800,000	286.0	0.181	0.797	0.310	7,280,000	3,160,000	20,000	5,410,000	339.6
150	800,000	292.2	0.185	0.803	0.315	7,190,000	3,120,000	20,000	5,310,000	346.3
160	700,000	299.0	0.189	0.805	0.320	7,090,000	3,070,000	19,000	5,200,000	353.5
170	700,000	304.6	0.192	0.807	0.324	7,000,000	3,020,000	19,000	5,110,000	359.3
180	700,000	312.0	0.195	0.817	0.329	6,870,000	2,950,000	18,000	4,970,000	367.5
190	700,000	318.9	0.199	0.821	0.333	6,750,000	2,880,000	17,000	4,840,000	374.8
200	600,000	325.1	0.202	0.826	0.337	6,640,000	2,820,000	17,000	4,720,000	381.4
210	600,000	331.5	0.205	0.830	0.341	6,520,000	2,760,000	16,000	4,600,000	388.2
220	600,000	339.4	0.209	0.834	0.346	6,370,000	2,690,000	16,000	4,450,000	396.5
230	600,000	347.2	0.213	0.838	0.351	6,210,000	2,620,000	15,000	4,310,000	404.8
240	500,000	356.3	0.218	0.849	0.355	6,030,000	2,530,000	14,000	4,120,000	414.6
250	500,000	366.6	0.224	0.855	0.361	5,830,000	2,440,000	14,000	3,930,000	425.5
300	400,000	422.7	0.256	0.836	0.368	4,910,000	2,040,000	10,000	2,940,000	481.7
350	300,000	471.5	0.289	0.781	0.380	4,230,000	1,780,000	7,000	2,340,000	528.8
400	200,000	514.4	0.320	0.714	0.394	3,660,000	1,560,000	5,000	1,920,000	569.3
450	200,000	558.2	0.366	0.731	0.433	2,990,000	1,350,000	4,000	1,590,000	616.1
500	100,000	612.9	0.432	0.717	0.513	2,330,000	1,120,000	3,000	1,330,000	673.7
1000	10,000	1187.1	1.312	0.857	1.535	490,000	370,000	-	430,000	1302.6

Inferred mixed material:

Cutoff g AgEq/t	Tonnes	Silver (g Ag/t)	Lead (% Pb)	Gold (g Au/t)	Zinc (% Zn)	Silver (oz)	Lead (lbs)	Gold (oz)	Zinc (lbs)	AgEq (g/t)
75	600,000	155.7	0.131	1.862	0.208	2,790,000	1,610,000	33,000	2,560,000	260.9
100	500,000	171.6	0.141	2.070	0.220	2,660,000	1,500,000	32,000	2,340,000	288.2
120	400,000	183.3	0.148	2.236	0.228	2,550,000	1,410,000	31,000	2,180,000	308.9
140	400,000	195.8	0.154	2.405	0.236	2,430,000	1,310,000	30,000	2,010,000	330.6
150	400,000	202.2	0.158	2.496	0.241	2,360,000	1,260,000	29,000	1,930,000	341.8
160	300,000	207.3	0.159	2.574	0.242	2,310,000	1,220,000	29,000	1,850,000	351.1
170	300,000	214.0	0.159	2.675	0.243	2,240,000	1,140,000	28,000	1,750,000	363.1
180	300,000	222.5	0.160	2.750	0.246	2,180,000	1,080,000	27,000	1,660,000	375.5
190	300,000	228.0	0.161	2.830	0.248	2,130,000	1,030,000	26,000	1,590,000	385.4
200	300,000	234.6	0.164	2.886	0.252	2,090,000	1,000,000	26,000	1,530,000	394.9
210	300,000	240.0	0.167	2.956	0.255	2,030,000	970,000	25,000	1,480,000	404.2
220	300,000	245.6	0.170	3.022	0.258	1,990,000	940,000	24,000	1,430,000	413.3
230	200,000	253.1	0.174	3.086	0.264	1,930,000	910,000	24,000	1,380,000	424.4
240	200,000	260.4	0.177	3.123	0.267	1,900,000	880,000	23,000	1,330,000	433.8
250	200,000	267.2	0.180	3.200	0.271	1,840,000	850,000	22,000	1,280,000	444.8
300	200,000	299.7	0.195	3.620	0.288	1,560,000	690,000	19,000	1,020,000	499.9
350	100,000	335.4	0.215	4.226	0.305	1,250,000	550,000	16,000	780,000	568.1
400	100,000	375.1	0.236	4.648	0.324	1,060,000	460,000	13,000	630,000	630.7
450	70,000	415.1	0.247	5.018	0.332	910,000	370,000	11,000	500,000	690.4
500	50,000	465.0	0.263	5.340	0.331	780,000	300,000	9,000	380,000	757.5
1000	10,000	926.5	0.484	6.746	0.381	230,000	80,000	2,000	60,000	1299

Table 14-9: Escobal Deposit AgEq Resource Tabulation (continued)

Inferred sulfide material:

Cutoff g AgEq/t	Tonnes	Silver (g Ag/t)	Lead (% Pb)	Gold (g Au/t)	Zinc (% Zn)	Silver (oz)	Lead (lbs)	Gold (oz)	Zinc (lbs)	AgEq (g/t)
75	7,200,000	162.2	0.304	0.216	0.569	37,290,000	47,910,000	50,000	89,660,000	195
100	5,200,000	197.4	0.343	0.258	0.663	33,070,000	39,380,000	43,000	76,210,000	235.7
120	4,300,000	219.3	0.364	0.287	0.715	30,640,000	34,910,000	40,000	68,460,000	260.8
140	3,700,000	240.0	0.387	0.316	0.762	28,350,000	31,360,000	37,000	61,720,000	284.8
150	3,400,000	250.8	0.400	0.332	0.788	27,180,000	29,730,000	36,000	58,550,000	297.4
160	3,100,000	261.8	0.413	0.348	0.813	26,040,000	28,170,000	35,000	55,450,000	310.2
170	2,900,000	271.9	0.424	0.362	0.836	25,040,000	26,750,000	33,000	52,780,000	321.8
180	2,700,000	282.1	0.436	0.377	0.861	24,030,000	25,440,000	32,000	50,260,000	333.7
190	2,500,000	291.8	0.446	0.392	0.885	23,110,000	24,220,000	31,000	48,070,000	345
200	2,300,000	302.5	0.457	0.406	0.910	22,140,000	22,930,000	30,000	45,680,000	357.3
210	2,100,000	312.0	0.467	0.419	0.931	21,310,000	21,850,000	29,000	43,580,000	368.2
220	2,000,000	321.6	0.474	0.431	0.945	20,520,000	20,720,000	28,000	41,360,000	379
230	1,800,000	333.0	0.488	0.450	0.976	19,530,000	19,610,000	26,000	39,260,000	392.5
240	1,700,000	342.9	0.495	0.464	0.997	18,760,000	18,580,000	25,000	37,410,000	403.9
250	1,600,000	352.2	0.503	0.477	1.020	18,050,000	17,680,000	24,000	35,850,000	414.6
300	1,200,000	398.5	0.552	0.546	1.152	14,750,000	14,000,000	20,000	29,220,000	468.9
350	900,000	442.4	0.590	0.613	1.266	12,110,000	11,070,000	17,000	23,760,000	519.9
400	700,000	476.4	0.616	0.665	1.332	10,290,000	9,120,000	14,000	19,720,000	558.9
450	500,000	520.2	0.595	0.738	1.292	8,360,000	6,560,000	12,000	14,240,000	605.1
500	400,000	568.4	0.572	0.817	1.228	6,630,000	4,580,000	10,000	9,830,000	655.3
1000	-	1263.2	1.185	0.675	1.054	340,000	220,000	-	190,000	1354.3

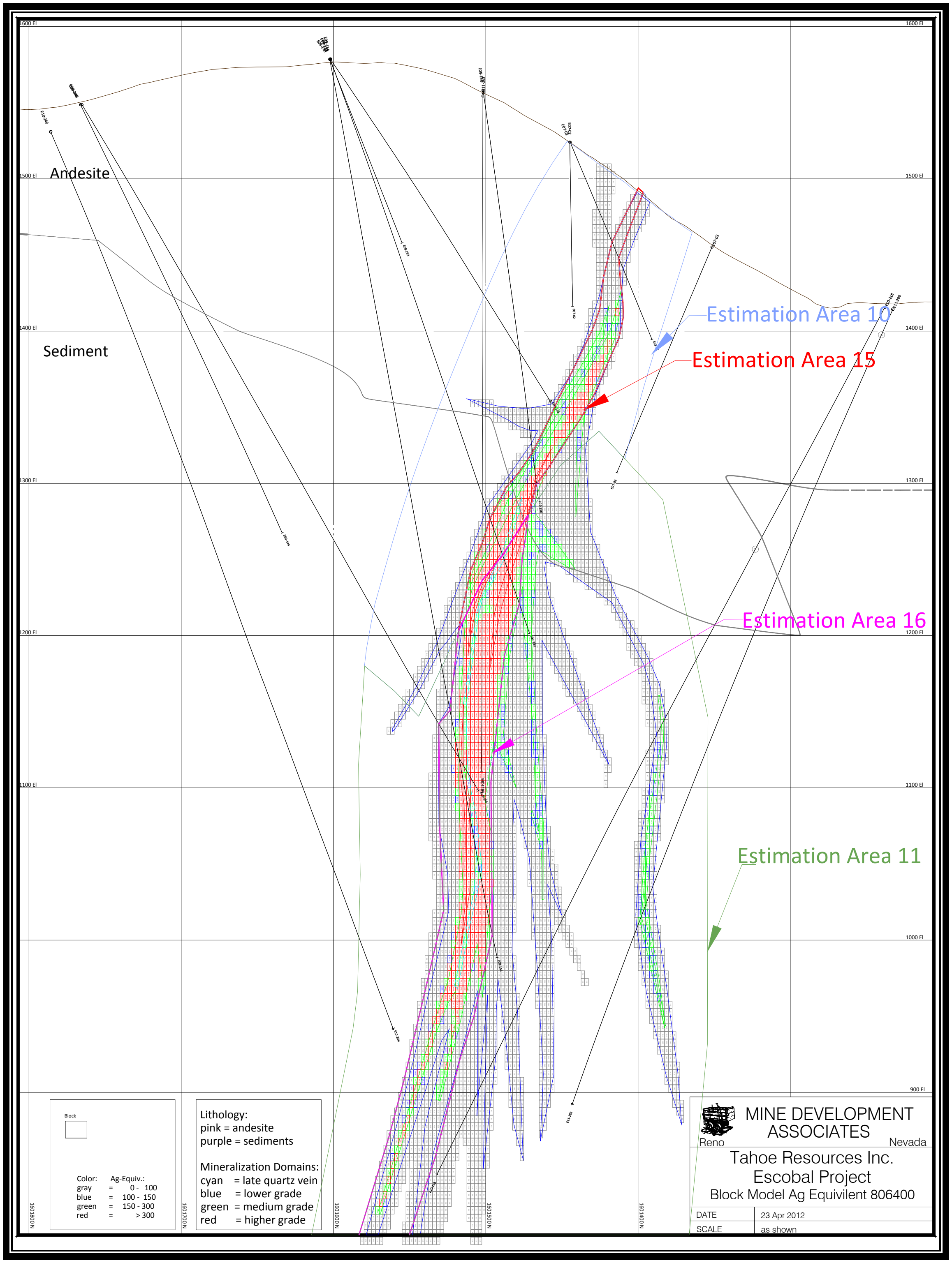


Figure 14-4: Section 806400 – Escobal Central Zone Block Model: AgEq Block Grades

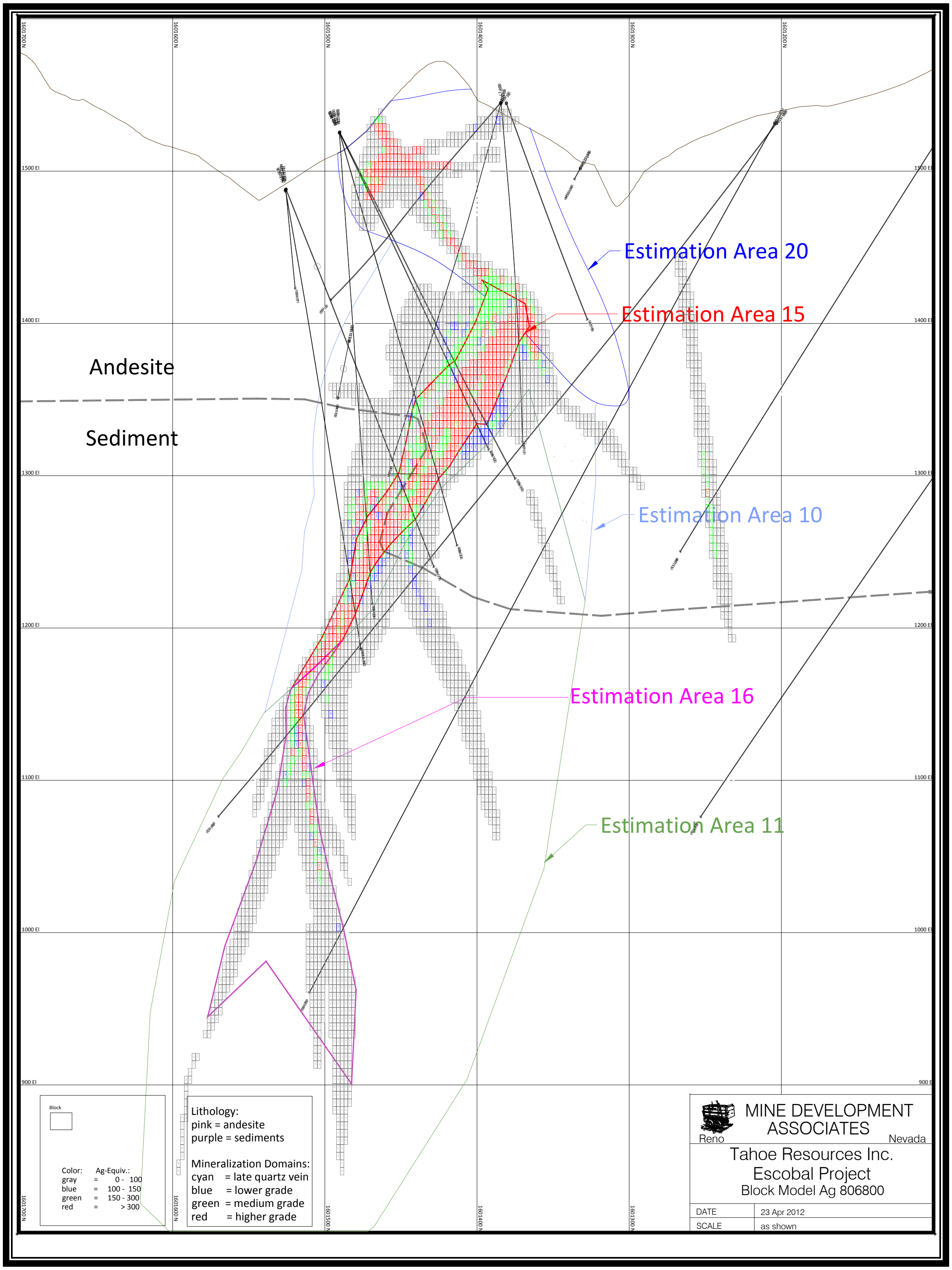


Figure 14-5: Section 806800 – Escobal Central Zone Block Model: AgEq Block Grades

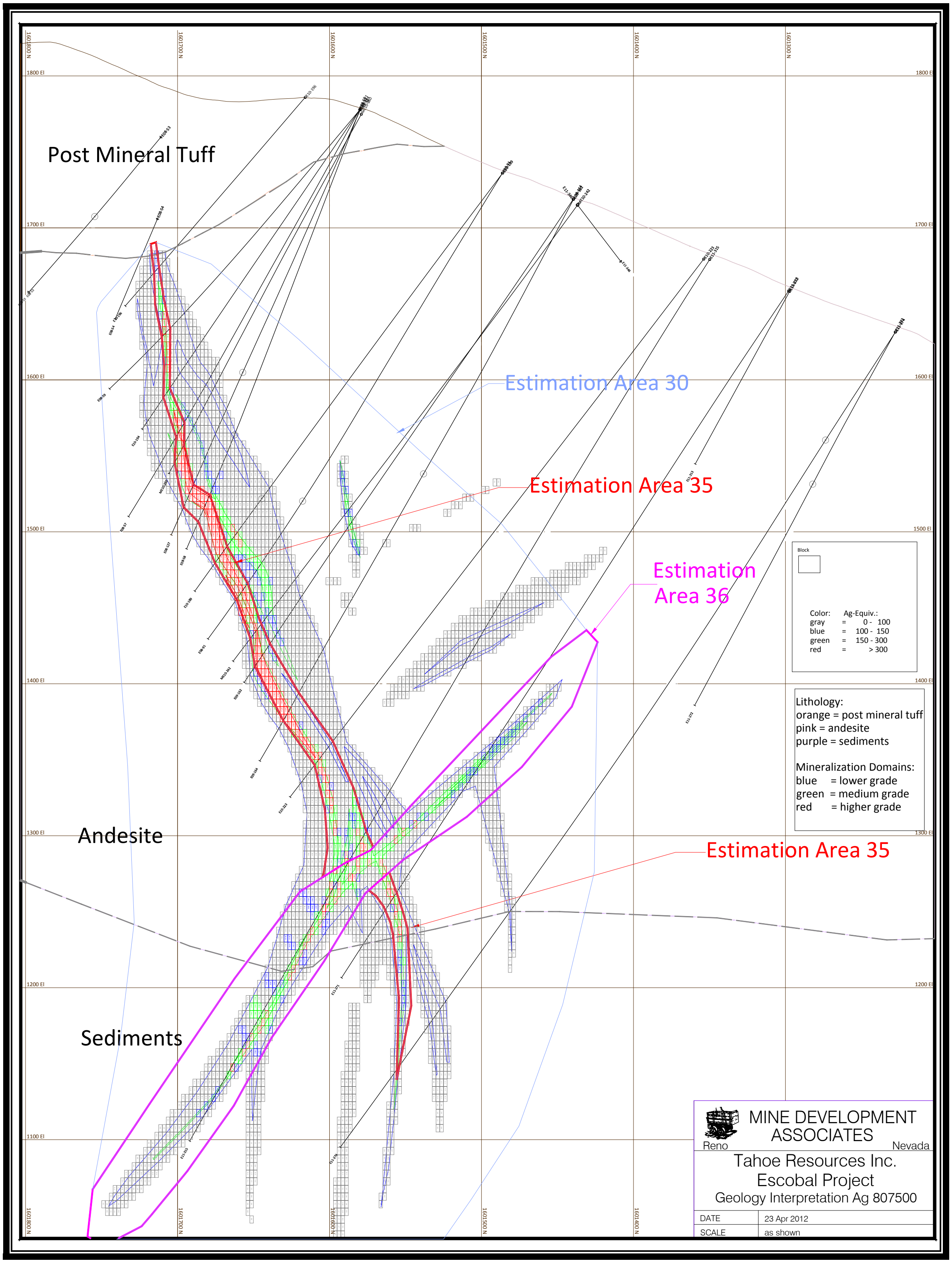


Figure 14-6: Section 807500 – Escobal East Zone Block Model: AgEq Block Grades

Checks were made on the Escobal resource model in the following manner:

1. Cross sections with the mineral domains, drill-hole assays and geology, topography, sample coding, and block grades with classification were reviewed for reasonableness;
2. Block-model information, such as coding, number of samples, and classification were checked visually by domain and lithology on sections and long-sections;
3. Cross-section mineral domain volumes to level plan mineral domain volumes, to block model mineral domain volumes were checked;
4. Nearest-neighbor and indicator kriging models were made for comparison;
5. Sectional polygonal models were calculated from the original modeled section domains; and
6. Quantile-quantile plots of assays, composites, and block-model grades were made to evaluate differences in distributions of metals throughout all domains and areas.

The resource estimate is considered reasonable, honors the geology, and is supported by the geologic model.

14.11 DISCUSSION, QUALIFICATIONS, RISK, AND RECOMMENDATIONS

The Escobal deposit's Central Zone hosts laterally continuous mineralization over a 1,200m strike and up to 900m down-dip. The East Zone mineralization is up to 850m along strike and 800m down-dip. Sulfide mineralization is dominant, with silver, lead, and zinc occurring in potentially economic grades throughout most of the sulfide mineralization. Gold distribution is more erratic within the sulfide mineralization but can be high grade (>10g Au/t) within the oxidized portions of the East Zone where gold is the dominant metal.

The Escobal resource estimate is based on sufficient drill-sample analytical and density measurements, detailed drill-hole lithology and alteration data, and preliminary metallurgical results, to support a classification of Indicated for much of the sulfide mineralization. The lack of metallurgical testing on the oxide material and some spatial uncertainty in the model have resulted in an Inferred classification for all of the oxide portions of the deposit.

Additional drilling to better characterize the gold mineralization within the upper levels of the East Zone is recommended where economic viability is dominated by gold. Both the Central and East zones are open at depth, and further extensional drilling is recommended.

15 MINERAL RESERVE ESTIMATES

There are no mineral reserves reported for the Escobal project.

16 MINING METHODS

16.1 CURRENT STATUS

Underground development commenced in May, 2011, with construction of the East Central and West Central decline portals; after which ramp development began. Figure 16-3, Figure 16-4, and Figure 16-5 are plan maps of the development advance as of April 1, 2012. Development is scheduled to reach the 1190 meter elevation where initial production is expected to commence in the second half of 2013. Tahoe intends to operate Escobal with a minimum of expatriates which requires training Guatemalans in all aspects of the mining processes. Tahoe is committed to accomplishing this with a focus on safety, preventing accidents, and achieving production goals with safety performance at or above the performance of the top North American mining operations. Achieving our training and safety performance goals coupled with the persistence of weak soils and blocky ground to depths in excess of those anticipated in the initial analysis have resulted in slower than anticipated initial ramp advance and higher development costs than anticipated in the November 2010 PEA. However, advance rates have been within the contingency allowance for this phase of the project. Methods of development advance are expected to continue as anticipated in the November 2010 PEA and overall development and equipment costs for the 3,500 tonne per day case will be within the budgeted levels and be completed as scheduled in the November 2010 PEA.

Figure 16-1: East Central Decline Portal Area

Figure 16-2: West Central Decline Portal Area

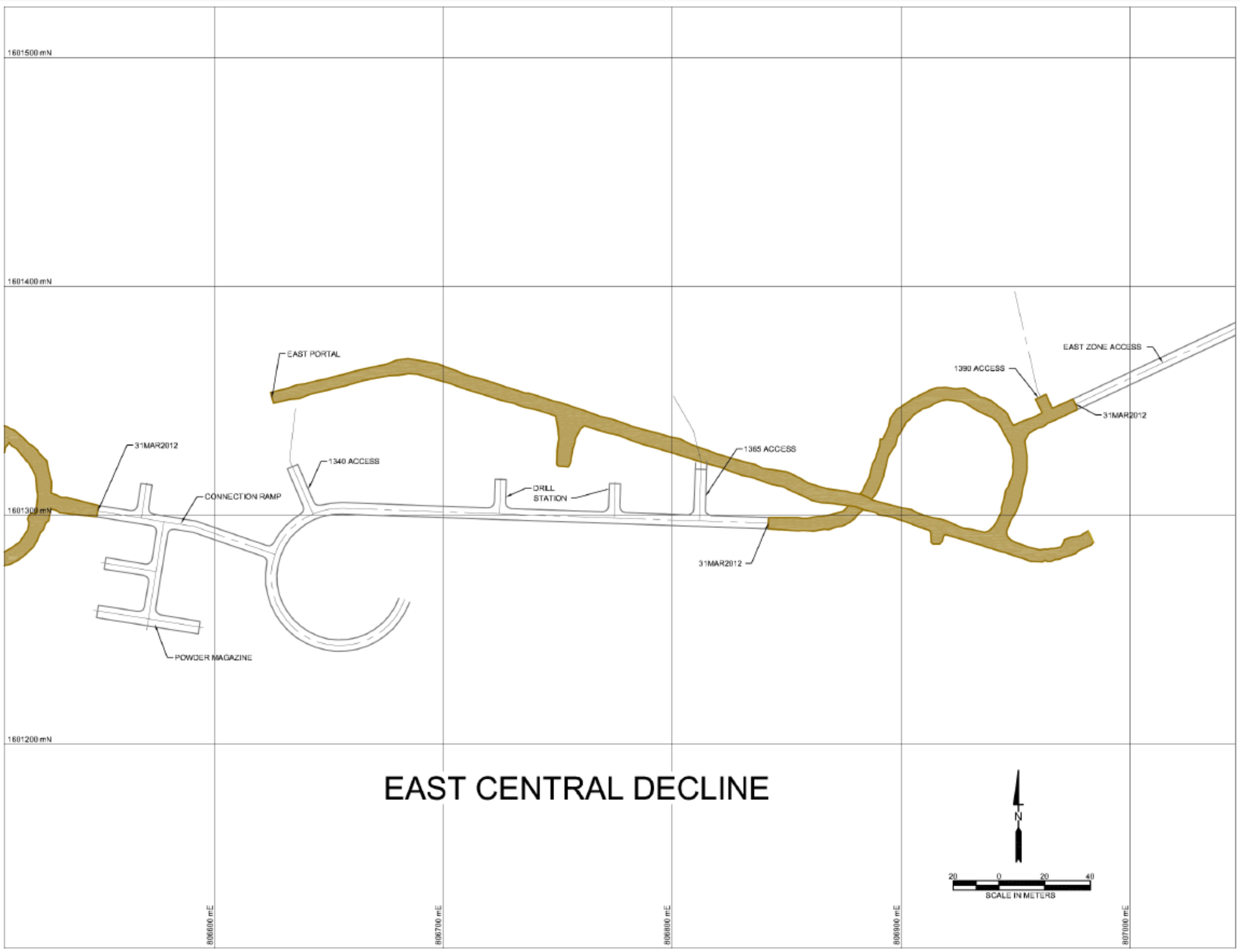


Figure 16-3: East Central Ramp

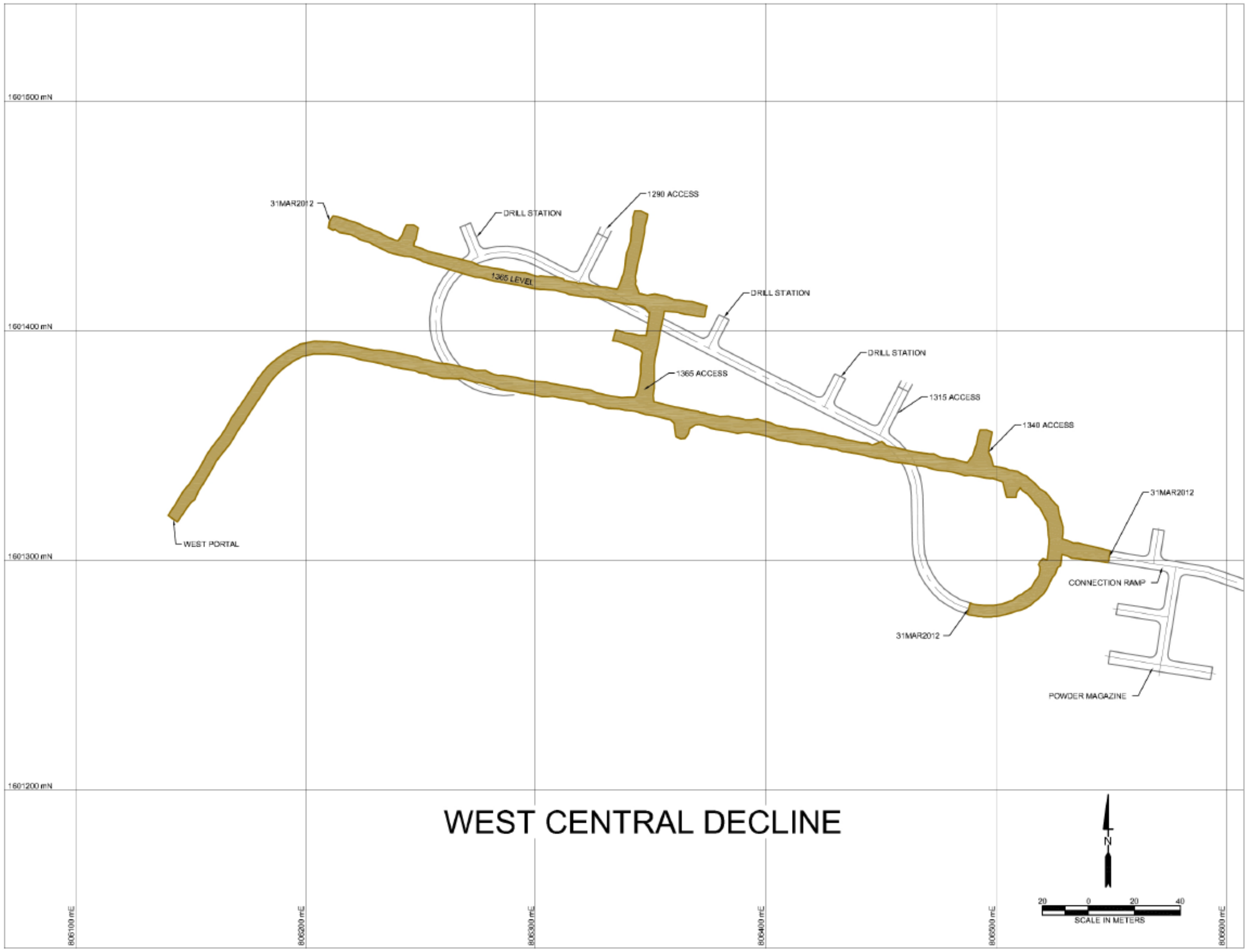


Figure 16-4: West Central Ramp

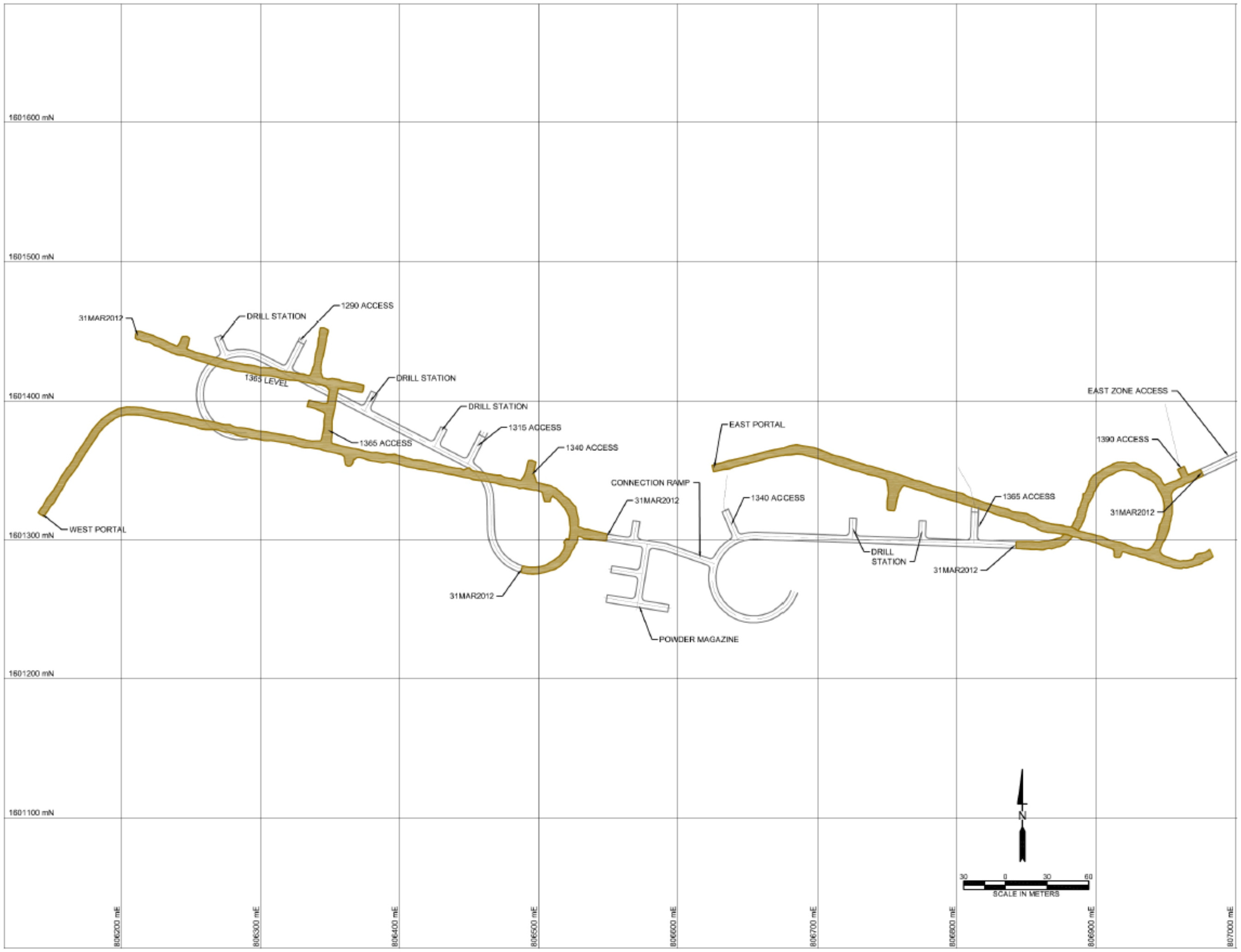


Figure 16-5: East and West Central Ramp

Exploration success since completion of the November 2010 PEA has added significant high quality resources to the Escobal inventory. The additions have prompted the need to evaluate the ability for expansion of the mine and process plant production capacity. With the additional resources the mine clearly has excess annual production capacity beyond that contemplated in the original PEA. In order to take advantage of that capacity, transverse long-hole stoping will replace longitudinal long-hole stoping in areas where the horizontal dimensions across the strike of the vein are greater than 15 meters. This will allow an increase in the number of active producing workplaces in the mine at any given time.

Approximately 15,000 meters of additional primary development ramps and 1,100 meters of raise will be required to access and develop the new resources. Primary development has been accelerated compared to the previous plan in order to access the new resources and allow increased production. This will be accomplished through the addition of development crews and equipment rather than an increase in productivity. Footwall laterals in waste will be used to access stoping areas in lieu of the individual spiral ramps that were utilized in the earlier study for stope development. Two cases have been analyzed in this study; increasing mine production to 4,500 tonnes per day and capping it at that rate throughout the mine life, or increasing mine production to 4,500 tonnes per day, making major modifications to the process plant and then further increasing mine production to 5,500 tonnes per day for the remainder of the mine life. The mine plan for the two cases only differ in the timing of development and a slightly smaller equipment fleet for the 4,500 tonne per day case.

16.2 LONG HOLE MINING

Production from the Escobal resource will be extracted by long-hole stoping methods. Two variations of this mining method will be utilized. Where the vein dimension across the strike is less than 15 meters, longitudinal long-hole stoping will be applied. This method consists of driving horizontal drifts on two different sublevels along the strike of the vein and then blasting the mineralized material vertically from the upper level or over-cut to the lower level or under-cut. As the vein is mined along strike, the stability of the stope walls decreases. This stability decrease is related to many factors but the key factors are rock strength and vein dip. As the size of the excavated opening approaches the point that instability in the stope walls begins to result in spalling or surface failure, mining must cease and the void is backfilled and the process continues. This instability point, the point where the maximum design hydraulic radius is reached, has been calculated for each individual stope utilizing the large geotechnical data base collected during exploration. In order to maximize the stability of the hanging wall along strike, the hanging wall will be cable bolted on 1.2 meter centers and a minimum of 8 meters into the hanging wall.

Productivities were calculated for each stope based on a stoping life cycle simulation. The time required for on-vein development of over- and under-cuts, followed by long-hole mining to the maximum safe strike length, preparation time for backfilling, backfilling and re-entry were estimated and productivities were then calculated. Productivities developed with these criteria were then reduced by 15% and used as the maximum productivity for the stope. In most cases other logistical factors or development requirements placed more restrictive productivity limits on individual stopes.

Stopes will be spaced 25 m vertically. Breaking slots will be established at the extreme ends of the stope to provide a void space for production blasting. Breaking slots will be excavated utilizing Cubex drills equipped with V-30 blind bore reaming heads to bore a 30-inch diameter raise between the upper-cut and under-cut for each stope. Once the breaking slots are complete, the stopes will advance towards the accesses by drilling holes between the over-cut and under-cut, charging the holes with ANFO or emulsion explosives, and blasting a ring or row of holes at the end of the stope. The broken material blasted from the end of the stope will be excavated from the under-cut with Caterpillar R1700 or R2900 load-haul-dump (LHD) machines equipped for remote operations. The material will be loaded into AD45 Caterpillar trucks and transported to the process plant. This process continues until the maximum hydraulic radius or design limit of the opening along strike is reached, at which time long-hole mining ceases and the void is filled with paste backfill. Once the stope is backfilled and the fill cured, a new breaking slot will be required to continue long-hole mining in the stope. This process continues until the entire strike length is mined and filled. Excavation lengths along strike prior to backfilling will vary depending on the Rock Mass Rating (RMR) of the hanging wall. In areas where the RMR of the vein will not allow excavation of the entire width of the vein in one pass, two or more panels will be utilized across the dip to complete excavation of the entire vein. Mining can progress vertically once mining has been completed on the level below and the stope has been backfilled and the fill allowed sufficient curing time. If mining has already taken place below, the stope can be filled with lower strength fill and or waste rock.

Figure 16-6 is an isometric view of the longitudinal long-hole stoping method.

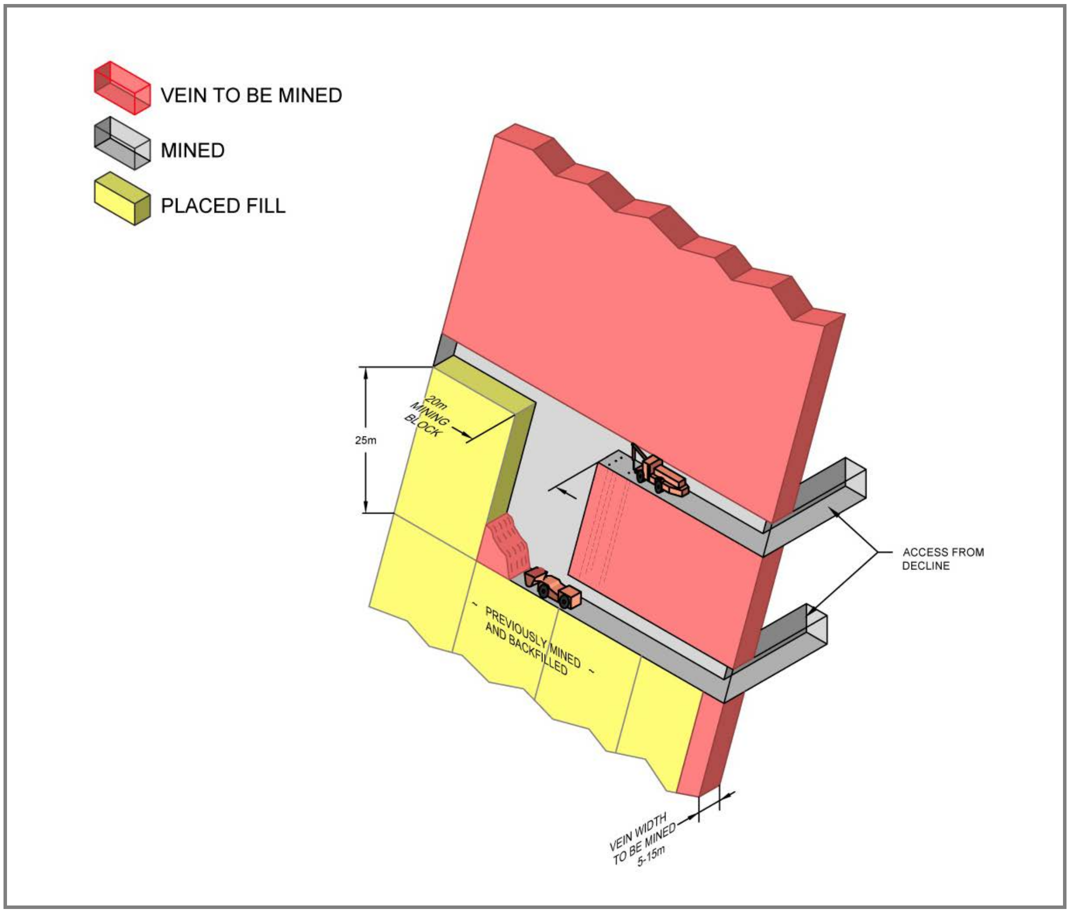


Figure 16-6: Longitudinal Long-hole Stoping Method (isometric view)

In areas where the vein width, the dimension perpendicular to the strike of the vein, exceeds 15 meters stopes will be developed perpendicular to the strike of the vein. This is commonly known as transverse long-hole stoping. In this case, 5 meter wide by 5 meter high footwall laterals will be developed approximately 20 meters to the south of and parallel to the vein. Access to the footwall laterals will be from the primary declines. Five (5) meter wide by 5 meter high over-cut and under-cut drifts will be developed from the south side of the vein to the north side of the vein spaced 25 meters vertically. Generally this will be the same as developing from the foot wall to the hanging wall of the vein but due to local change in dip, will occasionally be from hanging wall to footwall. Once the cross-cuts reach the hanging wall, a ”T” drift along the hanging wall will be excavated to a total stope width along strike of 20 meters. The hanging wall will be cable bolted on 1.2 meter centers a minimum of 8 meters into the hanging wall from this drift.

Breaking slots will be established on one end of the hanging wall drift of the stope to provide a void space for production blasting. Breaking slots will be excavated utilizing Cubex drills equipped with V-30 blind bore reaming heads to bore a 30 inch diameter raise between the over-cut and under-cut for each stope. Once the breaking slots are complete, the stopes will advance towards the accesses by drilling holes between the over-cut and under-cut in a ring pattern, charging the holes with ANFO or emulsion explosives, and blasting a ring or row of holes at the end of the stope. The broken material blasted from the end of the stope will be removed from the under-cut with Caterpillar R1700 or R2900 LHDs equipped for remote operations. The material will be loaded into an AD45 Caterpillar trucks and transport to the process plant. This process continues until all of the material between the hanging wall and the footwall has been excavated at which time hole mining ceases and the void is filled with paste backfill.

The transverse mining method allows from multiple stopes to be in production along strike simultaneously on any given sublevel. Stopes along strike will be split into primary stopes and secondary stopes. Each primary stope will be separated along strike by a secondary stope. This allows for a rock pillar to be maintained between the primary stopes while these stopes are being excavated increasing the overall stability of the stopes. Once two primary stopes are excavated, backfilled and the backfill is allowed to cure, the secondary stope between them can be excavated and subsequently filled with either lower strength fill or waste rock or a combination. Mining will progress from the lower level to the next level above as the stopes on the lower level are mined and backfilled. The over-cut from the lower level will become the undercut or mucking level for the next level above. Figure 16-7 is an isometric view of the transverse long-hole mining method and Figure 16-8 and Figure 16-9 are plan maps of the 1190 meter and the 1215 meter levels which are planned to be the initial over-cut and under-cut levels and depict plan views of both mining methods and the development access to the stopes.

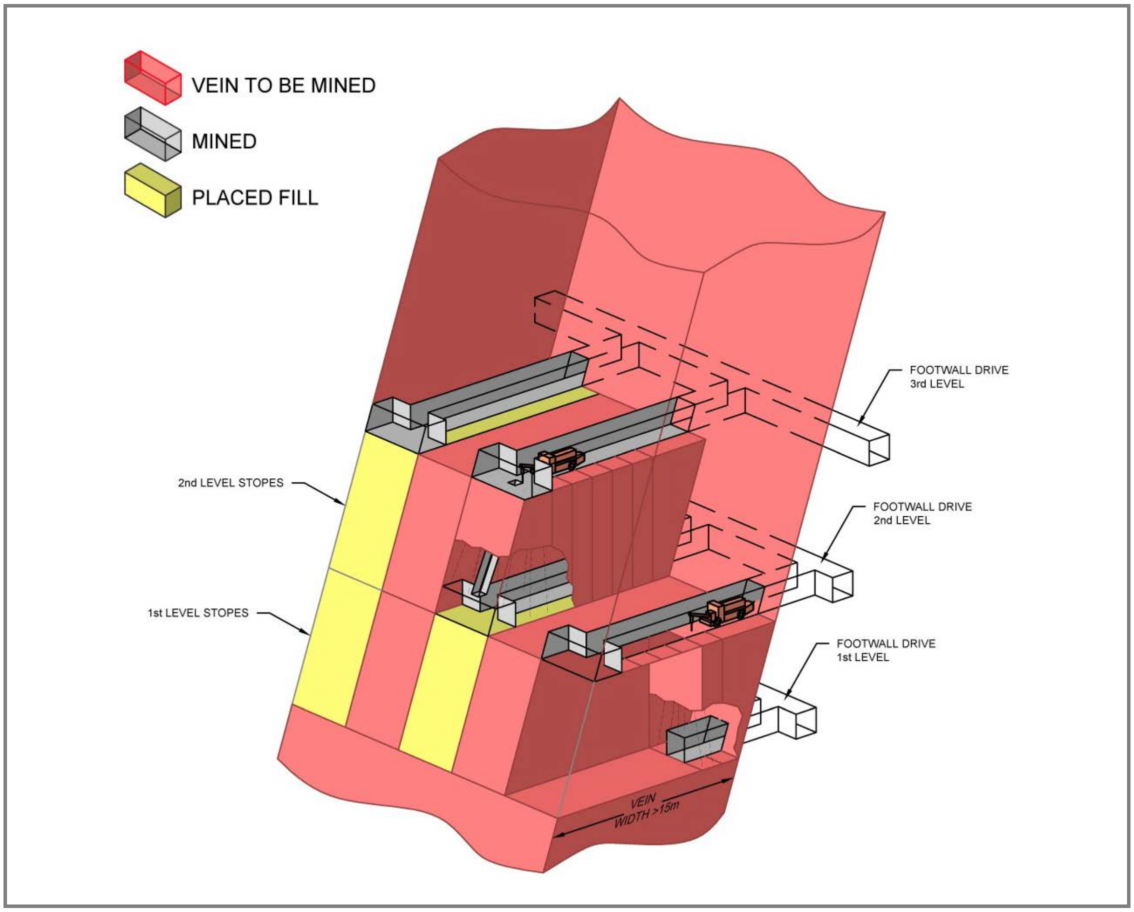


Figure 16-7: Transverse Long-hole Stoping Method (isometric view)

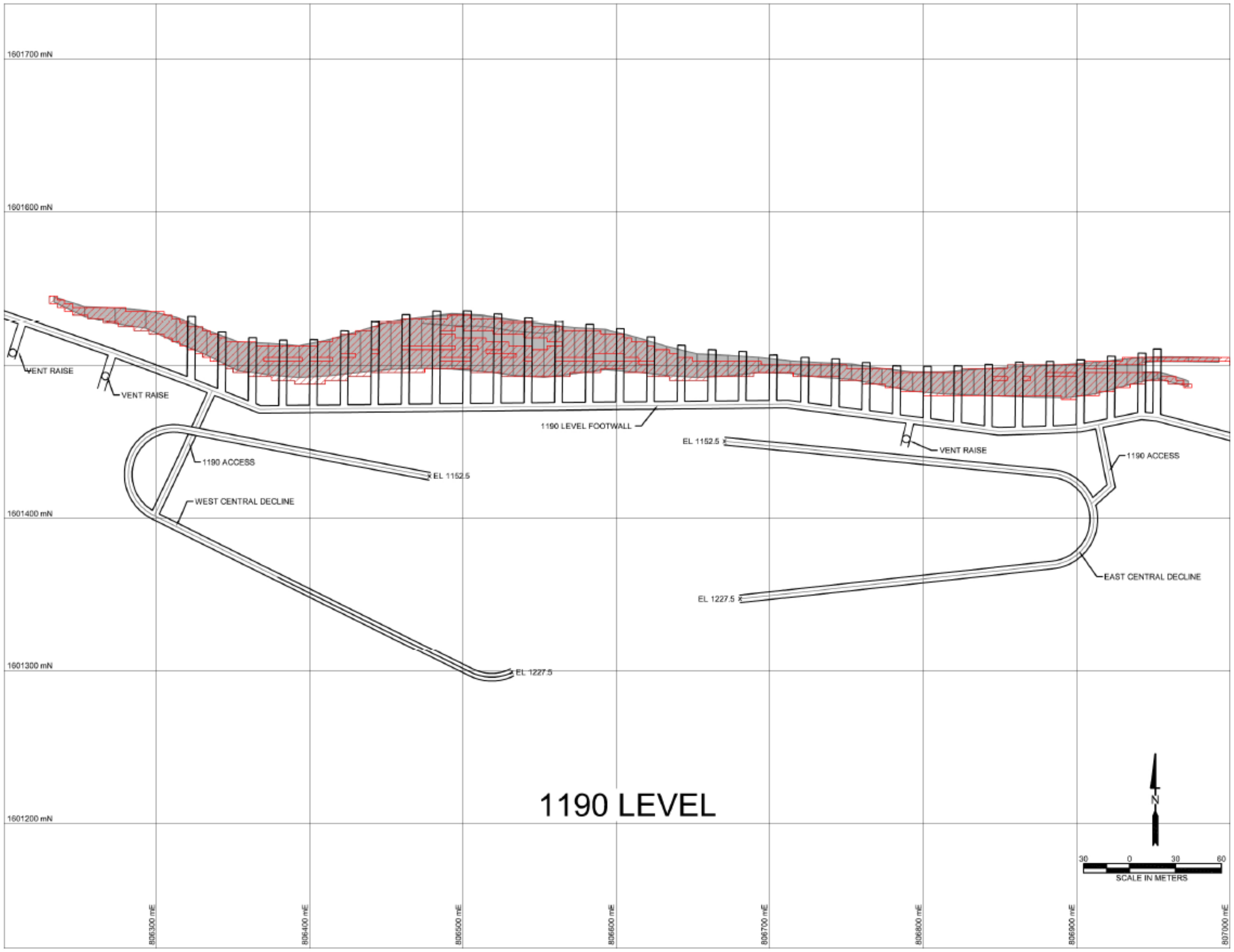


Figure 16-8: 1190 Meter Level

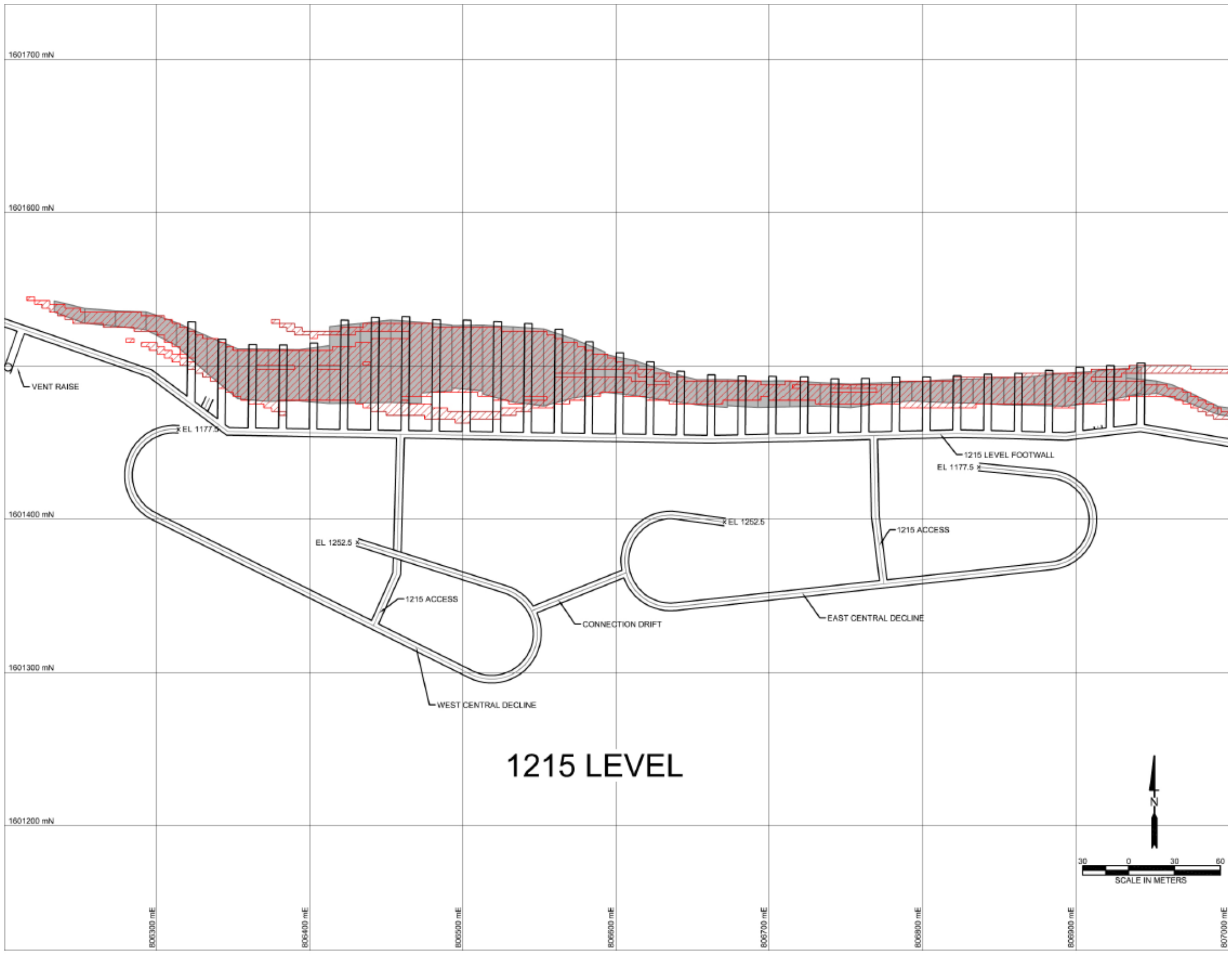


Figure 16-9: 1215 Meter Level

16.3 PASTE BACKFILL

The proposed mining methods at Escobal are transverse and longitudinal long-hole stoping. Both will require use of cemented backfill as an integral part of the mining cycle. The mined stope voids will require backfill to ensure stability before mining adjacent pillars.

Transverse stoping will be used in wider parts of the deposit. Typical stope dimensions are 20 meters in width measured along strike, 25 meters high and mined to the full width of the vein which is expected to reach 35 to, locally, 50 meters. A mining width of 15 to 35 meters is expected to be typical. Mining will take place in a series of primary and secondary stopes. The primary stopes will have two exposures of rock walls along the vein strike direction, both of which will be the width of the stope and the secondary stopes will have two exposures of paste backfill, both of which will be the width of the vein. At the base of each production panel, high strength fill will be placed to enable removal of the underlying sill pillar.

Longitudinal stoping will be used in narrower parts of the deposit up to 15 meters wide. Typical stope dimensions are 20 meters in length measured along strike, 25 meters high and mined the full width of the vein. A mining width of two to 16 meters was selected for longitudinal stoping and the longitudinal stopes will have one wall of backfill exposure, the width of the mined opening. In addition to vertical exposures, some longitudinal stopes will be undercut in sill pillars requiring increased fill strength.

The mining strategy at Escobal is aimed at 100% extraction of material above the cut-off grade. Cemented paste backfill will be critical to maximizing recovery of this material. Fill rate requirements have been calculated for mining rates ranging from 3,500 tonnes per day to 5,500 tonnes per day and the paste plant has been designed to achieve fill rates consistent with mine production of 5,000 tonnes per day with expansion capabilities beyond that.

The transverse stope mining schedule requires the vertical paste fill exposures in the primary stopes to achieve the target strength after 28 days of curing. The secondary stopes will not be exposed and therefore only require a minimum strength of 200 kPa to be achieved after 56 days. The longitudinal stope mining schedule requires the vertical paste fill exposures of the end walls to achieve the target strength after 14 days of curing. The table below presents the past fill mix designs for all the vertical exposures detailed within the mine plan.

Table 16-1: Paste Fill Mix Designs for Vertical Exposures

Stope Type	Number of Exposures	Width (m)	UCS Strength (kPa)	Cement % W/W	Cement kg/m^3	Curing Time
Transverse Primary	2	15≥26	300	6.1%	84	28 days
		26≥60	400	7.2%	99	
Transverse Secondary	0	n/a	200	3.9%	54	56 days
Longitudinal	1	2≥16	200	5.4%	54	14 days
		16≥25	300	7.0%	96	

Both transverse and longitudinal stope mining methods will require paste fill to be undercut at some stage within the mining schedule. The undercut exposures will have a long curing time so the 56 day results have been used to design these mixes. The paste fill mix designs required for the potential range of undercut widths within the mine plan range from a required strength of 700 kPa and 7.2% cement for undercuts of 2 meter widths to 900 kPa and 10.3% cement. At this time the plans limits undercut excavations to a maximum of 10 meters wide. Additional testing will be required for wider excavations which undercut fill.

Pumping will be required to deliver paste to the East Zone mining areas where stopes will be mined up to 85 meters above the paste plant elevation. A positive displacement pump rated at 60 Bar continuous and 100 Bar peak pumping through 200 mm diameter lines will be required for this area. For stopes in the Central Zone, paste backfill will be gravity delivered from the feed hopper to one of two surface 150 mm diameter cased boreholes. Both boreholes connect from surface to the various reticulation horizons on each sublevel.

16.4 DEVELOPMENT

The primary development ramps are designed at 5 meters wide by 6 meters high and will typically be driven at a maximum incline of 15%. The primary development ramps are sized to accommodate ventilation ducting which will allow ramp excavation up to 2,000 meters in length without the use of other ventilation systems. Secondary development heading which will allow access to the individual stopes are designed 5 meters wide by 5 meters high as these headings do not need to accommodate the large ventilation tubes included in the primary development design. Ventilation raises and ore passes are strategically located throughout the mine and included in the pre-production development schedule.

Table 16-2 shows the development required to ready the mine for production for each of the three production cases.

Table 16-2: Primary and Secondary Development – 3 production cases

	2012	2013	2014	2015	2016	2017	2018
3500 MTPD Meters Drift Development	4754	3404	1896	1896	1896	1896	1896
3500 MTPD Meters Raise Development	280	330	350	200	150	50	50
4500 MTPD Meters Drift Development	5989	5795	4614	5120	4630	4637	2162
4500 MTPD Meters Raise Development	365	218	0	60	510	550	200
5500 MTPD Meters Drift Development	5989	5795	4364	5020	4480	4737	2512
5500 MTPD Meters Raise Development	365	218	0	60	375	135	350

Annual and total development requirements for the 4,500 MTPD and 5,500 MTPD cases in excess of that required for the 3,500 MTPD case are composed of accelerated development included in the 3,500 MTPD case required to increase production in the early years and the additional development require to produce from the new resource areas.

Development waste that is not placed in the mined stopes as backfill will be trucked to surface for use in facilities construction or placed in a development rock storage facility. Modified Acid/Base Accounting (ABA) tests, acid generation potential/acid neutralization potential (AGP/ANP) tests, and long-duration kinetic tests (humidity cell tests) performed on samples of the waste rock from various zones demonstrate that the waste rock encountered in the development headings has a high net neutralizing potential and is unlikely to generate acid. Humidity cell effluent analyses and Meteoric Mobility tests demonstrate the mobility of metals contained in the development rock to be within regulatory limits. All of these test results indicate that the waste rock is unlikely to emit metals and is likely to neutralize meteoric waters that may come in contact with the rock. The results of these tests are the design basis for waste storage and handling programs. A program to continue testing development rock as it is being mined is being utilized to ensure that in the unlikely event that rock with the potential for liberating metals or generating acid is encountered it is properly identified and will be mixed with cemented fill and place underground or encapsulated in the dry stacked tails facility. In either case the material will be isolated from contact with water and oxygen to insure the rock does not generate acid or allow metals to be released into the environment.

16.5 GEOTECHNICAL CONSIDERATIONS

The geotechnical data collected by the exploration team in Guatemala is sufficient to utilize for mining method selection, opening size design, ground support design, and productivity estimates. Geotechnical data collected from the drill core includes core recovery, hardness, rock quality designation (RQD), joint number (Jn), joint roughness (Jr), joint alteration (Ja), joint water reduction factor (Jw), and the stress reduction factor (SRF). From this data, the tunneling quality index (Q rating) can be calculated to identify the rock quality to be anticipated during underground excavation. The rock mass rating (RMR) values for the Escobal vein, immediate hanging wall, and immediate footwall were determined from the Q rating by the formula $RMR = (9 \times \ln Q) + 44$. The equivalent RMR data is summarized in Table 16-3.

Table 16-3: Escobal RMR Values

Area	Location	Average	Median	Minimum	Maximum
East Zone	Vein	56	62	17	87
	Hanging Wall	52	57	7	85
	Footwall	50	47	12	89
Central Zone	Vein	61	64	0	80
	Hanging Wall	54	56	9	85
	Footwall	54	57	17	90

RMR data were plotted for the immediate hanging wall, footwall, and vein over the entire resource area. The data was contoured and overlaid on the stope outlines in long section. Depending on the width of the stope and the variability of the RMR, an average RMR was selected for each stope. In general, the vein material demonstrates a higher RMR than either wall of most stopes. In narrow stopes where a single panel will be adequate to mine the entire width of the vein, a design RMR was selected utilizing the hanging wall data. Vein dip and hanging wall rock strength are the most influential factors in determining the hydraulic radius for the opening. In wider stopes where multiple panels are required to extract the vein and the progression of extraction will be from footwall to hanging wall, the design RMR was typically selected from the vein data. In both cases locally weak areas were taken into consideration for the design.

Once a design RMR was selected for each stope, the RMR was used to calculate a maximum hydraulic radius and from that a maximum opening length along strike for each 5 meter wide by 25 meter high long hole panel. The maximum opening length then was utilized as the maximum length along strike that the stope could be mined before extraction is stopped and backfilling commenced. The design assumes that the over-cut and under-cut are fully supported with rock bolts but that no support is applied to the walls between. A simulation of the entire cycle time for drilling, blasting, mucking, preparation for and backfilling and re-establishing a breaking slot was performed to estimate the productivity of each stope based on the maximum opening length limitation. This productivity was used as a limiting factor in the stope production schedule.

Ground support in development headings will consist of 2.5 meter split set, grouted rebar, and swellex bolts depending on opening size and local ground conditions. The majority of the standard waste development can be properly supported with split set and grouted rebar bolts.

Shotcrete and longer bolts will be utilized where local ground conditions such as faulting warrant the additional support.

Over-cut and under-cut headings will primarily be support with 2.5 meter split set bolts up to a span of 5 meters wide and 5 meters high in typical vein conditions. Additional and longer bolts will be required with spans up to 10 meters wide. Where the vein is wider the stope will be mined in panels so as to maintain safe opening sizes and minimize dilution.

The capital and operating costs as well as the productivities and schedule have allowances for cable bolting where stope hanging walls are unusually weak or unusually large excavations are required for specific installations.

Preliminary analysis of the geomechanical data for stope design was conducted in 2010 by Dr. Rimas Pakalnis of Pakalnis and Associates. Dr. Pakalnis revisited the Project in August 2011 to provide recommendations for ground support and control procedures in the East Central and West Central declines. Dr. Pakalnis' analysis supports the methodology used in the mine design and development ground control programs. Dr. Pakalnis' report can be found in Appendix D.

16.6 MINE VENTILATION

The mining method selected for the Escobal deposit is highly mechanized utilizing a fleet of diesel powered equipment. There are no known natural contaminants such as radon or carbon monoxide and the mine is not in an unusual heat environment. The ventilation required to safely manage contaminants and heat introduced by the diesel equipment is therefore the governing factor in the ventilation design. U.S. Mine Safety and Health Administration guidelines are utilized as the standard in the Escobal ventilation design.

Ventilation modeling was conducted with standard methods in spreadsheets. Computerized modeling utilizing VNET PC or equivalent ventilation modeling software will be utilized for final design. Shock losses were considered using the equivalent length method. Standard, conservative K factors and nominal airway dimensions were used in the modeling, as no resistance data were available. K factors in the critical airways, specifically the boreholes to surface, were modeled over a range of values to ensure that unexpectedly high values would not create critical flow conditions.

Initially the mine will be developed via two declines into the Central Zone. For the first phase of primary development, the West Central and East Central declines will be driven 1,750 meters and 1,950 meters down slope, respectively, with drifts connecting the two declines on about 100 meter vertical spacing. The ventilation circuit for each decline will consist of two 48" (1.22 meter) diameter steel ducts installed in a nominal 5 meter wide by 6 meter high ramp. Each circuit will operate at 50,000 cfm (23.6 m^3/sec) at a static pressure of 9.6 mm Hg at an elevation of 1,420 meters elevation. Two 75kW, 100hp fans will be installed at each decline to provide the required air flow during development. Ventilation requirements will be reduced for the primary development heading once the two declines are connected. Once the declines are connected and prior to completion of the primary ventilation raise from surface the west decline will be converted to exhaust and ventilation will flow into the east decline and exhaust out the west decline. A single 75kW auxiliary fan and a single 1.22 meter duct will be required in the primary ramp into the East Zone and below the 1,130 meter elevation in the West Central decline and the 1,197 meter elevation in the East Central decline. Once the main exhaust bore holes are excavated, both declines will become intake airways and all mine air will exhaust through the main bore hole to the surface between the east and west declines and out the bore hole in the East Zone.

The primary production ventilation network will be placed into operation after the main exhaust bore holes in the Central Zone are excavated to the elevation of the initial production level undercut at the 1,190 meter elevation. An exhaust raise will be excavated on each end of the footwall laterals and daylight at the surface. A vane axial ventilation fan will be installed at the top of the borehole. Intake air will be drawn down the two declines, enter the active footwall laterals and drawn across the laterals to the exhaust boreholes where it will be drawn up to the surface through the bore hole. Fresh air will be directed from the declines to the production and development headings on the footwall laterals through a series air control doors. Air will be circulated from the footwall laterals through the active headings and returned to the footwall laterals to be exhausted to the surface.

As the mine is further developed, the bore holes in the Central Zone will be extended to the bottom of the mine and air will be drawn in a similar fashion to the bottom of the mine and exhausted through the bore holes. In addition, once the East Zone ramp reaches the 1,450 meter elevation, a bore hole from the surface will be excavated to connect the East Zone ramp with the surface and a vane axial fan will be installed at the top of this raise. Air will then be split from the East Central decline and funneled into the East Zone for ventilation of the production stopes in the East Zone. The ventilation scheme envisioned for the Central Zone will be sufficient to ventilate the additional resources in the lower west portion of the Central Zone. The additional resources in the lower East Zone Extension area will require a series of exhaust bore holes on each of the east and west ends of the strike length and extending vertically from top to bottom of the new zone.

Network modeling suggests that the system operating point for the 3,500 MTPD case would be approximately 120 m^3/sec, 250,000 cfm, a total pressure of 3.6 mmHg at an elevation of 1,420 meters. Ventilation requirements for the 4,500 and 5,500 MTPD cases will increase to 325,000 cfm and 400,000 cfm at the same velocity and total pressure as the base case. Air flow velocities in excess of 4.1 m/s or 800 ft/min will be avoided in areas where personnel will work or travel regularly as this is the point that respirable dust becomes airborne. The flow rate in the declines and stopes will be well below this critical velocity. Air velocities in the bore hole will reach 5.8 m/sec or 1,130 ft/min but travel will be limited to inspections and emergencies in which case the fans can be either turned off or velocities can be reduced.

Ventilation of the individual stopes will be accomplished through the use of auxiliary fans. A split of air from the main ventilation stream in the declines will be directed by the fan into the stope. Soft ventilation bag will be used to direct the air to the location where work is taking place after which the split of air will return to the main ventilation stream. In vein development headings the air will return via the route into the stope. During long-hole operations the air will be directed into the under-cut level and return to the main ventilation stream via the over-cut level. This flow direction reduces dust and improves visibility during mucking operations. Fan requirements for the individual stopes will be less than 26.3 m^3/sec, 50,000 cfm, and will be supplied by a 42kW or 60 hp auxiliary fan. Adequate volumes of air have been provided in the design to dilute contaminants in the air stream to acceptable levels at all stages of the network.

17 RECOVERY METHODS

17.1 MINING EQUIPMENT AND INFRASTRUCTURE

The following table summarizes the mobile equipment, fixed equipment, and infrastructure required for the three cases of the mine plan presented in this study. The initial capital will be spent in years 2011, 2012, and 2013 to achieve the 3,500 MTPD production plan. Capital for the expansion cases will be spent in 2012 through 2019 and consists of accelerating development in the Central and East zones as well as development of the new resources in the lower west portion of the Central Zone and the East Extension area in addition to the equipment and underground infrastructure required to support the additional development and production. Sustaining capital will be spent throughout the rest of the operating mine life. A freight and contingency of 15% has been applied to the mobile equipment. The 12% sales tax is applied to all items purchased inside Guatemala.

The costs are based on recent quotes of similar equipment. A schedule of sustaining capital is contemplated in the financial analysis and is based on estimates of equipment life and infrastructure requirements throughout the mine life. The following table shows a summary of capital requirements for equipment and mine development and underground infrastructure:

Table 17-1: Mine Capital

Mine Capital	Initial	Sustaining
3500 MTPD case	$ 77 million	$79.6 million
4500 MTPD case	$105.1 million	$125.6 million
5500 MTPD case	$105.6 million	$126.5 million

The major mine equipment will include R1700 and R2900 Caterpillar LHDs with 7.3 cubic meter buckets and equipped for remote operation production and development fleet. Forty-five (45) tonne Caterpillar trucks will be used for hauling ore and waste out of the mine. Atlas Copco two-boom electric hydraulic jumbos will be utilized to drive the development headings and stope development headings. Ground support will be installed in all headings with a fleet of Atlas Copco electric-hydraulic jumbos capable of installing split set, and swellex bolts. Cable bolting will be done using Atlas Copco Simba drills which will also be the primary production drill in the long-hole stopes. Cubex track-mounted drills equipped with a V-30 reaming head will be used for drilling breaking slots in the stopes. These drills will be equipped with top hammers and will be capable of drilling larger diameter holes for utilities as well as production drilling where larger diameter holes are desirable. Diamond drills will be utilized for stope definition to enhance production planning prior to stope production. The primary exhaust raise into the Central Zone and the East Zone will be initially developed using a contractor. Caterpillar 120 AWD graders will be used for road maintenance in the mine. Support equipment will include shotcrete remote spray jumbos and mixer trucks, scissor lifts, explosives trucks, and various materials handling vehicles are included in the fleet. A list of the equipment fleet and unit requirements including replacements over the life of the mine is included in the following table. The type of equipment is the same for each of the production cases. Additional units are required and included in the capital estimate for the higher production and accelerated development cases.

Table 17-2: Project Mobile Equipment List

Mobile Equipment	
R 2900 CAT LHD 7.3 mt^3	Stoper
R 1700 CAT LHD 7.3 mt^3	Shotcrete Spray Jumbo
	Underground transit Mixer Truck
	AWD 120 CAT Motor Grader
AD 45 CAT Truck 45t	LM55 and LM75 Diamond Drills
AC Jumbo Boomer 282 2-boom	Scissor Lift
AC Simba Long-hole Jumbo	ANFO Truck
AC Simba Drill for cables	Personnel Transport
AC Boltec MD Rock Bolt Jumbo	Telehandler
Cubex Long-hole Jumbo w/V-30 blind bore head	
Longhole DTH Jumbo/Compressor	Boom Truck
	Fan Truck
	Lube Truck
	HD Pickups
Jackleg	

17.2 MINING WORK FORCE

The mine is scheduled for two 11 hour shifts per day, 350 days per year. This will require 3 mine crews working a rotating schedule. The total mine personnel requirement is 298, including all management and supervisory personnel. A table listing the personnel breakdown is shown in the following table.

Table 17-3: Mine Personnel

Position	Per Shift	Shifts	Number	4500 MTPD	5500 MTPD
Staff					
UG Manager			1	1	1
Secretary			1	1	1
UG General Foreman			1	1	1
Production Foreman			1	1	1
Development Foreman			1	1	1
Backfill Foreman			1	1	1
Area Supervisors	5	3	15	15	21
Sub-total			***21***	***21***	***28***
Engineering					
Operations Manager			***1***	***1***	***1***
Chief Engineer			1	1	1

Position	Per Shift	Shifts	Number	4500 MTPD	5500 MTPD
Planning Engineers			4	4	4
Project Engineer			1	1	1
Geotech Engineer			1	1	1
Ventilation Tech			1	1	1
Production Engineer			1	1	1
Engineering Tech			2	2	2
Chief Surveyor			1	1	1
Surveyors			3	3	4
Survey Helpers			3	3	4
Sub-total			***18***	***18***	***20***
Geology					
Chief Geologist (Mine and Exploration)			1	1	1
Mine Geologists			8	8	10
Exploration Geologists			0	0	0
Geology Helpers			6	6	8
Data Processor			1	1	2
Samplers			3	3	6
Sub-total			***19***	***19***	***27***
Diamond Drilling					
Diamond Drill General Foreman			2	2	2
Diamond Drillers	3	3	9	9	9
Drill Helpers	6	3	18	18	18
Drill Mechanics			2	2	2
Sub-total			***3500 MTPD***	***4500 MTPD***	***5500 MTPD***
Production and Development					
Scoop Operators	6	3	18	21	24
Truck Drivers	7	3	21	24	27
Jumbo Operators	6	3	18	21	21
Bolter Operators	5	3	15	18	21
Stope Backfill Prep.	4	3	12	12	18
Shotcrete Operators	4	3	12	12	12
Grader Operator	1	3	3	6	6
Aux Equipment Operators	6	3	18	21	27
Lamp Room	1	3	3	3	3
Backfill Operators	2	3	6	6	6
Blasting Experts	1	3	3	6	9
Cleaning Personnel	1	3	3	3	6
Sub-total			***132***	***153***	***147***
Mine Maintenance					
Welders	2	3	6	6	9
Pump Mechanics	1	3	3	4	6
Electrical Engineer			1	1	1
Electrical General Foreman			1	1	1
Electricians	2	3	6	9	12
Electrical Helpers	1	3	3	3	6
Service Crew Leader	1	3	3	3	3

Position	Per Shift	Shifts	Number	4500 MTPD	5500 MTPD
Service Personnel	2	3	6	9	12
Pump Men	1	3	3	6	9
Sub-total			***32***	***42***	***59***
Mobile Equipment Maintenance					
Mobile Maintenance Manager			1	1	1
Maintenance Planner			1	1	2
Secretary			1	1	2
General Foreman			1	1	2
Supervisor	1	3	3	3	6
Mechanics	12	3	36	39	45
Helpers	4	3	12	15	18
Sub-total			***55***	***61***	***76***

17.3 MINE INFRASTRUCTURE AND FIXED EQUIPMENT

The mine infrastructure planned for the preproduction period is listed in the capital costs section of this report and includes establishment of mine portals, access roads to portals and vent rises, mobile maintenance offices and shops, mine dry, air compressors, fuel tanks, explosives magazines, pumps, electrical transformers and equipment.

Drilling to date has encountered limited and periodic quantities of ground water. It is anticipated from the available data that the mine will generate approximately 500 gpm of ground water and pumping will be required to transport this water to the surface. The pumping system contemplated for the mine will utilize pumps in the development headings and stopes to pump water to a central location. A series of mobile pump stations will pump the water to the surface during development. More permanent sumps and pump stations will be constructed below the active production levels and water will be routed to stations via a series of boreholes. From the main sump water will be pumped to the surface via a dewatering line located in boreholes to the surface and delivered to the surface settling ponds for treatment and discharge.

Compressed air requirements in the mine will be limited to the air required for the Cubex drills in the stopes and occasional use of jackleg drills for repair or specialized work and small tool use in the underground shops. A distribution system will supply compressed air to central locations in the footwall laterals. Routine communications will be accomplished through the installation of a leaky feeder radio system. The system will be installed throughout the mine. Mine foremen, lead men, mechanics, and other key underground personnel will be equipped with portable radios to facilitate mine wide communications. Additional radios will be distributed at key locations on the surface including the mine supervision offices, the safety director's office and the main office. Emergency communications will be provided through several systems. The first system is the leaky feeder system. In an emergency, this system functions the same way as in routine communications. The second system is the stench warning system. This system is used in an emergency to send a mine wide evacuation signal. The stench compound may be released into the compressed air and or the ventilation system. The third communication system that will be available in central locations of the mine is the paging telephone system. A limited number of paging telephones will distributed in key locations on the surface and underground. The primary function of the paging telephones will be to provide an independent emergency communication link to underground refuge chambers. This system will also serve as a standby system in the event of a failure of the leaky feeder system.

13.8 kV Power will be delivered to the mine from the main substation on the Escobal site. The primary underground loads during the pre-production development and production are associated with ventilation and the electric hydraulic mining equipment. Peak connected loads during the mine life are not expected to exceed 7,000 kW or 6,600 hp in the 5,500 MTPD case.

A schedule of sustaining capital is contemplated in the financial analysis.

17.4 PROCESSING

17.4.1 Process Overview – Sulfide

The process selected for recovering the gold, silver, lead and zinc can be classified as "conventional". The sulfide material will be crushed and ground to a fine size and processed through mineral flotation circuits. The following items summarize the process operations required to extract gold, silver, lead and zinc from the Escobal project ore.

1. Size reduction by a primary jaw crusher to reduce the material size from run-of-mine (ROM) to minus 200 millimeters.

2. Size reduction of the primary crushed material by secondary and tertiary crushing to reduce the ore size from 200 millimeters to minus 9 millimeters.

3. Grinding crushed material in a ball mill circuit to a size suitable for processing in a flotation circuit. The ball mill will operate in closed circuit with hydrocyclones to deliver an ore size of 80 percent passing 105 microns to the flotation circuit.

4. The flotation plant will consist of selective lead and zinc flotation circuits. The flotation circuits will each consist of rougher flotation and cleaner flotation to produce a high value gold, silver and lead concentrate and a lower value zinc concentrate with payable gold and silver values.

5. Final lead concentrate will be thickened, filtered, and loaded in super sacs for shipment. Final zinc concentrate will be also thickened, filtered and loaded in super sacs for shipment.

6. Flotation tailing will be thickened, filtered and either dry stacked in a tailing impoundment area or transported to the paste backfill plant. The paste backfill plant product will be used for backfill underground.

7. Water from tailing and concentrate dewatering will be treated and recycled for reuse in the process. The Escobal Project design strives to maximize recycling and reusing

process water in order to minimize treatment and discharge. Plant water stream types include: process water, raw water, and potable water.

8. Storing, preparing, and distributing reagents used in the process.

Mineral processing is shown in the flowsheet in Figure 17-1.

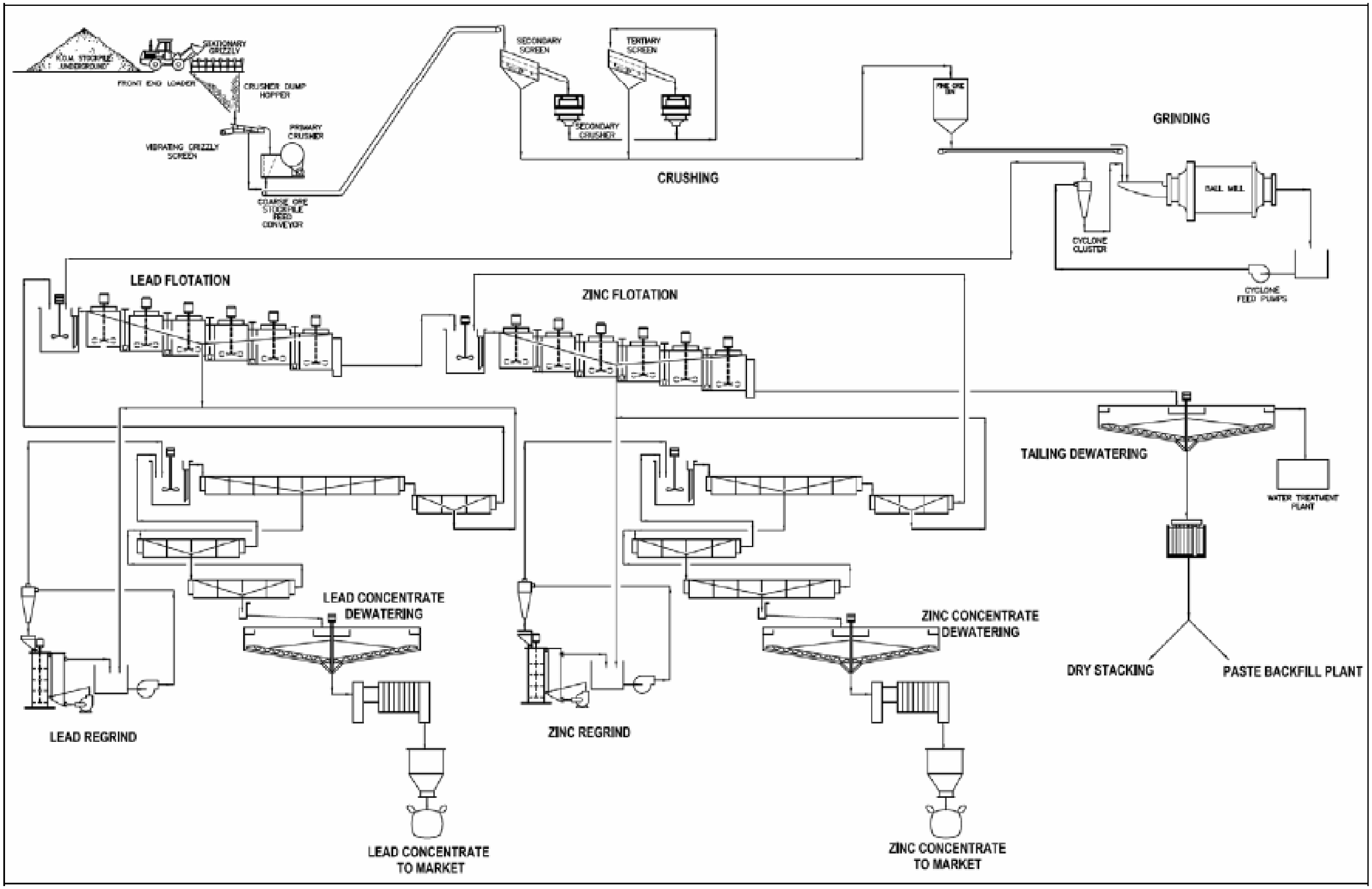


Figure 17-1: Overall Processing Flow Sheet

18 PROJECT INFRASTRUCTURE

Improvements to roads, bridges, and drainage structures, including ponds, may be removed or left in place whichever is most beneficial to the local community for post operation use. In the case of removal, each area will be regraded and revegetated.

19 MARKET STUDIES AND CONTRACTS

There are no market studies and contracts reported for the Escobal project.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 GEOCHEMICAL CHARACTERIZATION

Tahoe/Minera San Rafael has developed and implemented geochemical characterization programs to assess the potential for generation of acid rock drainage (ARD) from waste/development rock, mineralized material, and process residue (representing tailing material). The characterization programs include static and kinetic testing of samples collected from drill core, underground exploration excavations, and metallurgical testing.

As material excavated (waste/development rock) from the Escobal underground exploration project will be utilized to construct the pads (platforms) at the two portal locations, geochemical testing programs have been conducted to assess the potential for this material to generate ARD. Modified Acid/Base Accounting (ABA) tests were completed to determine the acid generation potential/acid neutralization potential (AGP/ANP) of the material. The results of these tests demonstrate that the rock to be excavated during the underground exploration has a high net neutralizing potential (average NNP of 87) and is unlikely to generate acid; therefore, no adverse impacts are predicted. Meteoric Water Mobility Procedure (MWMP) tests to evaluate the potential for dissolution and mobility of metals and other constituents from the development rock and tailings under natural precipitation conditions, i.e. rainwater, demonstrate leaching of deleterious constituents from waste rock and tails is unlikely to occur.

In addition to ABA and MWMP tests, a series of kinetic Humidity Cell (HC) tests have also been conducted on waste rock and tailings samples from Escobal. HC tests model the atmospheric and geological processes of weathering and are used to determine the rate (if any) of acid generation and variation of leachate (effluent) water quality over time. Samples of weakly mineralized waste rock, representing the rock types and alteration types that will be exposed in the development workings, and tailings from the metallurgical/flotation tests, were selected for HC testing. The ASTM procedure (D-5744) requires a minimum test duration of 20 weeks; the duration of the Escobal HC tests are currently beyond 54 continuous weeks with no indication of acid formation or deleterious metal concentrations that exceed regulatory limits in either the waste rock or tailings effluent, confirming the results of the ABA and MWMP tests.

Geochemical testing programs for waste/development rock were conducted by Goldcorp/Entre Mares in 2009 and by Tahoe/Minera San Rafael in 2010 and 2011 and continuing through the present. All ARD tests (ABA, MWM, HC) were performed by independent laboratories. McClelland Laboratories Inc (Sparks, Nevada USA) has conducted all ARD testing for Tahoe/Minera San Rafael; Goldcorp's tests were performed by SVL Analytical Inc. (Kellogg, Idaho USA).

The exploration decline is designed to be excavated approximately 75 meters or more from the Escobal vein and is not anticipated to intersect significantly mineralized material. Minera San Rafael is and will continue to conduct paste pH tests of the rock from the exploration declines regularly as they are being mined. Samples of materials suspected to be potentially acid generating based on paste pH test results will be laboratory-tested by ABA. To date, no samples have demonstrated characteristics that would make them candidates for further ABA testing. Paste pH is performed onsite by Minera San Rafael. ABA tests will be performed by a qualified independent laboratory. In the event isolated areas of mineralization are encountered during exploration, Minera San Rafael will store that material underground to mitigate any potential for acid rock drainage.

In addition to samples that do not pass the criteria established for the field test, ABA, MWMP, and humidity cell tests will be conducted periodically on samples to determine acid generating potential. Mineralized material that does not meet economic cut-off and is suspected to have the potential to generate ARD will be used as cemented backfill or encapsulated in the dry stack tailings facility.

20.2 TAILING AND DEVELOPMENT ROCK STORAGE FACILITY

20.2.1 Design Criteria

The above ground disposal of tailings will be designed to be “dry stacked”. Tailings that are thickened and filtered to 10 to 15% moisture content are commonly termed “dry tailings”. Dry tailings can be trucked and stacked at a relatively high density compared to conventional pipeline transported tailings slurry. Unlike conventional tailings impoundments that are designed to retain tailings and water, the design principles of dry stacked facilities are to create a self-supporting mound of tails rather than rely on the retaining forces of embankments to prevent mobilization of the tailings material. While this process can be considerably more expensive, dry stacked tailings have advantages over conventional tailings impoundments, particularly where water conservation is important and in regions with higher levels of seismic activity. Dry stacked facilities generally require a smaller surface footprint, are easier to reclaim, and have a higher long-term structural integrity and much lower long-term environmental impact as opposed to conventional tailings impoundments (Davies & Rice, 2004).

Over the life of mine, approximately 29.9 million tonnes of the dry tailings will be produced; of which approximately 14.5 million tonnes will be mixed with cement and go back underground as paste backfill. The remaining 14.5 million tonnes of dry tailings will be placed in the Tailings and Development Rock Storage Facility, along with approximately 2,300,000 tonnes of development rock.

20.2.2 Stormwater Management

Tailing stormwater surface run-off will report to a lined stormwater pond. Dry stack tailing facilities typically see limited infiltration. A coarse drain rock collection grid will be installed to capture potential seepage through the tailings. The seepage water will also discharge to the stormwater pond. This water will be utilized as process make-up water or be treated and released, as the site water balance dictates.

Stormwater run-on diversion will be accomplished by the construction of diversion channels upgradient of the active tailing placement area. The non-impacted water collected in these channels will report to down drains on either side of the facility which will in turn report to natural drainages.

20.2.3 Concurrent Reclamation

Before any tailing or development rock material is placed in the facility, topsoil from the facility footprint will be stripped and stored in a topsoil salvage stockpile.

A starter buttress will be constructed along the toe of the footprint behind which the dry stack tailings and waste rock will be placed. The starter buttress may be constructed of suitable development rock and/or native material from within or near the facility footprint.

Once the starter buttress is complete, portions of the outslopes will be covered with topsoil and seeded with a native seed mix. As each new outslope lift (comprised of compacted dry tailing and / or suitable development rock) is completed, it will also be covered and seeded. This will accelerate revegetation and provide useful data on the effectiveness and efficiency of the reclamation and revegetation methods at this particular site.

The facility will be designed so that the final contours blend in with the surrounding terrain. Geomorphic landform grading principles will be employed to the extent practicable in the design of the outslopes to provide greater resistance to erosion and promote revegetation. The tailings facility will be covered with an engineered evapotranspiration layer consisting of sand covered by a layer of topsoil, specifically designed to minimize infiltration from precipitation, and then revegetated.

20.2.4 Geotechnical

Preliminary site inspections and subsurface testing indicate that the tailings and waste rock storage facility location will be acceptable for the design as currently proposed. Robertson GeoConsultants, Inc. has been retained to conduct the geotechnical investigations and design the construction criteria for the facility. Site investigations will be completed in the second quarter of 2012 and the final design of the tailings facility is expected to be complete in the third quarter of 2012, at which time the final design will be submitted to MARN for approval.

20.2.5 Tailing Placement

The dry tailings will delivered to the tailings area via conveyor where the filtered tailings will be loaded into trucks, hauled to the tailings facility, and dumped behind the starter buttress. The tailings will be spread and compacted in thin lifts. Compaction will be accomplished by dozer, smooth drum roller, or similar type equipment.

20.2.6 Development Rock Placement

Development rock produced from exploration drifting and development operations will first be utilized to construct the pads immediately outside the portals as working platforms. Development rock of appropriate quality may also be utilized in the construction of roads, drainage structures, erosion protection and safety berms. Some development rock may be used as backfill in the underground workings with the remainder being placed in the Tailings and Development Rock Storage Facilities. The development rock can be segregated within a separate storage facility or co-mingled with the dry tailings.

20.3 PERMITTING

Overall activities at the Project are permitted through two primary agencies, the Ministry of Environment, Ministerio de Ambiente y Recursos Natural (MARN) and the Ministry of Energy and Mines, Ministerio de Energía y Minas (MARN). All permits for construction and development of the Project are in place. Receipt of the exploitation license for the commencement of production is pending.

MARN approval, including required environmental commitments, for surface exploration activities at the Project were granted to Entre Mares on December 23, 2008 by MARN Resolution 4590-2008/ELER/CG. The approval and requirements were transferred from Entre Mares to Minera San Rafael as specified in Resolution 1918-2010/ECM/GB, dated September 3, 2010.

An environmental impact study (EIS) addressing the additional activities associated with underground exploration was required prior to commencing excavation of the exploration declines. The underground exploration EIS was filed with MARN in November 2010 and an Environmental License filed on March 17, 2011. MEM notified the company of the receipt and acceptance of the work program for the exploration declines on April 5, 2011.

Development of an underground exploration program including the construction of two declines to gain access for additional drilling of the Escobal deposit is a permitted activity under the terms of the existing exploration license. An EIS for the exploitation phase of the project was prepared and submitted to MARN for approval in August of 2011. MARN approved the EIS for exploitation by issuing Resolution 3061-2011 in October of 2011. This approval allowed full construction of the mine, process plant and all surface and underground facilities to be conducted. Application for the Exploitation License was submitted to the MEM in November 2010 and is awaiting final approval by the agency. Approval of this license is required before production can commence.

The environmental impact statements require documentation of baseline conditions, a project description, and an analysis of potential impacts and their mitigation measures. Public disclosure and involvement has been required and developed throughout each stage of the project and the permitting.

All other permits have been acquired by Minera San Rafael, including tree-cutting permits as needed, issued by the National Institute of Forests (INAB). Land use changes in the project area have also been approved by INAB as required. Archeological clearances have been issued.

20.4 SOCIAL OR COMMUNITY IMPACTS

The Project is located approximately two kilometers from San Rafael Las Flores, a community of approximately 3,500 inhabitants. The Company is not aware of any significant indigenous population residing in the area of the Escobal Project. According to Guatemala's National Institute of Statistics (Census 2002) San Rafael Las Flores' population is 99.6% "Ladino", i.e., of Hispanic origin and non-indigenous. The area surrounding the community, including several small villages, is generally used by local farmers to grow vegetables in the valleys and coffee at higher elevations. There is no heavy industry in the immediate area. Tahoe/Minera San Rafael recognizes the potential impacts to the community's infrastructure due to increased industrial activities and the related influx of the growing workforce and is working directly with community leaders and community groups to minimize any potential negative impacts and maximize the numerous benefits related to the project for the betterment of the community and surrounding areas. Community support for the Project is very high and Minera San Rafael is committed to being an active and positive member of the local community.

Tahoe/Minera San Rafael has committed to a voluntary increase in the royalty of 4% for precious metals and 3% for base metals though its involvement with the Guatemalan Mining Association. The royalty will be shared equally between the local municipality and the federal government. Life of mine royalty payments are estimated to be $388.3 and $389.0 million for the 4,500 MTPD case and 5,500 MTPD case, respectively.

Tahoe has purchased all of the land necessary for the operation of the Project and is developing a profit sharing program to provide the ex-land owners benefits throughout the life of the Project. The concept is to pay an amount of 0.5% of net smelter returns to an Association of Land Owners and individual land owners. A certain percentage of this money will be deposited in a special fund, administrated by the association board of directors and used for improvements in local communities on behalf of the members of the association. Land purchase agreements include a provision that provides land owners the right to buy their land back from Tahoe at a significantly reduced price at the end of the life of the mine, once all reclamation has been completed.

The project currently employs approximately 450 people of which 430 are Guatemalan and 95% of those live in or near San Rafael. Once in the production phase, the project is expected to have direct employment of approximately 650 people, 95% or which are expected to reside in the local communities.

The company maintains a community relations department in San Rafael that focuses on working with the community to address concerns and provide information about the mining operation. In addition, the 10 person department works with the municipality and local villages to assist in community and school improvement projects. This department has been active for over 4 years. Some of the projects include clean water supply projects, assisting in the start of new business, partnering with government agencies to provide training and education, and working with the local schools to enhance education quality in the area.

21 CAPITAL AND OPERATING COSTS

Operating costs, including capital development costs, were developed on a unit cost and quantity basis utilizing both first principals and similar operation comparisons. Data used in the analysis was derived from internal data bases collected over a number of years. In some cases the data was factored and or escalated to 2010 dollars to better reflect the Escobal operating plan. Data obtained from other Guatemalan operations, Goldcorp's Marlin Mine in particular, received heavy weighting in the cost analysis due to similarities in location, mine size, ground conditions, and mining method. The term "ore" is used in this discussion of capital and operating costs to differentiate between mineralized material (including dilution) above an economic cutoff grade and waste rock; there is no inference of mineral reserves.

Table 21-1: Mine Operating Costs

		3500 MTPD case	4500 MTPD case	5500 MTPD case
Expensed Waste Development	Labor	$1.53	$1.43	$1.53
	Supplies	$3.94	$3.68	$3.94
Ore Development	Labor	$0.34	$.032	$0.34
	Supplies	$1.03	$0.96	$1.03
Production Drilling	Labor	$0.55	$0.51	$0.55
	Supplies	$1.64	$1.53	$1.64
Blasting	Labor	$0.47	$0.44	$0.47
	Supplies	$1.41	$1.32	$1.41
Haulage	Labor	$2.19	$2.06	$2.19
	Supplies	$4.43	$4.15	$4.43
Backfill	Labor	$0.59	$0.56	$0.59
	Supplies	$8.78	$8.23	$8.78
Ventilation & Services	Labor	$0.50	$0.50	$0.50
	Supplies	$1.64	$1.53	$1.64
Total	Labor	$6.17	$5.82	$6.17
	Supplies	$22.87	$21.40	$22.87
	Total	$29.04	$27.22	$29.04

21.1 DEVELOPMENT COST

The development cost in the financial analysis is based on analysis of similar mining operations and a first principal analysis. The costs have been adjusted to reflect pertinent information from experience to date at Escobal. Data from the Marlin Mine in Guatemala was considered quite reliable as the operations are similar in size, rock quality, and average haul distances. Development of 5 meter wide by 5 meter high declines with owner crews is estimated at $2,715/m and contractor development is estimated at $3,015/m. The cost of developing the primary declines 5 meters wide by 6 meters high is estimated at $3,017/m with owner crews and $3,620/m utilizing a contractor. These unit costs include the cost of labor and supplies, including ground support materials, explosives, definition diamond drilling, installed piping, electrical, and communications lines and ventilation and pumping systems integral to the unit operation. These costs are common to all three production cases. Labor estimates are based on crew size for each development heading and local labor rates. Sales tax or IVA is included at 12% of the non-owner labor costs.

21.2 MINING COST

Mining costs are also based on a detailed analysis of similar operations including the Marlin Mine as well as first principal analysis. The costs have been adjusted to reflect recent experience at Escobal. The Escobal cost for long-hole stoping is $18.56 per tonne mined for the 3,500 MTPD case, $20.06 per tonne for the 4,500 MTPD case, and $18.88 per tonne for the 5,500 MTPD case. This value does not include the access development or backfill placement. The long-hole mining costs include ore development, excavation of the over and under-cuts, as well as the cost of the excavation between the over and under-cut. These unit costs include the cost of labor and supplies, including ground support materials, explosives, installed piping, electrical, and communications lines and ventilation and pumping systems integral to the unit operation. Labor estimates are based on crew size for each development heading and local labor rates. Sales tax or IVA is included at 12% of the non-owner labor costs.

21.3 BACKFILL COST

The backfill costs include the operation and maintenance of the paste plant, pumping system, piping system, as well as electricity for the plant and pumps, cement as a binder, and the construction of bulkheads in the mine to seal areas to be filled. The total yearly backfill requirement is calculated to be 312,000 $m^3$for the 3,500 MTPD case, 350,000 m^3 380,000 m^3 for the 4,500 MTPD and 5,500 MTPD cases respectively. At a density of 2.0, cement content of 5% by weight and water to cement ratio of 4, every cubic meter will contain 1.5 t of tailings, 0.4t of water and 0.1 t of cement. The cement cost is $11.75/$m^3$. Additional costs are estimated to be about $5.24/$m^3$, bringing the overall backfill cost to $16.99/$m^3$. Estimated backfill cost per tonne mined is US$8.68 for the 3,500 MTPD case, $9.37 for the 4,500 MTPD case and $8.79 for the 5,500 MTPD case. This cost includes personnel for the paste backfill plant. Labor estimates are based on crew size for each development heading and local labor rates. Sales tax or IVA is included at 12% of the non-owner labor costs.

21.4 ENGINEERING AND GEOLOGY

Engineering and geology costs are included in the mining and development costs.

21.5 DEFINITION DRILLING

There will be two definition drills working full time to define stoping areas ahead of mining. The costs will be around $50/m drilled or about $0.45 per tonne ore and are included in the mining cost.

21.6 GENERAL AND ADMINISTRATIVE COSTS

The general and administrative costs to support the mining operations were developed on a unit cost and quantity basis and utilize data from Tahoe databases, the Marlin mine, and first principal estimates for Guatemalan labor, supplies and contracts listed in the following tables. Sales taxes or IVA is included at 12% of the non-owner labor costs.

Table 21-2: Escobal General and Administrative Operating Costs

					t/a	
G & A Workforce	# Shifts	Per Shift	**Number**		**Annual $**	**Total**
General Manager			1		$220,000	$220,000
Assistant			1		12,000	$12,000
Human Resources Manager			1		$90,000	$90,000
HR Supervisor			1		$24,000	$24,000
Clerks			4		$10,000	$40,000
						$0
Chief Accountant			1		$75,000	$75,000
Accountant			1		$24,000	$24,000
Clerks			4		$10,000	$40,000
Purchasing Manager			1		$100,000	$100,000
Purchasing Supervisor			1		$24,000	$24,000
Purchasing Assistant			3		$10,000	$30,000
Warehouse Supervisor			2		$36,000	$72,000
Warehouse Assistant			8		$10,000	$80,000
						$0
Community Relations						$0
Manager			1		$75,000	$75,000
Supervisor			2		$20,000	$40,000
Team			8		$10,000	$80,000
						$0
Environmental Manager			1		$125,000	$125,000
Supervisors			2		$30,000	$60,000
Technicians			8		$10,000	$80,000
						$0
Safety Manager			1		$150,000	$150,000
Safety Supervisors			2		$20,000	$40,000
Safety Assistant			2		$10,000	$20,000
Training Supervisor			1		$60,000	$60,000
Training Assistants			2		$15,000	$30,000
						$0
Security Superintendent			1		$60,000	$60,000
Security Supervisor			4		$20,000	$80,000
Security Officers - executive security			12		$15,000	$180,000
Front Gate	3	2	6		$8,000	$48,000

					t/a	
G & A Workforce	# Shifts	Per Shift	**Number**		**Annual $**	**Total**
Admin Office	3	2	6		$8,000	$48,000
Mine Area	3	3	9		$8,000	$72,000
Mill	3	3	9		$8,000	$72,000
Refinery	3	2	6		$8,000	$48,000
Total G&A Workforce			112			**$2,199,000**

Table 21-3: Escobal General and Administrative Operating Cost

	3500 MTPD Case	**4500 MTPD Case**	**5500 MTPD Case**
Accounting	$273,500	$312,571	$351,643
Human Resources	$946,600	$1,081,829	$1,217,057
IT	$122,225	$139,686	$157,146
Salaried Staff	$4,322,720	$4,940,251	$5,557,783
Environmental	$780,160	$891,611	$1,003,063
Sustainable Development	$1,104,200	$1,261,943	$1,419,686
Security	$705,900	$806,743	$907,586
Medical Services	$458,898	$524,455	$590,012
Warehouse	$173,020	$197,737	$222,454
Purchasing	$274,875	$314,143	$353,411
Total	$9,162,098	$10,470,970	$11,779,841

General and Administrative costs include employee withholdings and taxes as well as 12% IVA tax on services and supplies purchased in Guatemala.

21.7 OPERATING COST ESTIMATE

This section addresses the following costs:

- Mining Costs
- Process Plant Operating & Maintenance Cost
- General and Administrative Costs

The operating costs for the 4,500 MTPD and the 5,500 MTPD cases were calculated for each year during the life of the mine using the annual ore tonnage as a basis. Table 21-4 reflected the approximate production of zinc and lead concentrates and metal contained each concentrate.

Table 21-4: Approximate Concentrate Production and Content

4,500 MTPD Case	Tonnes (000's)	Zinc (klbs.)	Silver (kozs.)	Gold (kozs.)
Zinc Concentrate	515	596,372	15,807	15
Lead Concentrate	299	336,480	303,275	258
5,500 MTPD Case				
Zinc Concentrate	519	600,758	15,830	15
Lead Concentrate	300	337,734	303,708	258

Table 21-5 shows the unit cost per ore tonne for the life of the mine for both cases.

Table 21-5: Operating Costs by Area

	4,500 MTPD	5,500 MTPD
Life of Mine	$/tonne	$/tonne
Ore Tonnes	29,826,845	29,924,285
Mining Operations	$29.03	$27.22
Mill Operations		
Crushing & Conveying	$1.95	$1.85
Grinding & Classification	$5.06	$6.29
Flotation and Regrind	$4.20	$4.19
Concentrate Dewatering, Filtration & Dewatering	$1.05	$0.99
Tailing Disposal	$5.26	$4.92
Laboratory	$0.52	$0.50
Ancillary Services	$1.50	$1.42
Subtotal Processing	$19.54	$20.16
Supporting Facilities		
General and Administrative	$6.67	$6.87
Subtotal Supporting Facilities	$6.67	$6.87
Total Operating Cost	$55.24	$54.25

21.7.1 Process Plant Operating & Maintenance Costs

The process plant operating costs are summarized by areas of the plant and then by cost elements of labor, power, reagents, maintenance parts and supplies and services. In addition to the cost of these items an IVA tax of 12% was applied to materials and services required by the Guatemala government.

21.7.2 Process Labor & Fringes

Process labor costs were derived from a staffing plan and based on prevailing daily or annual labor rates in the area. Labor rates and fringe benefits for employees include all applicable social security benefits as well as all applicable payroll taxes. The staffing plan summary and gross annual labor costs are shown in Table 21-6 below.

Table 21-6: Process Plant Labor & Fringes

Department	Number of Personnel	
	4,500 MTPD Case	**5,500 MTPD Case**
Mill Operations	73	73
Laboratory	23	23
Mill Maintenance	47	47
Total	143	143

21.7.3 Electrical Power

Electrical power costs were provided by Tahoe. Power consumption was based on the equipment list connected kW, discounted for operating time per day and anticipated operating load level. The overall power rate is estimated at $0.140 per kWh. A summary of the base power consumption and cost and the additional power required for each case is shown below.

Table 21-7: Power Cost Summary

3500 MTPD Power Requirements	Annual kWhr	Annual Cost
Primary Crushing & Conveying	1,090,955	$171,062
Secondary and Tertiary Crushing	7,380,769	$1,157,305
Fine Ore Storage and Reclaim	679,032	$106,472
Grinding	26,668,629	$4,181,641
Flotation & Regrind	18,633,804	$2,921,780
Reagents	840,676	$131,818
Concentrate Thickening	3,945,741	$618,692
Tailings Disposal	36,605,529	$5,739,747
Dry Stack Area	394,631	$61,878
Fresh Water/Plant Water	3,499,064	$548,653
Ancillaries	272,238	$42,687
Total 3500 MTPD Power Requirements	**100,011,069**	**$15,681,736**
4500 MTPD Additional Power Requirements		
Grinding	1,052,350	$165,009
Flotation & Regrind	2,657,184	$416,646
Concentrate Thickening	134,175	$21,039
Tailings	263,088	$41,252
Fresh Water/Plant Water	368,323	$57,753
Total 4500 MTPD Additional Power Requirements	**4,475,119**	**$701,699**
Total 4500 MTPD Power Requirements	**104,486,188**	**$16,383,434**
5500 MTPD Additional Power Requirements		
Fine Ore Storage and Reclaim	210,470	$33,002
Grinding	25,676,617	$4,026,094
Flotation & Regrind	2,578,258	$404,271
Tailings	263,088	$41,252
Fresh Water/Plant Water	105,235	$16,501
Total 5500 MTPD Additional Power Requirements	**28,833,668**	**$4,521,119**
Total 5500 MTPD Power Requirements	**133,319,856**	**$20,904,553**

21.7.4 Reagents

Consumption rates were determined from the metallurgical test data or industry practice. Budget quotations were received for reagents supplied from local sources where available with an allowance for freight to site. In addition IVA tax of 12% was included.

A summary of process reagent consumption and costs are included in Table 21-8 below.

Table 21-8: Reagents Consumption Summary

Reagents	kg/tonne ore	$/kg
Lime	0.300	$ 0.11
Zinc Sulfate	0.060	$ 3.00
Zinc Cyanide	0.020	$ 2.20
Copper Sulfate	0.030	$ 1.06
PAX	0.020	$ 2.45
C-7931	0.040	$ 10.00
C-4132	0.020	$ 2.82
X-133	0.030	$ 2.50
Flocculant (Concentrate Thickeners)	0.010	$ 3.85
Flocculant (Tailings Thickeners)	0.090	$ 3.85

21.7.5 Maintenance Wear Parts and Consumables

Grinding media consumption and wear items (liners) were based on industry practice for the crusher and grinding operations. These consumption rates and unit prices are shown in Table 21-9. In addition IVA tax of 12% was included.

Table 21-9: Grinding Media and Wear Items

Grinding Media & Wear Parts	kg/tonne ore	$/kg
Primary Crusher - Liners	0.010	$ 4.85
Secondary Crusher - Liners	0.040	$ 4.85
Tertiary Crusher - Liners	0.021	$ 4.85
Ball Mill - Liners	0.039	$ 5.73
Zinc Regrind - Liners	0.001	$ 5.10
Lead Regrind - Liners	0.001	$ 5.10
Ball Mill - Balls	0.740	$ 1.25
Lead Regrind Mill- Balls	0.020	$ 5.10
Zinc Regrind Mill- Balls	0.020	$ 5.10

An allowance was made to cover the cost of maintenance of all items not specifically identified and the cost of maintenance of the facilities. The allowance was calculated using the direct capital cost of equipment times a percentage for each area, which totalled approximately $2.8 million for 4,500 MTPD case and $3.4 million for the 5,500 MTPD case. Also an annual allowance was made for outside maintenance services to be performed at approximately $0.5 million for both cases.

21.7.6 Process Supplies & Services

Allowances were provided in process plant for outside consultants, outside contractors, vehicle maintenance, and miscellaneous supplies. The allowances were estimated using M3's information from other operations and projects. Approximately $2.5 million will be spent annually for both cases.

21.8 CAPITAL COST ESTIMATE

Table 21-10 shows a summary of estimated initial capital expenses.

Table 21-10: Initial Capital Expense Estimate (3,500 MTPD)

Description	Cost
Direct Costs	
General Site	$14,045,472
Mine, West Portal – By Owner	$0
Mine, East Portal – By Owner	$0
Primary Crushing	$4,098,061
Secondary & Tertiary Crushing	$5,737,647
Fine Ore Storage & Reclaim	$5,775,776
Grinding	$15,074,923
Flotation & Regrind	$20,755,307
Reagents	$3,734,928
Concentrate	$9,613,887
Tailing Dewatering	$16,990,634
Paste Backfill Plant	$6,723,845
Tailing Dry Stack	$2,073,044
Water Systems and Well Field	$10,313,094
Sewage Treatment	$639,615
Main Substation	$6,065,325
Overhead Power Line	$2,402,365
Ancillaries	$20,893,022
Insurance/Capital Spares	$2,000,000
Freight	$10,388,696
Duties	$3,001,033
Subtotal DIRECT COST	**$160,326,674**
Indirect Costs	
CONTINGENCY	$26,646,797
Other Indirects Including EPCM,	$30,040,626
Contractor Power, Vendor Supervision	
and Commissioning	
IVA @ 12% (Eventually Refundable)	$10,697,853
TOTAL EPCM CAPITAL COST	**$227,711,950**
TOTAL MINE CAPITAL COST	**$78,494,050**
OWNER'S COST	**$20,443,000**
TOTAL	**$326,649,000**

The capital costs for the option to increase production to 4,500 MTPD are as follows:

Table 21-11: Capital Cost Estimate for the 4,500 MTPD Expansion

Total Costs for the 3,500 MTPD Project	**$326,649,000**
Additional Costs for the Expansion to 4,500 MPTD from 3,500 MTPD	
Mine Expansion Costs	$28,101,501
Direct Costs	$12,710,911
Indirect Costs	$5,323,588
Total Plant Expansion Costs	$18,034,499
Grand Total	**$372,785,000**

The 4,500 MTPD expansion will cost $18,034,499 in plant expansion and $28,101,501 in mine development and equipment in addition to the costs for the 3,500 MTPD project, for a total of $372,785,000.

A more detailed estimate for the 4,500 MPTD expansion is as follows:

Table 21-12: 4,500 MTPD Expansion Project

Description	Cost
Mining Costs	
Underground Mine Development	$8,436,001
Mine Equipment & Underground Infrastructure	$19,665,500
Subtotal Mine	$28,101,501
Direct Costs	
General Site	$25,384
Mine, West Portal - By Owner	$0
Mine, East Portal - By Owner	$0
Primary Crushing	$0
Secondary & Tertiary Crushing	$0
Fine Ore Storage & Reclaim	$0
Grinding	$591,525
Flotation & Regrind	$2,146,662
Reagents	$0
Concentrate	$215,737
Tailing Dewatering	$260,890
Paste Backfill Plant	$0
Tailing Dry Stack	$0
Water Systems and Well Field	$123,000
Sewage Treatment	$0
Main Substation	$8,000,000
Overhead Power Line	$0
Ancillaries	$0
Insurance/Capital Spares	$0
Freight	$1,035,187
Duties	$312,525
Subtotal DIRECT COST	**$12,710,911**
Indirect Costs	
CONTINGENCY	$2,307,701
Other Indirects Including EPCM,	$3,015,887
Contractor Power, Vendor Supervision	
and Commissioning	
IVA @ 12% (Eventually Refundable)	$342,128
TOTAL EPCM COSTS FOR THE EXPANSION	**$18,034,499**
TOTAL COSTS FOR THE 3,500 MPTD PROJECT	**$326,649,000**
GRAND TOTAL	**$372,785,000**

The capital costs for the option to increase production to 5,500 MTPD are as follows:

Table 21-13: Capital Cost Estimate for the 5,500 MTPD Expansion

Total Costs for the 3,500 MTPD Project	**$326,649,000**
Additional Costs for the Expansion to 4,500 MPTD from 3,500 MTPD	
Mine Expansion Costs	$28,101,501
Direct Costs	$35,147,120
Indirect Costs	$15,015,844
Total	$50,162,964
Grand Total	**$405,413,465**

The 5,500 MTPD expansion will cost $50,162,964 for plant expansion and $28,601,501 in mine development and equipment costs in addition to the costs for the 3,500 MTPD project, for a total of $405,413,465.

A more detailed estimate for the 5,500 MPTD expansion is as follows:

Table 21-14: 5,500 MTPD Expansion Project

Description	Cost
Mining Costs	
Underground Mine Development	$8,436,001
Mine Equipment & Underground Infrastructure	$19,665,500
Subtotal Mine	$28,101,501
Direct Costs	
General Site	$70,294
Mine, West Portal - By Owner	$0
Mine, East Portal - By Owner	$0
Primary Crushing	$0
Secondary & Tertiary Crushing	$0
Fine Ore Storage & Reclaim	$263,881
Grinding	$14,749,495
Flotation & Regrind	$4,943,932
Reagents	$0
Concentrate	$198,832
Tailing Dewatering	$252,908
Paste Backfill Plant	$0
Tailing Dry Stack	$0
Water Systems and Well Field	$124,896
Sewage Treatment	$0
Main Substation	$10,979,926
Overhead Power Line	$0
Ancillaries	$0
Insurance/Capital Spares	$0
Freight	$2,731,718
Duties	$831,238
Subtotal DIRECT COST	**$35,147,120**
Indirect Costs	
CONTINGENCY	$6,371,021
Other Indirects Including EPCM,	$7,326,349
Contractor Power, Vendor Supervision	
and Commissioning	
IVA @ 12% (Eventually Refundable)	$1,318,474
TOTAL EPCM COSTS FOR THE EXPANSION	**$50,162,964**
TOTAL COSTS FOR THE 3,500 MPTD PROJECT	**$326,649,000**
GRAND TOTAL	**$405,413,465**

21.8.1 Introduction

In general M3 based this capital cost estimate on its knowledge and experience of similar types of facilities and work in similar locations. Resources available to M3 included recent cost data collected for a nearby mining project and for similar process plants that have been constructed, are under construction, are being designed or studied in other locations.

21.8.2 Assumptions

The project is assumed to be constructed in a conventional EPCM format, e.g. Tahoe will retain a qualified contractor to manage and design the project; bid and procure materials and equipment as agent for Tahoe; bid and award construction contracts as agent; and manage the construction of the facilities as agent.

Tahoe will order major material supplies (e.g., structural and mechanical steelwork) as well as bulk orders (e.g., piping and electrical). These will be issued to construction contractors on site using strict inventory control.

All costs to date by Owner are considered as sunk costs.

“Initial Capital” is defined as all capital costs through to the end of construction. Capital costs predicted for later years are carried as sustaining capital in the financial model.

All costs are in 1st quarter 2012 US dollars.

21.8.3 Estimate Accuracy

The accuracy of this estimate for those items identified in the scope-of-work is estimated to be within the range of plus 20% to minus 15%; i.e., the cost could be 20% higher than the estimate or it could be 15% lower. Accuracy is an issue separate from contingency, the latter accounts for undeveloped scope and insufficient data (e.g., geotechnical data).

21.8.4 Contingency

Contingency is intended to cover unallocated costs from lack of detailing in scope items. It is a compilation of aggregate risk from all estimated cost areas. Contingency is not simply a “buffer” to cover estimate inaccuracy. Properly calculated contingency will be spent.

21.8.5 Documents

Documents available to the estimators include the following:

Design Criteria	(Yes)
Equipment List	(Yes)
Equipment Specifications	(No)
Construction Specifications	(No)
Flowsheets	(Yes)
P&IDs	(Yes)
General Arrangements	(Yes)
Architectural Drawings	(No)
Civil Drawings	(No)
Concrete Drawings	(No)
Structural Steel Drawings	(No)
Mechanical Drawings	(No)
Electrical Schematics	(No)
Electrical Physicals	(No)
Instrumentation Schematics	(No)
Instrument Log	(No)
Pipeline Schedule	(No)
Valve List	(No)
Cable and Conduit Schedule	(No)

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures and production cost and sales revenue. The sales revenue is based on the production of lead and zinc concentrate also containing gold and silver. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

The economic analysis of the Project includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserve. The PEA is preliminary in nature and there is no certainty that the PEA will be realized. The basis for the PEA is the Indicated mineral resources and Inferred mineral resources as reported herein. There is no pre-feasibility or feasibility study in respect to the Escobal project. The term “ore” is used in this economic analysis to differentiate between mineralized material (including dilution) above an economic cutoff grade and waste rock; there is no inference of mineral reserves.

22.2 MINE PRODUCTION STATISTICS

Mine production is reported as ore from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report.

The life of mine ore and waste quantities and ore grade are presented in the table below.

Table 22-1: Life of Mine Ore and Metal Grades

	Ore Tonnes (000's)	Zinc %	Lead %	Gold g/t	Silver g/t
4500 MTPD Case	29,872	1.1%	0.6%	0.4	383.2
5500 MTPD Case	29,924	1.1%	0.6%	0.4	382.6

22.3 PLANT PRODUCTION STATISTICS

Ore will be processed using crushing, grinding, and flotation technology to produce metals in a flotation concentrate. Two concentrate products will be produced, zinc concentrate and lead concentrate. Gold and silver will be recovered in both the zinc and lead concentrates.

The estimated metal recoveries in the lead and zinc concentrates are presented in the table below.

Table 22-2: Metal Recovery Factors

	Lead %	Zinc %	Gold %	Silver %
Lead Concentrate	82.5	-	71.0	82.5
Zinc Concentrate	-	82.6	4.1	4.3

Estimated life of mine lead and zinc concentrate production is presented below with the approximate metal contained.

Table 22-3: Life of Mine Concentrate Summary

4,500 MTPD Case	Tonnes (000's)	Zinc (klbs.)	Silver (kozs.)	Gold (kozs.)
Zinc Concentrate	515	596,372	15,807	15
Lead Concentrate	299	336,480	303,275	258
5,500 MTPD Case				
Zinc Concentrate	519	600,758	15,830	15
Lead Concentrate	300	337,734	303,708	258

22.3.1 Smelter Return Factors

Lead and zinc concentrates will be shipped from the site to lead and zinc smelting and refining companies. Smelter and refining treatment charges are negotiable at the time of agreement.

A smelter may impose a penalty either expressed in higher treatment charges or in metal deductions to treat concentrates that contain higher than specified quantities of certain elements. It is expected that this project will produce relatively clean concentrates that will not pose any special restrictions on smelting and refining and that the concentrates will be easily marketable.

The smelting and refining charges calculated in the financial evaluation include charges for smelting lead and zinc concentrates and refining precious metal from both the lead and zinc concentrates. Also included in these charges will be the transportation to get the concentrate from the site to the smelter. The off-site charges that will be incurred are presented in Table 22-4.

Table 22-4: Smelter Return Factors

Lead Concentrate	
Payable lead in concentrate	95.0 %
Payable gold in concentrate	95.0 %
Payable silver in concentrate	96.0 %
Lead deduction (minimum)	3.0 %
Gold deduction (oz/dmt)	0.032
Silver deduction (oz/dmt)	1.607
Treatment charge ($/tonne)	$300.00
Price Participation	
Add $0.04 for dollar increase in Pb price per metric ton	$2,000 to $2,300
Add $0.04 for dollar increase in Pb price per metric ton	>$2,300
Subtract $0.01 for dollar increase in Pb price per metric ton	$1,700 to $2,000
Subtract $0.01 for dollar increase in Pb price per metric	<$1,700
Refining charge – Au ($/oz)	$8.00
Refining charge – Ag ($/oz)	$1.50
Transportation Charges ($/wmt)	$100.00
Moisture (%)	8.0 %
Zinc Concentrate	
Payable zinc in concentrate	85.0 %
Payable gold in concentrate	85.0 %
Payable silver in concentrate	70.0 %
Zinc deduction (minimum)	8.0 %
Gold deduction (oz/dmt)	0.05
Silver deduction (oz/`dmt)	3.00
Treatment charge ($/tonne)	$191.00
Price Participation	
Add $0.05 for dollar increase in Zn price per metric ton	$2,000 to $2,300
Add $0.05 for dollar increase in Zn price per metric ton	>$2,300
Subtract $0.02 for dollar increase in Zn price per metric ton	$1,700 to $2,000
Subtract $0.02 for dollar increase in Zn price per metric	<$1,700
Refining charge – Au ($/oz)	$0.00
Refining charge – Ag ($/oz)	$0.00
Transportation Charges ($/wmt)	$100.00
Moisture (%)	8.0 %

22.4 CAPITAL EXPENDITURE

22.4.1 Initial and Expansion Capital

The base case financial indicators have been determined with 100% equity financing of the initial capital. Any acquisition cost or expenditures prior to the January, 2012 have been treated as “sunk” cost and have not been included in the analysis.

The total capital carried in the financial model for new construction, expansion capital and pre-production mine development is shown in the table below.

Table 22-5: Initial and Expansion Capital Summary

Period	4500 MTPD Case Amount	5500 MTPD Case Amount
Year 2012	$181,760	$181,760
Year 2013	$98,924	$99,274
Year 2014	$12,002	$11,448
Year 2015	$5,499	$5,247
Year 2016	$2,482	$3,439
Year 2017	$625	$600
Year 2018	$1,335	$1,335
Year 2019	$2,968	$19,032
Year 2020	$400	$16,464
Total	$305,995	$338,599

22.4.2 Sustaining Capital

A schedule of capital cost expenditures during the production period was estimated and included in the financial analysis under the category of sustaining capital. This capital will be expended during a 16 year period, starting in Year 1 and ending in Year 16. Table 22-6 shows the annual sustaining capital expenditures.

Table 22-6: Sustaining Capital Summary

Period	4500 MTPD Case	5500 MTPD Case
Year 2013	$10,450	$10,450
Year 2014	$12,556	$12,556
Year 2015	$14,037	$14,037
Year 2016	$15,622	$13,490
Year 2017	$18,660	$17,484
Year 2018	$11,841	$13,422
Year 2019	$16,850	$19,456
Year 2020	$7,190	$7,190
Year 2021	$5,557	$5,557
Year 2022	$6,942	$6,942
Year 2023	$5,313	$5,313
Year 2024	$6,375	$6,375
Year 2025	$6,284	$6,284
Year 2026	$6,821	$6,821
Year 2027	$3,059	$3,059
Year 2028	$766	$766
Total	$148,324	$149,204

22.4.3 Working Capital

A 60 day delay of receipt of revenue from sales is used for accounts receivables. A delay of payment for accounts payable of 30 days is also incorporated into the financial model. In addition, working capital allowance of $20.0 million for plant consumable inventory is estimated in year -1, year 1, year 4 and year 5. All the working capital is recaptured at the end of the mine life and the final value of these accounts is $0.

22.4.4 Salvage Value

No allowance for salvage value has been included in the cash flow analysis.

22.4.5 Revenue

Annual revenue is determined by applying estimated metal prices to the annual payable metal estimated for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The revenue is the gross value of payable metals sold before treatment charges and transportation charges. Metal sales prices used in the evaluation are as follows:

Zinc	$0.90/pound
Lead	$0.95/pound
Silver	$25.00/ounce

Gold $1,300.00/ounce

22.4.6 Operating Cost

Cash Operating Cost includes mine operations, process plant operations, general administrative cost, smelting and refining charges and shipping charges. The table below shows the estimated operating cost by area per metric ton of ore processed.

Table 22-7: Operating Cost

Operating Cost	4500 MTPD Case $/ore tonne	5500 MTPD Case $/ore tonne
Mine	$29.03	$27.22
Process Plant	$19.54	$20.16
General Administration	$6.67	$6.87
Smelting/Refining Treatment	$23.96	$23.96
Total Operating Cost	$79.20	$78.20

22.4.7 Total Cash Cost

The average Total Cash Cost over the life of the mine is estimated to be 92.50 and $91.49 per metric ton of ore processed for the 4500 MTPD case and 5,500 MTPD case, respectively. Total Cash Cost is the Total Cash Operating Cost plus royalties, property tax and tailings infrastructure and reclamation and closure costs.

22.4.7.1 Royalty

A royalty payment is based on 4.5% of the mineral content at market prices starting the first year of production. The life of mine royalty payments is estimated to be $388.3 and $389.0 million for the 4,500 MTPD case and 5,500 MTPD case, respectively.

22.4.7.2 Tailings Infrastructure and Reclamation & Closure

An allowance for the cost of reclamation and closure of the property has been included in the cash flow projection. Yearly concurrent reclamation and yearly tailing facility liner advancement are included in the cost. Years 2 – 18 shows an allowance of $0.15 per total material mined. An allowance of $4.0 million is included for end of mine life closure for both cases.

22.4.7.3 Depreciation

Depreciation is calculated using the straight line method starting with first year of production. The initial capital was depreciated using a 10 year life and the sustaining capital was depreciated using an 8 year life. The last year of production is the catch-up year if the assets are not fully depreciated by that time.

22.4.8 Taxation

22.4.8.1 Corporate Income Tax

The Escobal project is evaluated with a 7% gross corporate tax based on the metal value shipped outside of the country based on revenues returned from the smelters. The tax rate was increased by the Guatemalan congress in February 2012. It was assumed that all the metals were shipped out of the country.

Corporate income taxes paid is estimated to be $604.1 and $604.0 million for the life of the mine for the 4,500 MTPD case and 5,500 MTPD case, respectively.

22.4.9 Project Financing

For the purposes of this study it is assumed the project will be all equity financed. Therefore, no interest payments on debt are considered.

22.4.10 Net Income After Tax

Net Income after Tax is approximately $4.8 billion for the life of the mine for both cases.

22.4.11 NPV and IRR

The base case economic analysis indicates that the project has an NPV at 5% discount rate of $2.94 billion and $2.99 billion and an Internal Rate of Return (IRR) of 68.1% and 68.3% with a payback period of 1.5 years for the 4,500 MTPD case and the 5,500 MTPD case, respectively. This compares to the 3,500 MTPD case from the previous Preliminary Economic Assessment (29 November 2010) adjusted to the base case metal prices used in this new Preliminary Economic Assessment of an NPV at 5% discount rate of $ 2.76 billion an IRR of 69.8% and payback of 1.1 years.

Factors effecting the change in the NPV include increasing the royalties from 1.5% to 4.5%, increasing taxes from 5% of revenues net of smelter costs to 7%, considering 2011 costs sunk for all three cases, one less year of discounting, increased silver refining costs, increased power demand, higher cement prices and the effect of the two expansion cases. The cumulative effects can be seen in Figure 22-1. Sensitivity analyses for the 4,500 MTPD and 5,500 MTPD cases are presented in Table 22-8 and Table 22-9. Detailed financial models for each case are shown in Table 22-10 and Table 22-11.

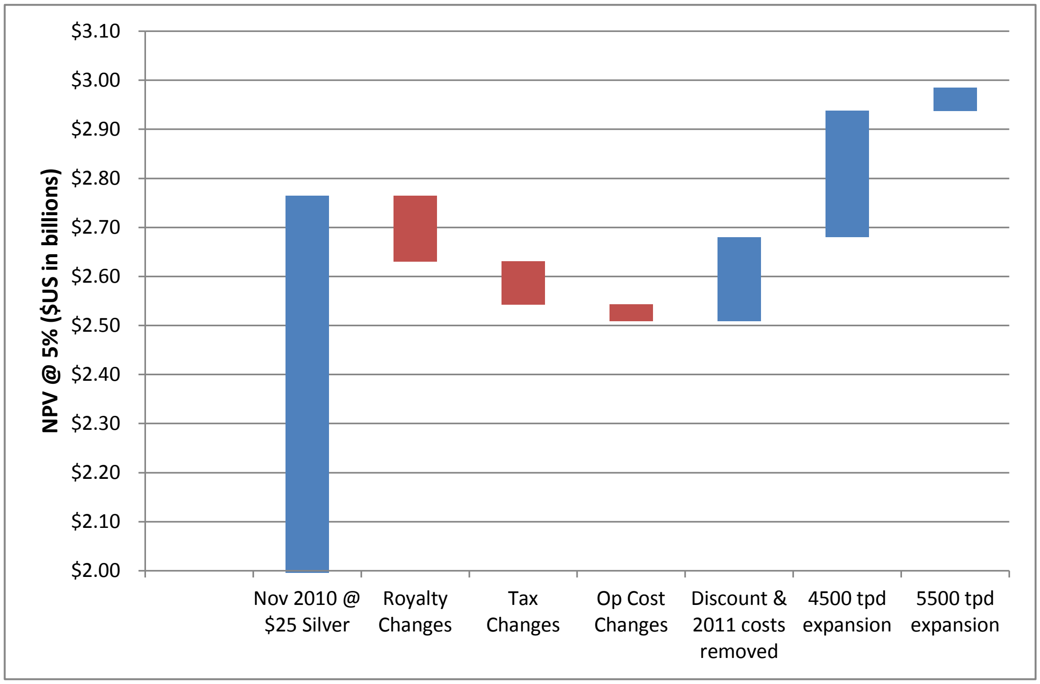


Figure 22-1: Changes in NPV @ 5% Due to Changes in Cost Structure

Table 22-8: 4500 MTPD Case – Sensitivity Analysis

Sensitivities - After Taxes					
Change in Metal Prices	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
20%	$6,339,750	$3,902,357	$2,568,846	83.6%	0.9
10%	$5,576,018	$3,420,385	$2,240,478	76.0%	1.1
0%	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
-10%	$4,048,553	$2,456,439	$1,583,741	59.8%	1.4
-20%	$3,284,821	$1,974,466	$1,255,373	51.2%	1.7
Change in Operating Cost	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
20%	$4,482,749	$2,736,312	$1,777,190	64.5%	1.3
10%	$4,647,517	$2,837,362	$1,844,650	66.3%	1.3
0%	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
-10%	$4,977,054	$3,039,462	$1,979,569	69.8%	1.2
-20%	$5,141,822	$3,140,512	$2,047,029	71.6%	1.1
Change in Initial Capital	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
20%	$4,760,314	$2,887,240	$1,861,664	60.2%	1.4
10%	$4,786,300	$2,912,826	$1,886,887	63.9%	1.3
0%	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
-10%	$4,838,271	$2,963,998	$1,937,332	72.9%	1.1
-20%	$4,864,257	$2,989,584	$1,962,555	78.7%	1.0
Change in Recovery	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
2.0%	$4,950,739	$3,025,832	$1,971,676	69.5%	1.2
1.0%	$4,881,512	$2,982,122	$1,941,893	68.8%	1.2
0.0%	$4,812,286	$2,938,412	$1,912,109	68.1%	1.2
-1.0%	$4,743,059	$2,894,702	$1,882,326	67.3%	1.2
-2.0%	$4,673,832	$2,850,992	$1,852,543	66.6%	1.3

Table 22-9: 5500 MTPD Case - Sensitivity Analysis

Sensitivities - After Taxes					
Change in Metal Prices	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
20%	$6,345,786	$3,963,392	$2,627,389	83.7%	1.0
10%	$5,580,592	$3,474,281	$2,292,385	76.1%	1.1
0%	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
-10%	$4,050,205	$2,496,061	$1,622,378	60.2%	1.4
-20%	$3,285,011	$2,006,950	$1,287,374	51.6%	1.7
Change in Operating Cost	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
20%	$4,490,725	$2,784,266	$1,822,839	64.9%	1.3
10%	$4,653,062	$2,884,719	$1,890,110	66.6%	1.3
0%	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
-10%	$4,977,736	$3,085,623	$2,024,653	70.0%	1.2
-20%	$5,140,073	$3,186,075	$2,091,924	71.7%	1.2
Change in Initial Capital	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
20%	$4,763,427	$2,933,999	$1,906,936	60.5%	1.4
10%	$4,789,413	$2,959,585	$1,932,159	64.1%	1.3
0%	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
-10%	$4,841,384	$3,010,757	$1,982,604	73.1%	1.2
-20%	$4,867,370	$3,036,343	$2,007,827	78.8%	1.1
Change in Recovery	NPV @ 0%	NPV @ 5%	NPV @ 10%	IRR%	Payback
Base Case	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
2.0%	$4,954,100	$3,073,868	$2,018,138	69.7%	1.2
1.0%	$4,884,749	$3,029,520	$1,987,760	69.0%	1.2
0.0%	$4,815,399	$2,985,171	$1,957,381	68.3%	1.2
-1.0%	$4,746,048	$2,940,822	$1,927,003	67.6%	1.3
-2.0%	$4,676,698	$2,896,473	$1,896,625	66.9%	1.3

Table 22-10: Detail Financial Model (4,500 MTPD)

		2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Base Case	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Mining Operations																										
Ore																										
Beginning Inventory (kt)	29,872	29,872	29,861	29,345	27,922	26,479	25,031	23,355	21,747	20,141	18,629	17,028	15,358	13,593	11,985	10,285	8,561	6,831	4,917	3,080	1,745	680	45	(0)	(0)	(0)
Mined (kt)	29,872	11	516	1,423	1,443	1,448	1,676	1,608	1,606	1,512	1,601	1,669	1,765	1,608	1,700	1,724	1,730	1,913	1,837	1,335	1,065	634	45	-	-	-
Ending Inventory (kt)	-	29,861	29,345	27,922	26,479	25,031	23,355	21,747	20,141	18,629	17,028	15,358	13,593	11,985	10,285	8,561	6,831	4,917	3,080	1,745	680	45	(0)	(0)	(0)	(0)
Gold Grade (g/t)	0.38	0.19	0.46	0.42	0.38	0.37	0.36	0.40	0.51	0.36	0.31	0.29	0.40	0.41	0.37	0.38	0.42	0.40	0.40	0.34	0.29	0.26	0.10	-	-	-
Silver Grade (g/t)	383.18	325.88	383.92	457.81	464.72	521.70	481.03	495.87	460.46	504.04	423.97	384.93	402.01	377.57	320.38	326.17	275.23	267.16	280.43	275.63	262.67	242.15	276.30	-	-	-
Lead Grade (%)	0.62%	0.01%	0.94%	0.86%	0.82%	0.73%	0.62%	0.56%	0.56%	0.59%	0.55%	0.47%	0.59%	0.59%	0.51%	0.51%	0.61%	0.65%	0.76%	0.64%	0.55%	0.44%	0.22%	0.00%	0.00%	0.00%
Zinc Grade (%)	1.10%	1.15%	1.54%	1.48%	1.32%	1.17%	0.99%	0.88%	0.92%	0.99%	0.93%	0.91%	1.03%	1.02%	0.98%	0.93%	1.27%	1.40%	1.38%	1.10%	0.95%	0.88%	0.51%	0.00%	0.00%	0.00%
Contained Gold (kozs)	364	0	8	19	18	17	19	21	27	17	16	16	23	21	20	21	24	24	24	15	10	5	0	-	-	-
Contained Silver (kozs)	368,008	113	6,370	20,944	21,558	24,291	25,926	25,636	23,769	24,510	21,827	20,660	22,816	19,516	17,511	18,082	15,309	16,435	16,565	11,831	8,997	4,939	402	-	-	-
Contained Lead (klbs)	408,073	1	10,729	26,836	25,932	23,267	22,836	19,820	19,795	19,752	19,491	17,266	22,984	20,745	19,257	19,541	23,434	27,310	30,804	18,877	13,034	6,143	219	-	-	-
Contained Zinc (klbs)	722,512	273	17,478	46,414	42,127	37,321	36,761	31,330	32,737	33,051	32,754	33,572	39,973	36,212	36,615	35,323	48,437	58,856	55,842	32,312	22,281	12,332	511	-	-	-
Waste																										
Beginning Inventory(kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Material Mined (kt)	29,872	11	516	1,423	1,443	1,448	1,676	1,608	1,606	1,512	1,601	1,669	1,765	1,608	1,700	1,724	1,730	1,913	1,837	1,335	1,065	634	45	-	-	-
Waste to Ore Ratio	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Process Plant Operations																										
Concentrator				3,600	4,000	4,000	4,110	4,178	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	2,455				
Beginning Ore Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined Ore to Concentrator (kt)	29,827	-	319	1,314	1,460	1,460	1,500	1,525	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	896	-	-	-	-
Mined Ore - Processed (kt)	29,827	-	319	1,314	1,460	1,460	1,500	1,525	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	1,643	896	-	-	-	-
Ending Ore Inventory		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Grade (g/t)	0.38	-	0.45	0.42	0.39	0.37	0.36	0.39	0.48	0.39	0.32	0.30	0.38	0.41	0.38	0.38	0.41	0.40	0.40	0.37	0.33	0.28	-	-	-	-
Silver Grade (g/t)	383.34	-	382.73	457.81	454.62	510.22	485.30	493.51	468.86	495.17	437.62	393.82	400.61	382.94	333.55	327.86	288.67	272.80	277.83	276.64	268.83	249.94	-	-	-	-
Lead Grade (%)	0.62%	0.00%	0.92%	0.86%	0.83%	0.75%	0.64%	0.58%	0.56%	0.59%	0.56%	0.48%	0.57%	0.58%	0.53%	0.52%	0.59%	0.63%	0.72%	0.68%	0.61%	0.49%	0.00%	0.00%	0.00%	0.00%
Zinc Grade (%)	1.10%	0.00%	1.53%	1.48%	1.36%	1.20%	1.02%	0.92%	0.92%	0.97%	0.94%	0.92%	1.01%	1.02%	0.99%	0.94%	1.19%	1.34%	1.37%	1.22%	1.07%	0.94%	0.00%	0.00%	0.00%	0.00%
Contained Gold (kozs)	363	-	5	18	18	17	17	19	26	21	17	16	20	22	20	20	22	21	21	19	17	8	-	-	-	-
Contained Silver (kozs)	367,606	-	3,930	19,341	21,340	23,950	23,404	24,197	24,759	26,148	23,110	20,797	21,155	20,222	17,614	17,313	15,244	14,406	14,671	14,609	14,196	7,200	-	-	-	-
Contained Lead (klbs)	407,855	-	6,505	24,782	26,732	24,017	21,054	19,381	20,409	21,188	20,222	17,533	20,724	21,085	19,127	18,731	21,351	22,894	25,954	24,485	22,029	9,651	-	-	-	-
Contained Zinc (klbs)	722,000	-	10,761	42,861	43,710	38,666	33,893	30,775	33,403	35,265	33,917	33,181	36,505	36,882	35,693	34,115	42,956	48,538	49,450	44,232	38,703	18,496	-	-	-	-
Zinc Concentrate																										
Recovery Zinc (%)	82.60%	0.00%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%
Recovery Gold (%)	4.10%	0.00%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%
Recovery Silver (%)	4.30%	0.00%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%
Zinc Concentrate (kt)	515	-	7,679	31	31	28	24	22	24	25	24	24	26	26	25	24	31	35	35	32	28	13	-	-	-	-
Zinc Concentrate - Grade (Zn %)	52.50%	0.00%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%
Recovered Zinc (klbs)	596,372	-	8,888	35,403	36,105	31,938	27,996	25,420	27,591	29,129	28,015	27,407	30,153	30,464	29,483	28,179	35,481	40,092	40,845	36,536	31,969	15,277	-	-	-	-
Recovered Gold (kozs)	15	-	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	-	-	-	-
Recovered Silver (kozs)	15,807	-	169	832	918	1,030	1,006	1,040	1,065	1,124	994	894	910	870	757	744	655	619	631	628	610	310	-	-	-	-
Lead Concentrate																										
Recovery Lead (%)	82.50%	0.00%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%
Recovery Gold (%)	71.00%	0.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%
Recovery Silver (%)	82.50%	0.00%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%
Lead Concentrate (kt)	299	-	5	18	20	18	15	14	15	16	15	13	15	15	14	14	16	17	19	18	16	7	-	-	-	-
Lead Concentrate - Grade (Pb%)	51.00%	0.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%
Recovered Lead (klbs)	336,480	-	5,367	20,445	22,054	19,814	17,370	15,990	16,837	17,480	16,683	14,465	17,098	17,395	15,780	15,453	17,614	18,888	21,412	20,200	18,174	7,962	-	-	-	-
Recovered Gold (kozs)	258	-	3	12	13	12	12	14	18	15	12	11	14	15	14	14	15	15	15	14	12	6	-	-	-	-
Recovered Silver (kozs)	303,275	-	3,242	15,956	17,605	19,758	19,308	19,962	20,426	21,573	19,066	17,157	17,453	16,683	14,532	14,283	12,576	11,885	12,104	12,052	11,712	5,940	-	-	-	-
			3,411	16,788	18,523	20,788	20,315	21,003	21,491	22,697	20,059	18,052	18,363	17,553	15,289	15,028	13,232	12,504	12,735	12,681	12,322	6,249				
Payable Metals																										
Zinc Concentrate																										
Payable Zinc (klbs)	505,497	-	7,534	30,008	30,603	27,072	23,730	21,547	23,386	24,690	23,746	23,231	25,558	25,822	24,990	23,885	30,075	33,983	34,621	30,968	27,097	12,949	-	-	-	-
Payable Gold (kozs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Payable Silver (kozs)	11,065	-	118	582	642	721	704	728	745	787	696	626	637	609	530	521	459	434	442	440	427	217	-	-	-	-
Lead Concentrate																										
Payable Lead (klbs)	316,687	-	5,051	19,242	20,757	18,649	16,348	15,049	15,847	16,452	15,702	13,614	16,092	16,372	14,852	14,544	16,578	17,777	20,152	19,012	17,105	7,494	-	-	-	-
Payable Gold (kozs)	245	-	3	12	12	12	12	13	17	14	11	11	14	15	13	13	15	14	14	13	12	5	-	-	-	-
Payable Silver (kozs)	291,144	-	3,113	15,318	16,901	18,968	18,536	19,164	19,609	20,710	18,303	16,471	16,755	16,016	13,950	13,712	12,073	11,410	11,620	11,570	11,243	5,702	-	-	-	-

Income Statement ($000)																										
Zinc ($/lb.)	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.80	0.80	0.80	0.80
Lead ($/lb)	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.70	0.70	0.70	0.70
Gold ($/oz)	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	1,300.00	825.00	825.00	825.00	825.00
Silver ($/oz)	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	13.00	13.00	13.00	13.00
Revenues																										
Zinc Concentrate - Zn	454,947	-	6,781	27,007	27,543	24,364	21,357	19,392	21,048	22,221	21,372	20,908	23,003	23,240	22,491	21,496	27,067	30,584	31,159	27,872	24,388	11,654	-	-	-	-
Zinc Concentrate - Au	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Zinc Concentrate - Ag	276,623	-	2,957	14,554	16,058	18,022	17,612	18,208	18,631	19,677	17,390	15,650	15,919	15,217	13,255	13,028	11,471	10,841	11,040	10,993	10,683	5,418	-	-	-	-
Lead Concentrate - Pb	300,853	-	4,798	18,280	19,719	17,716	15,531	14,297	15,054	15,629	14,916	12,933	15,287	15,553	14,109	13,817	15,749	16,888	19,145	18,061	16,250	7,119	-	-	-	-
Lead Concentrate - Au	318,723	-	4,094	15,403	16,198	15,286	15,157	16,824	22,371	18,059	14,891	13,745	17,673	18,853	17,382	17,414	19,064	18,554	18,559	17,079	15,076	7,040	-	-	-	-
Lead Concentrate - Ag	7,278,597	-	77,813	382,945	422,532	474,201	463,402	479,092	490,235	517,740	457,575	411,776	418,871	400,397	348,758	342,802	301,826	285,239	290,494	289,256	281,084	142,557	-	-	-	-
Total Revenues	8,629,743	-	96,444	458,190	502,050	549,590	533,058	547,813	567,340	593,326	526,145	475,012	490,753	473,261	415,995	408,558	375,177	362,106	370,397	363,262	347,480	173,788	-	-	-	-
Operating Cost	29.03																									
Mining	865,998	-	15,931	42,582	43,013	43,129	48,059	46,581	46,529	44,517	46,436	47,908	49,980	46,575	48,570	49,095	49,219	53,181	51,536	40,683	32,890	19,585	-	-	-	-
Process Plant	582,883	-	9,741	29,080	30,502	30,502	30,635	30,718	31,108	31,108	31,108	31,108	31,108	31,108	31,108	31,108	31,108	31,108	31,108	31,108	31,108	17,306	-	-	-	-
General Administration	198,801	-	4,581	9,162	9,293	9,817	9,817	9,960	10,050	10,471	10,471	10,471	10,471	10,471	10,471	10,471	10,471	10,471	10,471	10,471	10,471	10,471	-	-	-	-
Treatment & Refining Charges																										
Zinc Concentrates																										
Treatment Charges	98,251	-	1,464	5,833	5,948	5,262	4,612	4,188	4,546	4,799	4,615	4,515	4,968	5,019	4,857	4,642	5,845	6,605	6,729	6,019	5,267	2,517	-	-	-	-
Gold Refining Charges	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Silver Refining Charges	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Transportation	55,648	-	829	3,303	3,369	2,980	2,612	2,372	2,575	2,718	2,614	2,557	2,814	2,843	2,751	2,629	3,311	3,741	3,811	3,409	2,983	1,426	-	-	-	-
Penalties	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Lead Concentrates																										
Treatment Charges	89,763	-	1,432	5,454	5,883	5,286	4,634	4,266	4,492	4,663	4,450	3,859	4,561	4,640	4,210	4,122	4,699	5,039	5,712	5,389	4,848	2,124	-	-	-	-
Gold Refining Charges	1,961	-	25	95	100	94	93	104	138	111	92	85	109	116	107	107	117	114	114	105	93	43	-	-	-	-
Silver Refining Charges	436,716	-	4,669	22,977	25,352	28,452	27,804	28,746	29,414	31,064	27,455	24,707	25,132	24,024	20,926	20,568	18,110	17,114	17,430	17,355	16,865	8,553	-	-	-	-
Transportation	32,321	-	515	1,964	2,118	1,903	1,668	1,536	1,617	1,679	1,602	1,389	1,642	1,671	1,516	1,484	1,692	1,814	2,057	1,940	1,746	765	-	-	-	-
Penalties	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Operating Cost	2,362,341	-	39,188	120,449	125,579	127,425	129,935	128,469	130,467	131,130	128,844	126,598	130,784	126,466	124,515	124,228	124,572	129,187	128,968	116,479	106,270	62,790	-	-	-	-
Royalty	388,338	-	4,340	20,619	22,592	24,732	23,988	24,652	25,530	26,700	23,677	21,376	22,084	21,297	18,720	18,385	16,883	16,295	16,668	16,347	15,637	7,820	-	-	-	-
Property Tax	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Salvage Value	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reclamation & Closure	8,377	-	77	213	216	217	251	241	241	227	240	250	265	241	255	259	260	287	276	200	160	4,000	-	-	-	-
Total Production Cost	2,759,057	-	43,605	141,281	148,388	152,374	154,174	153,362	156,238	158,057	152,760	148,224	153,133	148,004	143,489	142,871	141,714	145,769	145,911	133,026	122,067	74,610	-	-	-	-
Operating Income	5,870,686	-	52,838	316,909	353,662	397,217	378,884	394,451	411,102	435,270	373,385	326,788	337,620	325,257	272,506	265,686	233,463	216,336	224,486	230,235	225,413	99,178	-	-	-	-
Initial Capital Depreciation	305,995		28,068	29,269	29,819	30,067	30,129	30,263	30,559	30,599	30,599	30,599	2,531	1,331	781	533	470	377	-	-	-	-	-	-	-	-
Sustaining Capital Depreciation	148,324		1,306	2,876	4,630	6,583	8,916	10,396	12,502	13,401	12,789	12,087	10,997	9,841	8,294	7,667	5,943	5,140	4,445	3,577	2,913	4,021	-	-	-	-
Total Depreciation	454,319	-	29,375	32,144	34,449	36,650	39,045	40,659	43,061	44,000	43,389	42,687	13,528	11,172	9,075	8,199	6,413	5,516	4,445	3,577	2,913	4,021	-	-	-	-
Net Income After Depreciation	5,416,368	-	23,464	284,765	319,213	360,567	339,839	353,793	368,041	391,269	329,996	284,101	324,092	314,085	263,431	257,487	227,050	210,820	220,041	226,658	222,500	95,157	-	-	-	-
Tax Loss Carry Forward Applied	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Taxable Income	5,416,368	-	23,464	284,765	319,213	360,567	339,839	353,793	368,041	391,269	329,996	284,101	324,092	314,085	263,431	257,487	227,050	210,820	220,041	226,658	222,500	95,157	-	-	-	-
Taxes at 7% of Gross Revenues	604,082	-	6,751	32,073	35,143	38,471	37,314	38,347	39,714	41,533	36,830	33,251	34,353	33,128	29,120	28,599	26,262	25,347	25,928	25,428	24,324	12,165	-	-	-	-
Net Income After Taxes	4,812,286	-	16,713	252,691	284,070	322,095	302,525	315,446	328,327	349,737	293,166	250,850	289,740	280,957	234,311	228,888	200,788	185,472	194,113	201,230	198,176	82,992	-	-	-	-

Cash Flow																										
Operating Income	$ 5,870,686	$ -	$ 52,838	$ 316,909	$ 353,662	$ 397,217	$ 378,884	$ 394,451	$ 411,102	$ 435,270	$ 373,385	$ 326,788	$ 337,620	$ 325,257	$ 272,506	$ 265,686	$ 233,463	$ 216,336	$ 224,486	$ 230,235	$ 225,413	$ 99,178	$ -	$ -	$ -	$ -
Working Capital																										
Account Recievable (60 days)	$ -	$ -	$ (15,854)	$ (59,465)	$ (7,210)	$ (7,815)	$ 2,718	$ (2,426)	$ (3,210)	$ (4,272)	$ 11,044	$ 8,405	$ (2,588)	$ 2,875	$ 9,413	$ 1,223	$ 5,487	$ 2,149	$ (1,363)	$ 1,173	$ 2,594	$ 28,552	$ 28,568	$ -	$ -	$ -
Accounts Payable (30 days)	$ -	$ -	$ 3,221	$ 6,679	$ 422	$ 152	$ 206	$ (120)	$ 164	$ 55	$ (188)	$ (185)	$ 344	$ (355)	$ (160)	$ (24)	$ 28	$ 379	$ (18)	$ (1,026)	$ (839)	$ (3,574)	$ (5,161)	$ -	$ -	$ -
Inventory - Parts, Supplies	$ -	$ (5,000)	$ (10,000)	$ -	$ -	$ (2,500)	$ (2,500)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 20,000	$ -	$ -	$ -	$ -	$ -
Total Working Capital	$ (0)	(5,000)	(22,633)	(52,786)	(6,788)	(10,163)	424	(2,546)	(3,046)	(4,217)	10,856	8,221	(2,244)	2,521	9,253	1,199	5,516	2,528	(1,381)	147	21,755	24,978	23,407	-	-	-
Capital Expenditures																										
Initial Capital																										
Mine	$ 45,662	$ 35,211	$ 10,450	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ 200,454	$ 131,542	$ 68,912	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Expansion Capital	$ 46,136	$ 5,844	$ 14,981	$ 12,002	$ 5,499	$ 2,482	$ 625	$ 1,335	$ 2,968	$ 400	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Owners Cost	$ 13,743	$ 9,162	$ 4,581	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital																										
Mining	$ 124,586	$ -	$ 10,450	$ 11,677	$ 12,279	$ 13,863	$ 16,901	$ 10,083	$ 15,091	$ 5,432	$ 3,799	$ 5,184	$ 3,555	$ 4,616	$ 4,526	$ 5,063	$ 1,300	$ 766	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ 23,738	$ -	$ -	$ 879	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Capital Expenditures	$ 454,319	$ 181,760	$ 109,375	$ 24,558	$ 19,536	$ 18,104	$ 19,285	$ 13,176	$ 19,817	$ 7,590	$ 5,557	$ 6,942	$ 5,313	$ 6,375	$ 6,284	$ 6,821	$ 3,059	$ 766	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Cash Flow before Taxes	$ 5,416,368	$ (186,760)	$ (79,169)	$ 239,565	$ 327,337	$ 368,950	$ 360,023	$ 378,729	$ 388,239	$ 423,462	$ 378,683	$ 328,067	$ 330,064	$ 321,403	$ 275,475	$ 260,064	$ 235,920	$ 218,098	$ 223,105	$ 230,382	$ 247,168	$ 124,157	$ 23,407	$ -	$ -	$ -
Cummulative Cash Flow before Taxes		$ (186,760)	$ (265,929)	$ (26,364)	$ 300,973	$ 669,923	$ 1,029,946	$ 1,408,675	$ 1,796,914	$ 2,220,377	$ 2,599,060	$ 2,927,126	$ 3,257,190	$ 3,578,592	$ 3,854,067	$ 4,114,131	$ 4,350,051	$ 4,568,149	$ 4,791,254	$ 5,021,636	$ 5,268,804	$ 5,392,960	$ 5,416,368	$ 5,416,368	$ 5,416,368	$ 5,416,368
			1.0	1.0	0.1	-	-	-	-	-	-															
Taxes																										
Income Taxes	$ 604,082	$ -	$ 6,751	$ 32,073	$ 35,143	$ 38,471	$ 37,314	$ 38,347	$ 39,714	$ 41,533	$ 36,830	$ 33,251	$ 34,353	$ 33,128	$ 29,120	$ 28,599	$ 26,262	$ 25,347	$ 25,928	$ 25,428	$ 24,324	$ 12,165	$ -	$ -	$ -	$ -
Cash Flow after Taxes	$ 4,812,286	$ (186,760)	$ (85,920)	$ 207,491	$ 292,194	$ 330,479	$ 322,709	$ 340,382	$ 348,526	$ 381,929	$ 341,853	$ 294,816	$ 295,711	$ 288,274	$ 246,355	$ 231,465	$ 209,657	$ 192,751	$ 197,177	$ 204,954	$ 222,844	$ 111,991	$ 23,407	$ -	$ -	$ -
Cummulative Cash Flow after Taxes		$ (186,760)	$ (272,680)	$ (65,188)	$ 227,006	$ 557,484	$ 880,193	$ 1,220,575	$ 1,569,101	$ 1,951,030	$ 2,292,883	$ 2,587,698	$ 2,883,409	$ 3,171,684	$ 3,418,039	$ 3,649,504	$ 3,859,161	$ 4,051,912	$ 4,249,089	$ 4,454,043	$ 4,676,887	$ 4,788,878	$ 4,812,286	$ 4,812,286	$ 4,812,286	$ 4,812,286
				1.0	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Economic Indicators before Taxes																										
NPV @ 0%	0%	5,416,368																								
NPV @ 5%	5%	3,323,035																								
NPV @ 10%	10%	2,176,297																								
IRR		74.9%																								
Payback		2.1																								
Economic Indicators after Taxes																										
NPV @ 0%	0%	4,812,286																								
NPV @ 5%	5%	2,938,412																								
NPV @ 10%	10%	1,912,109																								
IRR		68.1%																								
Payback	Years	1.2																								

Table 22-11: Detail Financial Model (5,500 MTPD)

		2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Base Case	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Mining Operations																										
Ore																										
Beginning Inventory (kt)	29,924	29,924	29,913	29,397	27,975	26,524	25,098	23,356	21,735	20,204	18,619	16,877	15,058	13,328	11,694	9,885	7,966	5,975	3,878	1,765	400	71	0	0	0	0
Mined (kt)	29,924	11	516	1,423	1,450	1,427	1,742	1,621	1,531	1,584	1,742	1,819	1,730	1,634	1,809	1,919	1,992	2,097	2,113	1,365	329	71	-	-	-	-
Ending Inventory (kt)	-	29,913	29,397	27,975	26,524	25,098	23,356	21,735	20,204	18,619	16,877	15,058	13,328	11,694	9,885	7,966	5,975	3,878	1,765	400	71	0	0	0	0	0
Gold Grade (g/t)	0.38	0.19	0.46	0.42	0.38	0.37	0.34	0.41	0.53	0.36	0.30	0.32	0.41	0.39	0.33	0.34	0.35	0.39	0.40	0.44	0.24	0.18	-	-	-	-
Silver Grade (g/t)	382.64	325.88	383.92	457.81	462.31	529.64	471.33	493.57	464.88	499.44	407.26	376.63	398.89	378.90	315.32	319.75	281.12	266.86	273.20	249.52	302.20	315.47	-	-	-	-
Lead Grade (%)	0.62%	0.01%	0.94%	0.86%	0.81%	0.74%	0.60%	0.56%	0.60%	0.61%	0.54%	0.47%	0.59%	0.58%	0.48%	0.48%	0.53%	0.66%	0.74%	0.70%	0.75%	0.76%	0.00%	0.00%	0.00%	0.00%
Zinc Grade (%)	1.10%	1.15%	1.54%	1.48%	1.32%	1.19%	0.97%	0.88%	0.99%	1.08%	0.94%	0.92%	1.04%	1.04%	0.94%	0.86%	1.05%	1.42%	1.36%	1.21%	1.01%	0.98%	0.00%	0.00%	0.00%	0.00%
Contained Gold (kozs)	363	0	8	19	18	17	19	21	26	18	17	19	23	20	19	21	22	27	27	19	3	0	-	-	-	-
Contained Silver (kozs)	368,130	113	6,370	20,944	21,558	24,291	26,391	25,719	22,889	25,442	22,808	22,029	22,184	19,907	18,344	19,723	18,000	17,988	18,559	10,950	3,197	722	-	-	-	-
Contained Lead (klbs)	409,374	1	10,729	26,836	25,932	23,267	23,178	19,911	20,126	21,242	20,738	18,693	22,321	20,741	19,265	20,449	23,218	30,666	34,388	21,006	5,469	1,198	-	-	-	-
Contained Zinc (klbs)	727,309	273	17,478	46,414	42,127	37,321	37,428	31,356	33,360	37,578	36,122	36,736	39,839	37,565	37,451	36,175	46,099	65,434	63,438	36,284	7,299	1,534	-	-	-	-
Waste																										
Beginning Inventory(kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Material Mined (kt)	29,924	11	516	1,423	1,450	1,427	1,742	1,621	1,531	1,584	1,742	1,819	1,730	1,634	1,809	1,919	1,992	2,097	2,113	1,365	329	71	-	-	-	-
Waste to Ore Ratio	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Process Plant Operations																										
Concentrator				3,500	4,000	4,100	4,247	4,247	4,500	4,658	4,959	5,000	5,137	5,137	5,137	5,137	5,479	5,479	5,479	3,817	902					
Beginning Ore Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined Ore to Concentrator (kt)	29,924	-	319	1,278	1,460	1,497	1,550	1,550	1,643	1,700	1,810	1,825	1,875	1,875	1,875	1,875	2,000	2,000	2,000	1,393	329	71	-	-	-	-
Mined Ore - Processed (kt)	29,924	-	319	1,278	1,460	1,497	1,550	1,550	1,643	1,700	1,810	1,825	1,875	1,875	1,875	1,875	2,000	2,000	2,000	1,393	329	71	-	-	-	-
Ending Ore Inventory		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Grade (g/t)	0.38	-	0.45	0.42	0.38	0.39	0.34	0.39	0.50	0.39	0.31	0.32	0.40	0.39	0.32	0.34	0.35	0.40	0.40	0.44	0.24	0.18	0.18	-	-	-
Silver Grade (g/t)	382.64	-	382.73	457.81	462.31	506.88	476.15	489.69	471.30	493.50	420.25	381.74	396.84	379.84	309.45	321.06	277.33	265.87	273.50	250.01	302.20	315.47	315.47	-	-	-
Lead Grade (%)	0.62%	0.00%	0.92%	0.86%	0.81%	0.77%	0.63%	0.57%	0.59%	0.60%	0.55%	0.48%	0.57%	0.58%	0.47%	0.48%	0.53%	0.68%	0.74%	0.70%	0.75%	0.76%	0.76%	0.00%	0.00%	0.00%
Zinc Grade (%)	1.10%	0.00%	1.53%	1.48%	1.32%	1.25%	1.01%	0.91%	0.97%	1.05%	0.96%	0.92%	1.03%	1.04%	0.93%	0.85%	1.07%	1.45%	1.36%	1.21%	1.01%	0.98%	0.98%	0.00%	0.00%	0.00%
Contained Gold (kozs)	363	-	5	17	18	19	17	20	26	21	18	19	24	23	20	21	22	26	26	20	3	0	-	-	-	-
Contained Silver (kozs)	368,130	-	3,930	18,803	21,701	24,388	23,728	24,403	24,888	26,973	24,456	22,398	23,923	22,898	18,655	19,354	17,833	17,096	17,587	11,198	3,197	722	-	-	-	-
Contained Lead (klbs)	409,374	-	6,505	24,093	26,104	25,421	21,402	19,568	21,365	22,647	21,934	19,145	23,653	23,790	19,614	20,031	23,502	29,798	32,665	21,467	5,469	1,198	-	-	-	-
Contained Zinc (klbs)	727,309	-	10,761	41,670	42,406	41,243	34,587	31,012	35,024	39,459	38,208	37,048	42,570	43,074	38,420	34,965	47,150	63,865	59,888	37,129	7,299	1,534	-	-	-	-
Zinc Concentrate																										
Recovery Zinc (%)	82.60%	0.00%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%	82.60%
Recovery Gold (%)	4.10%	0.00%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%	4.10%
Recovery Silver (%)	4.30%	0.00%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%	4.30%
Zinc Concentrate (kt)	519	-	7.679	30	30	29	25	22	25	28	27	26	30	31	27	25	34	46	43	26	5	1	-	-	-	-
Zinc Concentrate - Grade (Zn %)	52.50%	0.00%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%	52.50%
Recovered Zinc (klbs)	600,758	-	8,888	34,420	35,027	34,067	28,569	25,616	28,929	32,593	31,560	30,602	35,163	35,579	31,735	28,881	38,946	52,752	49,468	30,669	6,029	1,267	-	-	-	-
Recovered Gold (kozs)	15	-	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	-	-	-	-
Recovered Silver (kozs)	15,830	-	169	809	933	1,049	1,020	1,049	1,070	1,160	1,052	963	1,029	985	802	832	767	735	756	482	137	31	-	-	-	-
Lead Concentrate																										
Recovery Lead (%)	82.50%	0.00%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%
Recovery Gold (%)	71.00%	0.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%	71.00%
Recovery Silver (%)	82.50%	0.00%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%	82.50%
Lead Concentrate (kt)	300	-	5	18	19	19	16	14	16	17	16	14	17	17	14	15	17	22	24	16	4	1	-	-	-	-
Lead Concentrate - Grade (Pb%)	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%	51.00%
Recovered Lead (klbs)	337,734	-	5,367	19,877	21,536	20,972	17,657	16,143	17,626	18,684	18,096	15,795	19,514	19,627	16,182	16,526	19,389	24,583	26,949	17,710	4,512	988	-	-	-	-
Recovered Gold (kozs)	258	-	3	12	13	13	12	14	19	15	13	13	17	17	14	15	16	18	18	14	2	0	-	-	-	-
Recovered Silver (kozs)	303,708	-	3,242	15,513	17,903	20,120	19,576	20,132	20,533	22,253	20,176	18,479	19,736	18,891	15,390	15,967	14,712	14,104	14,509	9,238	2,638	596	-	-	-	-
Payable Metals																										
Zinc Concentrate																										
Payable Zinc (klbs)	509,214	-	7,534	29,175	29,690	28,876	24,216	21,713	24,521	27,626	26,750	25,938	29,805	30,157	26,899	24,480	33,011	44,714	41,930	25,995	5,110	1,074	-	-	-	-
Payable Gold (kozs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Payable Silver (kozs)	11,081	-	118	566	653	734	714	735	749	812	736	674	720	689	562	583	537	515	529	337	96	22	-	-	-	-
Lead Concentrate																										
Payable Lead (klbs)	317,867	-	5,051	18,708	20,269	19,739	16,618	15,194	16,590	17,585	17,031	14,866	18,366	18,473	15,230	15,554	18,249	23,137	25,364	16,668	4,247	930	-	-	-	-
Payable Gold (kozs)	245	-	3	12	12	13	12	13	18	14	12	12	16	16	13	14	15	17	18	13	2	0	-	-	-	-
Payable Silver (kozs)	291,559	-	3,113	14,892	17,187	19,315	18,793	19,327	19,712	21,362	19,369	17,740	18,947	18,135	14,775	15,329	14,123	13,540	13,929	8,869	2,532	572	-	-	-	-

Income Statement ($000)																										
Zinc ($/lb.)	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ 0.90	$ -	$ -	$ -	$ -
Lead ($/lb)	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ -	$ -	$ -	$ -
Gold ($/oz)	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00	$ 1,300.00
Silver ($/oz)	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00
Revenues																										
Zinc Concentrate - Zn	$ 458,292	$ -	$ 6,781	$ 26,257	$ 26,721	$ 25,988	$ 21,794	$ 19,541	$ 22,069	$ 24,864	$ 24,075	$ 23,345	$ 26,824	$ 27,141	$ 24,209	$ 22,032	$ 29,710	$ 40,243	$ 37,737	$ 23,396	$ 4,599	$ 967	$ -	$ -	$ -	$ -
Zinc Concentrate - Au	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Zinc Concentrate - Ag	$ 277,018	$ -	$ 2,957	$ 14,150	$ 16,330	$ 18,352	$ 17,856	$ 18,363	$ 18,729	$ 20,297	$ 18,403	$ 16,855	$ 18,002	$ 17,230	$ 14,038	$ 14,564	$ 13,419	$ 12,865	$ 13,234	$ 8,426	$ 2,406	$ 544	$ -	$ -	$ -	$ -
Lead Concentrate - Pb	$ 301,974	$ -	$ 4,798	$ 17,772	$ 19,255	$ 18,752	$ 15,787	$ 14,434	$ 15,760	$ 16,706	$ 16,180	$ 14,123	$ 17,448	$ 17,549	$ 14,469	$ 14,776	$ 17,336	$ 21,980	$ 24,096	$ 15,835	$ 4,034	$ 884	$ -	$ -	$ -	$ -
Lead Concentrate - Au	$ 318,643	$ -	$ 4,094	$ 14,975	$ 15,685	$ 16,267	$ 15,053	$ 17,163	$ 23,055	$ 18,557	$ 15,834	$ 16,244	$ 21,167	$ 20,573	$ 17,150	$ 18,175	$ 19,664	$ 22,452	$ 22,768	$ 17,271	$ 2,144	$ 354	$ -	$ -	$ -	$ -
Lead Concentrate - Ag	$ 7,288,980	$ -	$ 77,813	$ 372,308	$ 429,675	$ 482,882	$ 469,820	$ 483,176	$ 492,792	$ 534,061	$ 484,220	$ 443,488	$ 473,666	$ 453,373	$ 369,364	$ 383,214	$ 353,085	$ 338,503	$ 348,216	$ 221,717	$ 63,305	$ 14,302	$ -	$ -	$ -	$ -
Total Revenues	$ 8,644,907	$ -	$ 96,444	$ 445,463	$ 507,666	$ 562,241	$ 540,309	$ 552,678	$ 572,404	$ 614,484	$ 558,712	$ 514,053	$ 557,106	$ 535,867	$ 439,229	$ 452,760	$ 433,214	$ 436,043	$ 446,050	$ 286,645	$ 76,489	$ 17,051	$ -	$ -	$ -	$ -
Operating Cost	27.22																									
Mining	$ 814,564	$ -	$ 15,703	$ 41,530	$ 42,032	$ 41,596	$ 40,245	$ 45,142	$ 43,511	$ 44,479	$ 47,355	$ 48,766	$ 47,132	$ 45,387	$ 48,587	$ 50,579	$ 51,912	$ 53,829	$ 54,128	$ 40,470	$ 10,014	$ 2,167	$ -	$ -	$ -	$ -
Process Plant	$ 603,297	$ -	$ 9,741	$ 28,959	$ 30,502	$ 30,623	$ 30,801	$ 30,801	$ 31,108	$ 33,559	$ 36,766	$ 36,815	$ 36,981	$ 36,981	$ 36,981	$ 36,981	$ 37,396	$ 37,396	$ 37,396	$ 30,862	$ 9,925	$ 2,725	$ -	$ -	$ -	$ -
General Administration	$ 205,509	$ -	$ 4,581	$ 9,162	$ 9,162	$ 9,817	$ 9,947	$ 10,139	$ 10,139	$ 10,471	$ 10,677	$ 11,072	$ 11,125	$ 11,305	$ 11,305	$ 11,305	$ 11,305	$ 11,753	$ 11,753	$ 11,753	$ 9,577	$ 9,162	$ -	$ -	$ -	$ -
Treatment & Refining Charges																										
Zinc Concentrates																										
Treatment Charges	$ 98,973	$ -	$ 1,464	$ 5,671	$ 5,771	$ 5,612	$ 4,707	$ 4,220	$ 4,766	$ 5,370	$ 5,199	$ 5,042	$ 5,793	$ 5,862	$ 5,228	$ 4,758	$ 6,416	$ 8,691	$ 8,150	$ 5,053	$ 993	$ 209	$ -	$ -	$ -	$ -
Gold Refining Charges	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Silver Refining Charges	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Transportation	$ 56,057	$ -	$ 829	$ 3,212	$ 3,268	$ 3,179	$ 2,666	$ 2,390	$ 2,699	$ 3,041	$ 2,945	$ 2,855	$ 3,281	$ 3,320	$ 2,961	$ 2,695	$ 3,634	$ 4,922	$ 4,616	$ 2,862	$ 563	$ 118	$ -	$ -	$ -	$ -
Penalties	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Lead Concentrates																										
Treatment Charges	$ 90,114	$ -	$ 1,432	$ 5,304	$ 5,746	$ 5,596	$ 4,711	$ 4,307	$ 4,703	$ 4,985	$ 4,828	$ 4,214	$ 5,207	$ 5,237	$ 4,318	$ 4,409	$ 5,173	$ 6,559	$ 7,191	$ 4,725	$ 1,204	$ 264	$ -	$ -	$ -	$ -
Gold Refining Charges	$ 1,961	$ -	$ 25	$ 92	$ 97	$ 100	$ 93	$ 106	$ 142	$ 114	$ 97	$ 100	$ 130	$ 127	$ 106	$ 112	$ 121	$ 138	$ 140	$ 106	$ 13	$ 2	$ -	$ -	$ -	$ -
Silver Refining Charges	$ 437,339	$ -	$ 4,669	$ 22,338	$ 25,781	$ 28,973	$ 28,189	$ 28,991	$ 29,567	$ 32,044	$ 29,053	$ 26,609	$ 28,420	$ 27,202	$ 22,162	$ 22,993	$ 21,185	$ 20,310	$ 20,893	$ 13,303	$ 3,798	$ 858	$ -	$ -	$ -	$ -
Transportation	$ 32,441	$ -	$ 515	$ 1,909	$ 2,069	$ 2,015	$ 1,696	$ 1,551	$ 1,693	$ 1,795	$ 1,738	$ 1,517	$ 1,874	$ 1,885	$ 1,554	$ 1,587	$ 1,862	$ 2,361	$ 2,589	$ 1,701	$ 433	$ 95	$ -	$ -	$ -	$ -
Penalties	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Operating Cost	$ 2,340,255	-	38,960	118,176	124,427	127,510	123,055	127,646	128,329	135,858	138,659	136,991	139,944	137,305	133,202	135,419	139,004	145,960	146,854	110,835	36,520	15,600	-	-	-	-
Royalty	$ 389,021	$ -	$ 4,340	$ 20,046	$ 22,845	$ 25,301	$ 24,314	$ 24,870	$ 25,758	$ 27,652	$ 25,142	$ 23,132	$ 25,070	$ 24,114	$ 19,765	$ 20,374	$ 19,495	$ 19,622	$ 20,072	$ 12,899	$ 3,442	$ 767	$ -	$ -	$ -	$ -
Property Tax	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Salvage Value	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation & Closure	$ 8,478	$ 2	$ 77	$ 213	$ 218	$ 214	$ 261	$ 243	$ 230	$ 238	$ 261	$ 273	$ 259	$ 245	$ 271	$ 288	$ 299	$ 314	$ 317	$ 205	$ 49	$ 4,000	$ -	$ -	$ -	$ -
Total Production Cost	$ 2,737,754	$ 2	$ 43,377	$ 138,436	$ 147,489	$ 153,025	$ 147,630	$ 152,760	$ 154,317	$ 163,747	$ 164,062	$ 160,396	$ 165,274	$ 161,664	$ 153,238	$ 156,081	$ 158,798	$ 165,896	$ 167,243	$ 123,939	$ 40,011	$ 20,367	$ -	$ -	$ -	$ -
Operating Income	$ 5,907,153	$ (2)	$ 53,066	$ 307,027	$ 360,177	$ 409,216	$ 392,679	$ 399,918	$ 418,087	$ 450,737	$ 394,649	$ 353,658	$ 391,833	$ 374,202	$ 285,991	$ 296,679	$ 274,417	$ 270,147	$ 278,807	$ 162,706	$ 36,477	$ (3,317)	$ -	$ -	$ -	$ -
Initial Capital Depreciation	$ 338,599		$ 28,103	$ 29,248	$ 29,773	$ 30,117	$ 30,177	$ 30,310	$ 32,213	$ 33,860	$ 33,860	$ 33,860	$ 5,756	$ 4,612	$ 4,087	$ 3,743	$ 3,683	$ 5,196	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital Depreciation	$ 149,204		$ 1,306	$ 2,876	$ 4,630	$ 6,317	$ 8,502	$ 10,180	$ 12,612	$ 13,511	$ 12,899	$ 12,197	$ 11,107	$ 10,217	$ 8,818	$ 7,992	$ 5,943	$ 5,140	$ 4,445	$ 3,577	$ 2,913	$ 4,021	$ -	$ -	$ -	$ -
Total Depreciation	$ 487,803	$ -	$ 29,410	$ 32,124	$ 34,403	$ 36,433	$ 38,679	$ 40,490	$ 44,825	$ 47,371	$ 46,759	$ 46,057	$ 16,863	$ 14,829	$ 12,904	$ 11,736	$ 9,626	$ 10,336	$ 4,445	$ 3,577	$ 2,913	$ 4,021	$ -	$ -	$ -	$ -
Net Income After Depreciation	$ 5,419,350	(2)	23,656	$ 274,903	$ 325,773	$ 372,782	$ 354,000	$ 359,427	$ 373,262	$ 403,366	$ 347,890	$ 307,600	$ 374,969	$ 359,373	$ 273,086	$ 284,944	$ 264,791	$ 259,811	$ 274,362	$ 159,129	$ 33,564	$ (7,338)	$ -	$ -	$ -	$ -
Tax Loss Carry Forward Applied	$ (2)		-	(2)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Taxable Income	$ 5,419,348	$ (2)	$ 23,656	$ 274,901	$ 325,773	$ 372,782	$ 354,000	$ 359,427	$ 373,262	$ 403,366	$ 347,890	$ 307,600	$ 374,969	$ 359,373	$ 273,086	$ 284,944	$ 264,791	$ 259,811	$ 274,362	$ 159,129	$ 33,564	$ (7,338)	$ -	$ -	$ -	$ -
Taxes at 5% of Gross Revenues	$ 603,950	-	6,751	31,182	35,537	39,357	37,822	38,687	40,068	43,014	39,110	35,984	38,997	37,511	30,746	31,693	30,325	30,523	31,224	20,065	5,354	-	-	-	-	-
Net Income After Taxes	$ 4,815,399	(2)	16,905	243,719	290,237	333,425	316,179	320,740	333,194	360,352	308,781	271,617	335,972	321,863	242,340	253,250	234,466	229,288	243,138	139,064	28,210	(7,338)	-	-	-	-

Cash Flow																										
Operating Income	$ 5,907,153	$ (2)	$ 53,066	$ 307,027	$ 360,177	$ 409,216	$ 392,679	$ 399,918	$ 418,087	$ 450,737	$ 394,649	$ 353,658	$ 391,833	$ 374,202	$ 285,991	$ 296,679	$ 274,417	$ 270,147	$ 278,807	$ 162,706	$ 36,477	$ (3,317)	$ -	$ -	$ -	$ -
Working Capital																										
Account Recievable (60 days)	$ -	$ -	$ (15,854)	$ (57,373)	$ (10,225)	$ (8,971)	$ 3,605	$ (2,033)	$ (3,243)	$ (6,917)	$ 9,168	$ 7,341	$ (7,077)	$ 3,491	$ 15,886	$ (2,224)	$ 3,213	$ (465)	$ (1,645)	$ 26,204	$ 34,546	$ 9,771	$ 2,803	$ -	$ -	$ -
Accounts Payable (30 days)	$ -	$ -	$ 3,202	$ 6,511	$ 514	$ 253	$ (366)	$ 377	$ 56	$ 619	$ 230	$ (137)	$ 243	$ (217)	$ (337)	$ 182	$ 295	$ 572	$ 73	$ (2,960)	$ (6,108)	$ (1,719)	$ (1,282)	$ -	$ -	$ -
Inventory - Parts, Supplies	$ -	$ (5,000)	$ (10,000)	$ -	$ -	$ (2,500)	$ (2,500)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 20,000	$ -	$ -	$ -	$ -	$ -
Total Working Capital	$ (0)	(5,000)	(22,652)	(50,862)	(9,711)	(11,218)	739	(1,656)	(3,187)	(6,298)	9,398	7,204	(6,834)	3,275	15,548	(2,042)	3,508	107	(1,572)	23,243	48,438	8,051	1,521	-	-	-
Capital Expenditures																										
Initial Capital																										
Mine	$ 45,662	$ 35,211	$ 10,450	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ 200,454	$ 131,542	$ 68,912	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Expansion Capital	$ 78,740	$ 5,844	$ 15,331	$ 11,448	$ 5,247	$ 3,439	$ 600	$ 1,335	$ 19,032	$ 16,464	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Owners Cost	$ 13,743	$ 9,162	$ 4,581	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital																										
Mining	$ 125,466	$ -	$ 10,450	$ 11,677	$ 12,279	$ 11,732	$ 15,725	$ 11,664	$ 17,698	$ 5,432	$ 3,799	$ 5,184	$ 3,555	$ 4,616	$ 4,526	$ 5,063	$ 1,300	$ 766	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ 23,738	$ -	$ -	$ 879	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ 1,758	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Capital Expenditures	$ 487,803	$ 181,760	$ 109,725	$ 24,004	$ 19,285	$ 16,929	$ 18,084	$ 14,757	$ 38,488	$ 23,654	$ 5,557	$ 6,942	$ 5,313	$ 6,375	$ 6,284	$ 6,821	$ 3,059	$ 766	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Cash Flow before Taxes	$ 5,419,350	$ (186,761)	$ (79,310)	$ 232,161	$ 331,180	$ 381,069	$ 375,334	$ 383,505	$ 376,413	$ 420,784	$ 398,490	$ 353,919	$ 379,685	$ 371,102	$ 295,255	$ 287,816	$ 274,866	$ 269,487	$ 277,235	$ 185,949	$ 84,916	$ 4,734	$ 1,521	$ -	$ -	$ -
Cummulative Cash Flow before Taxes		$ (186,761)	$ (266,071)	$ (33,910)	$ 297,270	$ 678,339	$ 1,053,673	$ 1,437,178	$ 1,813,591	$ 2,234,375	$ 2,632,865	$ 2,986,784	$ 3,366,470	$ 3,737,572	$ 4,032,827	$ 4,320,643	$ 4,595,508	$ 4,864,995	$ 5,142,230	$ 5,328,180	$ 5,413,095	$ 5,417,829	$ 5,419,350	$ 5,419,350	$ 5,419,350	$ 5,419,350
				1.0	0.1	-	-	-	-	-	-															
Taxes																										
Income Taxes	$ 603,952	$ -	$ 6,751	$ 31,184	$ 35,537	$ 39,357	$ 37,822	$ 38,687	$ 40,068	$ 43,014	$ 39,110	$ 35,984	$ 38,997	$ 37,511	$ 30,746	$ 31,693	$ 30,325	$ 30,523	$ 31,224	$ 20,065	$ 5,354	$ -	$ -	$ -	$ -	$ -
Cash Flow after Taxes	$ 4,815,399	$ (186,761)	$ (86,061)	$ 200,977	$ 295,644	$ 341,712	$ 337,513	$ 344,817	$ 336,345	$ 377,770	$ 359,381	$ 317,936	$ 340,688	$ 333,591	$ 264,509	$ 256,122	$ 244,541	$ 238,964	$ 246,012	$ 165,884	$ 79,561	$ 4,734	$ 1,521	$ -	$ -	$ -
Cummulative Cash Flow after Taxes		$ (186,761)	$ (272,822)	$ (71,845)	$ 223,798	$ 565,511	$ 903,023	$ 1,247,840	$ 1,584,185	$ 1,961,955	$ 2,321,335	$ 2,639,271	$ 2,979,959	$ 3,313,551	$ 3,578,059	$ 3,834,182	$ 4,078,723	$ 4,317,687	$ 4,563,698	$ 4,729,582	$ 4,809,144	$ 4,813,878	$ 4,815,399	$ 4,815,399	$ 4,815,399	$ 4,815,399
				1.0	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Economic Indicators before Taxes																										
NPV @ 0%	0%	$ 5,419,350																								
NPV @ 5%	5%	$ 3,375,006																								
NPV @ 10%	10%	$ 2,226,717																								
IRR		75.1%																								
Payback		1.1																								
Economic Indicators after Taxes																										
NPV @ 0%	0%	$ 4,815,399																								
NPV @ 5%	5%	$ 2,985,171																								
NPV @ 10%	10%	$ 1,957,381																								
IRR		68.3%																								
Payback	Years	1.2																								

23 ADJACENT PROPERTIES

There are no properties immediately adjacent to the Project that are at the same stage of development.

This report focuses on the areas of the Escobal Project that contain resources. There are additional zones of mineralization within the Oasis concession however, that have been drilled; in 2009 five holes were drilled on the San Juan Bosco prospect six kilometers west of Escobal and three holes were drilled on the Morales prospect seven kilometers north of the Escobal vein in 2010-2011. Ongoing exploration continues on a regional level throughout Tahoe's other concession.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 MINING

This Preliminary Economic Assessment evaluates the potential economic viability of mineral resources at Tahoe Resources' Escobal project and includes material classified as Inferred mineral resources in the analysis, as permitted by Section 2.3(3) of National Instrument 43-101.

> ***NI 43-101 Required Disclosure:*** *The preliminary assessment is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary assessment will be realized. The basis for the PEA is the Indicated mineral resources and Inferred mineral resources, the effective date of which is 23 January 2012. There is no pre-feasibility or feasibility study with respect to the Escobal Project.*

The use of the term "ore" in the following discussion is to differentiate between mineralized material (including dilution) above an economic cutoff grade and waste rock; there is no inference of mineral reserves.

24.1.1 Cut-Off Grade

The use of cut-off grade or cut-off value is basically a defensive tactic to exclude unprofitable material from the production stream. The cut-off value calculation used to determine the minable portion of the resource at Escobal is predicated on the assumption that mine production will feed the mill at capacity throughout the mine life. Production of a discretionary increment of material will extend the life of the operation and therefore increase the total amount of fixed costs generated over the life of the operation. Production of the discretionary increment defers the realization of production from other increments. All costs that are incremental with production must therefore be covered in the cut-off value calculation. Costs in the cut-off value calculation include the variable and fixed costs directly related to ore production, expensed stope access development, smelting, refining, and concentrate transportation, general and administrative costs directly related to production, royalties, and project costs related to production and the plant facilities that do not have a measurable pay-back. Costs excluded from the basis include exploration, capitalized development costs, capital infrastructure costs, in mine projects having a measurable economic benefit, and non-cash charges.

Sustaining capital costs scheduled after the mine commences commercial production are excluded from the cut-off value cost basis as these costs are not incremental to a specific unit of production but rather common to large portions of the mineral deposit. Once the mine plan was completed, the net present value at a zero discount rate was calculated for each major area of the mine to insure the cut-off value is appropriate, allowing sustaining capital investments to reach pay-back. Table 24-1 lists the costs and other parameters utilized to calculate the Escobal cut off-value.

Table 24-1: Cut-Off Value

Mining Cost per tonne ore	$29.03
Processing Cost per tonne ore	$19.54
G&A Cost per tonne ore	$ 6.67
Smelting and Refining	$20.42
Concentrate Freight	$ 3.54
Royalties	$ 13.02
Total Cash Operating Cost per tonne ore	$92.22
Mill Recovery	87%
Smelter Payable	94%
Mining Recovery	95%
Silver Equivalent Cut-Off Grade	148 g/t

24.1.2 Mineral Resources for Mine Planning

The summary of Escobal resources shown in Table 24-2 is the basis for developing the portion of the resources available for production during the life of the mine. Indicated and Inferred resource blocks were plotted on long section and bench plans and used to evaluate mining method options. The primary goals in selecting the appropriate mining method were safe, complete extraction of the resource in the most productive possible way. Long-hole stoping was selected as the optimum mining method based on vein geometry, geomechanical properties, and productivity considerations. Design criteria were established for opening sizes for production and development excavations, excavation productivities, ground support including paste backfill, ventilation, pumping, equipment and other mine systems. Data available to provide the basis for the design criteria were collected during the exploration programs at Escobal since 2007, including data provided by Entre Mares based on experiences at its Marlin Mine in Guatemala, and the experience and databases of the M3 and Tahoe technical teams.

Table 24-2: Escobal Mineral Resources

Resource Classification	Tonnes (M)	Silver (g/t)	Gold (g/t)	Lead (%)	Zinc (%)	Silver (Moz)	Gold (koz)	Lead (kt)	Zinc (kt)
Indicated	27.1	422	0.43	0.71	1.28	367.5	373	192	347
Inferred	4.6	254	0.59	0.34	0.66	36.7	85	15	30

Individual production stopes and development headings were laid out on long section and plan. Primary development and stope access heading sizes were determined by the size of the production equipment selected, the ventilation requirements during development, and ground conditions. Stope sizes and maximum spans open prior to backfilling were determined by the geo-mechanical aspects in each stope. Stope and development productivities were determined based on geo-mechanical properties and unit operation estimated for each type of excavation and each specific stope. Costs were developed utilizing comparative analysis and adjusting data collected from similar mines and unit operations. Costs were also developed using first principal estimation techniques and compared to the experience based estimates. The final detailed cost estimates are a combination of experienced based and first principal based estimates.

Stope limits were located inside resource blocks having an estimated silver equivalent grade below the cut-off value and immediately adjacent to blocks of estimated gold grade equal to or greater than the cut-off value. The volume of material in the resource blocks having grades lower than the cut-off grade but included in the production plan represent the dilution which will be mined in the plan. This technique allows dilution to be modeled directly in the mine plan. The estimate of mineable resources is therefore based on an actual engineered stope design and practical mining experience rather than utilizing the traditional method of assigning an arbitrary numerical estimate of tonnes and grade for dilution which is then added to the resource estimate to determine the final production tonnes and grade.

Tons and grade, including dilution, were calculated for each stope throughout the entire ore body and used to complete a production schedule. The production schedule was detailed by month for the first two years of ore development and production and annually for the remainder of the mine life. The sequence of production from the individual stopes is designed to mine the highest grade stopes as early as possible in the mine life but is constrained by the development schedule. Development and stoping rates were scheduled to produce a sequence that maintains a minimum of six active ore development headings, three active long-hole fronts, and a duplicate set of workplaces developed and available at all times. Maintaining this number of available work places allows for on-going ore and waste development and backfill cycles and insures the ability of the mine to produce at the mill capacity of 1.28 million tonnes per year for the 3,500 MTPD case, 1.63 million metric tons per year for the 4,500 MTPD case, and 2.0 million tonner per year for the 5,500 MTPD case.

Once the initial mine plan was completed the cost estimate utilized to calculate the initial cut-off value was reviewed and adjusted to reflect a more detailed mine plan and design. The mine plan that is the basis for this PEA is the result of this iterative planning process and while optimization will continue throughout the final design and the mine life, the current mine plan has been shown to be both feasible and realistic. The 3,500 tonne per day mine plan extracts 22,651,000 metric tons at a silver grade of 415 g/t, gold grade of 0.47 g/t, 0.72% lead and 1.23% zinc. This total includes dilution of 1.422 MM tonnes at an average grade of 56 g/t silver, 0.15 g/t gold, 0.11% lead, and 0.24% zinc. The 4,500 and 5,500 MTPD mine plans extract 29.9 million metric tons at a silver grade of 383 g/t, gold grade of 0.38 g/t, 0.62% lead and 1.10% zinc. This total includes dilution of 4.7 million tonnes at an average grade of 71 g/t silver, 0.12 g/t gold, 0.10% lead, and 0.22% zinc.

24.1.3 Underground Mining

The deposit will be accessed through two main portals, called the East and West portals. These primary declines will access the Central Zone. A third primary ramp will be driven into the East Zone from the Central Zone. The three primary ramps will connect to a system of secondary access spirals and attack ramps to access stoping areas in the East Zone and the East Extension area. Footwall laterals will be driven parallel to the vein on 25 meter vertical centers and will be accessed from the primary ramps. Stopes in the Central Zone and East Extension area will be accessed from the footwall laterals. The access ramps are located nominally 75 and 150 m from the vein. There are also accesses leading to ventilation ingress and exhaust raises. Internal ventilation raises will be driven between the various ramps and accesses. The mining methods selected are transverse and longitudinal long-hole stoping. Development on vein to establish over-cut and undercut drifts for stoping will be excavated 5 meters wide by 5 meters high. Stopes located where the vein width exceeds 15 meters will be excavated utilizing the transverse stoping method. The longitudinal stoping method will be utilized where vein widths are less than 15 meters. Filtered tails from the process plant will be combined with cement and water to make a structural fill for use underground. A paste backfill plant located on the surface will produce backfill for delivery via a system of steel and HDPE pipe, installed in bore holes, into the mine for placement in the mined out stopes. Backfill will be required for all stopes for stability reasons and as a preferred place to store tailings. Resources will be hauled from the stopes and ore passes to the process plant by truck and development waste will be placed in mined stopes where possible, or trucked to a surface waste dump facility.

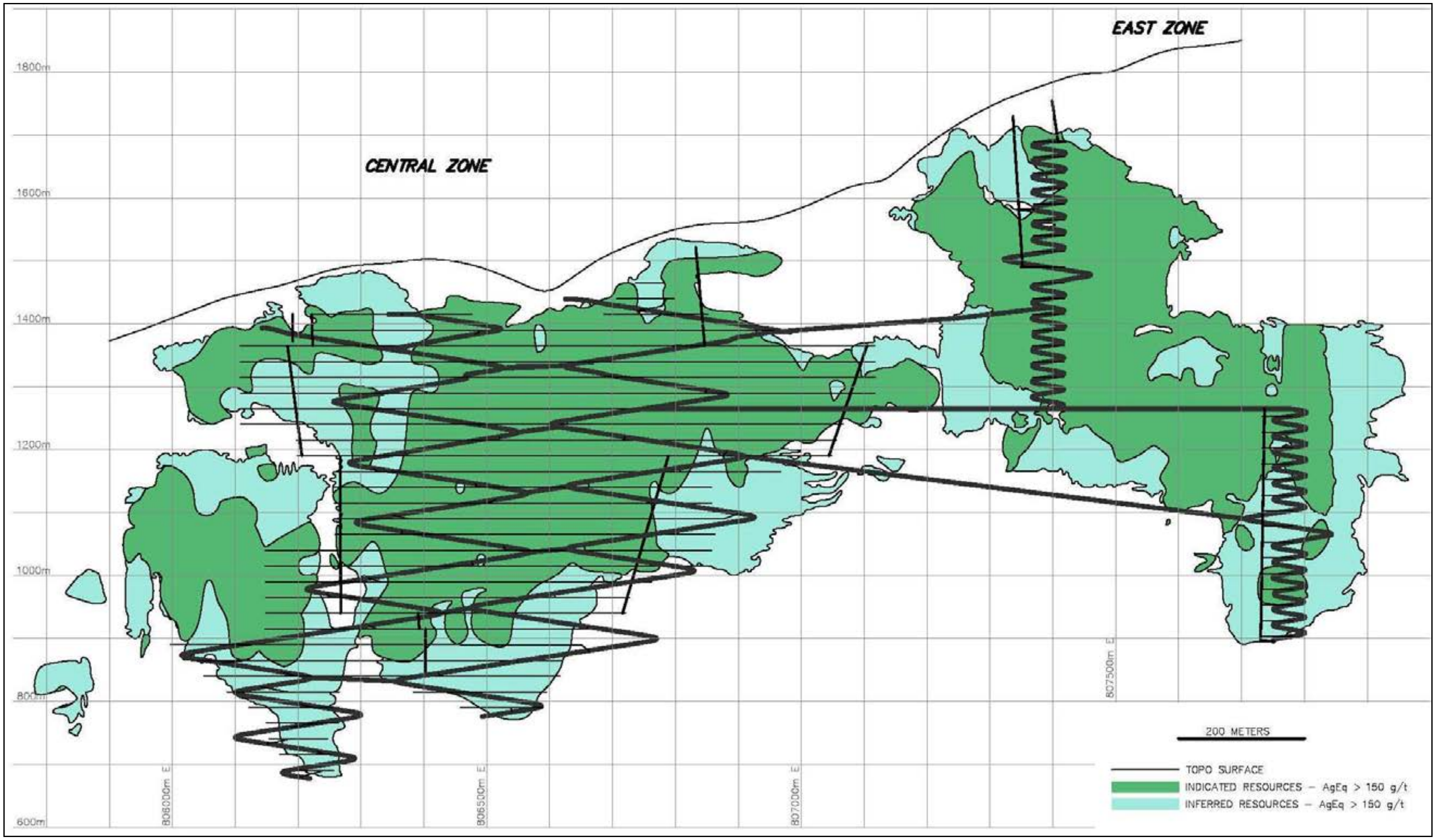


Figure 24-1: Escobal Main Ramp and Raise Layout

25 INTERPRETATION AND CONCLUSIONS

- Exploration work since 2010 has resulted in significant increase in the mineral resources of the Escobal site, leading to a new Preliminary Economic Assessment ("PEA") to analyze increased mine and plant throughput associated with extraction of the additional resources.

- The previous Preliminary Economic Assessment (November 2010) indicated that a 3,500 MTPD underground mine producing lead and zinc concentrates over a production life of 18 years is economically viable. The new Preliminary Economic Assessment demonstrates that increasing the throughput to 4,500 MTPD and/or 5,500 MTPD will enhance the economic results over the previous plan.

- The contemplated operations would provide direct employment of approximately 650 employees in Guatemala and provide a long-term revenue source to the local municipality.

- Permitting has progressed as anticipated in the previous PEA. The project has all necessary permits to develop the facilities necessary for exploitation of the mineral resources. Community support has been critical in obtaining the permits necessary to develop the project. The exploitation permit required for the extraction of the mineral resources is expected to be issued by MEM in the first half of 2012.

- The Escobal deposit holds considerable promise for successful exploitation given its size, grade, metallurgical characteristics, developed infrastructure, and the knowledge and experience of the individuals engaged in the project.

- The process plant estimate is very detailed for a PEA and estimated costs are considered to be an upper bound for the process facilities as now envisioned.

- The underground mine estimate is very detailed for a PEA and estimated costs are considered to be upper bound for the mine as envisioned in this study.

- If mining operating costs were to increase 50% from those currently estimated, the overall operating cost would increase approximately 19.6% and the project would still remain viable by interpolation of the sensitivity Table 1.20-23.

- If power costs were to double from $0.14 kWH to $.28 kWH (grid power to site generated power), operating costs would increase approximately 12.4% and the project would still remain viable by interpolation of the sensitivity Table 1.20-23.

- Escobal as defined at this point would be a relatively low environmental risk project.

- An independent verification program including a complete audit of the drill hole assay database, drill location and survey data, sample verification, sample handling and logging procedures, and QA/QC analysis support the estimation of the Escobal resource and the assignment of an Indicated classification to much of the stated resource.

- The following conclusions were drawn from the testwork conducted by McClelland Laboratories:
  - The Escobal sulfide and mixed oxide/sulfide composites did not respond particularly well to gravity concentration treatment, at an 80%-106µm feed size.
  - The Escobal sulfide ore composites responded well to conventional bulk sulfide flotation treatment for recovery of gold and silver, at an 80%-75µm feed size
  - The Escobal sulfide ore composites showed good potential for selective flotation of contained lead and zinc.
  - The Escobal mixed oxide/sulfide ore composite did not respond as well to conventional bulk sulfide flotation treatment.
  - The Escobal composites were moderately amenable to whole ore milling/cyanidation treatment, at an 80%-75µm feed size.
  - The EC08-127 composite may have displayed a moderate preg-robbing tendency during whole ore cyanidation.
  - Adding activated carbon during whole ore cyanidation (CIL) leaching generally was effective in significantly improving gold and silver recoveries.
  - Cyanidation of flotation products, including regrind/intensive cyanidation of flotation rougher concentrates, was not particularly effective in increasing overall leach recoveries, when compared to whole ore CIL/cyanidation leaching.
- The following conclusions are drawn from the testwork conducted so far by Dawson's Metallurgical:
  - The Escobal sulfide ore is amenable to selective flotation producing a clean lead concentrate with most of the silver and gold in the lead concentrate and a clean zinc concentrate with some precious metals.
  - Grinding the ore to 80 percent passing 105 microns produced enough mineral liberation suitable for the flotation process.
  - Ore floated well with normal flotation reagents such as; potassium amyl xanthate (PAX), sodium isopropyl xanthate (SIPX), copper sulfate ($CuSO_4$), zinc sulfate, Aerofloat 208 and Aerofroth X-133.
  - Very selective collectors and co-collectors can be used in the zinc circuit at lower pH to eliminate the use of lime.

- The Escobal resource estimate is considered reasonable, honors the geology, and is supported by the geologic model.

- The Escobal resource estimate is based on sufficient drill sample analytical and density measurements, detailed drill-hole lithology and alteration data, and preliminary metallurgical results, to support a classification of Indicated for much of the sulfide mineralization. The lack of metallurgical testing and some spatial uncertainty in the model has resulted in an Inferred classification for all of the oxide portions of the deposit.

26 RECOMMENDATIONS

- Based on financial and technical measures, exploration success and project advances to date it is recommended that Tahoe complete the detailed engineering and construction of the Escobal project and begin taking steps to increase mine and mill capacity to 4,500 metric tons per day.

- Based on these same factors it is recognized that positive economic benefits may be realized from expansion above 4,500 MTPD. It is recommended that Tahoe continue to explore adjacent to the know Escobal mineral resources and advance detailed engineering to further define and optimize potential mine and plant capacity beyond 4,500 MTPD.

- Presumably, the public utility company will continue with improvements to the national grid. However, this study is based on self-generation of incremental needs for power and this approach is prudent. As the grid gets refined, this additional ability to generate our power will serve as additional emergency backup.

- Baseline pre-mining hydrologic conditions have been established at the Project. Further detailed hydrologic study is needed to better understand the fracture-controlled distribution of groundwater and its effect on the contemplated mining operation. The extent and depths of the groundwater should be verified in conjunction with establishing both surface and groundwater monitoring stations in preparation for mine dewatering programs. Additional well information would be beneficial in verifying that an anomalous geothermal gradient does not exist.

- Rock mechanics investigations should be further detailed. Continued definition of the spatial distribution of rock quality will enable refinements to ground support estimates, and mine opening designs. This can be best accomplished by continuing the geotechnical mapping of ground conditions and rock quality in the East Central and West Central declines and subsequent geotechnical evaluations of the underground. RMR models of the Escobal deposit should be re-evaluated as additional information is obtained from underground definition drilling, Rock quality indices should continue to be compared to those of other mines using similar mining methods as a means to further evaluate confidence in the stope and extraction design.

- As recent and future drilling data become available, the block model should be refined to create resource estimate updates. Additional in-fill drilling will lend further confidence to the block model. Tahoe expects to commence definition drilling from underground drill platforms in mid-2012.

- The “mix design” for the paste fill to be placed underground needs final definition. Cement consumption and accompanying costs can be finalized based on the gradation of the tailings, percent fines and chemical make-up.

- Underground drilling should be utilized to test for additional mineralization on the Escobal structure along strike and dip, as well testing for parallel structures to the

Escobal vein. As nearly all of the surface exploration drilling has been oriented north-south, it is recommended that future drill campaigns include drill orientations subparallel to the Escobal vein to test for mineralized 'cross-structures' as are indicated by structural trends observed in the declines.

- Detailed design and exploitation permitting activities should continue and the project should be advanced to the feasibility study stage.

- The future drilling at the Escobal project should include more types of quality control sampling and analyses, including:

    - The continued use of suitable standards for each of the important metals;

    - The collection and analysis of field duplicates, processed and analyzed at the primary lab; and

    - The use of coarse reject or preparation duplicates, analyzed at the primary lab.

- Additional drilling to better characterize the gold mineralization within the upper levels of the East Zone is recommended in the areas where the economic viability is dominated by gold. Both the East and Central zones are open at depth and further extensional drilling is recommended.

27 REFERENCES

AMEC Americas Ltd., "Escobal Project Guatemala NI 43-101 Technical Report", prepared by Mr. Greg Kulla, 30 April, 2010.

Davies, Michael P., and Rice, Stephen, "An Alternative to Conventional Tailings Management – "Dry-Stack" Filtered Tailings", AMEC Earth and Environmental, 2004.

M3 Engineering and Technology Corporation, "Escobal Guatemala Project, NI 43-101 Technical Report Preliminary Economic Assessment", prepared under the guidance of Mr. Conrad Huss, 29 November 2010.

McClelland Laboratories, Inc., "Report on Scoping Metallurgical Testing – Escobal Drill Core Composites, MLI Job No. 3324", 20 May, 2009.

Kappes, Cassiday & Associates, "2,500 Tonne Per Day Flotation Plant and Cyanidation Plant Cost Comparisons", 22 July, 2009.

Appendix A

PEA Contributors and Professional Qualifications

CERTIFICATE of QUALIFIED PERSON

Conrad E. Huss

I, Conrad E. Huss, P.E., Ph.D., do hereby certify that:

1. I am Senior Vice President and Chairman of the Board of:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Rd., Suite 101
   Tucson, Arizona 85704
   U.S.A.

2. I graduated with a Bachelor's of Science in Mathematics and a Bachelor's of Art in English from the University of Illinois in 1963. I graduated with a Master's of Science in Engineering Mechanics from the University of Arizona in 1968. In addition, I earned a Doctor of Philosophy in Engineering Mechanics from the University of Arizona in 1970.

3. I am a Professional Engineer in good standing in the State of Arizona in the areas of Civil (No. 9648) and Structural (No. 9733) engineering. I am also registered as a professional engineer in the States of California, Illinois, Maine, Minnesota, Missouri, Montana, New Mexico, Oklahoma, Texas, Utah, and Wyoming.

4. I have worked as an engineer for a total of forty three years since my graduation from the University of Illinois. I have taught at the University level part-time for five years and as an assistant professor for one year.

5. 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 purposes of NI 43-101.

6. I am the principal author for the preparation of the technical report titled "Escobal Guatemala Project NI 43-101 Preliminary Economic Assessment" (the "Technical Report"), dated May 7, 2012, prepared for Tahoe Resources Inc.; and am responsible for Sections 1 through 3, 17 through 19, 21, 22, and 24 through 26. I have visited the project site on 1 December 2010.

7. I have prior involvement with the property that is the subject of the Technical Report. I was a contributing author of a previous Technical Report on the subject property entitled "Escobal Guatemala Project NI 43-101 Preliminary Economic Assessment" dated 29 November 2010. M3 Engineering & Technology Corporation is the EPCM contractor for the Escobal Project.

8. 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.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 7th day of May, 2012.

Conrad E. Huss

Signature of Qualified Person

Conrad E. Huss, P.E., Ph.D.

Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, Daniel Roth, P.E., do hereby certify that:

1. I am currently employed as a Civil Engineer/Project Manager at M3 Engineering & Technology Corporation located at 2051 West Sunset Road, Suite 101, Tucson, AZ, 85704.

2. I graduated with a Bachelor's of Science degree in Civil Engineering from the University of Manitoba in 1990.

3. I am a registered professional engineer in good standing in the following jurisdictions:

    - Alberta, Canada (No. 62310)
    - Ontario, Canada (No. 100156213)
    - New Mexico, USA (No. 17342)
    - Arizona, USA (No. 37319)

    I am also a member in good standing with the Society of Mining, Metallurgy and Exploration.

4. I have practiced engineering in the civil and environmental fields for 18 years and have experience in mine reclamation and impact water management.

5. 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 purposes of NI 43-101.

6. I am responsible for the review of the civil and environmental controls sections of this technical report titled *Escobal Guatemala Project NI 43-101 Preliminary Economic Assessment* for Tahoe Resources Inc. dated May 7, 2012 ("Technical Report").

7. I have prior involvement with the property that is the subject of the Technical Report. I was a contributing author of a previous Technical Report on the subject property entitled "Escobal Guatemala Project NI 43-101 Preliminary Economic Assessment" dated 29 November 2010. M3 Engineering & Technology Corporation is the EPCM contractor for the Escobal Project. I visited the Escobal project site on numerous occasions in 2010, 2011, and 2012.

8. 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.

9. I am independent of the Tahoe Resources Inc. and all their subsidiaries as defined in Section 1.5 of NI 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.

11. The Technical Report contains information relating to mineral titles, permitting, environmental issues, regulatory matters and legal agreements. I am not a legal, environmental or regulatory professional, and do not offer a professional opinion regarding these issues.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated May 7, 2012.

Daniel Roth

Signature of Qualified Person

Daniel Roth

Print Name of Qualified Person

1.0 CERTIFICATE OF AUTHOR

PAUL TIETZ, C.P.G.

I, Paul Tietz,C.P.G., do hereby certify that I am currently employed as Senior Geologist for Mine Development Associates, Inc. located at 210 South Rock Blvd., Reno, Nevada 89502 and:

1. I graduated with a Bachelor of Science degree in Biology/Geology from the University of Rochester in 1977, a Master of Science degree in Geology from the University of North Carolina, Chapel Hill in 1981, and a Master of Science degree in Geological Engineering from the University of Nevada, Reno in 2004.

2. I am a Certified Professional Geologist (#11004) with the American Institute of Professional Geologists.

3. 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 purposes of NI 43-101. I am independent of Tahoe Resources, Inc., applying all of the tests in section 1.5 of National Instrument 43-101.

4. I take responsibility for Sections 10.0, 11.0, 12.0, and 14.0 of this report entitled *Escobal Guatemala Project NI 43-101 Preliminary Economic Assessment* prepared for Tahoe Resources, Inc., and dated May 7, 2012.

5. I was a co-author of a previous Technical Report on this property entitled "Escobal Guatemala Project NI 43-101 Preliminary Economic Assessment" and dated 29 November 2010. I visited the Escobal project site on September 7th through the 10th, 2010 and again on February 6th through the 9th, 2012.

6. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains the necessary technical information to make the Technical Report not misleading.

7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

8. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated May 7, 2012.

"Paul G. Tietz"

Paul G. Tietz, C.P.G.

CERTIFICATE of QUALIFIED PERSON

I, Thomas L. Drielick, P.E., do hereby certify that:

1. I am currently employed as Sr. Vice President by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Ste. 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of Michigan Technological University and received a Bachelor of Science degree in Metallurgical Engineering in 1970. I am also a graduate of Southern Illinois University and received an M.B.A. degree in 1973.

3. I am a:

   - Registered Professional Engineer in the State of Arizona (No. 22958)
   - Registered Professional Engineer in the State of Michigan (No. 6201055633)
   - Member in good standing of the Society for Mining, Metallurgy and Exploration, Inc. (No. 850920)

4. I have practiced metallurgical and mineral processing engineering and project management for 41 years. I have worked for mining and exploration companies for 18 years and for M3 Engineering and Technology, Corporation for 23 years.

5. 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 purposes of NI 43-101.

6. I am responsible for the preparation of Section 13 "Mineral Processing and Metallurgical Testing", of the technical report titled "Escobal Guatemala Project NI 43-101 Preliminary Economic Assessment," dated May 7, 2012 (the "Technical Report").

7. I have prior involvement with the property that is the subject of the Technical Report. I was a contributing author of a previous Technical Report on the subject property entitled "Escobal Guatemala Project NI 43-101 Preliminary Economic Assessment" dated 29 November 2010. M3 Engineering & Technology Corporation is the EPCM contractor for the Escobal Project.

8. 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 required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 7th day of May, 2012.

Thomas L Drielick

Signature of Qualified Person

Thomas L. Drielick

Print name of Qualified Person

Appendix B

Escobal Project – Significant Drill Intercepts

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E07-04	104.0	114.0	10.0	7.9	502	0.30	0.20	0.06
incl.	*111.1*	*112.0*	*0.9*	*0.7*	*1061*	*0.95*	*0.77*	*0.11*
E07-05	168.5	169.5	1.0	0.7	191	0.21	0.10	0.24
	176.0	178.0	2.0	1.3	388	0.87	0.19	0.45
E07-06	95.0	96.0	1.0	0.9	135	1.95	0.01	0.02
E07-07	124.0	125.0	1.0	0.7	188	0.16	0.05	0.12
	131.0	132.0	1.0	0.7	182	0.19	1.70	0.18
E07-10	88.0	91.0	3.0	2.9	238	6.14	0.03	0.06
incl.	*89.0*	*90.0*	*1.0*	*1.0*	*372*	*13.92*	*0.04*	*0.06*
	97.0	106.0	9.0	8.6	111	1.96	0.02	0.03
E07-11	72.0	75.0	3.0	1.4	175	0.16	0.06	0.07
	100.0	114.0	14.0	6.4	222	0.34	0.06	0.14
	127.0	130.0	3.0	1.4	332	0.03	0.03	0.23
E07-13	148.5	154.5	6.0	5.9	129	0.14	0.18	0.39
	166.0	181.0	15.0	14.8	517	0.30	0.55	0.96
incl.	*166.0*	*167.5*	*1.5*	*1.5*	*1473*	*0.84*	*1.42*	*3.12*
	169.0	*170.5*	*1.5*	*1.5*	*1698*	*0.78*	*0.93*	*1.70*
E07-14	58.0	59.0	1.0	0.9	258	0.31	0.17	0.16
	72.6	73.5	0.9	0.8	171	0.07	0.06	0.07
E07-15	81.0	84.0	3.0	1.6	182	0.20	0.74	1.76
	90.0	92.0	2.0	1.1	927	0.90	0.22	0.33
incl.	*91.0*	*92.0*	*1.0*	*0.5*	*1207*	*0.98*	*0.31*	*0.41*
	130.5	133.0	2.5	1.5	308	0.34	0.05	0.16
	145.0	151.0	6.0	3.7	161	0.31	0.10	0.24
	161.0	162.0	1.0	0.6	171	0.33	0.18	0.45
	165.0	166.0	1.0	0.6	146	0.16	0.24	0.55
	169.0	170.0	1.0	0.6	175	0.17	0.20	0.57
	173.0	218.5	45.5	28.0	418	0.28	0.78	1.27
E07-16	54.5	56.0	1.5	1.3	1299	0.58	0.80	0.12
	76.0	79.0	3.0	2.6	563	0.30	0.60	0.32
incl.	*78.0*	*79.0*	*1.0*	*0.9*	*1034*	*0.63*	*1.10*	*0.63*
	87.0	88.0	1.0	0.9	89	1.58	0.01	0.08
E07-17	10.0	11.0	1.0	0.7	162	0.11	0.39	0.16
	121.0	126.0	5.0	3.5	247	0.28	0.25	0.30
	135.0	158.0	23.0	16.0	576	0.41	0.72	0.87
incl.	*149.0*	*150.0*	*1.0*	*0.7*	*1164*	*0.76*	*0.78*	*0.85*
	155.0	*156.0*	*1.0*	*0.7*	*1654*	*1.58*	*1.30*	*1.56*
E07-19	13.5	15.0	1.5	1.3	253	0.03	0.12	0.02
	101.8	103.3	1.6	1.3	1775	0.33	0.87	0.20
E07-20	130.5	132.0	1.5	0.8	196	0.22	0.07	0.17
	157.0	158.0	1.0	0.5	536	0.85	0.18	0.61
	173.0	176.0	3.0	1.5	172	0.33	0.12	0.39

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E07-21	98.0	100.5	2.5	2.3	226	0.71	0.16	0.29
E07-25	144.5	147.5	3.0	2.6	506	0.15	1.09	1.74
	152.5	153.5	1.0	0.9	72	1.51	0.02	0.07
	155.5	157.5	2.0	1.8	75	4.82	0.01	0.04
incl.	*156.5*	*157.5*	*1.0*	*0.9*	*74*	*6.86*	*0.01*	*0.03*
E07-31	207.0	208.5	1.5	1.1	172	-	0.00	0.01
E07-32	188.0	199.0	11.0	8.7	864	3.34	0.25	0.52
incl.	*190.0*	*191.0*	*1.0*	*0.8*	*462*	*24.75*	*0.47*	*0.26*
	195.0	*196.0*	*1.0*	*0.8*	*1811*	*0.73*	*0.34*	*0.82*
	196.0	*197.0*	*1.0*	*0.8*	*1183*	*1.09*	*0.48*	*0.65*
	197.0	*198.0*	*1.0*	*0.8*	*3823*	*2.01*	*0.43*	*1.41*
	201.0	202.0	1.0	0.8	51	1.93	0.00	0.01
E07-33	164.0	165.2	1.2	0.9	101	4.05	0.01	0.02
E07-34	200.0	215.0	15.0	13.4	720	1.54	0.35	1.04
incl.	*203.0*	*204.0*	*1.0*	*0.9*	*1130*	*17.07*	*0.20*	*0.35*
	205.0	*206.0*	*1.0*	*0.9*	*1082*	*0.14*	*0.26*	*1.68*
	207.0	*208.0*	*1.0*	*0.9*	*2380*	*0.05*	*1.38*	*2.23*
	208.0	*209.0*	*1.0*	*0.9*	*1417*	*0.08*	*0.49*	*1.82*
	212.0	*213.0*	*1.0*	*0.9*	*1695*	*1.11*	*1.30*	*2.95*
E07-35	26.0	28.0	2.0	1.9	174	0.30	0.10	0.20
	60.0	61.0	1.0	0.9	155	0.23	0.12	0.19
E08-36	484.5	486.0	1.5	1.3	167	0.06	0.17	0.23
E08-37	108.0	109.0	1.0	0.8	2608	167.65	0.16	0.33
E08-39	197.0	203.0	6.0	4.8	287	0.62	0.25	0.41
incl.	*200.0*	*201.0*	*1.0*	*0.8*	*1159*	*0.74*	*0.81*	*0.80*
E08-40	140.0	147.0	7.0	5.4	109	3.17	0.04	0.04
incl.	*140.0*	*141.0*	*1.0*	*0.8*	*94*	*5.49*	*0.01*	*0.02*
	141.0	*142.0*	*1.0*	*0.8*	*195*	*6.51*	*0.09*	*0.01*
	146.0	*147.0*	*1.0*	*0.8*	*122*	*6.58*	*0.01*	*0.05*
E08-43	143.0	147.0	4.0	3.4	177	4.21	0.06	0.31
incl.	*143.0*	*144.0*	*1.0*	*0.8*	*247*	*13.71*	*0.01*	*0.08*
E08-44	153.8	154.8	1.0	0.7	179	0.03	0.11	0.48
	177.5	179.5	2.0	1.4	512	0.19	1.17	0.95
E08-45	146.0	147.5	1.5	1.3	205	0.02	0.30	0.23
E08-46	98.0	100.0	2.0	1.6	243	21.87	0.01	0.02
incl.	*98.0*	*99.0*	*1.0*	*0.8*	*369*	*34.97*	*0.01*	*0.02*
	99.0	*100.0*	*1.0*	*0.8*	*116*	*8.78*	*0.01*	*0.02*
	102.0	103.5	1.5	1.2	161	0.51	0.22	0.25

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E08-47	127.0	128.0	1.0	0.7	348	24.27	0.03	0.03
	136.0	137.0	1.0	0.7	166	0.26	0.02	0.05
	143.0	146.0	3.0	2.0	119	6.03	0.01	0.10
incl.	*143.0*	*144.0*	*1.0*	*0.7*	*61*	*5.35*	*0.01*	*0.09*
	144.0	*145.0*	*1.0*	*0.7*	*157*	*6.99*	*0.03*	*0.15*
	145.0	*146.0*	*1.0*	*0.7*	*139*	*5.76*	*0.01*	*0.05*
E08-48	149.0	158.0	9.0	5.5	835	1.18	0.45	0.27
incl.	*152.0*	*153.0*	*1.0*	*0.6*	*2211*	*2.31*	*1.76*	*0.55*
	156.0	*157.0*	*1.0*	*0.6*	*3097*	*2.55*	*1.44*	*0.41*
E08-49	125.0	133.0	8.0	5.6	115	2.03	0.15	0.17
incl.	*129.0*	*130.0*	*1.0*	*0.7*	*68*	*5.76*	*0.03*	*0.04*
	130.0	*131.0*	*1.0*	*0.7*	*78*	*5.07*	*0.35*	*0.21*
E08-51	150.0	163.0	13.0	11.9	193	8.26	0.10	0.17
incl.	*150.0*	*151.0*	*1.0*	*0.9*	*164*	*8.71*	*0.02*	*0.16*
	156.0	*157.0*	*1.0*	*0.9*	*203*	*7.06*	*0.01*	*0.01*
	157.0	*158.0*	*1.0*	*0.9*	*388*	*26.61*	*0.01*	*0.04*
	161.0	*162.0*	*1.0*	*0.9*	*287*	*37.92*	*0.07*	*0.19*
	162.0	*163.0*	*1.0*	*0.9*	*193*	*18.51*	*0.10*	*0.25*
E08-52	173.5	188.5	15.0	9.6	1086	2.55	1.06	0.52
incl.	*176.5*	*177.5*	*1.0*	*0.6*	*482*	*18.51*	*0.06*	*0.07*
	177.5	*178.5*	*1.0*	*0.6*	*8629*	*2.23*	*2.15*	*0.41*
	178.5	*179.5*	*1.0*	*0.6*	*3797*	*1.05*	*10.20*	*3.35*
	181.5	*182.5*	*1.0*	*0.6*	*1052*	*0.45*	*0.35*	*0.65*
	196.5	197.5	1.0	0.6	186	0.32	0.34	0.46
E08-55	273.5	284.5	11.0	9.6	3642	3.92	1.12	1.49
incl.	*273.5*	*274.5*	*1.0*	*0.9*	*453*	*13.78*	*0.02*	*0.05*
	274.5	*275.5*	*1.0*	*0.9*	*888*	*19.20*	*0.07*	*0.06*
	275.5	*276.5*	*1.0*	*0.9*	*8862*	*0.73*	*2.15*	*2.62*
	276.5	*277.5*	*1.0*	*0.9*	*5272*	*0.81*	*1.58*	*1.60*
	277.5	*278.5*	*1.0*	*0.9*	*13961*	*1.13*	*2.23*	*4.06*
	278.5	*279.5*	*1.0*	*0.9*	*3777*	*0.47*	*0.68*	*2.97*
	279.5	*280.5*	*1.0*	*0.9*	*2122*	*1.81*	*1.10*	*1.58*
	280.5	*281.5*	*1.0*	*0.9*	*2284*	*0.18*	*2.38*	*2.10*
	281.5	*282.5*	*1.0*	*0.9*	*2375*	*0.23*	*2.13*	*1.30*
E08-56	292.5	296.5	4.0	3.3	433	1.30	0.26	0.46
E08-57	230.0	231.5	1.5	1.1	145	-	0.09	0.10
	237.0	245.0	8.0	6.0	369	1.68	0.50	0.62
incl.	*241.0*	*242.0*	*1.0*	*0.8*	*132*	*5.90*	*0.01*	*0.02*
	243.0	*244.0*	*1.0*	*0.8*	*1550*	*0.48*	*1.87*	*1.97*
	251.5	253.0	1.5	1.1	149	0.03	0.09	0.13

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E08-58	268.5	269.5	1.0	0.8	199	0.05	0.18	0.18
	272.5	282.5	10.0	8.1	2614	1.14	1.50	1.22
incl.	*273.5*	*274.5*	*1.0*	*0.8*	*6923*	*0.54*	*4.75*	*4.31*
	274.5	*275.5*	*1.0*	*0.8*	*5774*	*0.67*	*1.29*	*2.28*
	275.5	*276.5*	*1.0*	*0.8*	*1016*	*0.30*	*0.19*	*0.40*
	276.5	*277.5*	*1.0*	*0.8*	*2195*	*0.24*	*0.53*	*0.33*
	277.5	*278.5*	*1.0*	*0.8*	*6820*	*1.72*	*5.94*	*2.33*
	281.5	*282.5*	*1.0*	*0.8*	*1499*	*1.51*	*0.78*	*0.90*
E08-59	194.5	196.5	2.0	1.5	435	1.36	0.58	0.78
E08-60	267.0	268.0	1.0	0.9	142	-	0.15	0.16
	277.0	279.0	2.0	1.9	389	1.81	0.71	1.12
	288.0	289.5	1.5	1.4	157	0.19	0.05	0.09
E08-61	287.0	293.0	6.0	5.1	603	0.55	0.48	0.64
incl.	*290.0*	*291.0*	*1.0*	*0.8*	*2256*	*0.23*	*2.18*	*2.48*
	328.0	331.0	3.0	2.5	189	0.01	0.00	0.01
E08-62	282.0	302.0	20.0	18.1	894	0.28	0.37	0.65
incl.	*291.0*	*292.0*	*1.0*	*0.9*	*1293*	*0.18*	*0.41*	*0.72*
	294.0	*295.0*	*1.0*	*0.9*	*2739*	*1.24*	*0.47*	*0.75*
	296.0	*297.0*	*1.0*	*0.9*	*2346*	*0.53*	*0.54*	*2.33*
	297.0	*298.0*	*1.0*	*0.9*	*5037*	*0.48*	*1.67*	*2.10*
	298.0	*299.0*	*1.0*	*0.9*	*2623*	*0.42*	*1.76*	*1.58*
E08-63	246.0	249.0	3.0	2.1	180	0.03	0.10	0.14
	310.0	326.0	16.0	11.1	2581	0.66	1.71	1.89
incl.	*310.0*	*311.0*	*1.0*	*0.7*	*5106*	*0.53*	*4.41*	*3.32*
	315.5	*317.0*	*1.5*	*1.0*	*2575*	*0.12*	*1.66*	*0.56*
	318.0	*319.0*	*1.0*	*0.7*	*1307*	*0.14*	*0.59*	*0.46*
	320.0	*321.0*	*1.0*	*0.7*	*5232*	*0.64*	*3.61*	*5.10*
	322.0	*323.0*	*1.0*	*0.7*	*1968*	*3.70*	*1.72*	*3.47*
	323.0	*324.0*	*1.0*	*0.7*	*10551*	*1.94*	*8.07*	*9.11*
	324.0	*325.0*	*1.0*	*0.7*	*8088*	*0.60*	*2.52*	*4.11*
	325.0	*326.0*	*1.0*	*0.7*	*2566*	*0.32*	*1.37*	*1.68*
E08-64	266.0	267.0	1.0	0.9	865	-	0.33	0.66
E08-66	51.0	59.0	8.0	3.8	36	5.03	0.01	0.00
incl.	*53.0*	*54.0*	*1.0*	*0.5*	*20*	*6.65*	*0.00*	*0.00*
	54.0	*55.0*	*1.0*	*0.5*	*81*	*8.30*	*0.00*	*0.00*
	55.0	*56.0*	*1.0*	*0.5*	*85*	*13.58*	*0.00*	*0.00*
E08-68	96.0	97.0	1.0	0.8	73	3.84	0.00	0.01
	99.0	100.0	1.0	0.8	127	0.36	0.08	0.55

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E08-69	249.0	255.0	6.0	4.4	2338	0.29	1.17	1.08
incl.	*251.0*	*252.0*	*1.0*	*0.7*	*2411*	*0.40*	*0.86*	*1.32*
	252.0	*253.0*	*1.0*	*0.7*	*5899*	*0.52*	*3.61*	*2.23*
	253.0	*254.0*	*1.0*	*0.7*	*3013*	*0.25*	*1.75*	*0.70*
	254.0	*255.0*	*1.0*	*0.7*	*1266*	*0.39*	*0.10*	*0.10*
E08-70	253.0	254.0	1.0	0.7	259	0.15	0.24	0.47
E08-73	256.0	257.0	1.0	0.8	306	1.83	0.26	0.57
	338.0	341.0	3.0	2.3	133	0.03	0.09	1.37
	362.0	371.0	9.0	6.9	700	0.34	0.66	1.42
incl.	*367.0*	*368.0*	*1.0*	*0.8*	*1513*	*0.78*	*1.88*	*4.12*
	368.0	*369.0*	*1.0*	*0.8*	*1136*	*0.40*	*1.14*	*1.46*
	369.0	*370.0*	*1.0*	*0.8*	*1299*	*0.42*	*0.84*	*2.09*
E08-74	46.0	47.0	1.0	0.7	77	2.53	0.00	0.01
	59.0	62.0	3.0	2.0	150	0.96	0.05	0.03
E08-75	87.0	90.0	3.0	1.7	249	0.10	0.04	0.04
E08-76	383.0	384.0	1.0	1.0	143	0.02	0.44	1.31
	396.0	398.0	2.0	1.9	1842	1.48	2.75	4.10
incl.	*396.0*	*397.0*	*1.0*	*1.0*	*1034*	*0.71*	*0.95*	*2.05*
	397.0	*398.0*	*1.0*	*1.0*	*2650*	*2.25*	*4.55*	*6.14*
E08-79	133.0	134.0	1.0	0.9	308	0.30	0.15	0.30
	137.0	143.0	6.0	5.6	305	0.21	0.28	0.33
E08-80	148.0	187.0	39.0	37.5	371	0.20	0.79	1.90
incl.	*182.0*	*183.0*	*1.0*	*1.0*	*1626*	*0.57*	*5.45*	*14.20*
	184.0	*185.0*	*1.0*	*1.0*	*1767*	*0.77*	*2.60*	*9.10*
	185.0	*186.0*	*1.0*	*1.0*	*2724*	*1.12*	*4.40*	*8.25*
E08-82	154.0	157.0	3.0	2.2	147	0.19	0.18	0.39
	168.0	196.0	28.0	20.2	993	0.47	1.72	3.20
incl.	*190.0*	*191.0*	*1.0*	*0.7*	*1873*	*0.69*	*3.35*	*8.75*
	191.0	*192.0*	*1.0*	*0.7*	*1810*	*0.60*	*7.40*	*8.90*
	192.0	*193.0*	*1.0*	*0.7*	*10062*	*3.02*	*6.85*	*14.20*
	193.0	*194.0*	*1.0*	*0.7*	*3151*	*2.27*	*7.14*	*17.20*
	194.0	*195.0*	*1.0*	*0.7*	*4131*	*2.67*	*7.98*	*13.80*
	195.0	*196.0*	*1.0*	*0.7*	*2462*	*1.30*	*7.73*	*10.50*
E08-85	94.5	105.0	10.5	9.9	303	1.49	0.08	0.15
incl.	*101.0*	*102.0*	*1.0*	*0.9*	*443*	*9.12*	*0.04*	*0.07*
E08-86	114.0	118.0	4.0	3.9	232	0.20	0.32	1.20
E08-87	127.0	131.0	4.0	3.3	146	0.11	0.07	0.13
	134.0	139.5	5.5	4.6	174	1.47	0.15	0.31

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E08-90	90.5	96.0	5.5	3.8	539	0.23	1.03	1.82
	105.0	141.0	36.0	24.6	340	0.19	0.39	0.69
incl.	*112.0*	*113.0*	*1.0*	*0.7*	*3196*	*2.30*	*2.15*	*3.49*
	113.0	*114.0*	*1.0*	*0.7*	*1107*	*0.72*	*2.20*	*4.19*
	165.0	168.0	3.0	2.1	119	0.74	0.07	0.11
E08-91	301.5	307.5	6.0	5.1	249	0.12	0.47	0.97
	318.0	331.0	13.0	11.0	718	0.36	0.64	0.97
incl.	*323.0*	*324.0*	*1.0*	*0.8*	*1322*	*0.55*	*2.87*	*2.43*
	325.0	*326.0*	*1.0*	*0.8*	*3508*	*1.22*	*1.90*	*3.05*
	329.0	*330.0*	*1.0*	*0.8*	*1448*	*1.18*	*0.33*	*0.46*
	330.0	*331.0*	*1.0*	*0.8*	*1267*	*0.52*	*0.17*	*0.13*
E08-92	73.0	82.5	9.5	5.9	602	0.45	0.45	0.39
incl.	*73.0*	*74.0*	*1.0*	*0.6*	*1113*	*0.76*	*0.32*	*0.13*
	79.0	*80.0*	*1.0*	*0.6*	*1041*	*0.24*	*1.78*	*1.52*
	99.0	102.0	3.0	1.4	292	0.19	0.26	0.59
	106.0	107.0	1.0	0.5	224	0.15	0.16	0.43
	110.0	111.0	1.0	0.5	182	0.20	0.31	1.10
	127.5	132.0	4.5	2.0	154	0.06	0.14	0.42
	135.0	136.0	1.0	0.5	271	0.17	0.31	0.69
	140.0	141.0	1.0	0.5	323	0.15	0.27	0.58
	144.0	189.0	45.0	20.3	427	0.10	0.36	0.74
incl.	*177.0*	*178.0*	*1.0*	*0.5*	*1940*	*0.37*	*0.21*	*0.78*
	178.0	*179.0*	*1.0*	*0.5*	*2486*	*0.44*	*0.53*	*2.05*
	186.0	*189.0*	*3.0*	*1.4*	*1267*	*0.14*	*0.51*	*1.12*
	198.0	201.0	3.0	1.0	145	0.64	1.38	3.10
E08-93	184.5	186.0	1.5	1.5	443	0.19	0.47	0.47
	192.0	199.0	7.0	6.8	307	0.27	0.34	0.42
E08-94	313.5	315.0	1.5	1.2	134	-	0.33	0.41
	318.0	329.0	11.0	8.5	509	0.17	0.36	0.40
incl.	*328.0*	*329.0*	*1.0*	*0.8*	*1206*	*0.27*	*0.46*	*0.41*
E08-95	299.0	303.0	4.0	3.1	455	0.30	0.35	0.73
E08-96	204.0	215.0	11.0	9.5	396	0.25	0.55	0.38
incl.	*213.0*	*214.0*	*1.0*	*0.9*	*1631*	*0.63*	*1.04*	*0.59*
E08-97	343.0	346.0	3.0	2.0	560	0.20	0.71	0.68
incl.	*343.0*	*344.0*	*1.0*	*0.7*	*1474*	*0.18*	*2.05*	*1.93*
	349.0	350.0	1.0	0.7	431	0.14	0.10	0.17
	353.0	354.5	1.5	1.0	242	0.19	0.29	1.78
E08-98	111.0	148.5	37.5	34.5	234	0.18	0.36	0.69
incl.	*146.0*	*147.0*	*1.0*	*0.9*	*1758*	*1.27*	*2.02*	*4.01*

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E08-99	124.5	178.5	54.0	40.3	610	0.66	0.76	1.53
incl.	*145.5*	*147.0*	*1.5*	*1.1*	*1224*	*0.32*	*0.56*	*1.35*
	163.5	*165.0*	*1.5*	*1.1*	*1185*	*0.62*	*1.74*	*3.82*
	168.0	*169.0*	*1.0*	*0.7*	*1687*	*0.63*	*1.73*	*4.21*
	169.0	*170.0*	*1.0*	*0.7*	*2139*	*0.11*	*9.56*	*14.30*
	170.0	*171.0*	*1.0*	*0.7*	*3998*	*0.44*	*7.23*	*12.90*
	173.0	*174.0*	*1.0*	*0.7*	*7527*	*21.87*	*0.71*	*1.86*
	174.0	*175.0*	*1.0*	*0.7*	*1722*	*1.65*	*0.59*	*1.01*
E08-100	243.0	244.5	1.5	1.4	608	0.03	0.76	1.02
	253.5	267.0	13.5	13.0	278	0.02	0.60	0.66
	282.0	307.0	25.0	24.0	1102	0.14	0.74	0.73
incl.	*297.0*	*298.0*	*1.0*	*1.0*	*1280*	*0.12*	*1.02*	*1.04*
	299.0	*300.0*	*1.0*	*1.0*	*1977*	*0.18*	*2.13*	*1.50*
	300.0	*301.0*	*1.0*	*1.0*	*1405*	*0.15*	*1.49*	*1.29*
	301.0	*302.0*	*1.0*	*1.0*	*1003*	*0.24*	*0.57*	*1.30*
	302.0	*303.0*	*1.0*	*1.0*	*3531*	*0.27*	*0.38*	*1.10*
	303.0	*304.0*	*1.0*	*1.0*	*6504*	*0.35*	*0.49*	*1.16*
	304.0	*305.0*	*1.0*	*1.0*	*1603*	*0.35*	*0.17*	*0.37*
	305.0	*306.0*	*1.0*	*1.0*	*1830*	*0.53*	*0.23*	*0.86*
	306.0	*307.0*	*1.0*	*1.0*	*1513*	*0.58*	*0.09*	*0.16*
E08-101	120.0	166.0	46.0	41.7	465	0.23	0.86	1.92
incl.	*156.0*	*157.0*	*1.0*	*0.9*	*1936*	*0.81*	*5.90*	*8.92*
	157.0	*158.0*	*1.0*	*0.9*	*1412*	*0.76*	*2.77*	*9.91*
	158.0	*159.0*	*1.0*	*0.9*	*2701*	*0.79*	*7.14*	*17.40*
	159.0	*160.0*	*1.0*	*0.9*	*1741*	*0.56*	*6.09*	*13.10*
	162.0	*163.0*	*1.0*	*0.9*	*4567*	*1.66*	*6.39*	*13.40*
E08-102	276.0	279.0	3.0	2.7	150	0.05	0.06	0.08
	391.0	397.0	6.0	5.4	2220	0.36	2.24	1.99
incl.	*391.0*	*392.0*	*1.0*	*0.9*	*1546*	*0.21*	*2.14*	*1.88*
	392.0	*393.0*	*1.0*	*0.9*	*4322*	*1.11*	*4.80*	*3.70*
	393.0	*394.0*	*1.0*	*0.9*	*6833*	*0.68*	*5.50*	*5.20*
	404.0	405.0	1.0	0.9	184	0.08	0.15	0.29

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E08-103	177.0	215.0	38.0	27.1	1578	3.65	1.22	2.22
incl.	*179.0*	*180.0*	*1.0*	*0.7*	*4220*	*3.63*	*1.03*	*1.83*
	180.0	*181.0*	*1.0*	*0.7*	*10062*	*9.67*	*1.90*	*3.45*
	181.0	*182.0*	*1.0*	*0.7*	*14765*	*76.52*	*1.63*	*1.73*
	182.0	*183.0*	*1.0*	*0.7*	*2464*	*18.99*	*0.21*	*0.31*
	183.0	*184.0*	*1.0*	*0.7*	*1710*	*1.50*	*0.94*	*2.90*
	184.0	*185.0*	*1.0*	*0.7*	*3684*	*8.61*	*2.30*	*5.10*
	186.0	*187.0*	*1.0*	*0.7*	*3393*	*2.88*	*0.74*	*1.46*
	190.0	*191.0*	*1.0*	*0.7*	*2180*	*1.10*	*1.90*	*3.02*
	191.0	*192.0*	*1.0*	*0.7*	*1634*	*1.22*	*4.05*	*8.52*
	192.0	*193.0*	*1.0*	*0.7*	*1991*	*1.10*	*3.15*	*8.05*
	193.0	*194.0*	*1.0*	*0.7*	*1598*	*0.92*	*3.20*	*6.60*
	194.0	*195.0*	*1.0*	*0.7*	*1402*	*0.90*	*2.80*	*6.90*
	195.0	*196.0*	*1.0*	*0.7*	*2903*	*1.73*	*5.25*	*9.60*
	196.0	*197.0*	*1.0*	*0.7*	*1177*	*1.73*	*4.65*	*5.10*
E08-104	219.0	220.0	1.0	0.7	227	0.25	0.29	0.65
	226.0	260.6	34.6	24.5	352	0.37	0.49	1.15
incl.	*229.0*	*230.0*	*1.0*	*0.7*	*1115*	*0.54*	*0.82*	*2.01*
	265.0	266.5	1.5	1.1	336	0.25	0.27	0.56
	275.5	281.9	6.4	4.6	193	0.26	0.40	0.87
E08-105	205.0	206.0	1.0	0.8	355	0.07	0.89	1.62
E08-107	183.0	186.0	3.0	2.1	207	0.10	0.34	0.70
	200.0	210.0	10.0	7.0	238	0.21	0.18	0.36
	220.0	247.0	27.0	19.0	702	0.33	2.47	5.32
incl.	*227.0*	*228.0*	*1.0*	*0.7*	*1178*	*0.32*	*1.25*	*2.30*
	228.0	*229.0*	*1.0*	*0.7*	*1225*	*0.53*	*0.84*	*1.21*
	230.0	*231.0*	*1.0*	*0.7*	*2058*	*0.57*	*3.50*	*3.00*
	231.0	*232.0*	*1.0*	*0.7*	*2365*	*0.51*	*4.70*	*9.60*
	235.0	*236.0*	*1.0*	*0.7*	*3712*	*0.60*	*9.90*	*16.50*
	239.0	*240.0*	*1.0*	*0.7*	*1086*	*0.89*	*8.45*	*16.90*
E08-109	252.0	254.0	2.0	1.3	228	0.05	0.10	0.15
	256.0	258.0	2.0	1.3	192	0.04	0.22	0.36
	260.0	262.0	2.0	1.3	206	0.14	0.19	0.30

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E08-110	30.0	48.0	18.0	17.2	480	0.34	0.27	0.16
	141.0	142.5	1.5	1.4	617	0.26	0.47	0.68
	151.5	153.0	1.5	1.4	116	3.84	0.09	0.16
	163.0	172.0	9.0	8.6	168	0.10	0.20	0.36
	177.0	221.0	44.0	42.0	1161	0.94	0.98	1.20
incl.	*186.0*	*187.0*	*1.0*	*1.0*	*4668*	*6.45*	*2.00*	*0.84*
	187.0	*188.0*	*1.0*	*1.0*	*1481*	*1.16*	*0.80*	*2.00*
	188.0	*189.0*	*1.0*	*1.0*	*1646*	*1.48*	*1.21*	*1.59*
	198.0	*199.0*	*1.0*	*1.0*	*4476*	*3.57*	*0.51*	*0.46*
	200.0	*201.0*	*1.0*	*1.0*	*1220*	*1.46*	*0.46*	*0.45*
	202.0	*203.0*	*1.0*	*1.0*	*2070*	*1.45*	*2.75*	*1.60*
	203.0	*204.0*	*1.0*	*1.0*	*1022*	*0.43*	*0.66*	*1.58*
	204.0	*205.0*	*1.0*	*1.0*	*1053*	*0.45*	*2.65*	*1.60*
	205.0	*206.0*	*1.0*	*1.0*	*1831*	*0.57*	*2.20*	*2.35*
	206.0	*207.0*	*1.0*	*1.0*	*1733*	*1.28*	*2.80*	*1.85*
	207.0	*208.0*	*1.0*	*1.0*	*3198*	*1.43*	*0.36*	*0.11*
	208.0	*209.0*	*1.0*	*1.0*	*1118*	*0.49*	*1.62*	*1.20*
	209.0	*210.0*	*1.0*	*1.0*	*4166*	*2.32*	*2.50*	*3.10*
	210.0	*211.0*	*1.0*	*1.0*	*9819*	*5.62*	*6.00*	*7.45*
	211.0	*212.0*	*1.0*	*1.0*	*4296*	*7.82*	*3.00*	*3.55*
E08-111	167.0	168.5	1.5	0.8	308	0.03	0.26	0.56
E08-113	270.0	312.0	42.0	19.7	374	0.32	1.57	3.09
incl.	*277.0*	*278.0*	*1.0*	*0.5*	*1274*	*0.60*	*0.26*	*0.48*
	279.0	*280.0*	*1.0*	*0.5*	*1224*	*0.81*	*2.90*	*8.26*
	294.0	*295.0*	*1.0*	*0.5*	*1285*	*0.71*	*2.65*	*4.10*
	318.5	329.5	11.0	5.2	417	0.28	1.12	2.19
incl.	*327.5*	*328.5*	*1.0*	*0.5*	*1672*	*0.74*	*4.15*	*9.95*
	328.5	*329.5*	*1.0*	*0.5*	*1195*	*0.73*	*2.13*	*3.75*
E08-117	309.0	316.0	7.0	4.4	244	0.23	1.90	3.70
	326.0	333.5	7.5	4.7	581	1.59	1.25	1.75
incl.	*326.0*	*327.0*	*1.0*	*0.6*	*1664*	*5.21*	*1.30*	*1.95*
	329.0	*330.0*	*1.0*	*0.6*	*1625*	*4.32*	*1.25*	*2.09*
E08-119	330.0	336.0	6.0	5.1	219	0.10	0.15	0.22
E08-119	357.0	360.0	3.0	2.1	288	0.11	0.76	1.99
E08-119	454.0	455.5	1.5	1.1	262	0.11	0.23	0.50
E08-119	488.0	490.0	2.0	0.4	353	1.58	0.27	0.77
E08-121	183.0	187.0	4.0	3.1	263	0.12	0.32	0.57
	201.0	236.5	35.5	27.7	367	0.18	0.55	0.90
incl.	*202.0*	*203.0*	*1.0*	*0.8*	*1137*	*0.29*	*0.98*	*1.80*
	217.0	*218.5*	*1.5*	*1.2*	*1731*	*0.79*	*1.38*	*2.95*

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E08-122	180.9	197.5	16.6	14.6	734	0.76	-	-
incl.	*181.8*	*182.6*	*0.8*	*0.7*	*1180*	*0.03*	-	-
	185.4	*186.3*	*0.9*	*0.8*	*2700*	*3.43*	-	-
	186.3	*187.3*	*1.0*	*0.9*	*2770*	*3.44*	-	-
	201.2	210.4	9.2	8.1	1598	1.45	-	-
incl.	*202.1*	*203.0*	*0.9*	*0.8*	*4520*	*1.30*	-	-
	204.0	*204.9*	*0.9*	*0.8*	*4080*	*5.00*	-	-
	206.7	*207.6*	*1.0*	*0.8*	*1040*	*0.85*	-	-
	207.6	*208.6*	*1.0*	*0.8*	*2330*	*4.67*	-	-
	208.6	*209.6*	*1.0*	*0.8*	*1690*	*1.02*	-	-
	209.6	*210.4*	*0.9*	*0.7*	*1340*	*0.59*	-	-
E08-123	435.0	441.0	6.0	4.2	313	0.12	0.29	0.49
E08-124	231.0	259.0	28.0	21.3	241	0.16	0.39	0.49
	263.0	289.0	26.0	19.7	575	0.71	2.08	3.01
incl.	*275.0*	*276.0*	*1.0*	*0.8*	*1613*	*1.13*	*3.60*	*4.35*
	279.0	*280.0*	*1.0*	*0.8*	*1772*	*1.06*	*5.75*	*5.05*
	282.0	*283.0*	*1.0*	*0.8*	*1287*	*2.48*	*7.95*	*11.90*
	284.0	*285.0*	*1.0*	*0.8*	*2576*	*2.40*	*4.15*	*6.45*
	292.0	293.0	1.0	0.8	139	0.11	0.41	0.67
	296.0	297.0	1.0	0.8	367	0.18	0.14	0.32
E08-125	146.4	156.9	10.5	8.3	274	6.54	-	-
incl.	*147.3*	*148.2*	*1.0*	*0.8*	*237*	*11.80*	-	-
	150.0	*150.9*	*0.9*	*0.7*	*132*	*15.50*	-	-
	150.9	*151.8*	*0.9*	*0.7*	*347*	*34.90*	-	-
E08-127	270.2	280.4	10.2	8.7	1832	1.14	-	-
incl.	*272.0*	*272.9*	*0.9*	*0.8*	*5710*	*0.56*	-	-
	272.9	*273.7*	*0.9*	*0.7*	*3150*	*0.45*	-	-
	273.7	*274.7*	*1.0*	*0.8*	*3020*	*0.38*	-	-
	274.7	*275.5*	*0.8*	*0.7*	*6570*	*0.20*	-	-
	275.5	*276.4*	*0.9*	*0.8*	*1470*	*0.99*	-	-
E09-129	194.0	196.0	2.0	1.5	170	0.04	0.16	0.40
	200.0	242.0	42.0	31.5	349	0.30	0.50	1.10
incl.	*228.5*	*230.0*	*1.5*	*1.1*	*1217*	*2.81*	*1.60*	*2.40*
E09-131	45.0	48.0	3.0	2.2	1322	1.29	0.86	0.85
incl.	*45.0*	*46.5*	*1.5*	*1.1*	*2539*	*0.64*	*1.70*	*1.66*
	187.5	199.5	12.0	8.9	375	0.24	0.35	0.75
	208.5	223.5	15.0	11.2	841	0.37	1.35	1.80
incl.	*219.0*	*220.5*	*1.5*	*1.1*	*1372*	*0.73*	*2.90*	*1.04*
	220.5	*222.0*	*1.5*	*1.1*	*3970*	*1.95*	*2.05*	*4.45*
E09-133	312.0	336.0	24.0	15.1	324	0.16	1.20	2.67
incl.	*333.0*	*334.5*	*1.5*	*0.9*	*1048*	*0.53*	*2.60*	*7.65*

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E09-134	212.0	213.5	1.5	0.5	366	0.58	1.63	3.20
	234.5	236.0	1.5	0.5	219	0.38	0.70	1.95
	410.0	413.0	3.0	1.1	285	0.22	0.66	2.38
E09-135	381.5	383.0	1.5	0.8	623	0.39	0.64	2.00
	393.2	394.7	1.5	0.8	291	0.21	0.75	1.95
E09-136	241.5	243.0	1.5	0.9	151	0.19	0.11	0.37
	247.5	267.0	19.5	11.3	344	0.31	0.24	0.77
incl.	*256.5*	*258.0*	*1.5*	*0.9*	*1629*	*0.62*	*1.20*	*4.00*
E09-137	210.0	219.0	9.0	7.1	211	0.79	0.12	0.20
	225.0	235.5	10.5	8.3	206	0.38	0.15	0.38
E09-138	282.0	291.0	9.0	3.7	559	0.42	0.65	1.55
	297.0	303.0	6.0	2.5	167	0.28	0.28	0.95
E09-139	319.5	399.3	79.8	56.4	400	0.37	0.57	1.31
incl.	*336.0*	*337.5*	*1.5*	*1.1*	*1287*	*1.55*	*0.82*	*1.90*
	354.0	*355.5*	*1.5*	*1.1*	*1522*	*0.69*	*1.19*	*1.99*
	355.5	*357.0*	*1.5*	*1.1*	*1536*	*0.91*	*4.30*	*9.00*
	357.0	*358.5*	*1.5*	*1.1*	*1809*	*1.70*	*4.30*	*10.20*
	358.5	*360.0*	*1.5*	*1.1*	*3612*	*1.55*	*4.50*	*9.00*
E09-140	279.0	297.0	18.0	17.5	291	0.14	0.16	0.40
	312.0	315.0	3.0	2.9	190	0.16	0.08	0.25
E09-142	426.0	429.0	3.0	2.2	158	0.10	0.45	1.30
	450.0	453.0	3.0	2.2	157	0.04	0.01	0.02
	457.5	462.0	4.5	3.3	302	0.17	0.71	1.47
E09-143	300.0	313.5	13.5	11.3	801	0.20	0.64	0.99
incl.	*300.0*	*301.5*	*1.5*	*1.3*	*2811*	*0.52*	*0.22*	*0.45*
E09-145	499.5	507.0	7.5	5.1	294	0.28	0.18	0.35
	523.5	537.0	13.5	9.1	221	0.33	2.12	4.76
E09-146	481.5	483.0	1.5	1.1	271	0.67	0.75	1.24
	487.5	505.5	18.0	12.7	361	0.57	2.85	2.16
	510.0	513.0	3.0	2.1	235	0.63	0.21	0.46
E09-147	465.0	468.0	3.0	1.4	170	0.06	0.66	0.87
E09-148	225.0	228.0	3.0	2.4	368	0.31	0.26	0.75
E09-148	243.0	246.0	3.0	2.4	207	0.16	0.07	0.13
E09-148	262.5	264.0	1.5	1.2	245	0.33	0.35	0.94
E09-148	336.0	337.5	1.5	1.2	165	0.40	0.36	0.74
E09-148	340.5	342.0	1.5	1.2	334	0.78	1.97	3.05
E09-148	345.0	363.0	18.0	14.4	719	1.31	0.59	1.21
incl.	*345.0*	*346.5*	*1.5*	*1.2*	*5491*	*8.11*	*1.12*	*1.95*
E09-148	367.5	372.0	4.5	3.6	248	0.67	1.14	1.40
E09-148	378.0	385.5	7.5	6.0	435	1.32	1.43	2.09
E09-148	438.0	441.0	3.0	2.4	193	0.52	0.73	0.66
E09-148	453.0	459.0	6.0	4.8	262	0.22	0.11	0.13

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E09-150	210.0	229.5	19.5	14.2	258	0.20	0.61	1.36
E09-151	261.0	264.0	3.0	2.4	150	0.13	0.05	0.11
	321.0	324.0	3.0	2.4	155	0.22	0.36	0.70
	351.0	354.0	3.0	2.4	157	0.17	0.32	0.76
	385.5	390.0	4.5	3.7	275	0.60	0.44	1.03
	394.5	429.0	34.5	28.1	384	0.86	1.73	2.66
incl.	*402.0*	*403.5*	*1.5*	*1.2*	*1037*	*0.66*	*0.70*	*1.67*
	436.5	438.0	1.5	1.2	77	0.17	1.73	2.55
E09-152	247.5	282.0	34.5	17.4	283	0.40	0.31	0.63
incl.	*273.0*	*274.5*	*1.5*	*0.8*	*1305*	*1.47*	*1.14*	*2.05*
	309.0	324.0	15.0	7.6	194	0.19	0.32	0.67
E09-154	469.5	522.0	52.5	12.3	1146	1.92	6.40	10.62
incl.	*477.0*	*478.5*	*1.5*	*0.4*	*1358*	*1.46*	*2.75*	*3.85*
	480.0	*481.5*	*1.5*	*0.4*	*1877*	*2.55*	*3.25*	*5.75*
	481.5	*483.0*	*1.5*	*0.4*	*2757*	*4.00*	*1.92*	*3.55*
	483.0	*484.5*	*1.5*	*0.4*	*1675*	*1.49*	*3.25*	*4.55*
	484.5	*486.0*	*1.5*	*0.4*	*1311*	*0.77*	*8.80*	*13.50*
	486.0	*487.5*	*1.5*	*0.4*	*5550*	*5.79*	*7.43*	*14.90*
	487.5	*489.0*	*1.5*	*0.4*	*1700*	*1.96*	*6.65*	*12.30*
	489.0	*490.5*	*1.5*	*0.4*	*1739*	*2.56*	*7.48*	*15.10*
	490.5	*492.0*	*1.5*	*0.4*	*2227*	*2.35*	*3.88*	*6.20*
	492.0	*493.5*	*1.5*	*0.4*	*1507*	*1.61*	*2.88*	*5.10*
	493.5	*495.0*	*1.5*	*0.4*	*4747*	*7.45*	*17.30*	*22.50*
	495.0	*496.5*	*1.5*	*0.4*	*1303*	*1.46*	*26.00*	*29.50*
	498.0	*499.5*	*1.5*	*0.4*	*1662*	*2.57*	*43.60*	*23.00*
	499.5	*501.0*	*1.5*	*0.4*	*1620*	*3.67*	*13.10*	*28.00*
	501.0	*502.5*	*1.5*	*0.4*	*1136*	*3.10*	*7.15*	*32.06*
	528.0	529.5	1.5	0.4	189	0.24	0.16	0.26
	546.0	574.5	28.5	6.7	502	1.07	2.01	2.89
incl.	*568.5*	*570.0*	*1.5*	*0.4*	*1443*	*2.84*	*1.60*	*2.95*
E09-155	237.0	240.0	3.0	2.5	229	-	0.09	0.16
	270.0	282.0	12.0	10.1	323	-	0.41	0.68
	294.0	306.0	12.0	7.6	417	-	0.20	0.33
	330.0	333.0	3.0	1.9	175	0.20	0.33	0.50
	336.0	337.5	1.5	1.0	167	0.10	0.24	0.58
	346.5	348.0	1.5	1.0	153	0.24	0.25	0.68
E09-156	294.0	308.0	14.0	12.5	610	0.16	0.26	0.64
E09-157	336.0	339.0	3.0	2.3	163	0.26	0.30	0.58
	345.0	384.0	39.0	30.1	412	0.66	1.38	2.86
incl.	*357.0*	*358.5*	*1.5*	*1.2*	*1986*	*1.58*	*3.15*	*5.05*
	405.0	409.4	4.4	3.4	283	0.30	0.34	0.63

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E09-159	366.0	397.5	31.5	27.0	639	1.01	2.21	4.00
incl.	*384.0*	*385.5*	*1.5*	*1.3*	*1168*	*0.66*	*2.01*	*4.15*
	387.0	*388.5*	*1.5*	*1.3*	*2983*	*2.55*	*4.03*	*7.45*
	388.5	*390.0*	*1.5*	*1.3*	*3411*	*4.19*	*5.85*	*10.10*
	393.0	*394.5*	*1.5*	*1.3*	*1931*	*5.91*	*4.33*	*6.95*
	394.5	*396.0*	*1.5*	*1.3*	*1258*	*3.38*	*2.85*	*5.15*
	400.5	402.0	1.5	1.3	280	0.42	0.06	0.13
	417.0	426.0	9.0	7.4	259	0.22	0.36	0.79
E09-160	219.0	225.0	6.0	4.1	199	0.03	0.11	0.12
	289.5	291.0	1.5	1.0	288	0.24	0.28	0.64
E09-161	289.5	291.0	1.5	1.2	558	1.82	2.23	4.67
	297.0	306.0	9.0	6.9	436	1.24	1.92	3.52
incl.	*301.5*	*303.0*	*1.5*	*1.2*	*1414*	*3.68*	*1.65*	*4.25*
E09-162	258.0	259.5	1.5	1.2	278	0.11	0.14	0.35
	363.0	370.5	7.5	6.0	3109	1.52	2.58	3.63
	373.5	375.0	1.5	1.2	145	0.13	0.12	0.43
E09-163	357.0	358.5	1.5	1.1	153	0.23	0.79	1.09
E09-164	384.0	393.0	9.0	8.0	1420	0.86	1.17	1.84
incl.	*385.5*	*387.0*	*1.5*	*1.3*	*1255*	*0.47*	*1.35*	*2.70*
	387.0	*388.5*	*1.5*	*1.3*	*1330*	*1.39*	*0.74*	*1.25*
	390.0	*391.5*	*1.5*	*1.3*	*2632*	*1.53*	*2.32*	*3.65*
	391.5	*393.0*	*1.5*	*1.3*	*2950*	*0.90*	*1.94*	*2.75*
E09-165	190.5	226.5	36.0	23.3	1320	0.81	2.09	3.37
incl.	*195.0*	*196.5*	*1.5*	*1.0*	*2875*	*1.52*	*2.45*	*4.30*
	196.5	*198.0*	*1.5*	*1.0*	*1606*	*1.40*	*1.48*	*2.55*
	198.0	*199.5*	*1.5*	*1.0*	*4241*	*1.74*	*6.80*	*5.65*
	199.5	*201.0*	*1.5*	*1.0*	*3092*	*1.49*	*2.40*	*4.65*
	208.5	*210.0*	*1.5*	*1.0*	*1127*	*1.27*	*3.00*	*2.20*
	210.0	*211.5*	*1.5*	*1.0*	*1132*	*0.31*	*2.90*	*2.15*
	219.0	*220.5*	*1.5*	*1.0*	*2691*	*0.89*	*5.45*	*7.05*
	220.5	*222.0*	*1.5*	*1.0*	*2558*	*0.87*	*7.35*	*10.50*
	222.0	*223.5*	*1.5*	*1.0*	*3598*	*1.13*	*5.20*	*9.80*
	223.5	*225.0*	*1.5*	*1.0*	*1440*	*0.44*	*4.10*	*12.90*
	225.0	*226.5*	*1.5*	*1.0*	*1258*	*1.40*	*3.60*	*9.60*
E09-166	277.5	285.0	7.5	3.8	187	0.23	1.65	2.69
	294.0	295.5	1.5	0.8	383	0.40	0.09	0.13
	357.0	364.5	7.5	2.5	210	0.40	0.41	0.69
	390.0	399.0	9.0	1.2	688	1.20	0.58	1.17
incl.	*393.0*	*394.5*	*1.5*	*0.2*	*2864*	*3.72*	*2.20*	*4.50*

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E09-167	276.0	279.0	3.0	2.4	343	0.13	1.12	0.50
	379.5	388.5	9.0	7.1	1028	0.41	0.94	1.17
incl.	*382.5*	*384.0*	*1.5*	*1.2*	*1294*	*0.17*	*0.76*	*1.28*
	384.0	*385.5*	*1.5*	*1.2*	*1835*	*0.58*	*2.20*	*1.51*
	385.5	*387.0*	*1.5*	*1.2*	*1282*	*0.81*	*1.19*	*0.75*
E09-168	147.0	153.0	6.0	3.6	188	0.10	0.16	0.37
	162.0	202.0	40.0	24.2	1407	0.45	4.84	5.99
incl.	*172.5*	*174.0*	*1.5*	*0.9*	*4413*	*0.16*	*3.70*	*6.00*
	174.0	*175.5*	*1.5*	*0.9*	*4668*	-	*5.45*	*6.60*
	175.5	*177.0*	*1.5*	*0.9*	*3840*	-	*3.85*	*9.20*
	177.0	*178.5*	*1.5*	*0.9*	*1463*	*0.06*	*1.91*	*2.85*
	180.0	*181.5*	*1.5*	*0.9*	*1467*	-	*7.45*	*10.90*
	181.5	*183.0*	*1.5*	*0.9*	*4455*	*0.33*	*10.60*	*15.60*
	183.0	*184.5*	*1.5*	*0.9*	*1536*	*0.27*	*19.50*	*25.80*
	187.5	*189.0*	*1.5*	*0.9*	*3237*	*3.13*	*10.90*	*9.25*
	189.0	*190.5*	*1.5*	*0.9*	*2132*	*1.91*	*14.90*	*8.20*
	190.5	*192.0*	*1.5*	*0.9*	*1306*	*1.06*	*10.20*	*10.40*
	193.0	*194.5*	*1.5*	*0.9*	*1826*	*0.58*	*2.35*	*2.90*
	214.0	217.0	3.0	1.8	814	0.27	0.84	2.05
	232.0	233.5	1.5	0.9	207	0.67	0.60	1.56
	241.0	244.0	3.0	1.8	314	0.10	0.23	0.38
E09-169	160.5	198.0	37.5	31.9	584	0.48	0.83	0.92
incl.	*160.5*	*162.0*	*1.5*	*1.3*	*1177*	*0.13*	*0.52*	*1.48*
	163.5	*165.0*	*1.5*	*1.3*	*1101*	*0.37*	*1.08*	*1.90*
	165.0	*166.5*	*1.5*	*1.3*	*1291*	*0.42*	*1.53*	*2.15*
	187.5	*189.0*	*1.5*	*1.3*	*2778*	*0.47*	*3.05*	*4.45*
	196.5	*198.0*	*1.5*	*1.3*	*3506*	*4.28*	*2.05*	*0.87*
E09-170	195.0	196.5	1.5	1.0	656	0.13	0.88	2.30
	423.0	426.0	3.0	2.1	296	0.19	0.16	0.34
	442.5	444.0	1.5	1.0	202	-	0.23	0.53
E09-172	192.0	219.0	27.0	16.9	1078	0.48	2.95	3.55
incl.	*196.5*	*198.0*	*1.5*	*0.9*	*1816*	*0.92*	*1.79*	*2.60*
	205.5	*207.0*	*1.5*	*0.9*	*2048*	*0.49*	*1.62*	*2.20*
	207.0	*208.5*	*1.5*	*0.9*	*6407*	*1.03*	*4.45*	*4.95*
	210.0	*211.5*	*1.5*	*0.9*	*1413*	*0.80*	*1.77*	*1.88*
	214.5	*216.0*	*1.5*	*0.9*	*1056*	*0.58*	*27.50*	*21.50*
	216.0	*217.5*	*1.5*	*0.9*	*1364*	*1.55*	*9.75*	*17.80*
	237.0	240.0	3.0	1.9	193	0.21	0.48	0.65
	249.0	255.0	6.0	3.8	207	0.22	0.32	0.60
E09-173	271.5	282.0	10.5	4.4	566	0.31	3.20	2.01
incl.	*277.5*	*279.0*	*1.5*	*0.6*	*1265*	*0.69*	*8.15*	*3.80*
	279.0	*280.5*	*1.5*	*0.6*	*1461*	*0.70*	*11.20*	*5.20*

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E09-174	327.0	330.0	3.0	2.4	1330	1.24	0.77	0.98
incl.	*328.5*	*330.0*	*1.5*	*1.2*	*2438*	*1.31*	*1.47*	*1.80*
E09-175	201.0	204.0	3.0	2.0	462	-	0.11	0.22
	210.0	232.5	22.5	15.3	256	0.27	0.44	0.55
E09-176	250.5	255.0	4.5	3.7	470	1.30	0.19	0.49
E10-178	165.0	169.5	4.5	3.4	309	0.19	0.28	0.78
	175.5	180.0	4.5	3.4	152	0.22	0.20	0.40
	184.5	195.0	10.5	8.0	2507	0.99	1.08	1.25
incl.	*189.0*	*190.5*	*1.5*	*1.1*	*13028*	*5.04*	*2.90*	*2.05*
	190.5	*192.0*	*1.5*	*1.1*	*1786*	*0.67*	*1.12*	*1.44*
E10-179	367.5	370.5	3.0	2.6	170	-	0.13	0.34
	376.5	388.5	12.0	10.4	266	0.14	0.16	0.30
	396.0	409.5	13.5	11.7	317	0.21	0.11	0.12
	415.5	417.0	1.5	1.3	832	0.61	0.15	0.25
	441.0	445.6	4.6	4.0	305	0.15	0.21	0.48
E10-180	240.0	243.0	3.0	2.7	216	-	0.14	0.23
	253.5	273.0	19.5	17.5	544	0.12	0.35	0.61
incl.	*258.0*	*259.5*	*1.5*	*1.3*	*2625*	*-*	*0.30*	*0.61*
E10-181	135.0	160.5	25.5	22.5	688	0.43	0.38	0.74
incl.	*154.5*	*156.0*	*1.5*	*1.3*	*7168*	*3.52*	*1.23*	*2.00*
E10-182	190.5	193.5	3.0	1.6	146	1.26	0.13	0.18
	129.0	132.0	3.0	1.6	376	0.19	0.29	0.52
	138.0	142.5	4.5	2.5	200	0.22	0.45	0.60
	150.0	183.0	33.0	18.0	421	0.59	0.56	0.90
incl.	*166.5*	*168.0*	*1.5*	*0.8*	*1291*	*1.14*	*1.47*	*1.55*
	172.5	*174.0*	*1.5*	*0.8*	*1605*	*4.63*	*1.53*	*2.25*
	174.0	*175.5*	*1.5*	*0.8*	*1218*	*0.78*	*2.45*	*4.15*
E10-184	30.0	31.5	1.5	1.1	79	16.49	-	0.00
E10-185	178.5	186.0	7.5	6.0	284	1.05	0.19	0.40
E10-186	357.0	361.5	4.5	3.4	1090	0.46	2.86	3.14
incl.	*360.0*	*361.5*	*1.5*	*1.1*	*2548*	*0.79*	*3.45*	*2.50*
	364.5	366.0	1.5	1.1	180	0.14	1.58	1.56
E10-187	283.5	294.0	10.5	9.8	722	0.52	0.82	1.58
incl.	*291.0*	*292.5*	*1.5*	*1.4*	*1908*	*1.47*	*1.57*	*0.86*
E10-189	294.0	318.0	24.0	21.7	745	0.63	0.36	0.43
incl.	*309.0*	*310.5*	*1.5*	*1.4*	*1079*	*0.31*	*1.20*	*0.72*
	310.5	*312.0*	*1.5*	*1.4*	*1260*	*0.57*	*0.45*	*0.83*
	312.0	*313.5*	*1.5*	*1.4*	*1239*	*0.85*	*0.34*	*0.30*
	313.5	*315.0*	*1.5*	*1.4*	*6129*	*2.02*	*1.37*	*1.17*

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E10-191	96.0	99.0	3.0	1.8	188	0.17	0.43	1.31
	268.5	288.0	19.5	11.4	429	0.20	0.32	0.63
incl.	*271.5*	*273.0*	*1.5*	*0.9*	*1679*	*0.88*	*0.33*	*0.62*
	291.0	294.0	3.0	1.8	203	0.17	0.06	0.12
E10-194	222.0	225.0	3.0	2.1	94	2.19	0.01	0.01
E10-195	283.5	318.0	34.5	31.4	429	0.22	0.35	0.73
incl.	*292.5*	*294.0*	*1.5*	*1.4*	*1033*	*0.34*	*0.34*	*0.65*
	295.5	*297.0*	*1.5*	*1.4*	*1294*	*0.30*	*0.54*	*1.81*
	315.0	*318.0*	*3.0*	*2.7*	*1233*	*0.15*	*1.22*	*1.75*
E10-196	147.0	150.0	3.0	2.3	42	2.27	0.01	0.03
E10-197	204.0	211.5	7.5	4.6	670	1.00	0.71	0.80
incl.	*204.0*	*205.5*	*1.5*	*0.9*	*1779*	*0.30*	*2.05*	*2.50*
	205.5	*207.0*	*1.5*	*0.9*	*1291*	*0.25*	*1.39*	*1.24*
E10-198	276.0	277.5	1.5	1.4	291	0.12	0.17	0.42
	282.0	301.5	19.5	17.7	310	0.17	0.64	0.95
E10-199	277.5	285.0	7.5	6.9	846	0.72	0.46	0.33
incl.	*277.5*	*279.0*	*1.5*	*1.4*	*3173*	*0.25*	*1.76*	*0.69*
E10-200	253.5	259.5	6.0	5.5	1243	3.06	0.88	1.39
incl.	*253.5*	*255.0*	*1.5*	*1.4*	*1789*	*0.65*	*1.04*	*0.60*
	255.0	*256.5*	*1.5*	*1.4*	*805*	*9.53*	*0.55*	*0.97*
	256.5	*258.0*	*1.5*	*1.4*	*1372*	*0.38*	*1.15*	*2.80*
	258.0	*259.5*	*1.5*	*1.4*	*1005*	*1.67*	*0.79*	*1.19*
E10-201	327.0	357.0	30.0	21.5	396	0.35	0.29	0.61
incl.	*333.0*	*334.5*	*1.5*	*1.1*	*1366*	*0.07*	*0.70*	*0.89*
	349.5	*351.0*	*1.5*	*1.1*	*1398*	*0.55*	*0.10*	*0.27*
E10-203	265.5	267.0	1.5	1.2	138	0.10	0.12	0.33
	381.0	387.0	6.0	4.8	246	0.26	1.91	1.41
E10-204	309.0	312.0	3.0	2.2	160	0.05	0.08	0.16
	334.5	343.5	9.0	6.5	596	0.16	0.81	1.32
incl.	*337.5*	*339.0*	*1.5*	*1.1*	*1005*	*0.22*	*0.85*	*1.52*
	360.0	363.0	3.0	2.2	512	-	0.13	0.35
E10-205	300.0	303.0	3.0	2.2	334	0.03	0.07	0.17
	318.0	321.0	3.0	2.2	500	0.14	0.59	0.86
E10-206	289.5	291.0	1.5	0.8	189	-	0.15	0.20
	294.0	295.5	1.5	0.8	288	0.03	0.08	0.15
E10-207	306.0	334.5	28.5	20.6	394	0.06	0.35	0.56
incl.	*313.5*	*315.0*	*1.5*	*1.1*	*1137*	*0.09*	*1.04*	*1.70*
	327.0	*328.5*	*1.5*	*1.1*	*1595*	*0.24*	*1.16*	*1.89*
E10-208	313.5	334.5	21.0	17.5	278	0.08	0.40	0.59

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E10-210	249.0	252.0	3.0	2.3	152	0.05	0.12	0.21
	274.5	280.5	6.0	4.5	2821	0.23	0.38	0.38
incl.	*276.0*	*277.5*	*1.5*	*1.1*	*1354*	*0.12*	*0.15*	*0.19*
	277.5	*279.0*	*1.5*	*1.1*	*3172*	*0.15*	*0.54*	*0.59*
	279.0	*280.5*	*1.5*	*1.1*	*5962*	*0.59*	*0.58*	*0.48*
E10-211	333.0	334.5	1.5	1.1	196	0.13	0.26	0.56
	337.5	340.5	3.0	2.3	186	0.12	0.19	0.36
	348.0	367.5	19.5	14.9	1780	0.20	1.13	1.96
incl.	*348.0*	*349.5*	*1.5*	*1.1*	*1440*	*0.23*	*0.70*	*1.10*
	349.5	*351.0*	*1.5*	*1.1*	*7454*	*0.15*	*3.73*	*4.50*
	351.0	*352.5*	*1.5*	*1.1*	*3563*	*0.45*	*2.28*	*3.21*
	352.5	*354.0*	*1.5*	*1.1*	*3128*	*0.36*	*1.00*	*3.07*
	357.0	*358.5*	*1.5*	*1.1*	*1468*	*0.44*	*1.44*	*2.05*
	360.0	*361.5*	*1.5*	*1.1*	*1126*	*0.11*	*0.43*	*0.95*
	363.0	*364.5*	*1.5*	*1.1*	*1376*	*0.22*	*0.91*	*2.69*
	366.0	*367.5*	*1.5*	*1.1*	*1006*	*0.20*	*0.26*	*0.80*
E10-213	384.0	385.5	1.5	1.2	227	0.18	0.42	1.41
E10-214	222.0	225.0	3.0	2.2	192	0.05	0.15	0.20
	339.0	340.5	1.5	1.1	231	0.14	0.39	1.06
	357.0	366.0	9.0	6.7	150	0.07	0.76	0.76
E10-216	384.0	385.5	1.5	0.9	159	0.15	0.14	0.43
E10-217	342.0	345.0	3.0	2.9	284	0.14	0.25	0.65
	409.5	412.5	3.0	2.9	268	0.16	0.09	0.17
E10-219	396.0	406.5	10.5	8.4	453	0.10	0.32	0.65
	438.0	457.5	19.5	13.1	183	0.14	0.20	0.26
	481.5	483.0	1.5	1.0	321	0.10	1.36	1.80
E10-220	450.0	451.5	1.5	1.0	211	0.03	0.12	0.25
E10-221	327.0	330.0	3.0	1.7	59	0.30	1.03	2.56
	414.0	417.0	3.0	1.7	136	0.11	0.11	0.26
	513.0	517.5	4.5	2.6	291	1.31	0.79	1.55
	532.5	534.0	1.5	0.9	275	0.98	0.06	0.11
	538.5	540.0	1.5	0.9	175	0.59	0.46	0.94
	570.0	579.0	9.0	5.2	445	0.97	0.32	0.83
incl.	*576.0*	*577.5*	*1.5*	*0.9*	*1289*	*2.85*	*0.38*	*1.24*
	583.5	585.0	1.5	0.9	308	0.50	0.26	0.64
E10-222	375.0	381.0	6.0	3.4	271	0.28	0.32	0.65
	387.0	393.0	6.0	3.4	466	0.18	0.30	0.61
incl.	*388.5*	*390.0*	*1.5*	*0.8*	*1076*	*0.26*	*0.53*	*1.07*
E10-223	414.0	418.5	4.5	4.2	495	0.48	1.01	1.01
E10-224	420.0	423.0	3.0	2.0	234	0.19	1.33	1.94
	429.0	441.0	12.0	8.1	350	0.64	0.98	3.34

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E10-225	297.0	303.0	6.0	3.2	711	0.24	0.12	0.20
incl.	*300.0*	*303.0*	*3.0*	*1.6*	*1150*	*0.31*	*0.19*	*0.36*
	465.0	466.5	1.5	0.8	188	0.19	0.67	1.18
	471.0	472.5	1.5	0.8	109	0.33	0.82	1.18
	522.0	525.0	3.0	1.6	161	0.15	0.16	0.26
E10-228	327.0	330.0	3.0	2.3	282	0.44	0.31	1.07
ME10-229	322.9	330.1	7.3	5.0	4145	0.76	2.32	2.84
incl	*324.7*	*325.9*	*1.2*	*0.8*	*12650*	*0.86*	*7.51*	*5.78*
	326.4	*327.3*	*0.9*	*0.6*	*6690*	*1.27*	*2.71*	*4.36*
	327.8	*330.1*	*2.3*	*1.6*	*3200*	*0.94*	*2.21*	*3.20*
E10-231	3.0	4.5	1.5	0.8	156	0.24	0.07	0.02
	93.0	100.5	7.5	3.8	150	0.33	0.02	0.15
E10-232	327.0	330.0	3.0	2.0	520	0.55	0.61	1.86
	369.0	372.0	3.0	2.0	154	0.10	0.04	0.06
	387.0	390.0	3.0	2.0	311	0.41	0.26	0.54
	393.0	394.5	1.5	1.0	453	0.60	0.74	1.69
	418.5	423.0	4.5	3.0	188	0.62	0.48	0.99
E10-233	213.0	276.0	63.0	25.9	322	0.26	0.48	1.17
incl.	*261.0*	*262.5*	*1.5*	*0.6*	*1238*	*1.26*	*3.32*	*7.33*
	285.0	297.0	12.0	4.9	576	0.81	0.97	1.98
incl.	*286.5*	*288.0*	*1.5*	*0.6*	*1292*	*1.82*	*1.49*	*2.83*
	301.5	322.5	21.0	8.6	181	0.37	0.87	2.22
E10-235	288.0	291.0	3.0	2.3	168	0.14	0.38	0.78
	385.5	387.0	1.5	1.2	162	0.14	0.54	0.63
	396.0	412.5	16.5	12.8	208	0.49	0.70	1.27
	417.0	418.5	1.5	1.2	112	0.37	0.34	0.50
E10-236	469.5	474.0	4.5	4.0	297	0.56	0.43	0.91
	520.5	534.5	14.0	9.0	591	2.38	0.75	1.68
incl.	*520.5*	*522.0*	*1.5*	*1.0*	*1259*	*3.57*	*0.58*	*1.19*
	523.5	*525.0*	*1.5*	*1.0*	*1143*	*2.34*	*1.51*	*2.30*
	526.5	*528.0*	*1.5*	*1.0*	*706*	*5.25*	*1.84*	*2.74*
	551.0	554.0	3.0	1.9	204	0.57	0.04	0.09
	560.0	563.0	3.0	1.9	293	1.12	0.29	0.52
E10-239	408.0	417.0	9.0	6.3	191	0.27	0.42	0.75
	430.5	432.0	1.5	1.0	150	0.30	0.20	0.37
	436.5	438.0	1.5	1.0	202	0.27	0.52	1.44
	444.0	451.5	7.5	5.0	277	0.59	0.43	0.89
ME10-241	258.3	264.8	6.6	4.6	135	0.07	0.10	0.12
ME10-242	349.8	351.9	2.1	1.7	419	0.23	0.49	0.47

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
ME10-243	180.5	207.7	27.2	13.5	2936	5.73	2.17	3.45
incl.	*180.5*	*182.5*	*2.0*	*1.0*	*11050*	*17.70*	*2.33*	*2.61*
	182.5	*184.5*	*2.0*	*1.0*	*11200*	*43.05*	*0.73*	*1.07*
	184.5	*185.6*	*1.1*	*0.5*	*3645*	*6.18*	*2.67*	*4.38*
	190.8	*192.8*	*2.0*	*1.0*	*2935*	*2.27*	*6.12*	*8.20*
	192.8	*194.8*	*2.0*	*1.0*	*3045*	*4.23*	*6.04*	*8.41*
	194.8	*196.8*	*2.0*	*1.0*	*2725*	*1.51*	*4.09*	*7.61*
	196.8	*198.6*	*1.8*	*0.9*	*3740*	*2.05*	*4.55*	*7.95*
ME10-244	264.0	298.6	34.6	22.3	589	0.74	1.51	3.30
incl.	*284.5*	*286.5*	*2.0*	*1.3*	*1230*	*1.12*	*4.01*	*8.86*
	296.2	*298.6*	*2.4*	*1.5*	*2520*	*3.98*	*2.98*	*5.88*
	324.5	333.8	9.3	6.0	138	0.33	0.52	1.13
ME10-245	280.8	293.3	12.5	11.9	512	0.78	0.52	1.66
incl.	*289.4*	*290.8*	*1.4*	*1.3*	*1375*	*0.64*	*0.47*	*1.76*
ME10-247	121.6	123.6	2.0	1.9	173	0.09	0.13	0.30
	130.3	166.7	36.4	35.1	558	0.31	1.05	2.44
incl.	*158.2*	*159.8*	*1.6*	*1.5*	*1515*	*0.41*	*6.82*	*11.90*
	161.5	*162.8*	*1.3*	*1.2*	*4160*	*1.17*	*7.94*	*21.30*
	162.8	*164.0*	*1.3*	*1.2*	*1350*	*0.71*	*2.44*	*8.19*
	164.0	*164.7*	*0.7*	*0.7*	*3710*	*1.12*	*5.49*	*15.50*
E10-248	669.0	670.5	1.5	1.0	250	0.39	0.07	0.23
E10-252	24.0	25.5	1.5	1.0	74	2.15	0.00	0.01
E10-253	603.0	604.5	1.5	1.0	249	0.09	0.31	0.91
E10-256	559.5	565.5	6.0	4.3	288	0.18	0.15	0.30
ME10-257	143.0	158.4	15.4	14.5	200	0.09	0.35	0.51
	164.5	179.3	14.8	14.0	1325	0.50	1.76	3.24
incl.	*170.5*	*172.5*	*2.0*	*1.9*	*2960*	*0.56*	*4.53*	*8.53*
	174.8	*176.4*	*1.7*	*1.6*	*1370*	*0.83*	*1.85*	*3.54*
	178.0	*179.3*	*1.3*	*1.2*	*3650*	*1.79*	*5.03*	*8.98*
ME10-258	108.0	114.6	6.6	6.5	199	0.10	0.20	0.45
	122.9	146.6	23.7	23.3	803	0.24	0.81	1.07
incl.	134.4	136.4	2.0	2.0	1815	0.54	1.14	2.16
	140.4	142.4	2.0	2.0	1315	0.23	2.04	0.94
	144.2	146.6	2.4	2.3	3015	0.56	0.93	1.35
ME10-259	170.6	189.3	18.7	9.4	768	0.38	1.06	3.60
incl.	*172.1*	*174.5*	*2.3*	*1.2*	*2085*	*1.03*	*2.62*	*10.20*
	178.3	*180.3*	*2.0*	*1.0*	*3205*	*0.93*	*4.58*	*13.30*
ME10-260	245.0	259.0	14.0	10.5	311	0.67	0.22	0.53

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
ME10-261	199.5	205.3	5.8	2.9	306	0.22	0.09	0.19
	213.3	214.5	1.3	0.6	175	0.10	0.17	0.22
	225.0	245.4	20.4	10.2	1672	0.64	5.02	6.71
incl.	*225.0*	*227.0*	*2.0*	*1.0*	*3055*	*0.08*	*0.02*	*0.02*
	228.3	*229.8*	*1.5*	*0.8*	*2350*	*1.05*	*14.00*	*4.40*
	232.2	*234.0*	*1.8*	*0.9*	*5515*	*0.51*	*7.37*	*7.81*
	234.0	*236.2*	*2.2*	*1.1*	*3085*	*0.34*	*2.84*	*4.86*
	244.2	*245.4*	*1.2*	*0.6*	*1105*	*1.64*	*8.66*	*9.75*
ME11-262	269.5	283.0	13.5	10.0	736	0.66	2.51	2.06
incl.	*276.8*	*278.9*	*2.1*	*1.6*	*1595*	*0.44*	*9.46*	*3.93*
	278.9	*280.9*	*2.0*	*1.5*	*1145*	*1.76*	*0.42*	*0.42*
	280.9	*283.0*	*2.1*	*1.6*	*1195*	*1.26*	*0.84*	*1.70*
ME11-263	309.3	328.9	19.6	3.3	788	0.61	1.12	2.50
incl.	*309.3*	*311.6*	*2.3*	*0.4*	*1650*	*1.98*	*0.56*	*1.07*
	321.3	*323.3*	*2.0*	*0.3*	*1310*	*0.62*	*2.97*	*6.48*
	323.3	*324.9*	*1.6*	*0.3*	*2230*	*1.19*	*2.62*	*5.69*
	338.5	443.5	105.0	18.0	396	1.26	2.73	6.98
incl.	*437.0*	*439.0*	*2.0*	*0.3*	*1440*	*2.67*	*6.61*	*15.10*
E11-264	303.0	306.0	3.0	2.6	151	0.01	0.10	0.24
	429.0	430.5	1.5	1.3	141	0.01	0.14	0.38
	451.5	453.0	1.5	1.3	207	0.02	0.22	0.26
	457.2	483.0	25.8	22.2	380	0.09	0.68	1.17
incl.	*465.0*	*466.5*	*1.5*	*1.3*	*1167*	*0.27*	*2.39*	*3.41*
	478.5	*480.0*	*1.5*	*1.3*	*1027*	*0.11*	*1.44*	*2.30*
ME11-265	253.0	276.0	23.0	11.0	1975	0.46	1.14	2.24
incl.	*253.0*	*255.0*	*2.0*	*1.0*	*1335*	*0.31*	*0.88*	*2.76*
	255.0	*257.4*	*2.4*	*1.1*	*1205*	*1.12*	*0.37*	*0.81*
	266.0	*268.0*	*2.0*	*1.0*	*6230*	*0.37*	*5.27*	*5.66*
	268.0	*270.0*	*2.0*	*1.0*	*4870*	*0.20*	*1.42*	*3.91*
	270.0	*272.0*	*2.0*	*1.0*	*6275*	*0.33*	*3.34*	*7.91*
	310.7	316.1	5.4	2.6	147	0.22	0.24	0.74
E11-267	702.5	704.0	1.5	1.0	332	0.25	0.12	0.27
	710.0	711.5	1.5	1.0	174	0.13	0.02	0.05
E11-268	359.0	360.7	1.6	1.2	575	0.24	2.26	6.01
E11-269	408.0	411.0	3.0	2.1	142	0.03	0.21	0.61
	427.5	438.0	10.5	7.4	255	0.32	1.37	2.76
	442.5	450.0	7.5	5.3	478	0.48	0.84	1.89
incl.	*447.0*	*448.5*	*1.5*	*1.1*	*1279*	*1.35*	*1.79*	*3.50*
	463.5	474.0	10.5	7.4	131	0.26	0.37	0.76
	478.5	481.5	3.0	2.1	984	1.54	0.81	1.58
incl.	*478.5*	*480.0*	*1.5*	*1.1*	*1488*	*2.52*	*1.34*	*2.40*
	487.5	496.5	9.0	6.4	81	0.33	1.16	2.67

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E11-270	476.5	479.5	3.0	2.7	524	0.02	0.21	0.61
	488.5	493.0	4.5	4.1	601	0.04	0.12	0.33
E11-270A	54.0	57.0	3.0	2.7	305	0.02	0.14	0.27
	96.0	99.0	3.0	2.7	141	0.03	0.18	0.44
	127.5	129.0	1.5	1.4	995	0.24	0.14	0.25
	174.0	177.0	3.0	2.7	154	0.08	0.53	1.29
	183.0	184.5	1.5	1.4	308	0.22	0.86	1.41
	207.0	208.5	1.5	1.4	279	0.06	0.05	0.12
E11-271	360.0	372.0	12.0	10.0	151	0.11	0.15	0.39
	378.0	381.0	3.0	2.5	201	0.13	0.60	1.51
	475.5	480.0	4.5	4.1	470	0.27	0.18	0.52
incl.	*475.5*	*477.0*	*1.5*	*1.4*	*1056*	*0.58*	*0.10*	*0.30*
E11-273	330.0	385.5	55.5	35.0	229	0.92	1.10	2.30
incl.	*354.0*	*355.5*	*1.5*	*0.9*	*1020*	*1.45*	*2.32*	*5.10*
	355.5	*357.0*	*1.5*	*0.9*	*1349*	*1.28*	*0.80*	*1.36*
	376.5	*378.0*	*1.5*	*0.9*	*238*	*6.92*	*4.85*	*11.81*
E11-274	585.0	586.5	1.5	1.0	199	0.09	0.06	0.21
E11-277	315.0	316.5	1.5	1.3	256	0.04	1.32	4.22
	394.5	396.0	1.5	1.3	180	0.02	0.26	0.54
	411.0	412.5	1.5	1.3	323	-	0.22	0.52
	441.0	459.0	18.0	15.5	610	0.12	0.48	0.99
incl.	*442.5*	*444.0*	*1.5*	*1.3*	*1437*	*0.18*	*0.58*	*0.95*
	444.0	*445.5*	*1.5*	*1.3*	*2405*	*0.33*	*1.05*	*1.66*
	465.0	466.5	1.5	1.3	224	-	0.17	0.40
	478.5	486.0	7.5	6.5	199	0.05	0.45	1.02
E11-278	514.5	516.0	1.5	1.3	167	0.21	0.08	0.02
E11-280	432.0	435.0	3.0	1.7	157	0.08	0.35	0.86
	447.0	450.0	3.0	1.7	135	0.08	0.12	0.32
	459.0	463.5	4.5	2.6	304	0.12	0.53	0.96
	469.5	471.0	1.5	0.9	93	0.23	0.81	1.81
	496.5	510.0	13.5	7.8	136	2.62	0.26	0.52
incl.	*499.5*	*501.0*	*1.5*	*0.9*	*55*	*8.14*	*0.24*	*0.49*
	514.5	516.0	1.5	0.9	138	4.41	0.11	0.22
	543.0	546.0	3.0	1.8	120	0.67	0.58	1.07
	550.5	552.0	1.5	0.9	195	0.96	1.02	1.47
	579.0	580.5	1.5	0.9	216	1.18	0.31	0.62

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E11-281	178.5	180.0	1.5	1.0	303	0.07	0.44	1.64
	241.5	243.0	1.5	1.0	273	0.03	0.24	0.66
	299.0	300.5	1.5	1.0	154	0.12	0.22	0.68
	303.5	309.5	6.0	3.8	579	0.10	1.24	3.04
	356.0	359.0	3.0	1.9	512	0.15	1.26	1.93
	396.5	398.0	1.5	0.9	151	0.02	0.09	0.24
	438.5	440.0	1.5	0.9	435	0.25	1.11	1.67
	468.5	476.0	7.5	4.6	184	0.09	0.54	0.75
	482.0	483.5	1.5	1.0	845	0.23	0.15	0.43
	497.0	510.5	13.5	8.7	232	0.06	0.37	0.74
	516.5	518.0	1.5	1.0	379	0.07	0.92	0.81
	531.5	534.5	3.0	2.0	2247	15.20	0.30	0.57
incl.	*533.0*	*534.5*	*1.5*	*1.0*	*4323*	*29.92*	*0.32*	*0.50*
E11-282	312.0	313.5	1.5	1.0	153	0.06	0.16	0.29
	318.0	321.0	3.0	2.0	272	0.23	0.08	0.17
	330.0	333.0	3.0	2.0	160	0.21	0.00	0.01
	373.5	375.0	1.5	1.0	304	0.15	0.24	0.75
	387.0	396.0	9.0	6.0	416	0.48	0.52	1.17
	403.5	405.0	1.5	1.0	162	0.05	0.10	0.28
E11-283	303.0	304.5	1.5	1.3	737	0.11	0.41	0.54
	360.0	361.5	1.5	1.3	543	0.04	0.35	1.04
	454.5	456.0	1.5	1.3	405	0.14	0.64	2.61
	514.5	517.5	3.0	2.5	271	0.07	0.57	1.47
	535.5	537.0	1.5	1.3	761	0.13	0.40	0.66
	543.0	546.0	3.0	2.5	283	0.12	0.12	0.38
	556.5	567.0	10.5	8.5	375	0.18	0.30	0.88
E11-284	505.5	507.0	1.5	1.0	206	0.21	0.08	0.18
E11-285	391.5	411.0	19.5	16.0	401	1.66	0.80	1.00
incl.	*408.0*	*409.5*	*1.5*	*1.2*	*1662*	*10.22*	*1.04*	*1.04*
	424.5	426.0	1.5	1.2	177	0.34	0.69	2.36
E11-286	366.0	367.5	1.5	1.0	165	0.29	0.11	0.29
E11-287	339.0	340.5	1.5	0.9	229	0.10	0.19	0.45
	426.0	441.0	15.0	9.2	289	0.33	0.70	1.27
E11-288	423.0	427.5	4.5	2.3	187	0.47	0.25	0.63
	432.0	435.0	3.0	1.5	213	1.46	0.14	0.33
	780.0	781.5	1.5	0.8	68	0.18	2.24	3.52
E11-289	327.0	330.0	3.0	1.6	163	0.04	0.05	0.02
	387.0	388.5	1.5	0.8	1664	0.80	0.92	2.71
E11-290	289.5	291.0	1.5	1.0	387	1.03	0.35	0.87

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E11-291	286.5	288.0	1.5	0.8	445	0.08	0.48	1.42
	370.5	373.5	3.0	1.5	581	0.04	0.41	0.52
	378.0	379.5	1.5	0.8	178	0.01	0.13	0.10
	387.0	388.5	1.5	0.8	243	0.02	0.26	0.35
	391.5	393.0	1.5	0.8	229	0.03	0.16	0.34
	424.5	448.5	24.0	12.0	720	0.35	1.01	1.25
incl.	*436.5*	*438.0*	*1.5*	*0.8*	*1634*	*0.55*	*0.87*	*1.26*
	438.0	*439.5*	*1.5*	*0.8*	*2971*	*0.56*	*1.33*	*1.99*
	439.5	*441.0*	*1.5*	*0.8*	*1103*	*0.90*	*0.85*	*1.43*
	447.0	*448.5*	*1.5*	*0.8*	*1004*	*1.69*	*0.19*	*0.23*
	457.5	463.5	6.0	3.0	283	0.53	0.33	0.48
	478.5	480.0	1.5	0.8	107	2.77	0.01	0.03
	514.5	516.0	1.5	0.8	252	0.14	0.24	1.17
E11-293	376.5	378.0	1.5	0.8	136	0.02	0.18	0.42
	444.0	445.5	1.5	0.8	139	0.03	0.15	0.37
	471.0	472.5	1.5	0.8	185	0.01	0.18	0.51
	631.5	646.5	15.0	9.0	450	0.15	0.25	0.73
incl.	*636.0*	*637.5*	*1.5*	*0.9*	*1299*	*0.25*	*0.28*	*0.75*
E11-294	234.0	235.5	1.5	0.9	24	2.98	0.00	0.01
	324.0	328.5	4.5	2.6	487	3.84	0.01	0.03
incl.	*325.5*	*327.0*	*1.5*	*0.9*	*1105*	*7.48*	*0.01*	*0.02*
E11-296	294.0	297.0	3.0	2.0	180	0.17	0.04	0.05
	313.5	316.5	3.0	2.0	326	0.38	0.14	0.31
	321.0	325.5	4.5	3.0	382	0.30	0.22	0.66
E11-297	417.0	420.0	3.0	1.5	150	0.31	2.31	4.08
	430.5	433.5	3.0	1.5	198	0.38	0.85	3.12
	514.5	516.0	1.5	1.0	145	0.10	0.01	0.02
	534.0	537.0	3.0	2.0	357	0.66	1.24	2.19
	541.5	543.0	1.5	1.0	113	0.33	0.79	1.69
	546.0	549.0	3.0	2.0	116	0.55	1.50	2.88
	556.5	558.0	1.5	1.0	176	0.49	0.14	0.36
	588.0	616.5	28.5	19.0	482	0.35	1.28	1.71
incl.	*588.0*	*589.5*	*1.5*	*1.0*	*1239*	*0.51*	*5.71*	*6.26*
	606.0	*607.5*	*1.5*	*1.0*	*1852*	*0.57*	*0.76*	*1.76*
	607.5	*609.0*	*1.5*	*1.0*	*1063*	*0.47*	*0.48*	*1.12*
E11-298	240.0	240.8	0.8	0.6	127	0.11	0.40	1.92
E11-300	382.5	384.0	1.5	0.9	189	0.04	0.36	0.89
	475.5	477.0	1.5	0.9	313	0.75	0.09	0.23
	502.5	505.5	3.0	1.8	457	0.28	1.05	1.81
	508.5	510.0	1.5	0.9	361	0.50	0.15	0.33

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E11-301	433.5	436.5	3.0	2.0	222	0.03	0.27	0.40
	466.5	469.5	3.0	2.0	265	0.11	0.40	0.82
	516.0	517.5	1.5	1.0	495	0.10	0.06	0.10
E11-302	147.0	148.5	1.5	0.9	146	0.19	0.17	0.35
	264.0	265.5	1.5	0.9	343	0.11	0.55	1.49
	268.5	270.0	1.5	0.9	103	0.06	0.54	1.32
	651.0	652.5	1.5	0.9	233	0.11	0.12	0.37
	688.5	696.0	7.5	4.5	576	0.12	0.50	1.83
	700.5	703.5	3.0	1.8	225	0.19	0.42	0.94
	715.5	720.0	4.5	2.8	281	0.14	0.12	0.30
E11-304	646.5	649.5	3.0	1.4	227	0.12	0.13	0.27
	660.0	663.0	3.0	1.3	198	0.33	0.15	0.29
E11-305	354.0	357.0	3.0	1.7	271	0.17	0.05	0.08
	384.0	387.0	3.0	1.7	248	0.34	0.15	0.32
	417.0	420.0	3.0	1.7	425	0.56	0.54	1.53
	426.0	427.5	1.5	0.9	362	0.24	0.47	1.02
	436.5	447.0	10.5	6.0	321	0.71	0.19	0.37
	463.5	469.5	6.0	3.4	835	1.18	1.02	2.45
incl.	*463.5*	*465.0*	*1.5*	*0.9*	*1131*	*0.95*	*0.62*	*1.92*
	466.5	*468.0*	*1.5*	*0.9*	*1215*	*1.94*	*1.47*	*3.17*
	499.5	501.0	1.5	0.9	130	0.12	0.70	0.69
	532.5	534.0	1.5	1.0	317	0.61	1.42	2.76
	567.0	568.5	1.5	1.0	183	1.76	0.08	0.19
	625.5	627.0	1.5	0.9	201	0.26	0.74	1.45
	637.5	645.0	7.5	4.3	176	0.16	1.26	2.56
	655.5	657.0	1.5	0.9	109	0.41	0.64	1.53
	661.5	664.5	3.0	1.8	236	1.14	0.14	0.36
E11-306	597.0	598.5	1.5	0.8	196	0.49	0.07	0.16
E11-307	466.5	477.0	10.5	6.8	712	0.28	0.92	1.47
incl.	*468.0*	*469.5*	*1.5*	*1.0*	*2262*	*0.56*	*0.74*	*1.70*
	469.5	*471.0*	*1.5*	*1.0*	*1912*	*0.54*	*0.59*	*1.56*
	484.5	487.5	3.0	1.9	126	0.18	0.35	0.83

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E11-309	184.5	186.0	1.5	1.0	205	0.12	0.53	0.96
	394.5	396.0	1.5	1.0	276	0.06	0.26	0.93
	450.0	451.5	1.5	1.0	722	0.12	0.12	0.34
	459.0	460.5	1.5	1.0	150	0.05	0.12	0.08
	474.0	477.0	3.0	2.0	289	0.14	0.18	0.43
	520.5	546.0	25.5	17.0	776	0.41	0.20	0.43
incl.	*526.5*	*528.0*	*1.5*	*1.0*	*2290*	*0.08*	*0.83*	*1.74*
	528.0	*529.5*	*1.5*	*1.0*	*1407*	*0.14*	*0.00*	*0.00*
	541.5	*543.0*	*1.5*	*1.0*	*2948*	*1.39*	*0.42*	*0.73*
	543.0	*544.5*	*1.5*	*1.0*	*3301*	*3.98*	*0.21*	*0.35*
	562.5	564.0	1.5	1.0	172	0.08	0.12	0.17
E11-311	198.0	199.5	1.5	0.5	136	0.03	0.30	0.82
	507.0	523.5	16.5	5.5	251	0.51	0.29	0.63
	543.0	588.0	45.0	15.0	329	0.11	0.35	0.86
incl.	*570.0*	*571.5*	*1.5*	*0.5*	*1810*	*0.17*	*0.47*	*1.14*
	580.5	*582.0*	*1.5*	*0.5*	*1119*	*0.49*	*0.30*	*0.52*
	597.0	598.5	1.5	0.5	411	0.24	0.16	0.22
	613.5	618.0	4.5	1.5	154	0.12	0.09	0.26
E11-312	798.0	799.5	1.5	1.0	208	0.05	0.05	0.16
E11-315	226.5	228.0	1.5	0.7	151	0.10	0.30	0.68
	442.5	451.5	9.0	3.9	277	0.11	0.45	0.61
	465.0	475.5	10.5	4.6	284	0.11	0.34	0.36
	483.0	495.0	12.0	5.2	356	0.10	0.84	1.10
	502.5	504.0	1.5	0.7	466	0.20	0.39	0.80
	529.5	531.0	1.5	0.7	376	0.20	0.35	0.44
	558.0	561.0	3.0	1.3	892	0.30	0.16	0.25
incl.	*558.0*	*559.5*	*1.5*	*0.7*	*1541*	*0.55*	*0.24*	*0.31*
	577.5	580.5	3.0	1.3	477	0.07	0.20	0.29
	594.0	595.5	1.5	0.7	356	0.12	0.23	0.58
	601.5	603.0	1.5	0.7	261	0.02	0.09	0.21
	613.5	619.5	6.0	2.6	387	0.05	0.14	0.21
incl.	*613.5*	*615.0*	*1.5*	*0.7*	*1130*	*0.08*	*0.09*	*0.24*
	628.5	637.5	9.0	22.6	170	0.01	0.12	0.29
E11-316	334.5	343.5	9.0	4.6	146	0.21	0.27	0.55
	355.5	357.0	1.5	0.8	138	-	0.29	1.84
	361.5	363.0	1.5	0.8	171	-	0.06	0.13
	376.5	378.0	1.5	0.8	349	0.19	0.56	1.43
	424.5	439.5	15.0	7.7	361	0.88	2.29	2.49
incl.	*430.5*	*432.0*	*1.5*	*0.8*	*1207*	*4.55*	*2.00*	*2.90*

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E11-317	258.0	261.0	3.0	2.0	316	0.41	0.59	1.45
	273.0	276.0	3.0	2.0	317	0.19	0.01	0.02
	391.5	393.0	1.5	1.1	367	0.79	0.07	0.12
	408.0	411.0	3.0	2.1	219	0.35	0.50	1.11
	430.5	432.0	1.5	1.1	147	0.49	0.07	0.12
	438.0	439.5	1.5	1.1	340	0.24	1.03	1.43
E11-318	660.0	661.5	1.5	0.8	183	0.12	0.15	0.57
E11-319	292.5	295.5	3.0	0.7	1700	1.25	1.42	3.99
incl.	*292.5*	*294.0*	*1.5*	*0.3*	*3041*	*1.25*	*2.41*	*6.66*
	558.0	559.5	1.5	0.8	376	0.01	0.37	0.42
E11-321A	370.5	372.0	1.5	1.1	254	0.15	0.04	0.02
	381.0	382.5	1.5	1.1	451	0.10	0.09	0.19
	387.0	388.5	1.5	1.1	323	0.18	0.04	0.05
E11-322	574.5	589.5	15.0	8.6	306	0.68	1.83	3.27
	609.0	643.5	34.5	11.9	362	0.41	0.52	1.20
incl.	*609.0*	*610.5*	*1.5*	*0.5*	*1222*	*0.17*	*0.87*	*1.91*
	615.0	*616.5*	*1.5*	*0.5*	*1023*	*0.38*	*0.58*	*2.19*
	616.5	*618.0*	*1.5*	*0.5*	*1334*	*0.79*	*0.83*	*2.09*
	648.0	649.5	1.5	0.5	167	0.29	0.10	0.22
	652.5	654.0	1.5	0.5	310	0.59	0.29	1.18
	658.5	660.0	1.5	0.5	539	0.48	0.20	0.69
	667.5	673.5	6.0	2.1	218	0.20	0.18	0.63
	682.5	690.0	7.5	2.6	561	0.91	0.39	1.47
incl.	*687.0*	*688.5*	*1.5*	*0.5*	*1624*	*3.35*	*1.21*	*4.83*
E11-323	342.0	349.5	7.5	2.5	198	0.12	0.10	0.30
	441.0	442.5	1.5	0.5	285	0.31	0.01	0.02
	478.5	480.0	1.5	0.5	147	0.19	0.01	0.01
	496.5	498.0	1.5	0.5	147	0.10	0.07	0.19
	522.0	523.5	1.5	0.5	496	0.18	0.11	0.40
E11-324	175.5	177.0	1.5	0.8	223	0.23	0.12	0.43
	184.5	186.0	1.5	0.8	175	0.06	0.10	0.15
	526.5	528.0	1.5	0.8	541	0.32	0.60	1.63
	555.0	558.0	3.0	1.5	220	0.33	0.03	0.06
	642.0	645.0	3.0	1.5	136	0.05	0.15	0.57
E11-329	309.0	310.5	1.5	0.9	130	0.01	0.31	0.74
	382.5	397.5	15.0	9.0	325	0.17	0.54	0.66
	414.0	418.5	4.5	2.7	200	0.12	0.21	0.45
	427.5	438.0	10.5	6.3	379	0.66	0.49	0.43
	457.5	459.0	1.5	0.9	125	1.78	0.12	0.33
E11-330	342.0	357.0	15.0	7.5	171	0.06	0.28	0.62
	369.0	370.5	1.5	0.8	305	0.22	0.18	0.40

Escobal - Significant Drill Intercepts (150 AgEq g/t cutoff)

Hole ID	From (m)	To (m)	Drilled Length (m)	Est True Width (m)	Ag (g/t)	Au (g/t)	Pb (%)	Zn (%)
E11-333	273.0	274.5	1.5	1.0	152	0.13	0.24	0.87
	334.5	339.0	4.5	3.1	522	0.22	0.38	0.99
	468.0	474.0	6.0	4.1	376	0.13	0.86	0.87
	513.0	514.5	1.5	1.0	213	-	0.19	0.50
	523.5	525.0	1.5	1.0	204	0.05	0.20	0.51
E11-335	507.0	511.5	4.5	2.2	231	0.41	1.14	1.62
E11-337	487.5	493.5	6.0	3.2	227	0.57	1.49	3.34
	501.0	519.0	18.0	9.5	199	0.32	1.95	1.64
E11-338	354.0	355.5	1.5	0.9	258	0.11	0.28	0.64
	565.5	567.0	1.5	0.9	154	0.15	0.24	0.38
E11-339	649.5	651.0	1.5	1.5	248	0.33	0.07	0.24
E11-340	441.0	481.5	40.5	23.8	222	0.68	1.80	3.15
	492.0	493.5	1.5	0.9	162	0.18	1.32	2.24
	498.0	543.0	45.0	26.5	204	0.31	0.37	0.81
E11-341	400.5	403.5	3.0	1.8	605	0.45	0.28	0.43
	420.0	423.0	3.0	1.8	340	0.12	0.06	0.11
	444.0	447.0	3.0	1.8	371	0.22	0.07	0.18
	564.0	565.5	1.5	0.9	410	0.50	0.75	1.49
	760.5	762.0	1.5	0.8	121	0.17	1.12	1.85
	772.5	774.0	1.5	0.8	234	0.50	3.89	11.55
	796.5	801.0	4.5	2.4	62	0.27	4.22	6.91
E11-344	243.0	244.5	1.5	0.8	154	0.06	0.14	0.39
	423.0	426.0	3.0	1.3	273	0.11	0.11	0.21
	438.0	439.5	1.5	0.6	139	0.06	0.68	1.34
	472.5	474.0	1.5	0.6	546	0.22	0.53	1.37
	481.5	493.5	12.0	5.1	460	0.14	0.66	1.02
incl.	*492.0*	*493.5*	*1.5*	*0.6*	*1240*	*0.39*	*0.72*	*0.91*
E11-345	375.0	378.0	3.0	2.4	223	0.08	0.30	1.33
	454.5	456.0	1.5	1.2	189	0.07	0.30	0.63
	487.5	493.5	6.0	4.7	352	0.08	1.02	2.29
	504.0	505.5	1.5	1.2	291	0.20	0.28	0.62
PZ11-01	36.0	40.5	4.5	3.0	239	0.32	0.24	0.19
	78.0	79.5	1.5	1.0	185	0.22	0.27	0.58
	82.5	87.0	4.5	3.0	419	0.23	0.31	0.39
	96.0	103.5	7.5	5.0	288	0.19	0.29	0.75
	114.0	123.0	9.0	6.0	816	1.02	0.87	1.23
incl.	*114.0*	*115.5*	*1.5*	*1.0*	*3526*	*0.59*	*3.81*	*4.14*
PZ11-02	245.5	247.0	1.5	1.0	186	0.14	0.05	0.12
	260.5	262.0	1.5	1.0	311	0.08	0.05	0.36

Appendix C

Escobal Project – Descriptive Statistics – Drill Samples

Silver Assays

Silver Assays **Areas 10 - 11- 20**

Domain	Valid N	Mean (g/t)	Median (g/t)	Std.Dev. (g/t)	CV	Minimum (g/t)	Maximum (g/t)
100	2196	26.3	12.3	40.5	1.54	0.0	617.4
100 (capped)	2196	26.3	12.3	40.5	1.54	0.0	617.4
200	290	261.3	186.0	259.1	0.99	1.3	2538.8
200 (capped)	290	261.3	186.0	259.1	0.99	1.3	2538.8
300	2	1720.5	1775.2	67.9	0.04	1663.9	1775.2
300 (capped)	2	1720.5	1775.2	67.9	0.04	1663.9	1775.2

Silver Assays **Areas 15 - 16**

Domain	Valid N	Mean (g/t)	Median (g/t)	Std.Dev. (g/t)	CV	Minimum (g/t)	Maximum (g/t)
110	258	81.1	30.3	243.1	3.00	0.0	3055.0
110 (capped)	258	67.7	30.3	110.0	1.62	0.0	610.0
100	1525	40.4	26.0	44.1	1.09	0.0	539.1
100 (capped)	1525	40.2	26.0	42.6	1.06	0.0	310.0
200	1580	357.3	248.3	320.0	0.90	5.7	2864.4
200 (capped)	1580	356.0	248.3	312.3	0.88	5.7	1900.0
300	185	2663.2	1985.7	2262.7	0.85	81.0	14765.1
300 (capped)	185	2617.1	1985.7	2106.7	0.81	81.0	14765.1

Silver Assays **Area 30**

Domain	Valid N	Mean (g/t)	Median (g/t)	Std.Dev. (g/t)	CV	Minimum (g/t)	Maximum (g/t)
100	1633	24.7	12.9	32.1	1.30	0.0	494.7
100 (capped)	1633	24.4	12.9	29.6	1.21	0.0	200.0
200	123	297.1	199.8	299.2	1.01	27.2	3040.7
200 (capped)	123	283.6	199.8	206.3	0.73	27.2	1000.0
300	4	1553.3	941.6	1490.1	0.96	731.7	4323.3
300 (capped)	4	988.6	941.6	287.3	0.29	731.7	1500.0

Silver Assays **Area 35**

Domain	Valid N	Mean (g/t)	Median (g/t)	Std.Dev. (g/t)	CV	Minimum (g/t)	Maximum (g/t)
100	1167	41.9	23.1	70.0	1.67	0.0	1541.0
100 (capped)	1167	40.7	23.1	53.3	1.31	0.0	500.0
200	518	294.9	228.6	227.5	0.77	7.5	1810.2
200 (capped)	518	294.0	228.6	222.0	0.76	7.5	1350.0
300	179	2179.1	1299.0	2205.3	1.01	82.2	13960.7
300 (capped)	179	2096.0	1299.0	1893.2	0.90	82.2	7500.0

Gold Assays

Gold Assays — **Areas 10 - 11- 20**

Domain	Valid N	Mean (g/t)	Median (g/t)	Std.Dev. (g/t)	CV	Minimum (g/t)	Maximum (g/t)
10	1507	0.020	0.014	0.032	1.590	0.000	0.798
10 (capped)	1507	0.020	0.014	0.024	1.239	0.000	0.200
100	995	0.128	0.093	0.119	0.927	0.000	1.756
100 (capped)	995	0.127	0.093	0.111	0.871	0.000	0.750
200	61	0.926	0.701	0.627	0.677	0.152	2.977
200 (capped)	61	0.926	0.701	0.627	0.677	0.152	2.977
300	2	3.242	3.241	0.733	0.226	2.643	3.840
300 (capped)	2	3.242	3.241	0.733	0.226	2.643	3.840

Gold Assays — **Areas 15 - 16**

Domain	Valid N	Mean (g/t)	Median (g/t)	Std.Dev. (g/t)	CV	Minimum (g/t)	Maximum (g/t)
110	258	0.207	0.090	0.312	1.508	0.000	1.500
110 (capped)	258	0.253	0.090	0.583	2.308	0.000	4.834
10	651	0.025	0.021	0.023	0.935	0.000	0.190
10 (capped)	651	0.025	0.021	0.023	0.935	0.000	0.190
100	1855	0.162	0.125	0.132	0.817	0.000	1.128
100 (capped)	1855	0.162	0.125	0.132	0.817	0.000	1.128
200	656	0.796	0.673	0.555	0.697	0.000	4.608
200 (capped)	656	0.796	0.673	0.555	0.697	0.000	4.608
300	125	4.495	2.800	7.553	1.680	0.453	76.524
300 (capped)	125	3.608	2.800	2.647	0.734	0.453	12.000

Gold Assays — **Area 30**

Domain	Valid N	Mean (g/t)	Median (g/t)	Std.Dev. (g/t)	CV	Minimum (g/t)	Maximum (g/t)
10	1598	0.020	0.012	0.028	1.389	0.000	0.243
10 (capped)	1598	0.020	0.012	0.027	1.361	0.000	0.180
100	178	0.149	0.124	0.099	0.667	0.000	0.937
100 (capped)	178	0.147	0.124	0.089	0.607	0.000	0.450
200	15	0.590	0.454	0.335	0.568	0.198	1.389
200 (capped)	15	0.590	0.454	0.335	0.568	0.198	1.389
300	8	5.553	1.245	10.791	1.943	0.948	29.918
300 (capped)	8	5.553	1.245	10.791	1.943	0.948	29.918

Gold Assays — **Area 35**

Domain	Valid N	Mean (g/t)	Median (g/t)	Std.Dev. (g/t)	CV	Minimum (g/t)	Maximum (g/t)
10	1052	0.034	0.023	0.038	1.113	0.000	0.547
10 (capped)	1052	0.033	0.023	0.033	1.001	0.000	0.180
100	472	0.170	0.150	0.102	0.599	0.000	0.812
100 (capped)	472	0.169	0.150	0.100	0.591	0.000	0.600
200	190	0.582	0.538	0.334	0.573	0.020	2.769
200 (capped)	190	0.576	0.538	0.299	0.519	0.020	1.750
300	156	5.173	2.020	13.857	2.679	0.363	167.654
300 (capped)	156	4.441	2.020	6.806	1.533	0.363	40.000

Lead Assays

Lead Assays **Areas 10 - 11- 20**

Domain	Valid N	Mean (%)	Median (%)	Std.Dev. (%)	CV	Minimum (%)	Maximum (%)
100	2808	0.020	0.009	0.038	1.85	0.000	1.030
100 (capped)	2808	0.020	0.009	0.035	1.73	0.000	0.500
200	672	0.227	0.130	0.277	1.22	0.000	2.240
200 (capped)	672	0.226	0.130	0.271	1.20	0.000	2.000
300	6	2.350	2.260	0.608	0.26	1.630	3.320
300 (capped)	6	2.350	2.260	0.608	0.26	1.630	3.320

Lead Assays **Areas 15 - 16**

Domain	Valid N	Mean (%)	Median (%)	Std.Dev. (%)	CV	Minimum (%)	Maximum (%)
110	258	0.071	0.032	0.114	1.62	0.000	0.904
110 (capped)	258	0.071	0.032	0.114	1.62	0.000	0.904
100	996	0.027	0.015	0.037	1.37	0.000	0.478
100 (capped)	996	0.027	0.015	0.034	1.26	0.000	0.220
200	1949	0.361	0.244	0.367	1.02	0.001	4.150
200 (capped)	1949	0.359	0.244	0.353	0.98	0.001	2.800
300	520	3.385	2.080	3.954	1.17	0.068	43.595
300 (capped)	520	3.214	2.080	2.970	0.92	0.068	15.000

Lead Assays **Area 30**

Domain	Valid N	Mean (%)	Median (%)	Std.Dev. (%)	CV	Minimum (%)	Maximum (%)
100	2273	0.021	0.013	0.025	1.16	0.000	0.315
100 (capped)	2273	0.021	0.013	0.024	1.13	0.000	0.200
200	406	0.168	0.128	0.166	0.99	0.002	2.410
200 (capped)	406	0.165	0.128	0.135	0.82	0.002	1.000
300	10	1.471	1.340	0.607	0.41	0.621	2.630
300 (capped)	10	1.471	1.340	0.607	0.41	0.621	2.630

Lead Assays **Area 35**

Domain	Valid N	Mean (%)	Median (%)	Std.Dev. (%)	CV	Minimum (%)	Maximum (%)
100	818	0.026	0.017	0.028	1.08	0.000	0.283
100 (capped)	818	0.026	0.017	0.025	0.98	0.000	0.120
200	948	0.191	0.147	0.158	0.82	0.001	1.360
200 (capped)	948	0.191	0.147	0.154	0.81	0.001	1.000
300	190	1.444	1.070	1.414	0.98	0.013	10.200
300 (capped)	190	1.402	1.070	1.223	0.87	0.013	6.000

Zinc Assays

Zinc Assays **Areas 10 - 11- 20**

Domain	Valid N	Mean (%)	Median (%)	Std.Dev. (%)	CV	Minimum (%)	Maximum (%)
100	2808	0.054	0.027	0.090	1.67	0.000	1.430
100 (capped)	2808	0.053	0.027	0.078	1.47	0.000	0.800
200	626	0.367	0.245	0.398	1.08	0.003	3.520
200 (capped)	626	0.362	0.245	0.364	1.01	0.003	2.200
300	57	2.251	1.830	1.396	0.62	0.690	7.330
300 (capped)	57	2.162	1.830	1.139	0.53	0.690	5.000

Zinc Assays **Areas 15 - 16**

Domain	Valid N	Mean (%)	Median (%)	Std.Dev. (%)	CV	Minimum (%)	Maximum (%)
110	258	0.138	0.069	0.204	1.48	0.001	1.760
110 (capped)	258	0.131	0.069	0.167	1.27	0.001	0.900
100	996	0.075	0.044	0.098	1.31	0.000	1.350
100 (capped)	996	0.074	0.044	0.093	1.25	0.000	0.800
200	1443	0.455	0.378	0.343	0.75	0.002	3.050
200 (capped)	1443	0.455	0.378	0.343	0.75	0.002	3.050
300	1021	3.643	1.960	4.685	1.29	0.035	36.100
300 (capped)	1021	3.643	1.960	4.685	1.29	0.035	36.100

Zinc Assays **Area 30**

Domain	Valid N	Mean (%)	Median (%)	Std.Dev. (%)	CV	Minimum (%)	Maximum (%)
100	2273	0.060	0.037	0.068	1.13	0.001	0.748
100 (capped)	2273	0.060	0.037	0.065	1.09	0.001	0.500
200	387	0.342	0.273	0.259	0.76	0.012	2.610
200 (capped)	387	0.341	0.273	0.250	0.73	0.012	2.000
300	27	2.130	1.540	1.528	0.72	0.919	6.660
300 (capped)	27	2.130	1.540	1.528	0.72	0.919	6.660

Zinc Assays **Area 35**

Domain	Valid N	Mean (%)	Median (%)	Std.Dev. 0.000	CV	Minimum (%)	Maximum (%)
100	818	0.072	0.047	0.077	1.07	0.001	0.629
100 (capped)	818	0.071	0.047	0.074	1.04	0.001	0.500
200	899	0.352	0.294	0.253	0.72	0.003	2.760
200 (capped)	899	0.349	0.294	0.235	0.67	0.003	1.500
300	241	1.759	1.370	1.420	0.81	0.124	11.200
300 (capped)	241	1.727	1.370	1.260	0.73	0.124	7.000

Appendix D

Escobal Project – Geotechnical Assessment

PAKALNIS & ASSOCIATES
Mining Engineers

Preliminary Geotechical Assessment – Escobal Project
to
Tahoe Resources Inc
No. Tahoe 1/10

October 21, 2010

Phone: (604) 988-5076 Fax: (604) 988-5029 Cell: (604) 671-6114 e-mail: pakalnis@direct.ca
1441 McNair Drive, North Vancouver, B.C., V7K 1X4

IIPAKALNIS & ASSOCIATES
Mining Engineers

October 21, 2010

Charlie Muerhoff/Technical Services Director
Tahoe Resources Inc.
5190 Neil Road, Suite 460
Reno Nevada
89502 USA

Ref. : Geotechnical Review of Escobal Conceptual Mining Method

This report summarizes a review of the geotechnical parameters as compiled by Tahoe Resources under guidance of R. Pakalnis. The purpose was to provide initial estimates in terms of design dimensions for planning purposes. The preliminary assessment and compilation is summarized in Appendix I. The initial data provided was obtained from geotechnical core logging performed by Goldcorp and Tahoe Resources' personnel. The corelogs/photos/sections/plans were compiled/assessed by Tahoe Resources and are summarized in Figure 1. The initial estimates for purposes of design in terms of rock mass assessment are as follows:

	DESIGN RMR_{76}
HW	52%-54%
ORE	56%-64%
FW	50%-54%

This was employed in providing initial stoping dimensions as summarized in Figure 2 whereby the HW dimensions for a 70° dipping HW and RMR_{76} of 54% would dictate that a Hydraulic Radius (HR) of 5m should be employed for ~1m of HW slough. This largely is a stope that is 30m in height (20m + 5m + 5m) x 15m in strike length. It must be noted a shallow 45° HW would result in HW slough approaching 2m if similar stope dimensions were employed. The back span largely within the ore is shown in Figure 2 whereby the design RMR_{76} has been estimated to be 56% less 10% due to shallow joints or 45%+. This would dictate that for a 5m span that only local support is required.

The proposed mining sequence is detailed in Appendix I and summarized in Figure 3A (C. Muerhoff/Tahoe). It largely shows mining along strike retreating from the HW to the FW with stope spans along strike being 20m and heights of 30m. This will result in a HR of 6m which should result in wall slough of ~1m+ for the 70° dipping HW. It is recommended to support the upper and lower drill drives to reduce the potential for wall slough as well as the potential for migration above the level. The subsequent longitudinal stope will have its HW be located in paste with the FW in ore of similar RMR_{76} quality to that of the HW.

Phone: (604) 988-5076 Fax: (604) 988-5029 Cell: (604) 671-6114 e-mail: pakalnis@direct.ca
1441 McNair Drive, North Vancouver, B.C., V7K 1X4

The stopes will be backfilled as shown in Figure 3B with consolidated paste. The schematic shown in Figure 3B shows mining from HW to FW which will result in the filled stope to be hanging over the immediate FW stope.. This requires a competent fill in order to minimize any fill dilution. It is recommended that a vertical fill wall be maintained. An alternative is to mine from FW to HW and have the HW in ore which of similar rock quality to the waste HW.

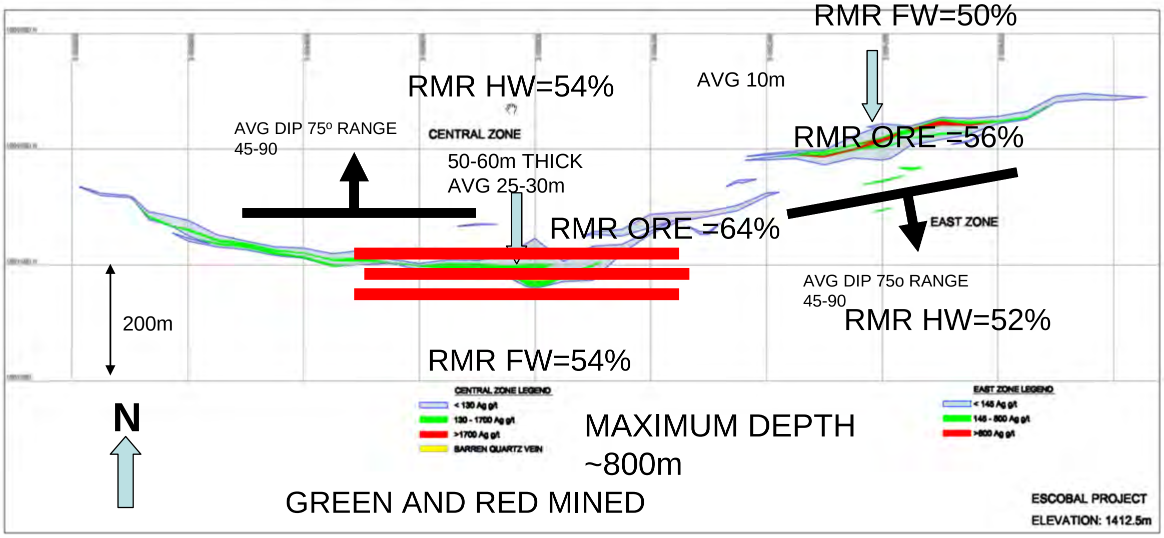
The overall observations and conclusions are summarized in Figure 4 and largely show that the concept as proposed by Tahoe Resources will result in a workable and safe mining method. The review is largely conceptual at this stage and should be augmented when a detailed mining plan/geology/rock mass model is available for purposed of providing a comprehensive detail feasibility assessment.

Please contact me if any questions or comments arise.

LICENSED PROFESSIONAL ENGINEER
R. T. PAKALNIS
PROVINCE OF ONTARIO

Pakalnis & Associates

Dr. Rimas Pakalnis, P.Eng.



POZO E10-195
HW 68% RMR

POZO E10-195
ORE 61% RMR

POZO E09-154 FW 62% RMR

TAHOE RESOURCES INC. – ESCOBAL PROJECT

FIGURE 1: PRELIMINARY DESIGN PARAMETERS - GEOTECH

PAKALNIS & ASSOCIATES

ASSUMPTIONS

N' = Q' X A X B X C

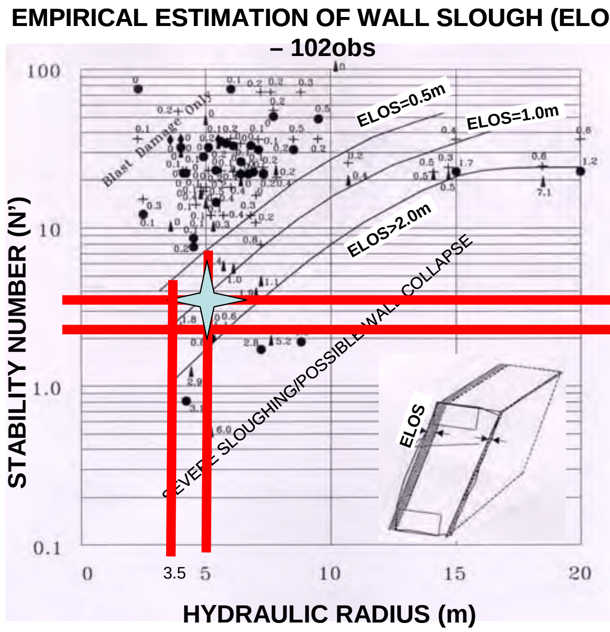
A = 1 (RELAXED)

B = 0.2

C_{HW} = 8-6cosφ

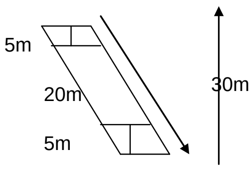
C_{FW} = 8

* ELOS LINES APPLY TO UNSUPPORTED STOPE SURFACE



DESIGN RMR=54% AND DIP 70°

RMR=54% AND DIP 45°



H=30/SIN 70 = 32m

HR =5m = H X L/(2H + 2L)

= 32 X L/(64 + 2L)

L=15m

RMR design=54% Q=3

DIP=70° C=5.9

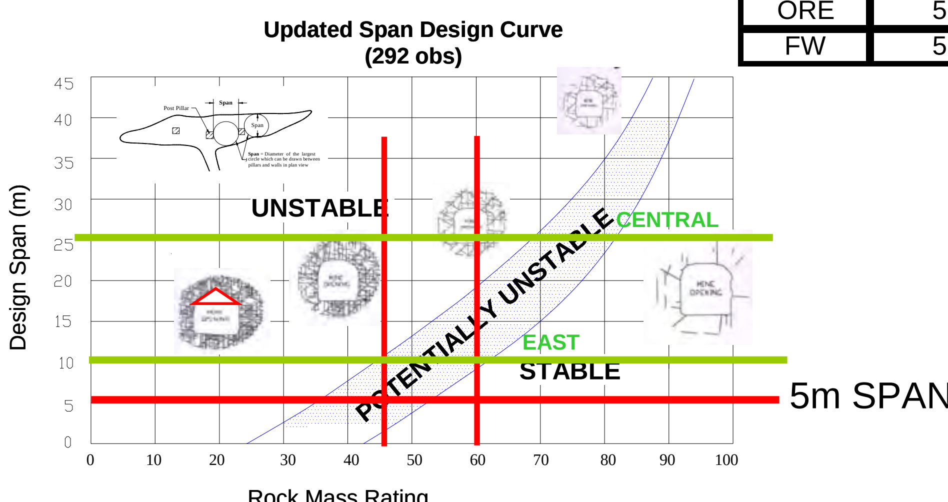
N'=3 X 1 X 0.2 X 5.9=3.5

RMR design=54% Q=3

DIP=45° C=3.8

N'=3 X 1 X 0.2 X 3.8=2.3

	DESIGN RMR_{76}
HW	52%-54%
ORE	56%-64%
FW	50%-54%

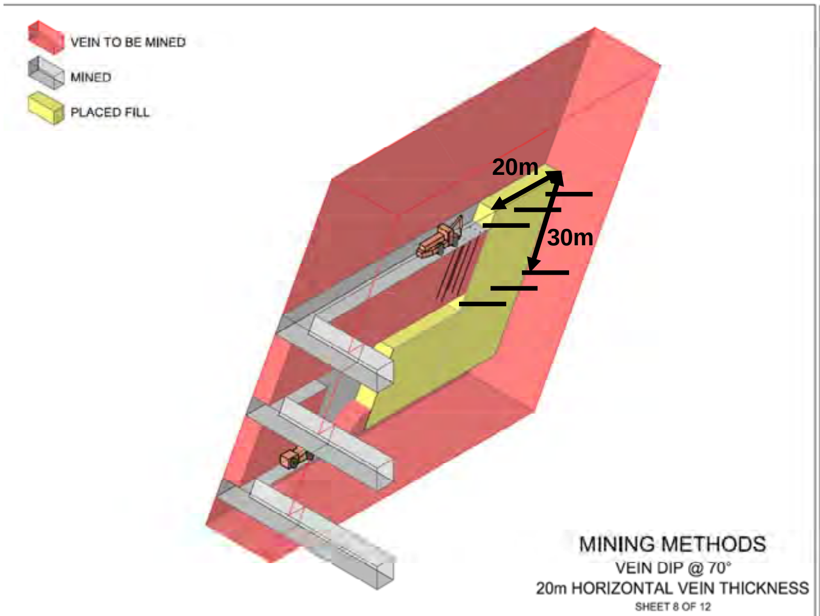


DESIGN RMR REDUCED BY 10% FOR ORE DUE TO FLAT JOINTS. RMR DESIGN 45%-60%

TAHOE RESOURCES INC. – ESCOBAL PROJECT

FIGURE 2: PRELIMNARY DESIGN CONSTRAINTS

PAKALNIS & ASSOCIATES



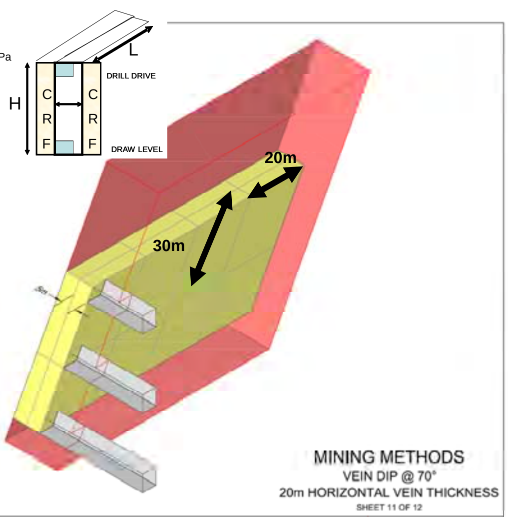
A) LONGHOLE MINING

EXPOSED 20m ON STRIKE X 30m VERTICAL (AT 70º). BACK SPAN IS 5m, RMR OF 54%. HR=6m ESTIMATE ~1m+ OF WALL SLOUGH. NOTE BOLT UPPER AND LOWER DRILL DRIVES AT HW TO MINIMIZE

$$\sigma_{ucs} = \gamma H / (1 + H/L)$$

σ_{ucs} = FILL COMPRESSIVE STRENGTH, MPa
γ = FILL DENSITY, MN/m3
H = HEIGHT OF EXPOSURE, m
L = WIDTH OF EXPOSURE, m

ESTIMATE FILL DENSITY AT SG=2



B) PASTE/BACKFILL OF STOPES

H=30m, L=20m

UCS=2t/m3 X 30/(1+30/20)=

24t/m2 = 0.24MPa = 35psi.

GAURANTEED 10%. MOST LIKELY 5% CEMENT CONTENT. NEED ONLY 0.5MPa UCS.

MINE IN 5m SLICES FROM HW TO FW OR VICE VERSA. STRIKE WITH FILL WALL . MINE AGAINST FILL WALL OF 20m ALONG STRIKE. NOTE VERTICAL WALL OF PASTE IS PREFERRED AS IF IS HW AT 70o WILL BE LESS STABLE AND OVERHANGING SUBSEQUENT STOPE TO BE MINED RESULTING IN POTENTIAL DILUTION AND/OR INCREASED STRENGTHS REQUIRED.

TAHOE RESOURCES INC. – ESCOBAL PROJECT

FIGURE 3: PRELIMINARY MINING METHOD OPTION(MM-70-FULL)

PAKALNIS & ASSOCIATES

OBSERVATIONS/CONCLUSIONS

• THE CONCEPT PROPOSED FOR THE ESCOBAL PROJECT BY TAHOE(C. MUERHOFF) IS WORKABLE AT A CONCEPTUAL STAGE (MM-70-FULL)

• THE RMR AS DETERMINED FROM LOGS AND COMPILED BY C. MUERHOFF SHOULD BE VERIFIED UPON EXPOSURE/ASSESSMENT

• PRELIMINARY SPANS FOR THE LONGHOLE ARE BASED UPON AN INCLINED HW OF ~70º AND RMR_{76} OF 54% THIS WOULD INDICATE THAT ~1m+ OF WALL SLOUGH FROM THE HW WILL OCCUR (HR=6m). NOTE TO REDUCE THE WALL SLOUGH TO 1m THE HR SHOULD APPROACH 5m OR SPAN OF 15m vs 20m STRIKE. SUPPORT ALL DRILL/DRAW LEVELS IN HW ROCK.

• PASTE AS OUTLINED IN CONCEPTUAL IS WORKABLE. IT IS RECOMMENDED THAT THE PASTE WALL BE VERTICAL IN ORDER TO MAXIMIZE STABILITY.

• THE BACK SPAN LIMITED TO 5m AND THE ORE RMR ESTIMATED AT 45%-60% WILL RESULT IN STABLE CONDITIONS EMPLOYING LOCAL SUPPORT.

• DEPTH OF MINING APPROACHES 800m AND SHOULD BE EVALUATED UPON MOVING TOWARDS DETAILED FEASIBILITY. THE METHOD PROPOSED LARGELY DOES NOT INCORPORATE PILLARS.

• THE STUDY IS LARGELY PRELIMARY AND CONCEPTUAL AND SHOULD BE REASSESSED UPON A MINE PLAN HAS BEEN DEVELOPED AND CONCERNS IDENTIFIED.

TAHOE RESOURCES INC. – ESCOBAL PROJECT

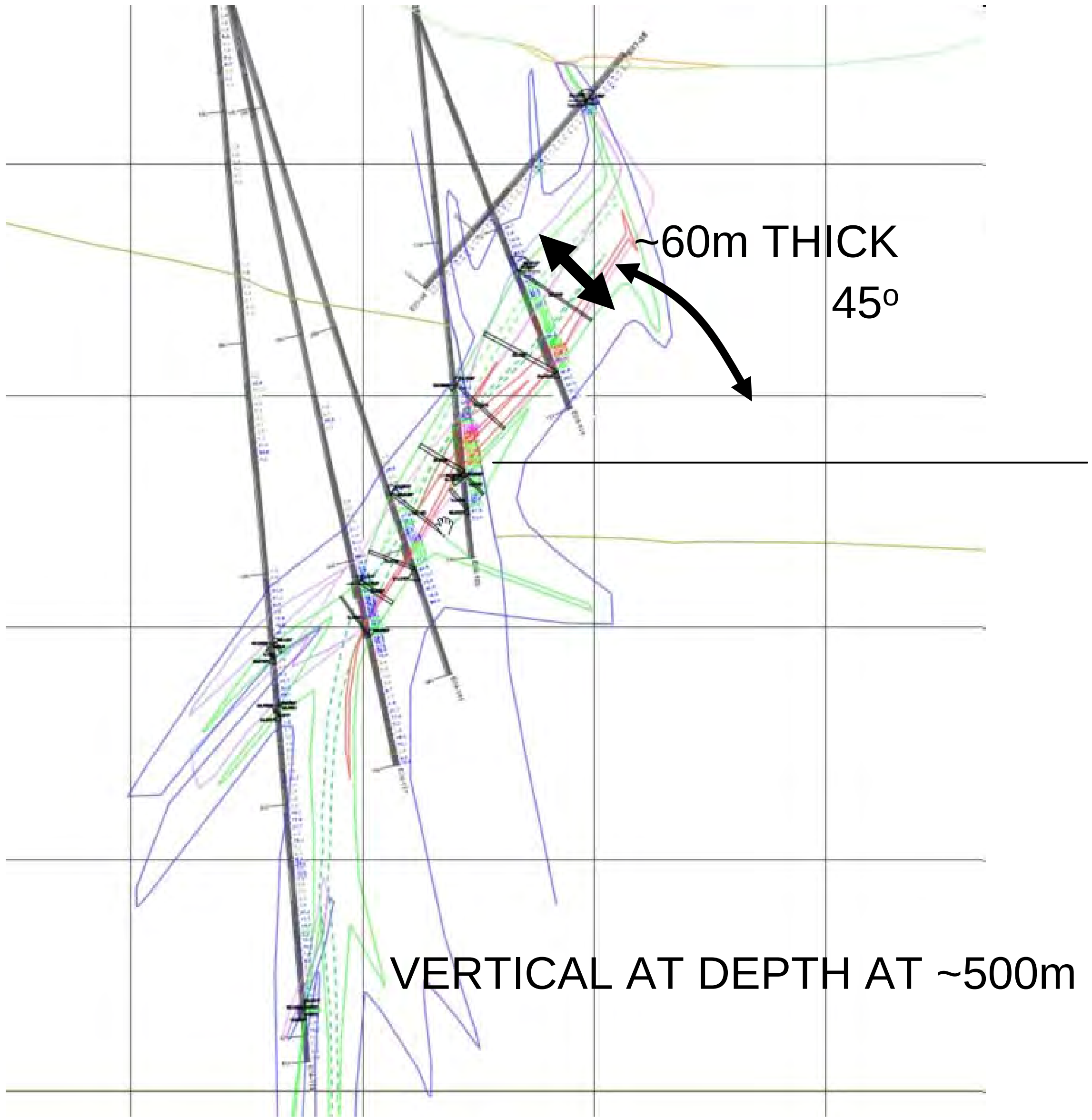
FIGURE 4: OBSERVATIONS/CONCLUSIONS

PAKALNIS & ASSOCIATES

APPENDIX I: ESCOBAL PROJECT - ANALYSIS

DIP IS TO THE NORTH

CENTRAL ZONE SECTION 806600 (FURTHER TO WEST)



HOLE E09-154 CENTRAL ZONE

HW (NORTH). NO EXPOSURE UNDERGROUND. DISCOVERED 2007

HW

ORE IS

RQD=5%, Q'=0.35 ,
RMR=34% HW

ORE NO CLAY.

AVG. 46% RMR, Q'=3.96

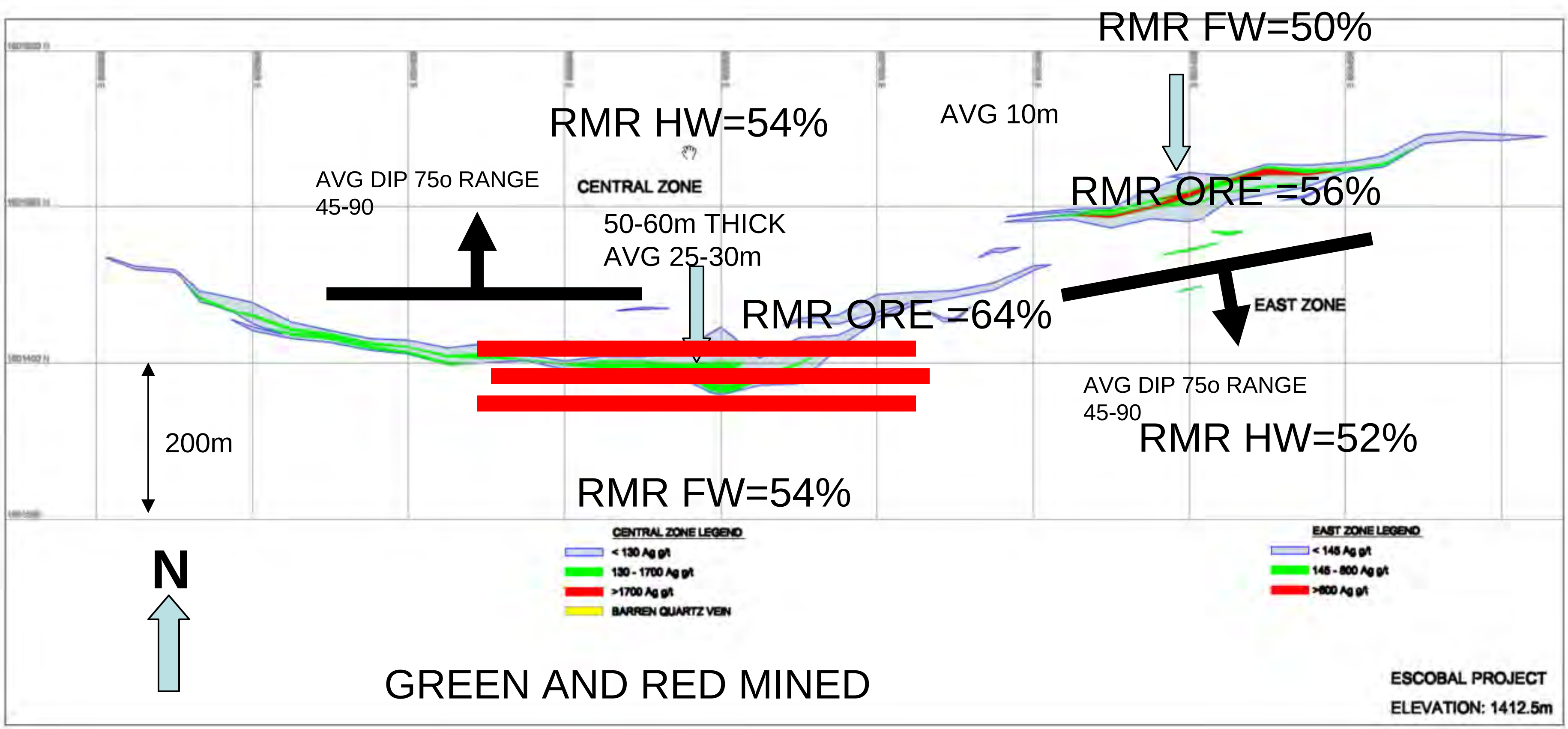
FW

Q'=7.5, RMR=62%

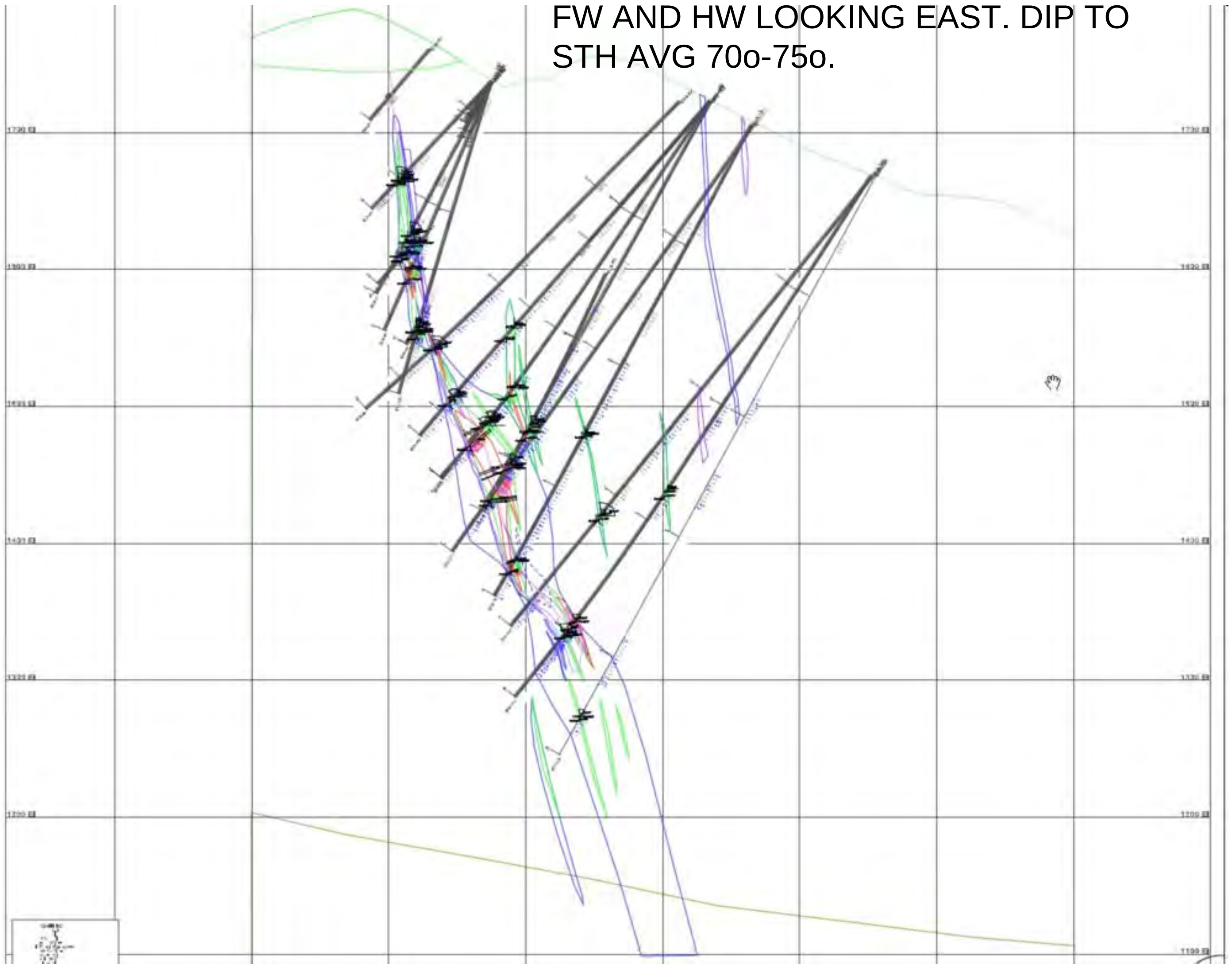
HW 68% RMR

ORE

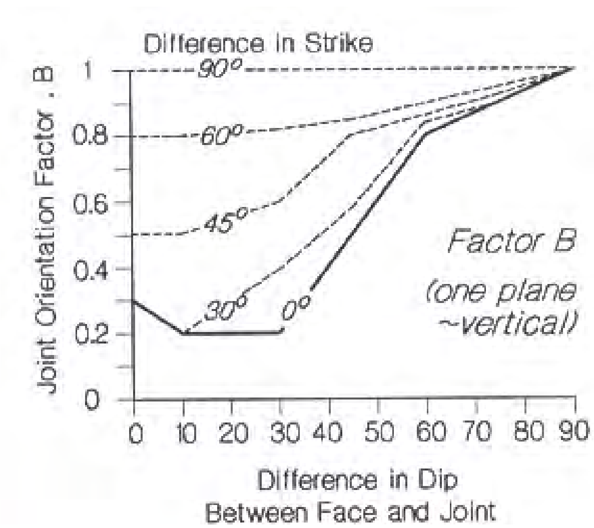
RMR IS 61%

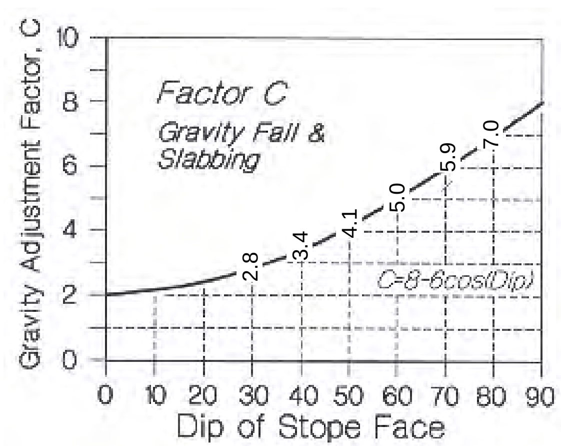

RMR FW=50%
RMR HW=54%
AVG 10m
AVG DIP 75o RANGE 45-90
CENTRAL ZONE
RMR ORE =56%
50-60m THICK
AVG 25-30m
RMR ORE =64%
EAST ZONE
AVG DIP 75o RANGE 45-90
200m
RMR HW=52%
RMR FW=54%
CENTRAL ZONE LEGEND
< 130 Ag g/t
130 - 1700 Ag g/t
>1700 Ag g/t
BARREN QUARTZ VEIN
EAST ZONE LEGEND
< 145 Ag g/t
145 - 800 Ag g/t
>800 Ag g/t
N
GREEN AND RED MINED
ESCOBAL PROJECT
ELEVATION: 1412.5m

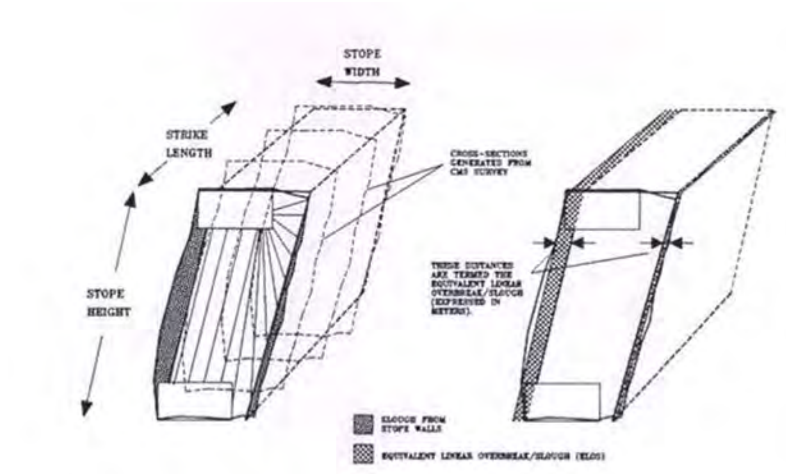
EAST ZONE HOSTED IN ANDESITE IN FW AND HW LOOKING EAST. DIP TO STH AVG 70o-75o.



A = 1 RELAXED







PROCEDURE FOR CALCULATING EQUIVALENT LINEAR OVERBREAK/SLOUGH (ELOS)

1) CUT SECTIONS THROUGH THE CMS SURVEY ALONG THE LONGHOLE SECTION LINES.

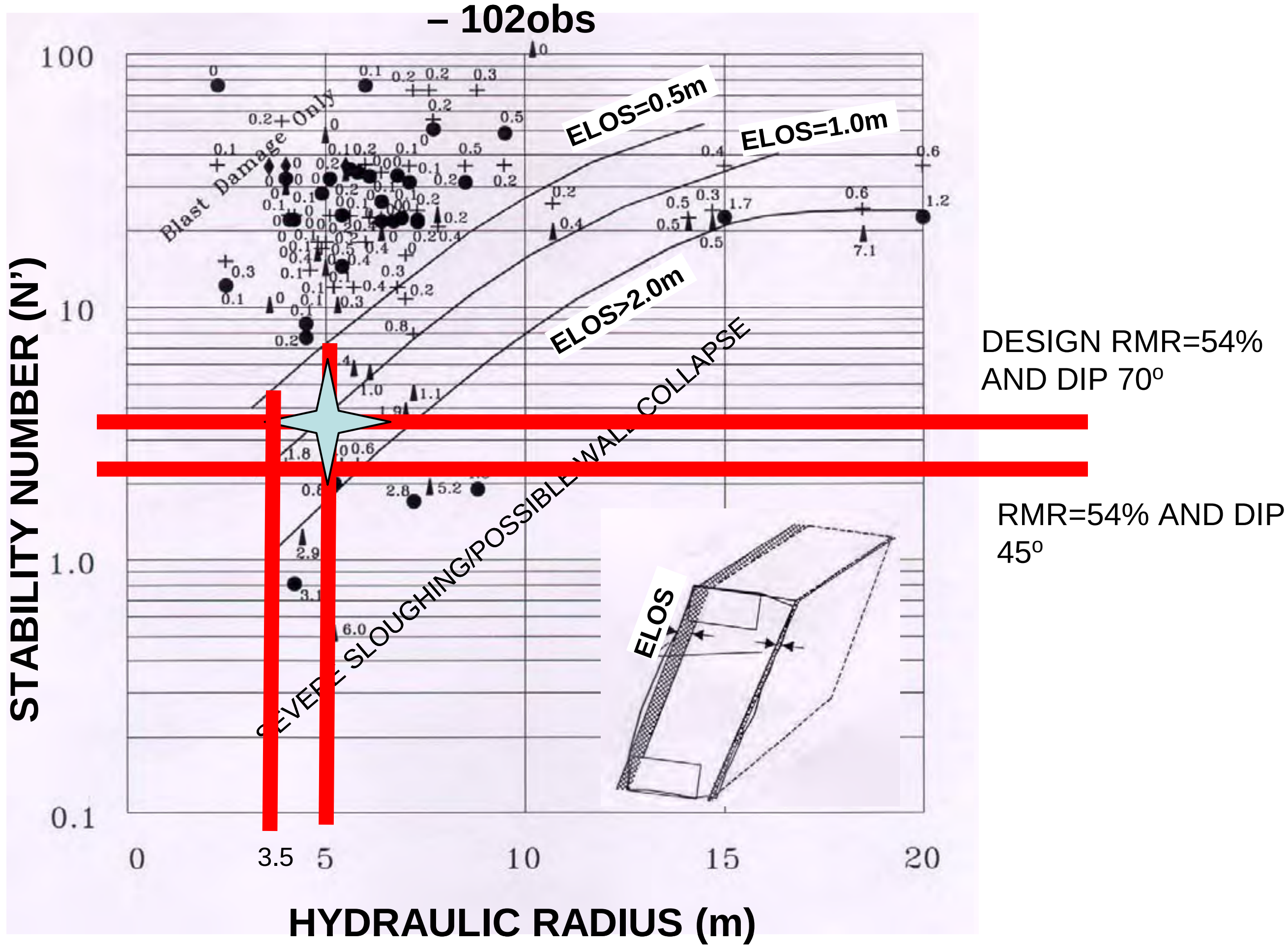
2) ON EACH SECTION CALCULATE THE AREA (m-2) OF OVERBREAK/SLOUGH FROM EACH OF THE STOPE SURFACES. THE OVERBREAK/SLOUGH SHOULD BE MEASURED RELATIVE TO THE ORE CONTACTS OR MINING LINES. UNPLANNED DILUTION TAKEN DURING DEVELOPMENT OF THE ORE DRIFTS SHOULD NOT BE INCLUDED.

3) BASED ON THE AREAS CALCULATED FROM THE SECTIONS, CALCULATE A VOLUME OF SLOUGH (m-3) FOR EACH STOPE SURFACE. THIS WILL REQUIRE THAT A THICKNESS BE ASSIGNED TO EACH SECTION.

4) THE EQUIVALENT LINEAR OVERBREAK/SLOUGH (m) FOR A GIVEN SURFACE CAN BE CALCULATED USING THE EQUATION GIVEN BELOW:

$$\text{EQUIVALENT LINEAR OVERBREAK/SLOUGH} = \frac{\text{VOLUME OF SLOUGH FROM STOPE SURFACE}}{\text{STOPE HEIGHT} \times \text{WALL STRIKE LENGTH}}$$

EMPIRICAL ESTIMATION OF WALL SLOUGH (ELOS) – 102obs



ASSUMPTIONS

N' = Q' X A X B X C

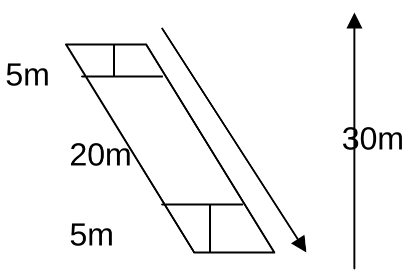
A = 1 (RELAXED)

B = 0.2

C_{HW} = 8-6cosφ

C_{FW} = 8

* ELOS LINES APPLY TO UNSUPPORTED STOPE SURFACE



H=30/SIN 70 = 32m

HR =5m = H X L/(2H + 2L)

= 32 X L/(64 + 2L)

L=15m

RMR design=54% Q=3

DIP=70º C=5.9

N'=3 X 1 X 0.2 X 5.9=3.5

RMR design=54% Q=3

DIP=45º C=3.8

N'=3 X 1 X 0.2 X 3.8=2.3

ASSUMPTIONS

N' = Q' X A X B X C

A = 1 (RELAXED)

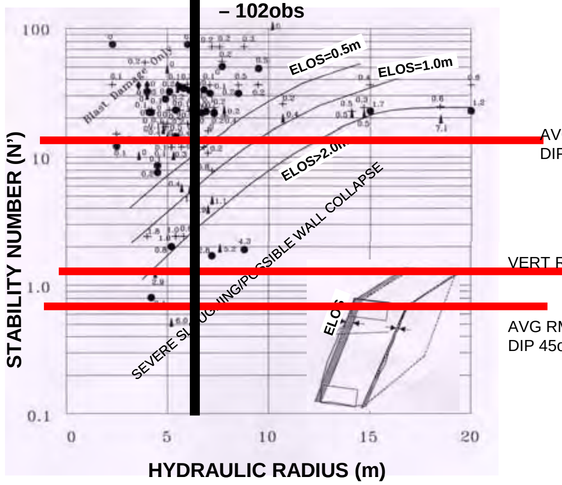
B = 0.2

C_{HW} = 8-6cos

C_{FW} = 8

* ELOS LINES APPLY TO UNSUPPORTED STOPE SURFACE

EMPIRICAL ESTIMATION OF WALL SLOUGH (ELOS) – 102obs



Hw=35%-65% RMR or Q'=0.8 to 10.3

DIP = 75o C=6.4

DIP=45o C=3.8

DIP=90o C=8

LOW N'=0.8 X 1 X 0.2 X 3.8=0.6 PT A

AVG N' = 10.3 X 0.2 X 6.4 = 13.2 PT B

VERT N' = 0.8 X 1 X 0.2 X 8 = 1.3

HR = 8m = L X H/(2L + 2H)

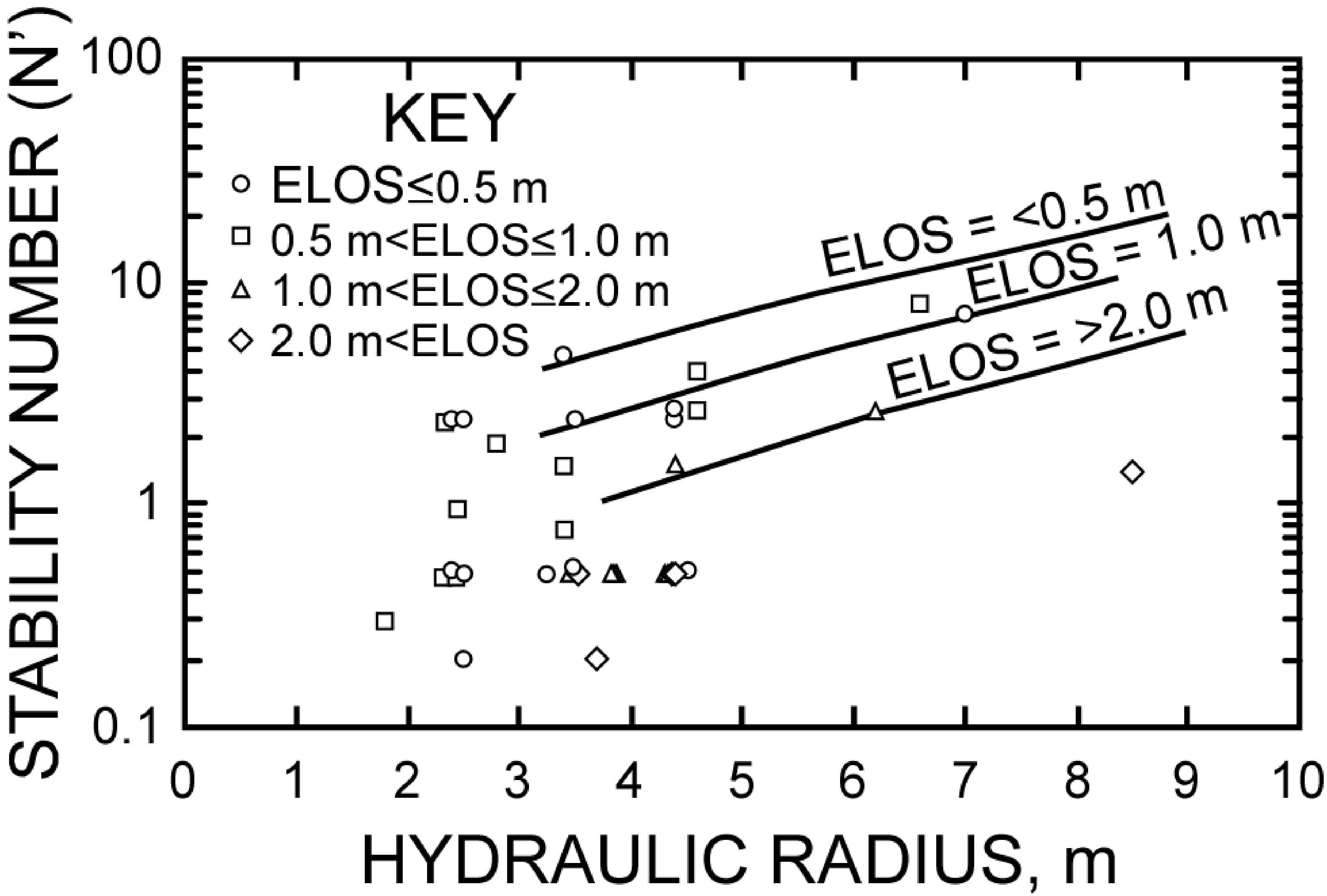
SUB LEVEL 20m SLOPE = H

L= 30m

HR= 30 X 20/(60 + 40) = 6m

8m = 20L/(2L + 40)

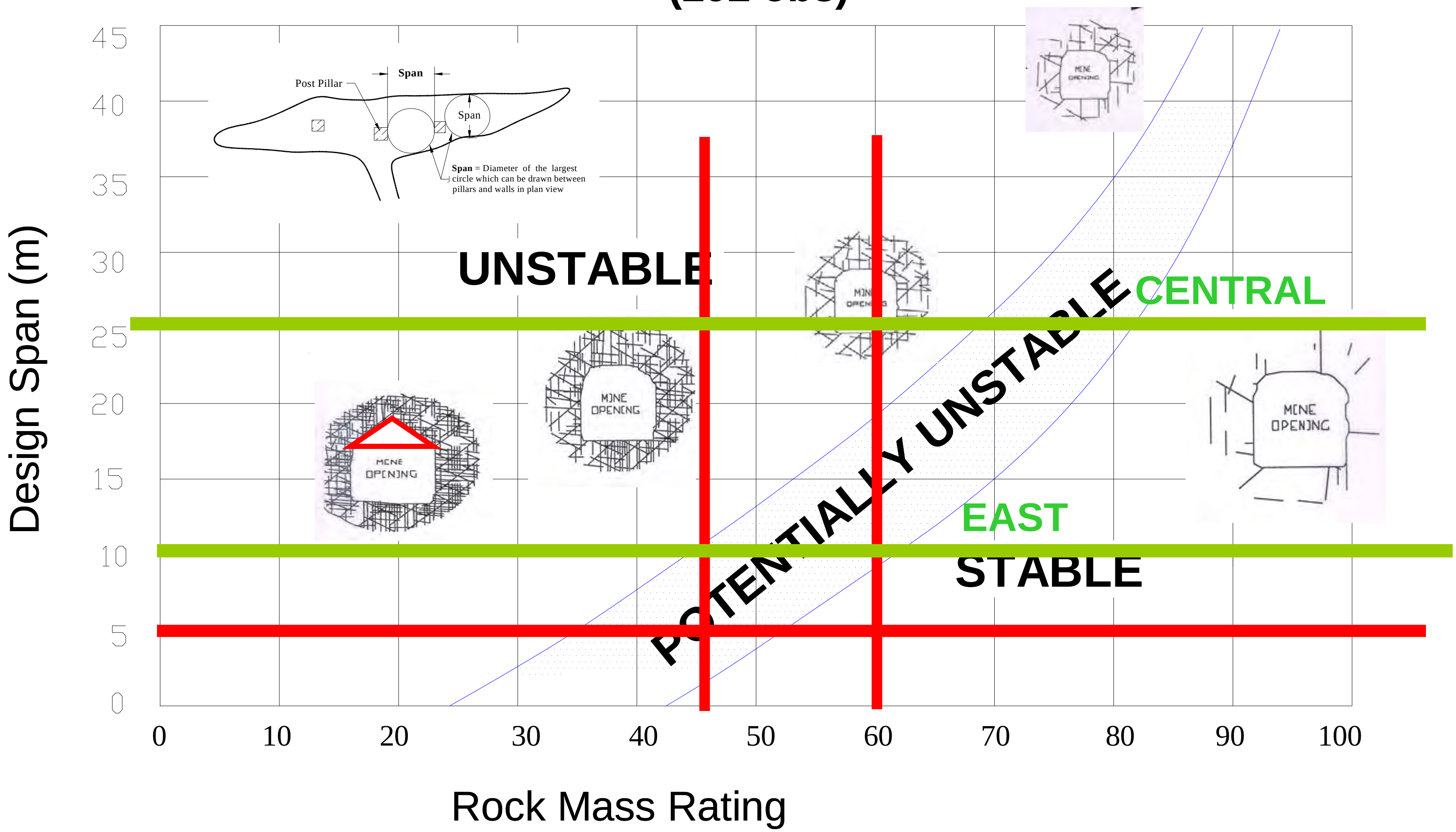
L= 80m



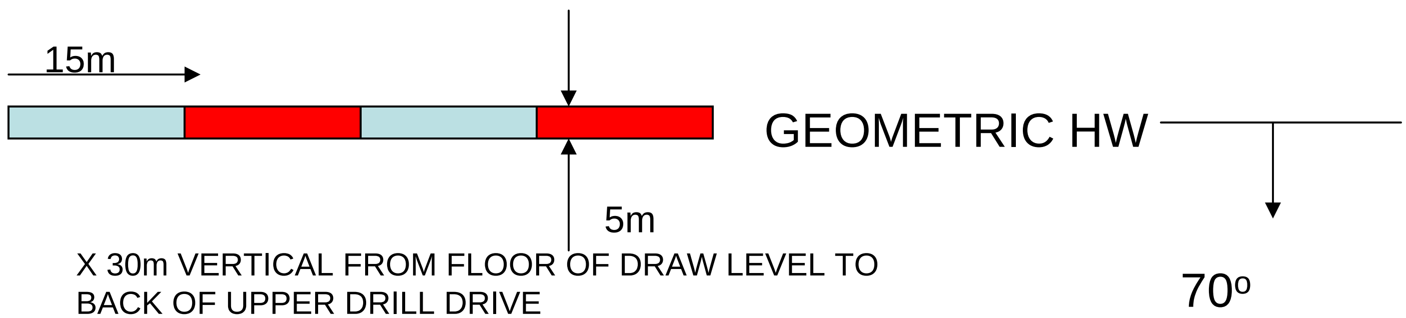
KEY
ELOS≤0.5 m
0.5 m<ELOS≤1.0 m
1.0 m<ELOS≤2.0 m
2.0 m<ELOS
ELOS = <0.5 m
ELOS = 1.0 m
ELOS = >2.0 m
STABILITY NUMBER (N')
HYDRAULIC RADIUS, m
100
10
1
0.1
0
1
2
3
4
5
6
7
8
9
10

HR=3m DESIGN

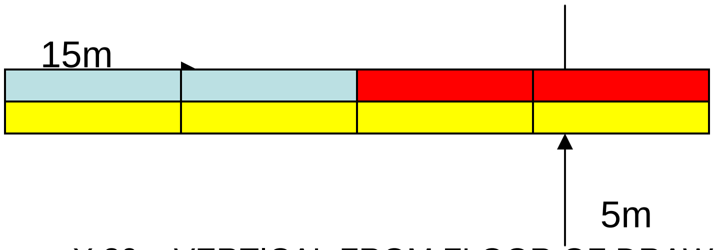
Updated Span Design Curve
(292 obs)



ORE RMR = 45%-60%

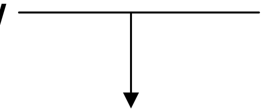


SCHEMATIC PLAN



X 30m VERTICAL FROM FLOOR OF DRAW LEVEL TO
BACK OF UPPER DRILL DRIVE

GEOMETRIC HW



70^{o}

HW IS PASTE WHICH IS VERTICAL. USE 30m STRIKE IN PASTE

2 * 32/(1 + 32/30) = 31t/m2=0.31MPa
~5% CEMENT

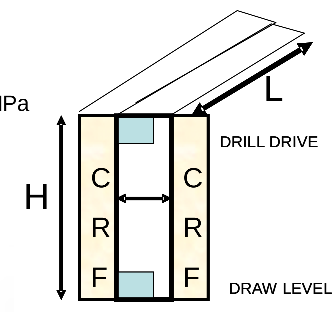
$$\sigma_{ucs} = \gamma H /(1 + H/L)$$

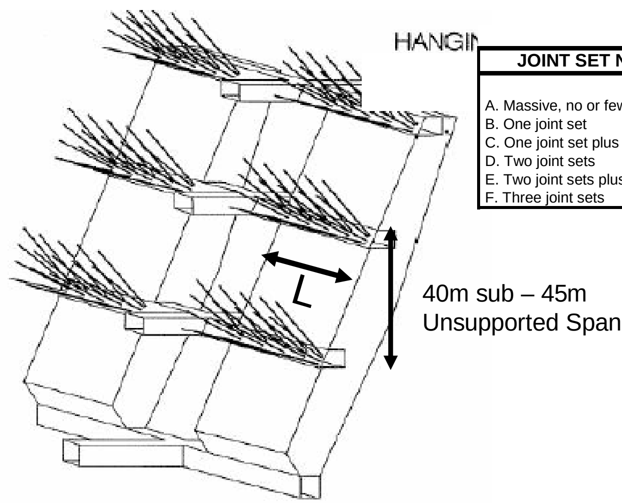
σ_{ucs} = FILL COMPRESSIVE STRENGTH, MPa
γ = FILL DENSITY, MN/m3
H = HEIGHT OF EXPOSURE, m
L = WIDTH OF EXPOSURE, m

ESTIMATE FILL DENSITY AT SG=2

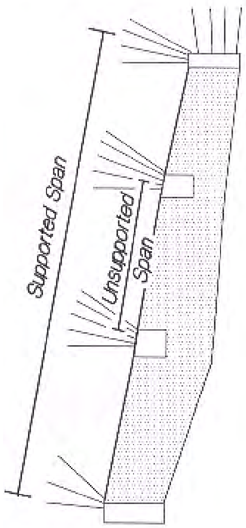
Parameters used for modelling the fills were based on Canadian experience as summarized in Table 5.

Table 5. Rock Fill Strength Parameters





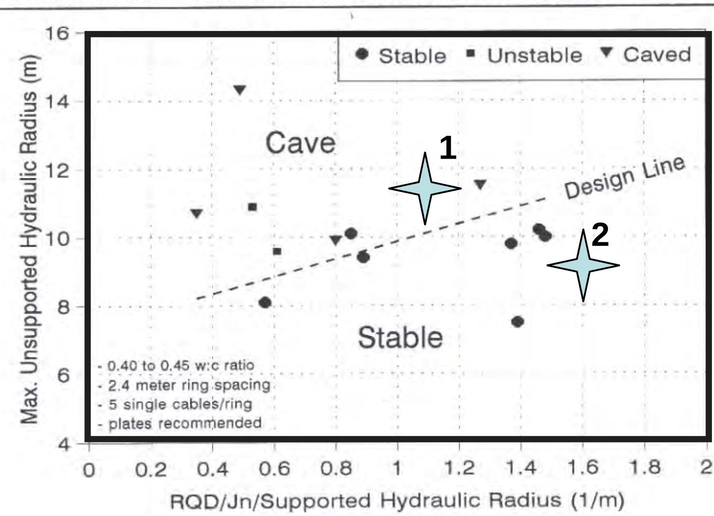
JOINT SET NUMBER	Jn
A. Massive, no or few joints	0.5 - 1.0
B. One joint set	2
C. One joint set plus random	3
D. Two joint sets	4
E. Two joint sets plus random	6
F. Three joint sets	9



RQD=80%, Jn=4 (TWO JOINT SETS), SUB HT=40m EFFECTIVE 45m

	HEIGHT OF WALL					
L X HT	**UNSUPP**	**SUPP**	**STRIKE**	**HR unsup**	**HRsupp**	**RQD/Jn/HRsupp**
1) 45m X 180m	45m(1 SUB)	180m(4 SUBS)	45m	11.3m	18m	1.1
2) 30m X 180m	45m(1 SUB)	180m(4 SUBS)	30m	9m	12.8m	1.6

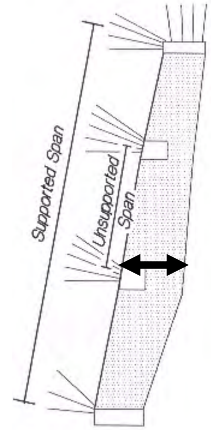
DESIGN CHART FOR POINT ANCHOR HW CABLE SUPPORT



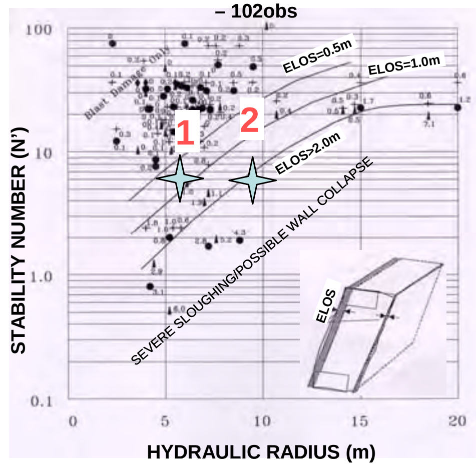
BACK SPAN - CONCERNS

Refer Golder Analysis Backs – Figure 3A. Where
1) 15m transverse x 45m strike HR=5.6m and N=6.0 (stable)

2) 30m transverse x 45m strike HR=9m and N=6.0 (cave)

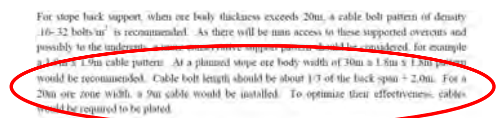


EMPIRICAL ESTIMATION OF WALL SLOUGH (ELOS) – 102obs

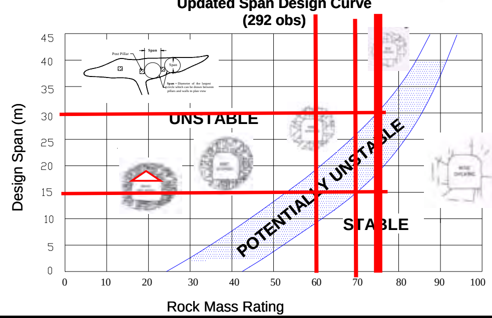


RMR = 70%-75% DESIGN (~ Q'=18-31)

BACK SPAN WITHIN STOPES.


For stope back support, when ore body thickness exceeds 20m, a cable bolt pattern of density .16-.32 bolts/m² is recommended. As there will be man access to these supported overcuts and possibly to the undercuts, a more conservative support pattern should be considered, for example a [illegible] x 1.9m cable pattern. At a planned stope ore body width of 30m a 1.6m x 1.8m pattern would be recommended. Cable bolt length should be about 1/3 of the back span + 2.0m. For a 20m ore zone width, a 9m cable would be installed. To optimize their effectiveness, cables would be required to be plated.

Updated Span Design Curve (292 obs)



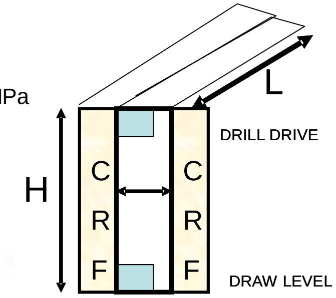
GOLDER GEOTECH 2007

NOTE 30m X-VERSE (45m STRIKE) WILL CAVE TO 15m ABOVE BACK. 15m X-VERSE WILL REQUIRE BACK SUPPORT IF FLAT JOINTS PRESENT.

* NOTE: DISCUSSION WITH R. PARKER(JULY, 2010) RMR ~84% WITH FLAT JOINTS RMR~74% AS OBERVED FROM UNDERGROUND AND STOPE EXPLORATORY DIAMOND CORE HOLE GT-SKUG-10-006.

$\sigma_{ucs} = \gamma H / (1 + H/L)$

σ_{ucs} = FILL COMPRESSIVE STRENGTH, MPa
γ = FILL DENSITY, MN/m3
H = HEIGHT OF EXPOSURE, m
L = WIDTH OF EXPOSURE, m



LONGHOLE STOPE

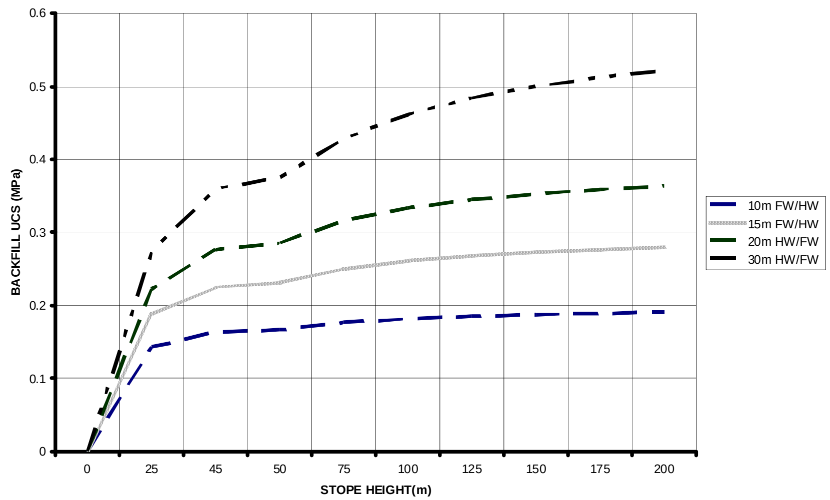
ESTIMATE FILL DENSITY AT SG=2

Parameters used for modelling the fills were based on Canadian experience as summarized in Table 5.

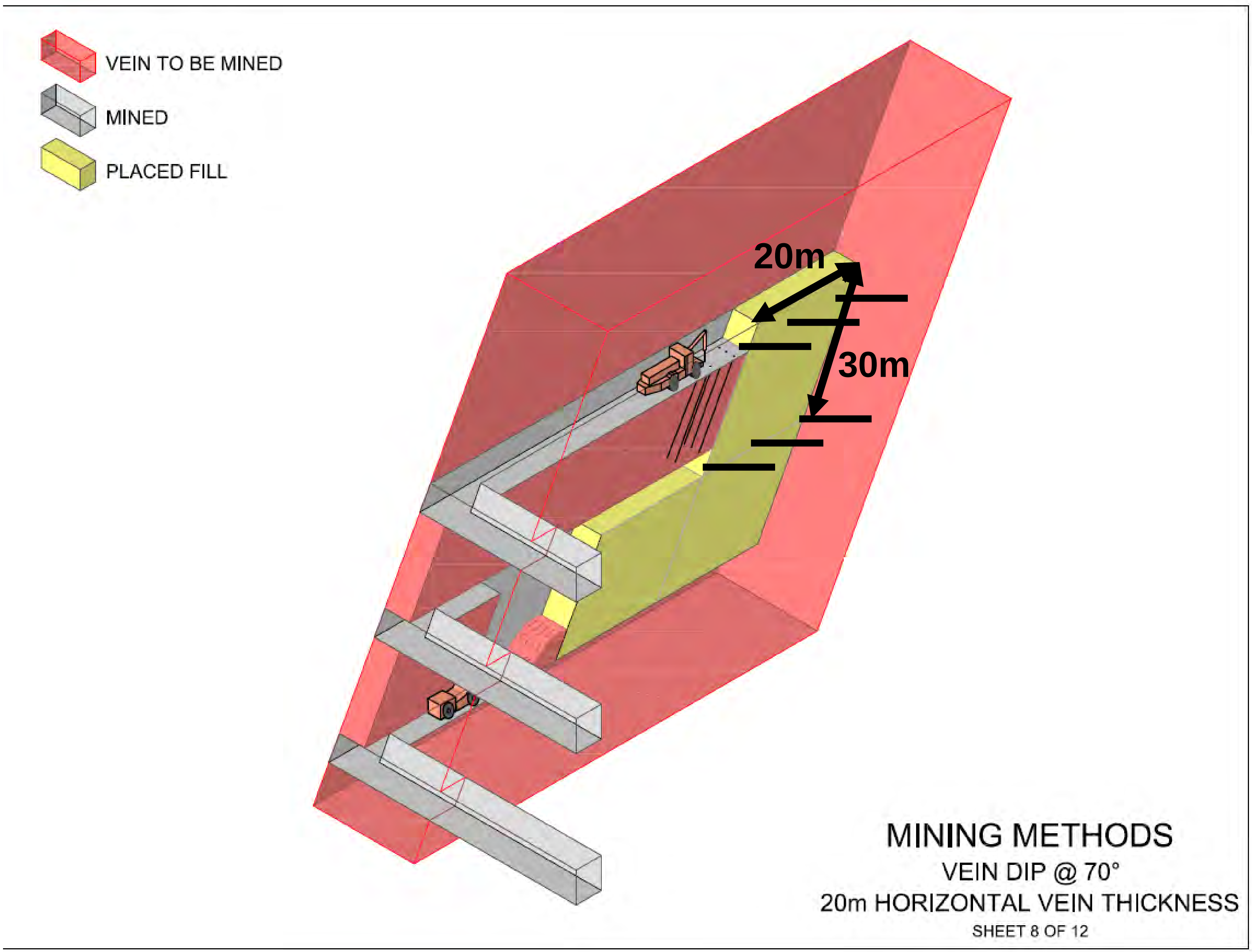
Table 5. Rock Fill Strength Parameters

Parameter	Cemented Rock Fill	Un-Cemented Rock Fill
Young's Modulus	600 MPa	100 MPa
Poisson's ratio	0.3	0.3
Tensile Strength	0.1 MPa	0
Pre-peak strength Friction Angle (degrees)	38	38
Pre-peak Cohesion	1.2 MPa/m	0
Residual Strength Friction Angle (degrees)	38	38
Residual Strength Cohesion	0	0

GOLDER 2009



REQUIRE 6% CEMENT AT 6MPa UCS SAMPLE



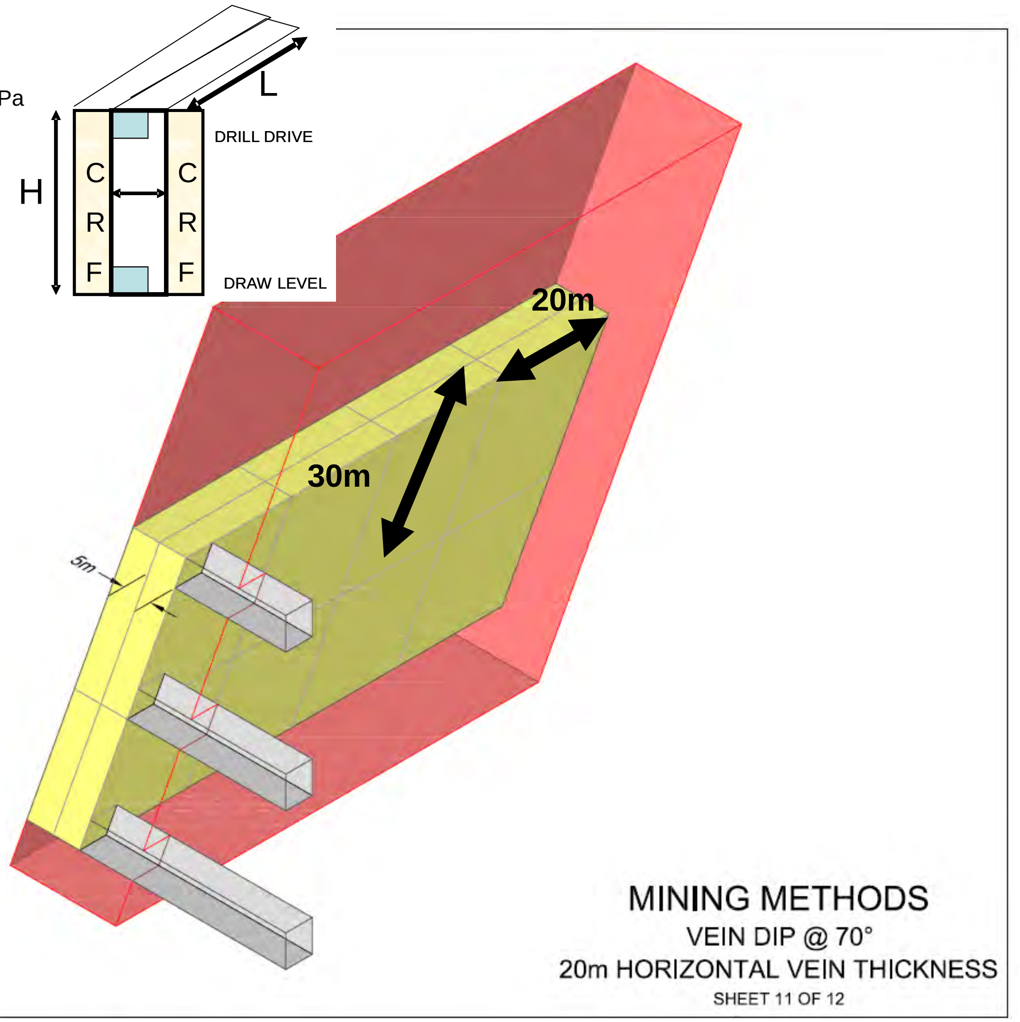
EXPOSED 20m ON STRIKE X 30m VERTICAL (AT 70º). BACK SPAN IS 5m, RMR OF 54%. HR=6m ESTIMATE ~1m+ OF WALL SLOUGH. NOTE BOLT UPPER AND LOWER DRILL DRIVES AT HW TO MINIMIZE

$$\sigma_{ucs} = \gamma H / (1 + H/L)$$

σ_{ucs} = FILL COMPRESSIVE STRENGTH, MPa
γ = FILL DENSITY, MN/m3
H = HEIGHT OF EXPOSURE, m
L = WIDTH OF EXPOSURE, m

ESTIMATE FILL DENSITY AT SG=2

Parameters used for modelling the fills were based on Canadian experience as summarized in Table 5.



H=30m, L=20m

UCS=2t/m3 X 30/(1+30/20)=

24t/m2 = 0.24MPa = 35psi.

GAURANTEED 10%. MOST LIKELY 5% CEMENT CONTENT. NEED ONLY 0.5MPa UCS.

MINE IN 5m SLICES FROM HW TO FW OR VICE VERSA. STRIKE WITH FILL WALL . MINE AGAINST FILL WALL OF 20m ALONG STRIKE. NOTE VERTICAL WALL OF PASTE IS PREFERRED AS IF IS HW AT 70o WILL BE LESS STABLE AND OVERHANGING SUBSEQUENT STOPE TO BE MINED RESULTING IN POTENTIAL DILUTION AND/OR INCREASED STRENGTHS REQUIRED.

ESCOBAL MINE SITE VISIT

AUGUST 20-25, 2011

MINERA SAN RAPHAEL - ESCOBAL MINE AUGUST 20-25/11

STOP	LOCATION
1	EAST DECLINE (FACE)
2	EAST DECLINE
3	WEST DECLINE
4	WEST DECLINE (FACE)

2

Tahoe RESOURCES INC

MINE SITE VISIT - ESCOBAL 2011 – WORK SCOPE

- UNDERGROUND TOUR
  - EAST RAMP
  - WEST RAMP
- GROUND SUPPORT UPDATE
- TIGHT FILL
- TRAINING
- MINE PLAN

Minera San Rafael GUATEMALA

Tahoe RESOURCES INC

ESCOBAL MINE SITE VISIT

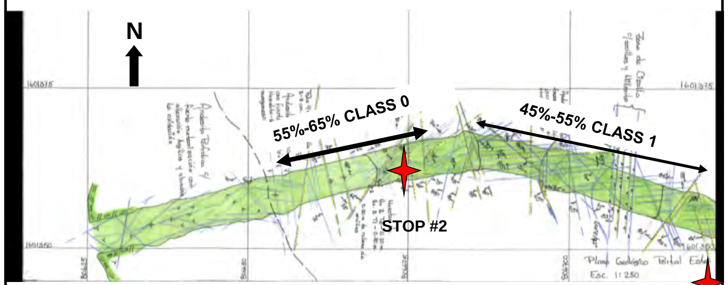
UNDERGROUND TOUR
AUGUST 20-25, 2011

C. MUERHOFF (TECHNICAL SERVICES – TAHOE)

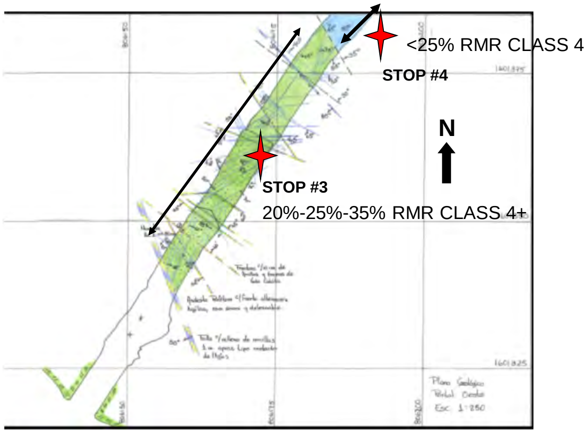
V. ISLAS (CHIEF GEOLOGIST ESCOBAL)

M. MOLLINEDO (GROUND CONTROL GEOLOGIST ESCOBAL)

Minera San Rafael GUATEMALA



EAST RAMP – ESCOBAL MINE

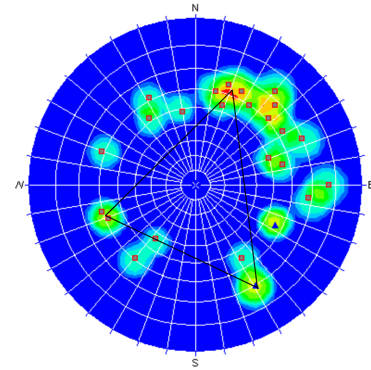


WEST RAMP – ESCOBAL MINE

5

MAPPING AS PER VICTOR ISLAS

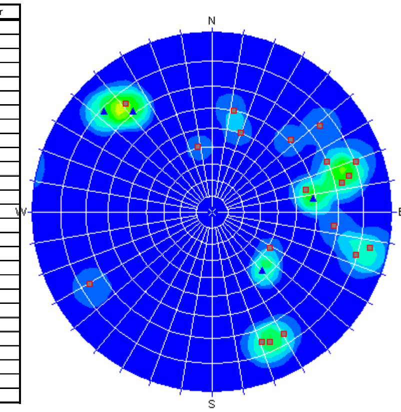
	Rumbo	Inclinación	Dip	DipDir
1	340	50SW	50	250
2	328	62SW	62	238
3	25	55NW	55	295
4	20	60SE	60	110
5	310	73SW	73	220
6	315	68SW	68	225
7	309	60NE	60	39
8	22	60SE	60	72
9	25	55NW	55	295
10	126	65SW	65	206
11	305	45NE	45	35
12	304	60SW	60	214
13	240	70NW	70	330
14	55	50SE	50	145
15	78	50SE	50	168
16	61	60SE	60	151
17	0	76W	76	270
18	60	70NW	70	330
19	57	55NW	55	327
20	110	65SW	65	200
21	108	54SW	54	198
22	338	58NE	58	68



EAST RAMP

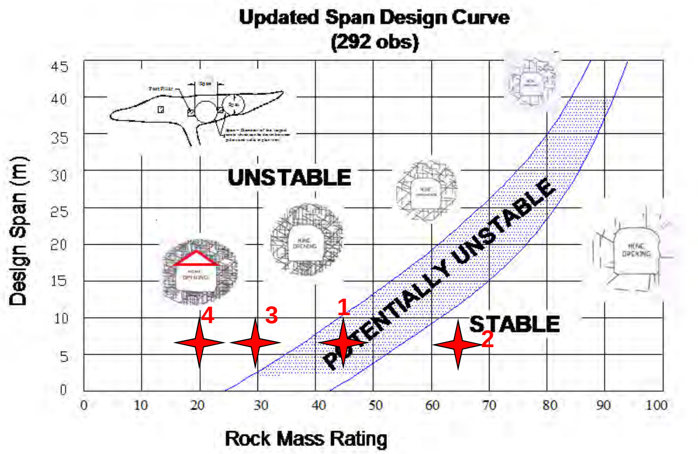
22 JOINTS/STRUCTURES

	Rumbo	Inclinación	Dip	DipDir
1	40	79SE	40	169
2	62	75NW	75	232
3	160	80SW	80	250
4	17	80NW	80	287
5	166	75SW	75	256
6	53	70SE	70	143
7	240	75NW	75	330
8	70	75NW	75	340
9	66	78NW	78	336
10	193	85NW	85	283
11	51	76SE	76	141
12	174	60SW	60	264
13	171	60SW	60	261
14	108	50SW	50	198
15	42	78SE	78	132
16	44	78SE	78	134
17	229	46NW	46	319
18	214	42NW	42	304
19	328	76NE	76	58
20	228	48NW	48	318
21	52	72SE	72	142
22	166	72SW	72	256
23	167	55SW	55	257
24	188	70NW	70	278
25	158	70SW	70	248
26	100	60SW	60	190
27	137	60SW	60	227



WEST RAMP

27 JOINTS/STRUCTURES



ESCOBAL SITE VISIT AUGUST 20-25/11

STOP	LOCATION	RMR	BACK SPAN
1	EAST DECLINE (FACE)	45%	6.2m
2	EAST DECLINE	65%	5.9m
3	WEST DECLINE	25%-30%	6m
4	WEST DECLINE (FACE)	20%	6m

7

INSTALLING SWELLEX FROM BASKET.

SWELLEX PLATES NOT TIGHT TO FACE/SCALE THEREAFTER IS QUESTIONABLE. BOLT THROUGH SHOTCRETE.

8

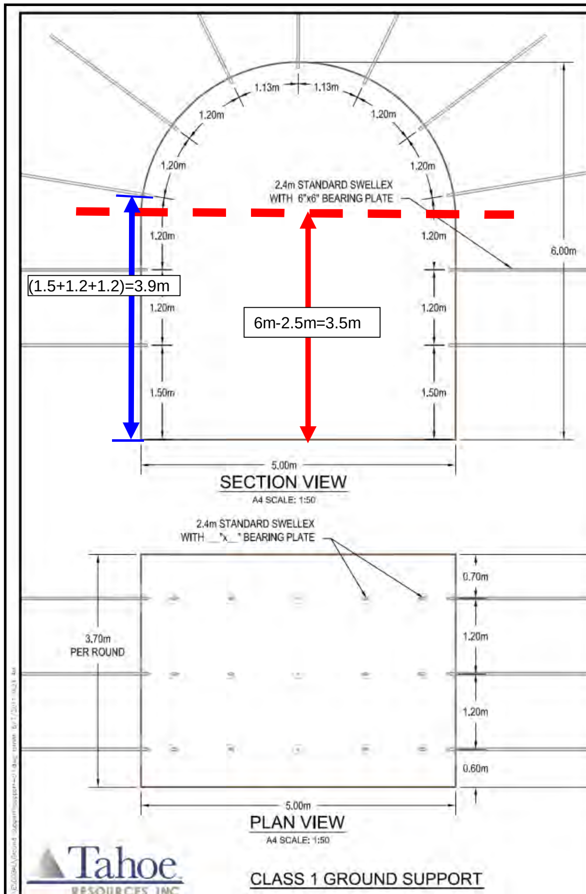
STOP #1: EAST DECLINE PROXIMITY OF FACE. SPAN IS 6.2m WITH RMR OF BACK OF 45%. WALL IS 55%.

RMR BACK/WALL ANDESITE		
1) STRENGTH	50-100MPa-	7-4
2) RQD	75%-50%	13
3) SPACING	0.3-1m-	20-15-10
4) CONDITON	SLT OPN	12
5) GRNWTR	DRY	10
	RATING(WALL)	62%-49%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	45%

SPAN IS 6.2m. DESIGN IS 5m. BACK HEIGHT IS 6.5m. SHOULD BRING SPRINGLINE DOWN TO ENSURE ARCHED BACK.

STOP #1: EAST DECLINE PROXIMITY OF FACE. SPAN IS 6.2m WITH RMR OF BACK OF 45%. WALL IS 55%.

9



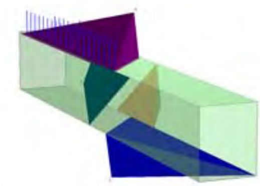
10

SPRING LINE – SHOULD BE LOWERED OR AT LEAST BOLT RIB LEFT AND RIGHT AT THIS ELEV.

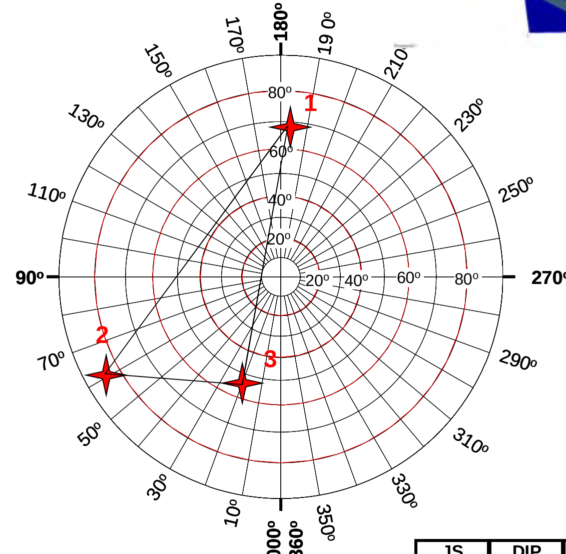
STOP #1: EAST DECLINE PROXIMITY OF FACE. RMR OF BACK IS 45%-55% FOR SPAN OF 6.2m WITH RMR OF BACK OF 45%. WALL IS 55%. CHAINLINK IN BACK HELD BY 2.4m LONG SWELLEX BOLTS ON 1.2m X 1.2m PATTERN. NOTE SHOTCRETE BUT MINOR. NOTE NO NEED FOR SHOTCRETE IF SCREEN USED AS REPLACEMENT.

11

STOP #1: EAST RAMP WEDGE



POLE PLOT



FS SUPP=1.2. NOTE BACK ARCHED

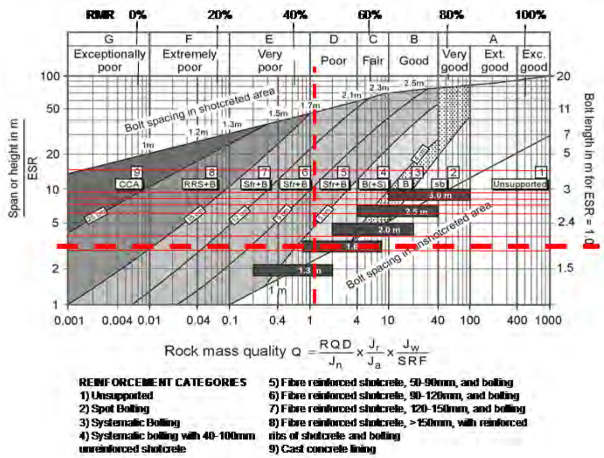
JS	DIP	DDR
1	68°	185°
2	87°	060°
3	52°	020°

WALL TREND=110°

NOTE JUST AFTER BEND. TREND OF TUNNEL CHANGED. HAVE SLIVERS NOT "DEAD WEIGHT" WEDGES (V.I.)

12

RMR=45% SUPPORT CLASS 2 (SCREEN)



RMR BACK = 45%-55%, Q=1.1-3.4

ESR=1.6 PERMANENT. SPAN=5m/1.6=3.1

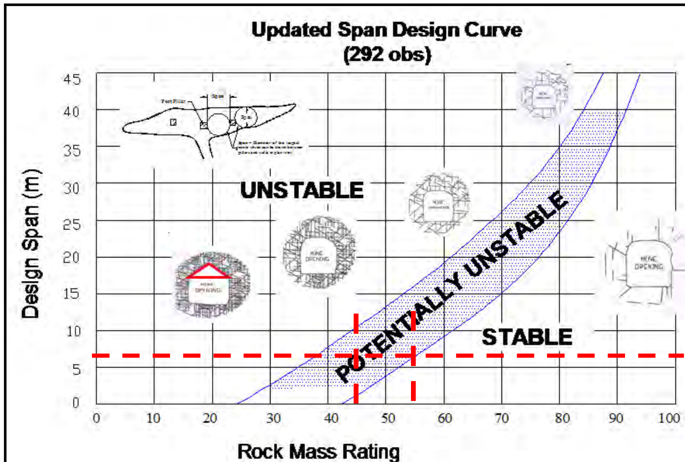
REQUIRE SHOTCRETE 40-100mm(UNREINFORCED) THICK + BOLTING OR SCREEN + BOLTING.
SPAN=6.2m

RMR BACK = 45%-55%, Q=1.1-3.4

ESR=1.6 PERMANENT. SPAN=6m/1.6=3.8

REQUIRE SHOTCRETE 40-100mm (UNREINFORCED) THICK + BOLTING OR SCREEN + BOLTING.

STOP #1: EAST DECLINE PROXIMITY OF FACE. SPAN IS 6.2m WITH RMR OF BACK OF 45%. WALL IS 55%.



POTENTIALLY UNSTABLE – REQUIRE SUPPORT

STOP #1: EAST DECLINE PROXIMITY OF FACE. SPAN IS 6.2m WITH RMR OF BACK OF 45%. WALL IS 55%.

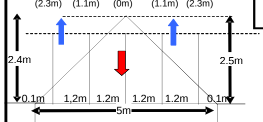
SUPPORT PATTERN: 2.4m(8ft) LONG STD (PM12) SWELLEX BOLTS ON 1.2m X 1.2m PATTERN

DEAD WEIGHT ANALYSIS: WEDGE HEIGHT = 0.5 X SPAN

SPAN=5m

SG=2.6 (INTRUSIVE)

BOND PAST WEDGE



X 1.2m DEPTH SLICE

DEAD WEIGHT WEDGE
= (0.5 * 5m * 2.5m * 1.2m) *2.6t/m 3 = 20tonnes
* SPECIFIC GRAVITY(SG) = 2.6

SUPPORT CAPACITY-NO CORNER BOLTS
= 11t + 11t = 22t (GOOD - RMR>55%)
** BREAKING STRENGTH=11tonne
BOND STRENGTH = GOOD ROCK=12t/m OR
CRITICAL BOND=11/12=0.9m

= (1.1m * 8t/m) * 2 = 18t (WEAK RMR<45%)
BOND STRENGTH = WEAK ROCK=8t/m OR
CRITICAL BOND=11t/8t/m=1.4m

SUPPORT CAPACITY – WITH CORNER BOLTS
= (11t) *4 = 44t (GOOD - RMR>55%)
** BREAKING STRENGTH=11tonne
BOND STRENGTH = GOOD ROCK=12t/m OR
CRITICAL BOND=11/12=0.9m

= (11t+1.1m * 8t/m) * 2 = 40t (WEAK RMR<45%)
BOND STRENGTH = WEAK ROCK=8t/m OR
CRITICAL BOND=11t/8t/m=1.4m

FS = CAPACITY/DEAD WEIGHT(SG=2.6)
=22t/20t = 1.1 (GOOD RMR>55%)
=18t/20t = 0.9 (WEAK RMR<45%)
* IGNORE CORNER BOLTS

FS = CAPACITY/DEAD WEIGHT(SG=2.6)
=44t/20t = 2.1 (GOOD RMR>55%)
=40t/20t = 2.0 (WEAK RMR<45%)
* WITH CORNER BOLTS

*** FS>1.5 (PERMANENT)
FS>1.2 (TEMPORARY)

SWELLEX – GROUND SUPPORT

TAHOE RESOURCES INC. – ESCOBAL PROJECT

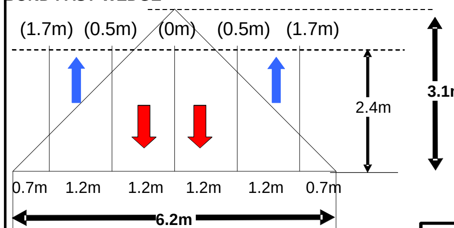
FIGURE 2B: SUPPORT – DEVELOPMENT FLAT BACK(4ft x 4ft)

PAKALNIS & ASSOCIATES

"DEAD WEIGHT" ANALYSIS: WEDGE HEIGHT = 0.5 X SPAN

SUPPORT PATTERN: 2.4m(8ft) LONG SWELLEX BOLTS ON 1.2m X 1.2m PATTERN

BOND PAST WEDGE



X 1.2m DEPTH SLICE

DEAD WEIGHT WEDGE
= (0.5 * 6.2m * 3.1m * 1.2m) * 2.6t/m³ = 30tonnes
* SPECIFIC GRAVITY(SG) = 2.6

SUPPORT CAPACITY (GOOD RMR >55%)
= 11t + 0.5m*12t/m + 0 + 0.5m*12t/m + 11t = 34t
** BREAKING STRENGTH=11tonne
BOND STRENGTH = 12t/m (>55% RMR)
CRITICAL BOND= 0.9m (11t/12t/m)

SUPPORT CAPACITY (WEAK RMR <45%)
= 11t + 0.5m*8.1t/m + 0 + 0.5m*8.1t/m + 11t = 30t
** BREAKING STRENGTH=11tonne
BOND STRENGTH = 8.1t/m (<45% RMR)
CRITICAL BOND=1.4m (11t/8.1t/m)

FS = CAPACITY/DEAD WEIGHT(SG=2.6)

=34t/30t = 1.1 (GOOD RMR>55%)

=30t/30t = 1.0 (WEAK RMR<45%)

*** FS>1.5 (PERMANENT)

FS>1.2 (TEMPORARY)

NOTE HAVE FS ~1 DUE TO SPAN OF 6.2m. POTENTIAL FOR WEDGES EXIST FOR EAST RAMP. BACK IS ARCHED SO FS>>4. CRITICAL TO KEEP ARCH, BOLT PERPENDICULAR TO WALL, MAINTAIN SPAN TO 5m.

RMR BACK/WALL ANDESITE		
1) STRENGTH	50-100MPa	7
2) RQD	90%-75%	17
3) SPACING	0.3-1m	20
4) CONDITON	SLT OPN	12
5) GRNWTR	DRY	10
STRUCTURE	RATING(WALL)	66%
	FLAT	
	DESIGN (BACK)	65%

RMR IS 65% WITH NO FLAT JOINTS. SPAN IS 5.9m. HAVE POTENTIAL FOR "DEAD WEIGHT" WEDGES IN BACK. SUPPORT LARGELY CALLS FOR SUPPORT CLASS 1 ie. NO NEED FOR SCREEN OR SHOTCRETE ONLY BOLTS BUT REQUIRES "RAMP INSPECTION" PERIODICALLY TO ENSURE PROPERLY SCALED.

3mm SHOTCRETE. NOTE CAN PULL OFF ROCK. SHOULD BOLT THRU SHOTCRETE IF SHOTCRETE REQUIRED. NOTE NO NEED FOR SHOTCRETE

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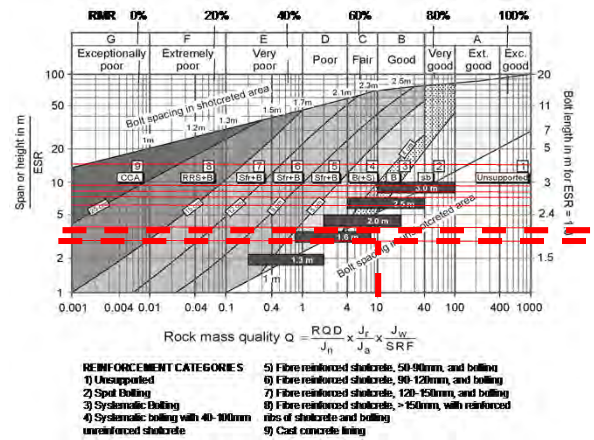
STOP #2: EAST RAMP – TUNNEL PRIOR TO TURN.

RMR=65% SUPPORT CLASS 1 (NO SCREEN)



RMR BACK = 65%, Q=10.3

ESR=1.6 PERMANENT. SPAN=5m/1.6=3.1

REQUIRE BOLTING ONLY – NOTE OWNESS IS UPON OPERATOR TO SCALE BACK/WALLS. RAMP INSPECTION IF NO SCREEN/SHOTCRETE. SCREEN TO ~1m DOWN HAUNCHES.

SPAN=5.9m

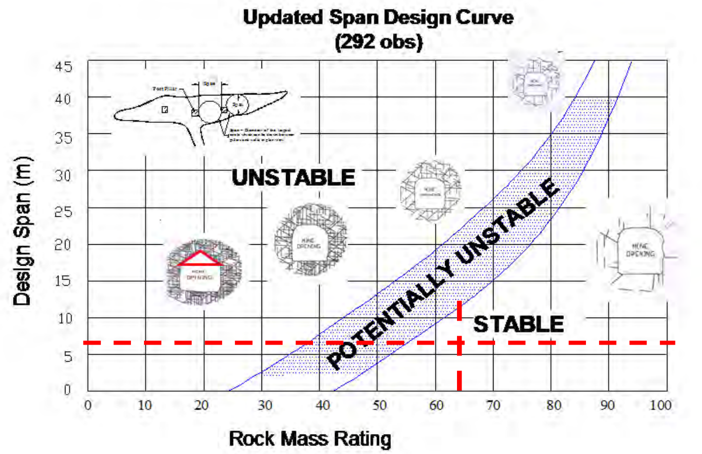
RMR BACK = 65%, Q=10.3

ESR=1.6 PERMANENT. SPAN=5.9m/1.6=3.7

REQUIRE BOLTING ONLY – NOTE OWNESS IS UPON OPERATOR TO SCALE BACK/WALLS. RAMP INSPECTION IF NO SCREEN/SHOTCRETE. SCREEN TO ~1m DOWN HAUNCHES.

18

STOP #2: EAST RAMP – TUNNEL PRIOR TO TURN.



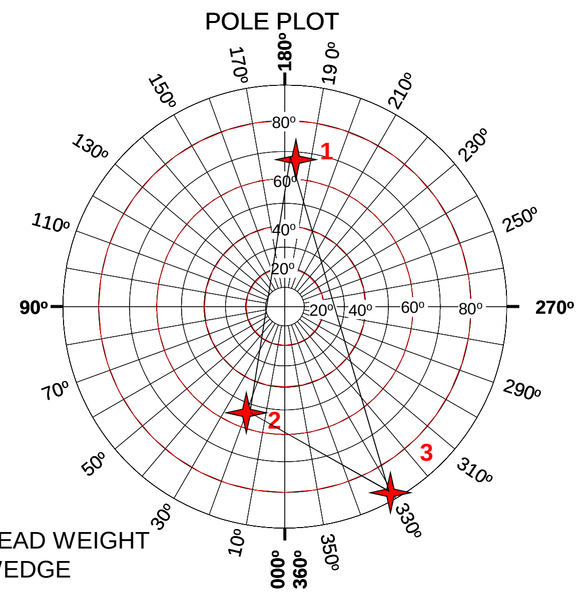
STABLE – LOCAL SUPPORT ONLY (1.8m LONG BOLTS)

HOWEVER, STRUCTURE EXISTS – "DEAD WEIGHT"

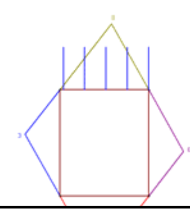
19

STOP #2: EAST RAMP – TUNNEL PRIOR TO TURN.

STOP #2: EAST RAMP WEDGE



DEAD WEIGHT WEDGE



JS	DIP	DDR
1	68°	185°
2	52°	020°
3	87°	330°

WALL TREND=110°

FS=0.8, 100tonne + WEDGE, BACK ARCHED – STABLE.

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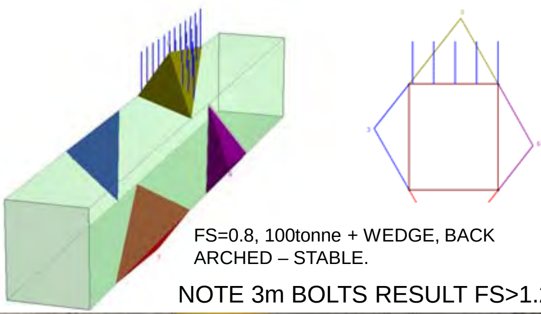
NOTE 3m BOLTS RESULT FS>1.2.

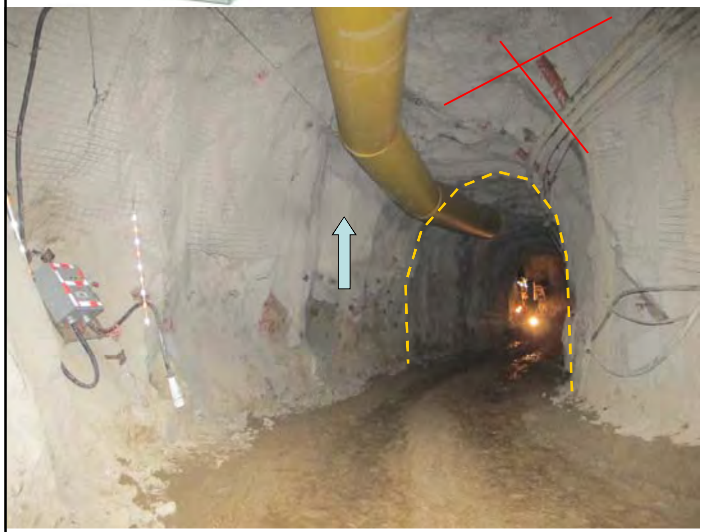
STOP #2: EAST RAMP WEDGE

ROOF WEDGE EXISTS.



FS=0.8, 100tonne + WEDGE, BACK ARCHED – STABLE.

NOTE 3m BOLTS RESULT FS>1.2.

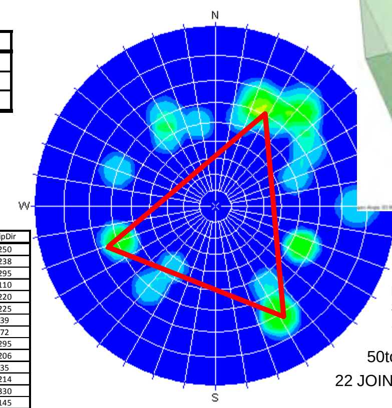


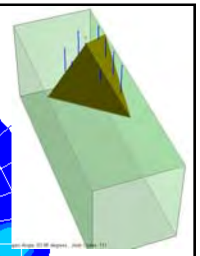
EXCELLENT ARCH. HAVE WEDGES IN BACK.

MAPPING BY VICTOR ISLA (ESCOBAL)

JS	DIP	DDR
1	60°	210°
2	70°	330°
3	65°	070°

WALL TREND=110°





FS=0.9 2.4m LONG SWELLEX ON 1.2m X 1.2m PATTERN.

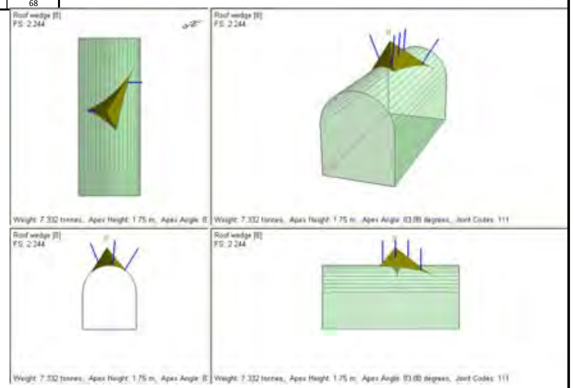
50tonne WEDGE.

22 JOINTS/STRUCTURES

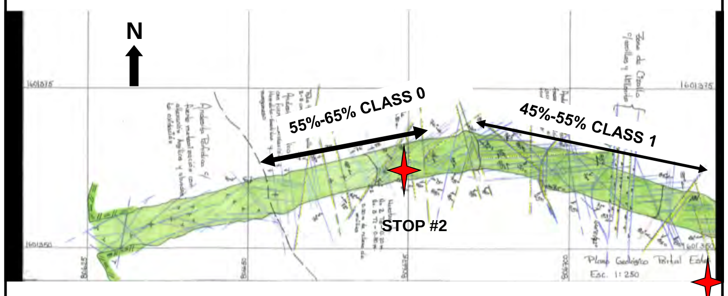
Rumbo	Inclinación	Dip	DipDir
340	50SW	50	250
328	62SW	62	238
25	55NW	55	295
20	60SE	60	110
310	73SW	73	220
315	68SW	68	225
309	60NE	60	39
22	60SE	60	72
25	55NW	55	295
126	65SW	65	206
305	45NE	45	35
304	60SW	60	214
240	70NW	70	330
55	50SE	50	145
78	50SE	50	168
61	60SE	60	151
0	76W	76	270
60	70NW	70	330
57	55NW	55	327
110	65SW	65	200
108	54SW	54	198
338	58NE	58	68

EAST RAMP

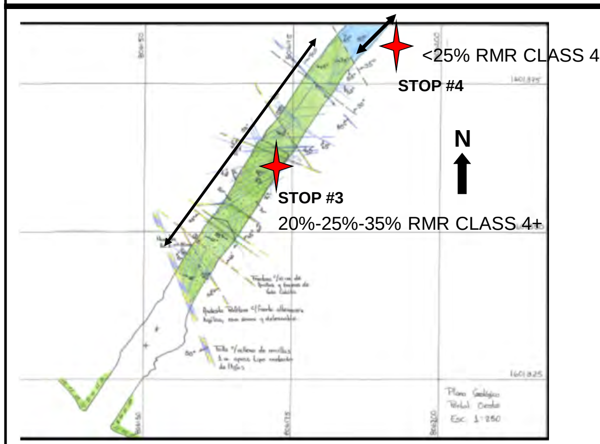
ARCH CRITICAL!!



7 tonne WEDGE, FS>2 WITH 2.4m LONG BOLTS ON 1.2m X 1.2m PATTERN



EAST RAMP – ESCOBAL MINE

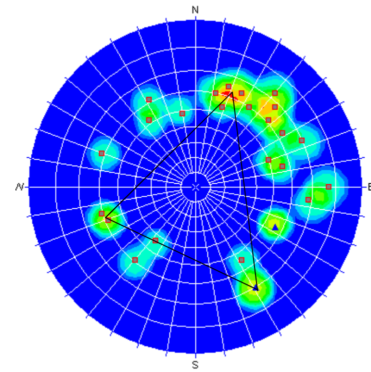


WEST RAMP – ESCOBAL MINE

23

MAPPING AS PER VICTOR ISLAS

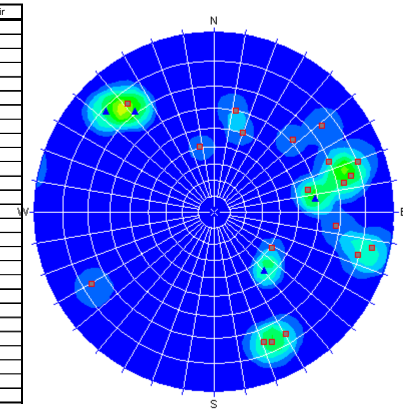
	Rumbo	Inclinación	Dip	DipDir
1	340	50SW	50	250
2	328	62SW	62	238
3	25	55NW	55	295
4	20	60SE	60	110
5	310	73SW	73	220
6	315	68SW	68	225
7	309	60NE	60	39
8	22	60SE	60	72
9	25	55NW	55	295
10	126	65SW	65	206
11	305	45NE	45	35
12	304	60SW	60	214
13	240	70NW	70	330
14	55	50SE	50	145
15	78	50SE	50	168
16	61	60SE	60	151
17	0	76W	76	270
18	60	70NW	70	330
19	57	55NW	55	327
20	110	65SW	65	200
21	108	54SW	54	198
22	338	58NE	58	68



EAST RAMP

22 JOINTS/STRUCTURES

	Rumbo	Inclinación	Dip	DipDir
1	40	79SE	40	169
2	62	75NW	75	232
3	160	80SW	80	250
4	17	80NW	80	287
5	166	75SW	75	256
6	53	70SE	70	143
7	240	75NW	75	330
8	70	75NW	75	340
9	66	78NW	78	336
10	193	85NW	85	283
11	51	76SE	76	141
12	174	60SW	60	264
13	171	60SW	60	261
14	108	50SW	50	198
15	42	78SE	78	132
16	44	78SE	78	134
17	229	46NW	46	319
18	214	42NW	42	304
19	328	76NE	76	58
20	228	48NW	48	318
21	52	72SE	72	142
22	166	72SW	72	256
23	167	55SW	55	257
24	188	70NW	70	278
25	158	70SW	70	248
26	100	60SW	60	190
27	137	60SW	60	227



WEST RAMP

27 JOINTS/STRUCTURES

STEEL ARCHES AT ~1.5m SPACING FROM PORTAL TO FACE.

STEEL ARCHES NOT LAGGED AT RIB (0.3m VOID) AND MINOR AT BACK WITH TIMBER LAG. PURPOSE IS TO CONFINE SPILES 25mm DIAM.

RMR BACK/WALL TUFFS

1) STRENGTH	10-25MPa	2
2) RQD	50%-25%	8-3
3) SPACING	50-300mm-	10-5
4) CONDITON	GOUGE(DRY)	6
5) GRNWTR	DRY	10
	RATING(WALL)	36%-26%
STRUCTURE	FLAT	
	DESIGN (BACK)	25%-35%

NOTE IF WET/MOIST WOULD BE 15% - 25% RMR.

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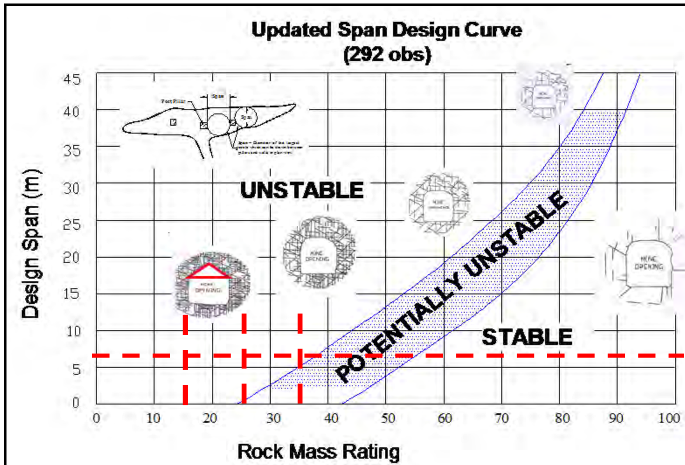
STOP #3: WEST RAMP. ALTERED ANDESITE. SPAN IS 6m. SPILING. SAN CARLOS UNIV. TESTED UCS=14MPa.

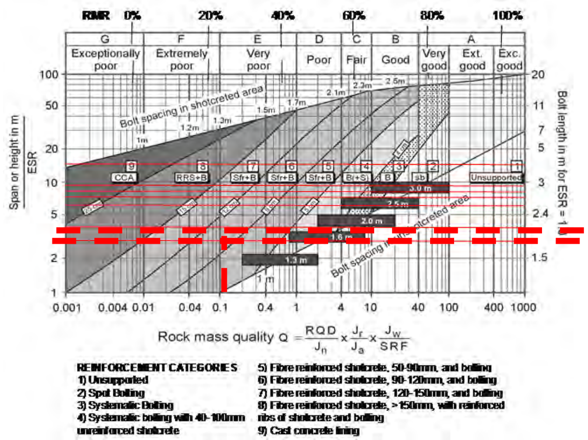


WILL CAVE ABOVE BACK. SPAN OF 6m WITH RMR 25%-35% (DRY) AND 15%-25% (MOIST/WET).

26

STOP #3: WEST RAMP. ALTERED ANDESITE (PORPHYRYTIC). SPAN IS 6m. SPILING. SAN CARLOS UNIV. TESTED UCS=14MPa.

RMR=25% SUPPORT CLASS 4



RMR BACK = 25%, Q= 0.12

ESR=1.6 PERMANENT. SPAN=5m/1.6=3.1

FIBRE REINFORCED SHOTCRETE 90-120mm THICK(~100mm) + BOLTING.

SPAN=6m

RMR BACK = 25%, Q=0.12

ESR=1.6 PERMANENT. SPAN=6m/1.6=3.8

FIBRE REINFORCED SHOTCRETE 90-120mm THICK(~120mm) + BOLTING.

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STOP #3: WEST RAMP

RMR BACK/WALL TUFFS

1) STRENGTH	10-25MPa	2
2) RQD	<25%	3
3) SPACING	<50mm	5
4) CONDITON	GOUGE/MST	6-0
5) GRNWTR	MOIST	7
	RATING(WALL)	23%-17%
STRUCTURE	FLAT	
	DESIGN (BACK)	20%

CORE WEST RAMP AREA. HOLE E11-314 FROM 278.89m-277.36m HAVING RMR OF <25%.

~50m EAST OF EAST PORTAL. NOTE RMR ~15%.

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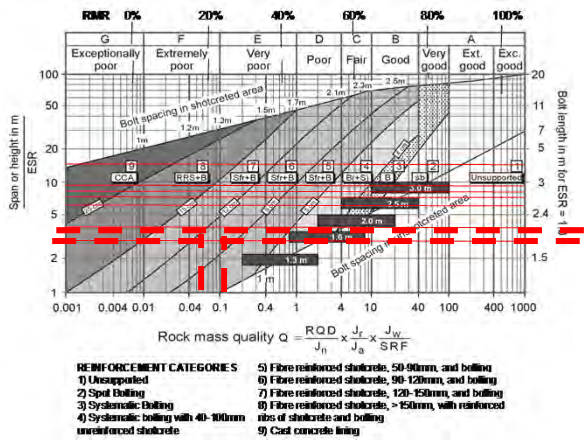
STOP #4: WEST RAMP AT FACE. MOIST RMR 15%-25% (~20%). SPAN IS 6.2m. SHOTCRETE THICKNESS IS <3mm IN AREAS. STEEL SETS ARE 1.5m APART. CRUMBLES IN HAND.

RMR=20% SUPPORT CLASS 4



RMR BACK = 20%, Q= 0.06

ESR=1.6 PERMANENT. SPAN=5m/1.6=3.1

FIBRE REINFORCED SHOTCRETE 90-120mm THICK(~100mm) + BOLTING.

SPAN=6m

RMR BACK = 25%, Q=0.12

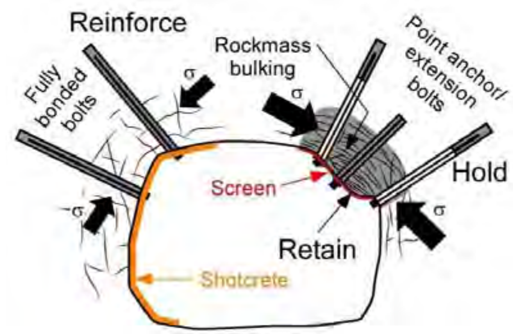
ESR=1.6 PERMANENT. SPAN=6m/1.6=3.8

FIBRE REINFORCED SHOTCRETE 90-120mm THICK(~120mm) + BOLTING.

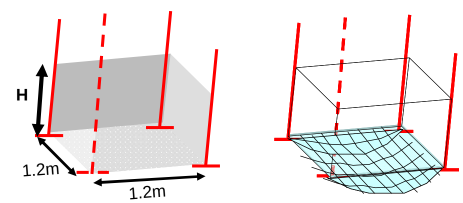
STOP #4: WEST RAMP

TABLE 1: FABRIC SUPPORT RECOMMENDATIONS - BARTON FOR 6m SPAN(20ft)

Q	RMR	PERMANENT SUPPORT ESR=1.3	TEMPORARY SUPPORT ESR=2
0.07	<20%	SPILING + (ACTUAL)	SPILING + (ACTUAL)
0.07-0.4	20%-35%	9-12cm (3"-5") FIBRE REINFORCED SHOTCRETE	5-9cm (2"-4") FIBRE REINFORCED SHOTCRETE
0.4-1.1	35%-45%	5-9cm (2"-4") FIBRE REINFORCED SHOTCRETE	4-10cm(2"-4") UNREINFORCED SHOTCRETE



SUPPORT – BAG SCREEN #6GAUGE WELD WIRE MESH



BAG STRENGTH #6 GAUGE (0.2" DIAM) WELDMESH 4"X4" 3.3tonne

H X 1.2m X 1.2m * 2.6t/m³ = 3.3tonne BAG CAPACITY RESULTING IN HT OF BAG (DEPTH OF BAG) THAT CAN A HEIGHT 0.9m. (NOTE INTACT UNIT WEIGHT USED SO CONSERVATIVE ie. BULK WEIGHT LESS). RECOMMEND THAT WHEN BAG IS 0.3-0.6m SHOULD BE CUT/REHABBED.

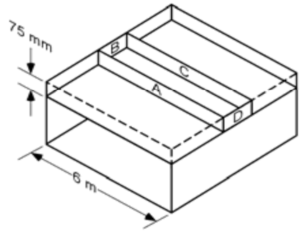
* BASED UPON SG OF ROCK OF 2.6

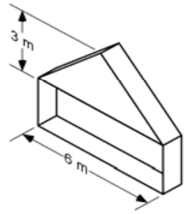
Support analysis: Shotcrete (2 MPa shear strength)

Support capacity = Shear strength x Area

Sides A, C: Support capacity = 200 t/m^2 x 6 m x 0.075 m = 90 tonnes

Sides B, D: Support capacity = 200 t/m^2 x 1 m x 0.075 m = 15 tonnes



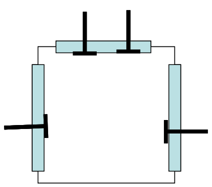


STEEL SETS 1.5m APART

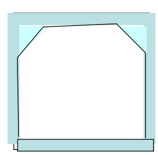
DEAD WEIGHT ON 6m SPAN
= 0.5 X 6m X 3m X 1.5m X 2.6t/m^3= 35tonne

HORIZONTAL SLAB OF 90t= 6m X Ht X 1.5m X 2.6t/m3

HORZ SLAB ~4m THICK. NOTE STEEL SETS ~16-20tonnes SUPPORT.

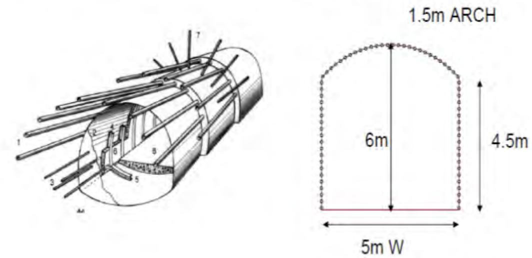


VS



INITIAL RECOMMENDATION TO ADVANCE (RPSITE)

APPROXIMATE SCHEMATIC



SPILES 200mm SPACING

ADVANCE BY 6m SPAN , SHORT CUTS 1-2m, TYPE III SUPPORT - SPILES 4.5m LONG ON 200mm SPACING FROM WALL/BACK/WALL. SUPPORT BY SHOTCRETE ARCHES EVERY 2m AS WELL A TYPE III SUPPORT.

SHOTCRETE ARCH Specification Standard

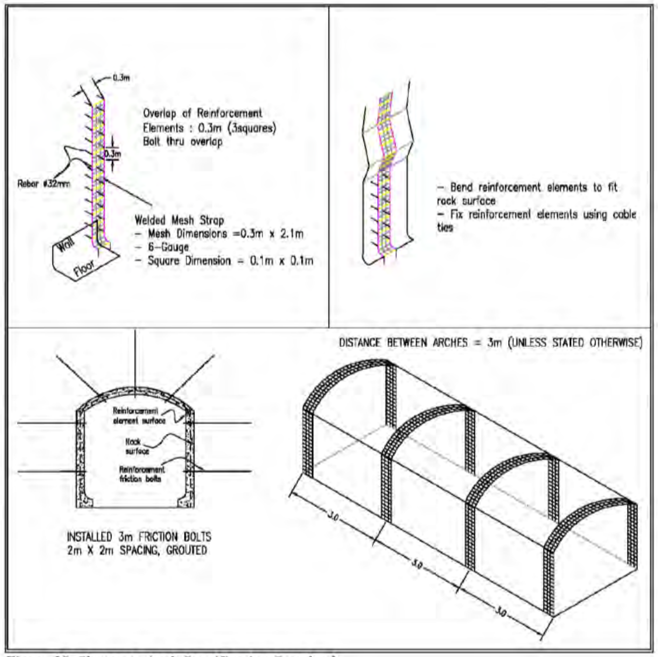


Figure 28 Shotcrete Arch Specification Standard

- Shotcrete arch to be installed in specified POOR to VERY POOR ground
- Pin the mesh using 3m long friction bolt, 2m x 2m spacing, grouted.
- Shotcrete should be sprayed over the mesh arch to minimum thickness 300mm
- Sequence of application in support cycle are : shotcrete and shotcrete fillets, bolt and mesh, shotcrete arches, grout bolts, spiling (if prescribed), mesh face, than continue taking advance.

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SUPPORT TYPE 4: VERY POOR ROCK (RMR<25)

- Spray floor-to-backs with 25-50mm 'flash coat' of fibrecrete
- Install weld mesh down to the floor with a spot bolting pattern using ungrouted 2.4m friction bolts
- Re-spray floor-to-backs with 50-75mm of fibrecrete (giving a total thickness of 100mm)
- Depending on the specific profile and location, install 2.4m or 3.0m friction bolts (47mm), 1.0m internal spacing x 1.0m ring spacing (installed after the final spray of fibrecrete
- Lowest row of bolts must be collared within 0.5m from the floor
- Spile the perimeter, if required prior to firing the next cut
- Cut lengths should be reduced to about 2.0m

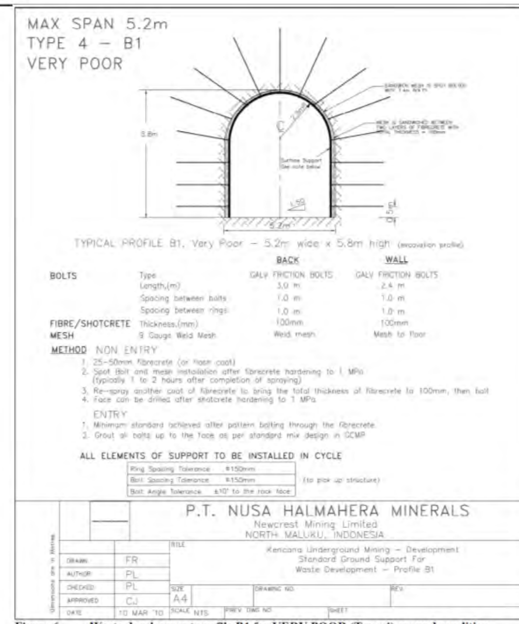

Figure 6 Waste development profile B1 for VERY POOR (Type 4) ground conditions

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SPILING Specification Standard

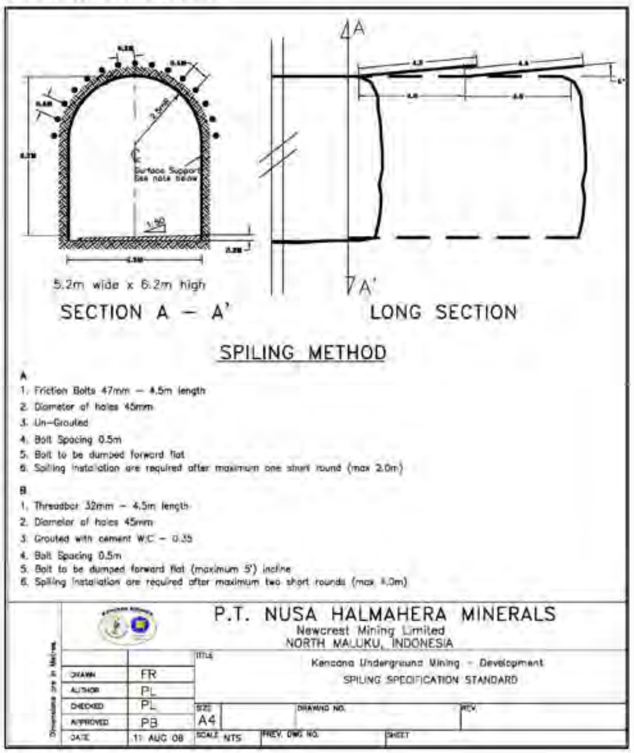

Figure 27 Spiling Specification Standard

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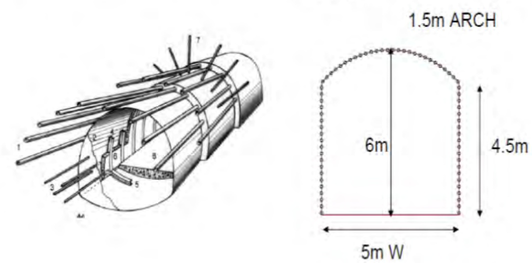
INITIAL RECOMMENDATION TO ADVANCE (RPSITE)

APPROXIMATE SCHEMATIC



SPILES 200mm SPACING

ADVANCE BY 6m SPAN , SHORT CUTS 1-2m, TYPE III SUPPORT - SPILES 4.5m LONG ON 200mm SPACING FROM WALL/BACK/WALL. SUPPORT BY SHOTCRETE ARCHES EVERY 2m AS WELL A TYPE III SUPPORT.

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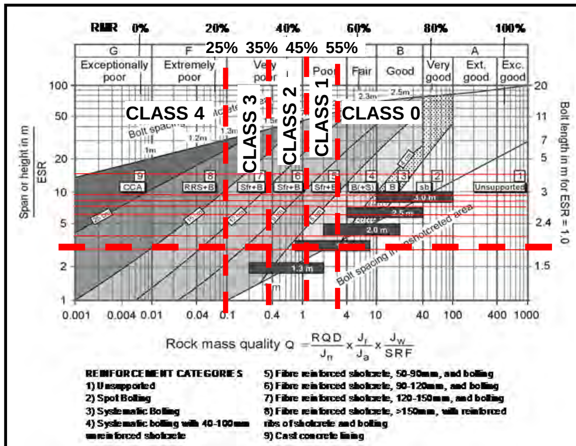
GENERAL GROUND SUPPORT REQUIREMENTS

SUPPORT TYPE 2: FAIR ROCK RMR≥35)
➢ Spray floor-to-backs with 50mm fibrecrete
➢ Meshing over the fibrecrete: always mesh backs and shoulders; mesh to gradeline as required/directed
➢ Install 2.4m or 3.0m friction bolts (47mm), 1.2m internal spacing x 1.2m ring spacing
➢ Lowest row of bolts must be collared within 1.0m from the floor
SUPPORT TYPE 3: POOR ROCK (25≤RMR<35)
➢ Spray floor-to-backs with 100mm fibrecrete
➢ Meshing over the fibrecrete: always mesh backs and shoulders; mesh to gradeline as required/directed
➢ Lowest row of bolts must be collared within 1.0m from the floor
SUPPORT TYPE 4: VERY POOR ROCK (RMR<25)
➢ Spray floor-to-backs with 25-50mm 'flash coat' of fibrecrete
➢ Install weld mesh down to the floor with a spot bolting pattern using ungrouted 2.4m friction bolts
➢ Re-spray floor-to-backs with 50-75mm of fibrecrete (giving a total thickness of 100mm)
➢ Depending on the specific profile and location, install 2.4m or 3.0m friction bolts (47mm), 1.0m internal spacing x 1.0m ring spacing (installed after the final spray of fibrecrete)
➢ Lowest row of bolts must be collared within 0.5m from the floor
➢ Spile the perimeter, if required prior to firing the next cut
➢ Cut lengths should be reduced to about 2.0m
FACE MESH FOR ALL GROUND CONDITIONS
➢ Install mesh from the top of the face down to grade line, using 1.2m un-grouted friction bolts on a 1.2m x 1.2m pattern. Overlap mesh minimum 0.2m over the fibrecrete where the back meets the face.

FRICTION BOLTS
➢ 47mm diameter x 3.0m or 2.4m friction bolts (depending upon the specific drive profile)
➢ Minimum bond strength: 3.3 tonne per metre (after grouting)
➢ 47mm x 1.2m friction bolts: minimum bond strength = 2-tonnes (un-grouted)
➢ Black friction bolts and plates are to be used in ore development (service life < 1 year)
➢ Galvanised bolts and plates are to be used in waste development (service life > 1 year)
WELD MESH
➢ Minimum #9 gauge wire with 10cm x 10cm grid
➢ Black mesh to be used in ore development (service life < 1 year)
➢ Galvanised mesh to be used in waste development (service life > 1 year)
FIBRECRETE
➢ Thickness as specified in the standard patterns
➢ Fibre addition at Batch Plant using 48mm (minimum) poly fibres at 3 kg per cubic metre
➢ 'Non-entry' work such as remote bolting: fibrecrete UCS ≥ 1 MPa
➢ 'Entry' work such grouting bolts to face, face mark-up, and sampling: fibrecrete must be pattern bolted.
➢ Long term stability: 28-day UCS ≥ 30 MPa

Table 1 General Minimum Ground Support Requirements—Friction Bolts and Fibrecrete Method



BACK SPAN=5m, ESR=1.6 (PERMANENT), SPAN/ESR=5m/1.6=3.1m

	RMR_{76}	Q'
CLASS 1	45% +	>1.1
CLASS 2	35%-45%	0.4-1.1
CLASS 3	25%-35%	0.1-0.4
CLASS 4	<25%	<0.1

	RMR_{76}	Q'
CLASS 0	55% +	>3.4
CLASS 1	45%-55%	1.1-3.4
CLASS 2	35%-45%	0.4-1.1
CLASS 3	25%-35%	0.1-0.4
CLASS 4	<25%	<0.1

TAHOE RESOURCES INC. – ESCOBAL PROJECT

FIGURE : SUPPORT – DEVELOPMENT ROCK MASS

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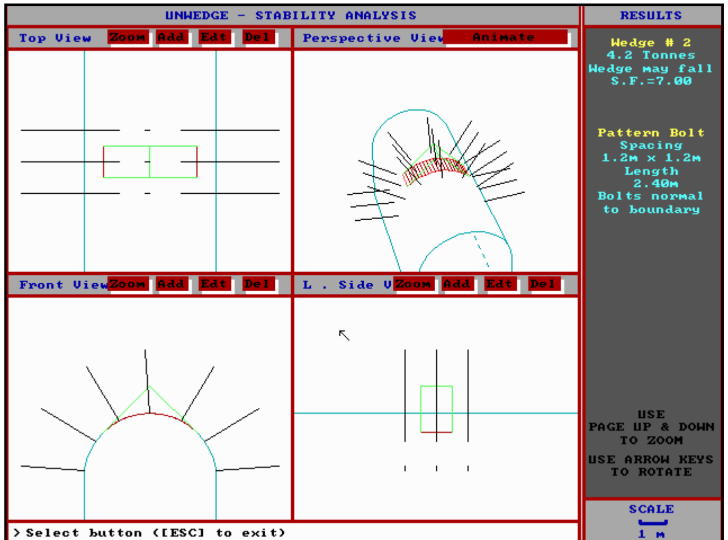

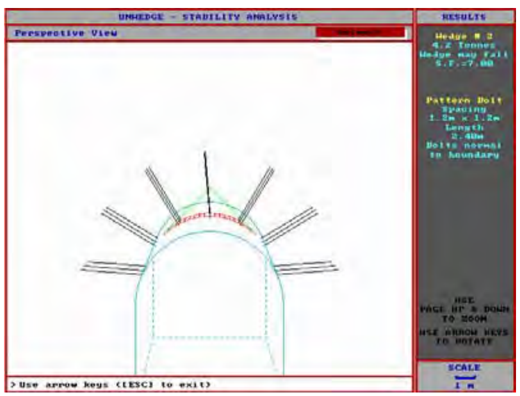

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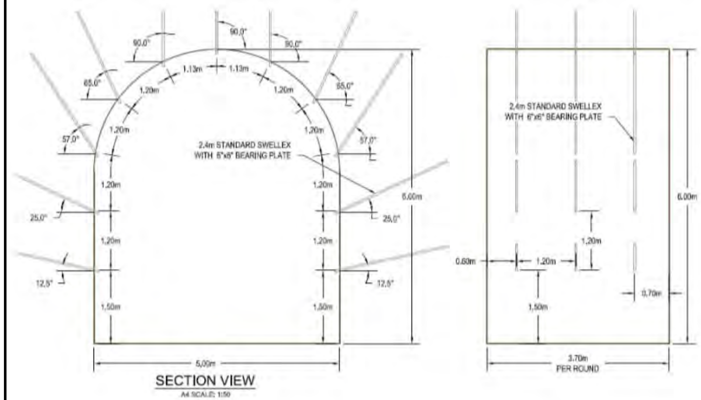

SHOULD BOLT PERPENDICULAR TO WALL.

CLASS 1 GROUND SUPPORT

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SUPPORT DISCUSSION - UPDATE

	RMR$_{76}$	Q'
CLASS 1	45% +	>1.1
CLASS 2	35%-45%	0.4-1.1
CLASS 3	25%-35%	0.1-0.4
CLASS 4	<25%	<0.1

PERMANENT SUPPORT – 5m WIDE OPENINGS X 6m HIGH

CLASS 1: 45% + RMR REQUIRE 2.4m LONG STD SWELLEX ON 1.2m X 1.2m PATTERN + SPOTBOLTS ON RIBS. NOTE 45% LESS REQUIRE SCREEN AND/OR UNREINFORCED SHOTCRETE(2")

CLASS II: 35%-45%: 2.4m LONG STD SWELLEX ON 1.2m BETWEEN BOLTS X 1.8m BETWEEN ROWS + 1.8m SPLITSETS(5-3-5 PATTERN) + (5cm X 5cm) CHAINLINK(2.7mm) IN BACK WALLS . RIBS BOLTED/CHAINLINK TO 1.5m OF FLOOR. 1.8m SPLITS IN BACK. 2.4m SWELLEX 1.5m OFF FLOOR IN RIB AUGMENTED BY SPLITS. NOTE AT <40% REQUIRE FIBRE SHOTCRETE (50mm-75mm) OR SHOTCRETE+SCREEN.

CLASS III: 25%-35%: CLASS I SUPPORT + 50mm-75mm SHOTCRETE. BOLT THROUGH SHOTCRETE. ADVANCE LIMITED TO 2m SHORT ROUNDS. BACK SHOULD BE ARCHED IN THIS GROUND(~1m). FIBRE SHOTCRETE OR SHOTCRETE + SCREEN

CLASS IV: <25%: SPILING + CLASS I. SPILES 4m WITH 2m ADVANCE. SPILES NO FURTHER THAN 0.3m APART. (32mm DIAM). BACK SHOULD BE ARCHED IN THIS GROUND. NOTE SHOULD SHOTCRETE BACK TO CONFINE POTENTIAL FALLOUT BETWEEN SPILES. 50mm-75mm-100mm.

WILL EVALUATE UPON EXPOSURE.

TAHOE RESOURCES INC. – ESCOBAL PROJECT

FIGURE 8: SUPPORT – DEVELOPMENT – ROCK MASS -

WASTE DEVELOPMENT – PERMANENT RAMP SUPPORT – 5m(W) X 6m(H) ARCHED TO 0.5 X SPAN. UPDATED AUGUST 2011.

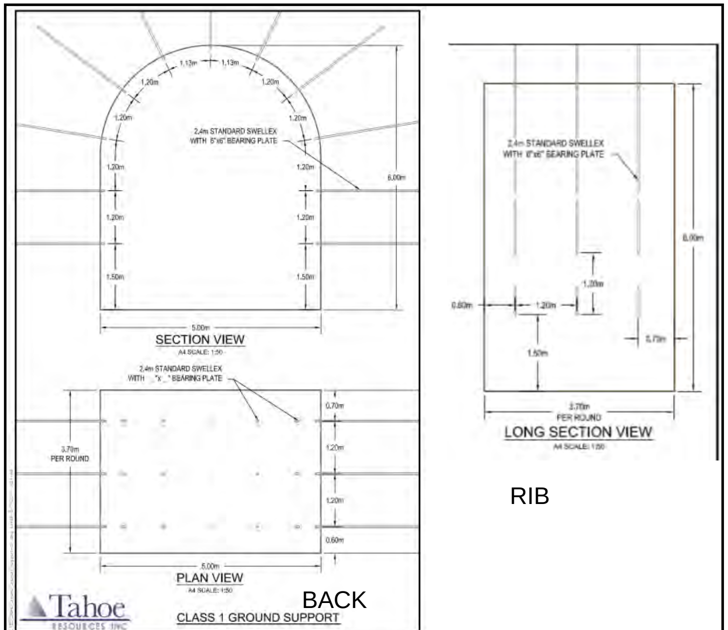
CLASS 0: 55% + RMR. REQUIRE 2.4m LONG STD SWELLEX ON 1.2m X 1.2m PATTERN + SPOTBOLTS ON RIBS. NO SCREEN/SHOTCRETE REQUIRED. MUST ENSURE SCALED/RAMP INSPECTION PERIODICALLY TO REMOVE LOOSE/POTENTIAL SCALE THAT WILL RESULT DUE TO FURTHER MINING/ACTIVITY.

CLASS 1: 45% - 55% RMR. REQUIRE 2.4m LONG STD SWELLEX ON 1.2m X 1.2m PATTERN + SPOTBOLTS ON RIBS. NO SCREEN/SHOTCRETE REQUIRED. MUST ENSURE SCALED/RAMP INSPECTION PERIODICALLY TO REMOVE LOOSE/POTENTIAL SCALE THAT WILL RESULT DUE TO FURTHER MINING/ACTIVITY. SHOULD CONSIDER SCREEN.

CLASS 2: 35% - 45% RMR. REQUIRE 2.4m LONG STD SWELLEX ON 1.2m BETWEEN BOLTS X 1.8m BETWEEN ROWS +1.8m SPLITSETS (5-3-5 PATTERN) + MESH IN BACK/WALLS. RIBS BOLTED/MESHED TO 1.5m OF FLOOR. 1.8m SPLITS IN BACK. 2.4m SWELLEX 1.5m OFF FLOOR IN RIB AUGMENTED BY SPLITS. FIBRE SHOTCRETE(50mm-75mm) OR SHOTCRETE + SCREEN.

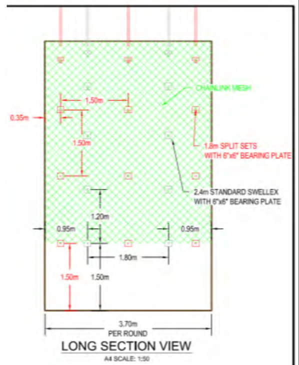
CLASS 3: 25% - 35% CLASS 1 SUPPORT +50mm-75mm SHOTCRETE. BOLT THROUGH SHOTCRETE. ADVANCE LIMITED TO 2m SHORT ROUNDS. BACK SHOULD BE ARCHED IN THIS GROUND (~1m). FIBRE SHOTCRETE OR SHOTCRETE + SCREEN.

CLASS 4: <25%: SPILING + CLASS 1. SPILES 4m WITH 2m ADVANCE. SPILES NO FURTHER THAN 0.3m APART. (32mm DIAM). BACK SHOULD BE ARCHED IN THIS GROUND. NOTE SHOULD SHOTCRETE BACK TO CONFINE POTENTIAL FALLOUT BETWEEN SPILES. APPLY 35mm FLASHCOAT, MESH, BOLT THEN APPLY FURTHER 75mm OF SHOTCRETE AND BOLT THROUGH THE SHOTCRETE. SPILE NEXT PRODUCTION ROUND IF REQUIRED.



CLASS 1 GROUND SUPPORT

CLASS 1: 45% - 55% RMR. REQUIRE 2.4m LONG STD SWELLEX ON 1.2m X 1.2m PATTERN + SPOTBOLTS ON RIBS. NO SCREEN/SHOTCRETE REQUIRED. MUST ENSURE SCALED/RAMP INSPECTION PERIODICALLY TO REMOVE LOOSE/POTENTIAL SCALE THAT WILL RESULT DUE TO FURTHER MINING/ACTIVITY. SHOULD CONSIDER SCREEN TO MINIMIZE NEED FOR SCALE/INSPECTION.



RIB

BACK

CLASS 2 GROUND SUPPORT

CLASS 2: 35% - 45% RMR. REQUIRE 2.4m LONG STD SWELLEX ON 1.2m BETWEEN BOLTS X 1.8m BETWEEN ROWS +1.8m SPLITSETS (5-3-5 PATTERN) + MESH IN BACK/WALLS. RIBS BOLTED/MESHED TO 1.5m OF FLOOR. 1.8m SPLITS IN BACK. 2.4m SWELLEX 1.5m OFF FLOOR IN RIB AUGMENTED BY SPLITS. FIBRE SHOTCRETE(50mm-75mm) OR SHOTCRETE + SCREEN.

* NOTE BETWEEN ROWS IS 1.8m AUGMENTED BY SPLITSETS. THIS WORKS ONLY BECAUSE ARCHED. NOTE HAVE BLOCKY GROUND "DEAD WEIGHT WEDGES". CONSIDER 1.2m X 1.2m SWELLEX PATTERN.

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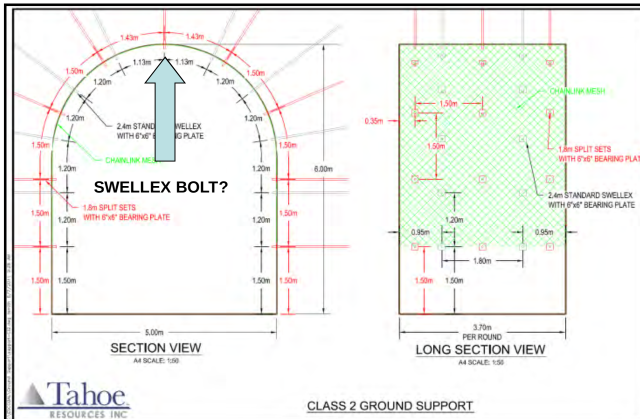
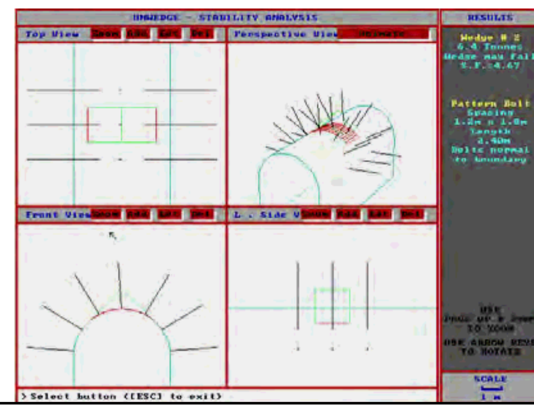
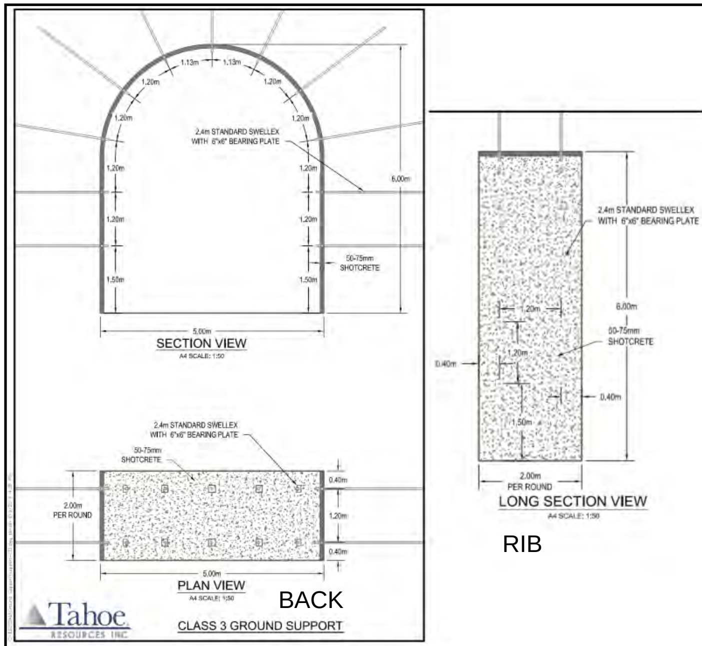


CLASS 2 GROUND SUPPORT

• 2.4m SWELLEX BOLTS ON 1.2m PATTERN IN RING AND 1.8m OUT OF RING. NOTE CENTRE BOLT SHOULD BE SWELLEX BOLT.



46



RIB

BACK

CLASS 3 GROUND SUPPORT

CLASS 3: 25% - 35% CLASS 1 SUPPORT +50mm-75mm SHOTCRETE. BOLT THROUGH SHOTCRETE. ADVANCE LIMITED TO 2m SHORT ROUNDS. BACK SHOULD BE ARCHED IN THIS GROUND (~1m). FIBRE SHOTCRETE OR SHOTCRETE + SCREEN. BOLT THROUGH SHOTCRETE.

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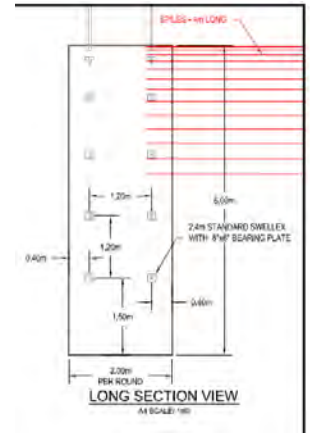

RIB

BACK

CLASS 4 GROUND SUPPORT

CLASS 4: <25%: SPILING + CLASS 1. SPILES 4m WITH 2m ADVANCE. SPILES NO FURTHER THAN 0.3m APART. (32mm DIAM). BACK SHOULD BE ARCHED IN THIS GROUND. NOTE SHOULD SHOTCRETE BACK TO CONFINE POTENTIAL FALLOUT BETWEEN SPILES. APPLY 35mm FLASHCOAT, MESH, BOLT THEN APPLY FURTHER 75mm OF SHOTCRETE AND BOLT THROUGH THE SHOTCRETE. SPILE NEXT PRODUCTION ROUND IF REQUIRED. BOLT THRU SHOTCRETE.

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SUPPORT EFFICIENCIES

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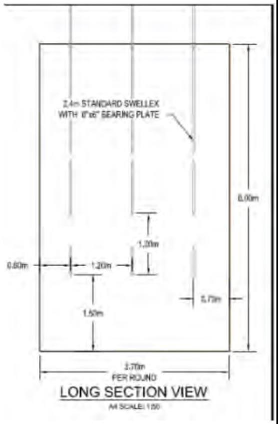


RIB

MESH BACK TO SPRINGLINE.

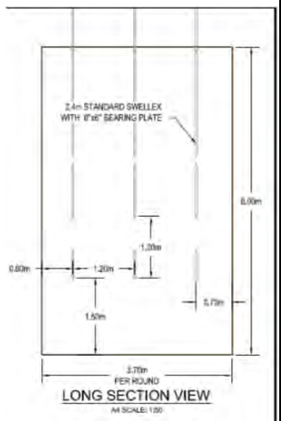
BACK

CLASS 0 GROUND SUPPORT

CLASS 0: 55% + RMR. REQUIRE 2.4m LONG STD SWELLEX ON 1.2m X 1.2m PATTERN + SPOTBOLTS ON RIBS. NO SCREEN/SHOTCRETE REQUIRED. MUST ENSURE SCALED/RAMP INSPECTION PERIODICALLY TO REMOVE LOOSE/POTENTIAL SCALE THAT WILL RESULT DUE TO FURTHER MINING/ACTIVITY.

NOTE: SCREEN DECIDED TO BE PLACED BLANKET.
FOR CLASS 0 AND CLASS 1.

50



RIB

MESH BACK TO SPRINGLINE.

BACK

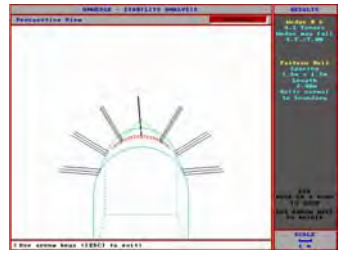
CLASS 1 GROUND SUPPORT

CLASS 1: 45% - 55% RMR. REQUIRE 2.4m LONG STD SWELLEX ON 1.2m X 1.2m PATTERN + SPOTBOLTS ON RIBS. NO SCREEN/SHOTCRETE REQUIRED. MUST ENSURE SCALED/RAMP INSPECTION PERIODICALLY TO REMOVE LOOSE/POTENTIAL SCALE THAT WILL RESULT DUE TO FURTHER MINING/ACTIVITY. SHOULD CONSIDER SCREEN TO MINIMIZE NEED FOR SCALE/INSPECTION.

51

CLASS 1 RMR 45%-55%. STD 2.4m LONG SWELLEX ON 1.2m X 1.2m PATTERN.

BRK STRENGTH=11tonne, BOND STRENGTH = 12t/m.



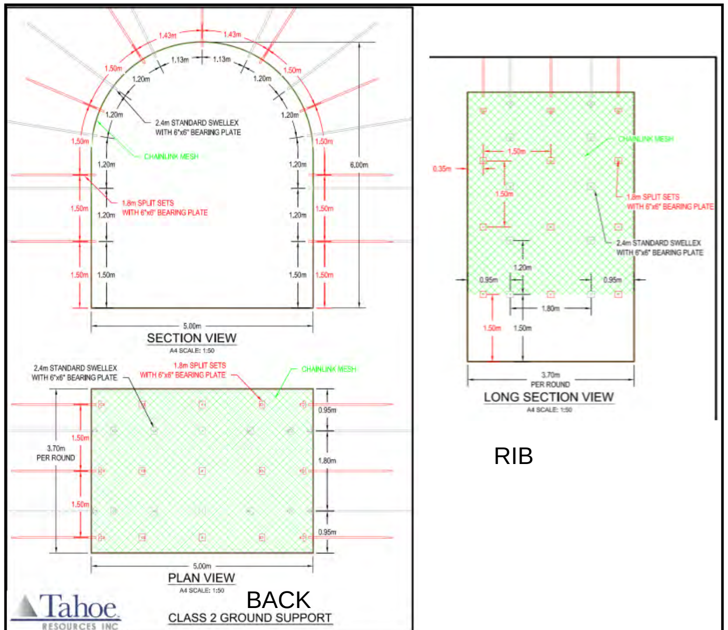
ARCHED
FS=7.0 (5m SPAN)

FS=6.6 (6m SPAN)

FLAT BACK
FS=2.3 (5m SPAN)

FS=1.3 (6m SPAN)

52



CLASS 2 GROUND SUPPORT

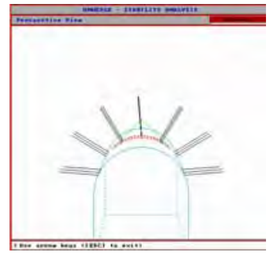
CLASS 2: 35% - 45% RMR. REQUIRE 2.4m LONG STD SWELLEX ON 1.2m BETWEEN BOLTS X 1.8m BETWEEN ROWS +1.8m SPLITSETS (5-3-5 PATTERN) + MESH IN BACK/WALLS. RIBS BOLTED/MESHED TO 1.5m OF FLOOR. 1.8m SPLITS IN BACK. 2.4m SWELLEX 1.5m OFF FLOOR IN RIB AUGMENTED BY SPLITS. FIBRE SHOTCRETE(50mm-75mm) OR SHOTCRETE + SCREEN.

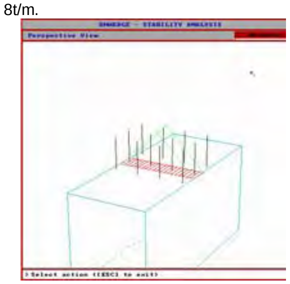
* NOTE BETWEEN ROWS IS 1.8m AUGMENTED BY SPLITSETS. THIS WORKS ONLY BECAUSE ARCHED. NOTE HAVE BLOCKY GROUND "DEAD WEIGHT WEDGES". CONSIDER 1.2m X 1.2m SWELLEX PATTERN.

53

CLASS 2 RMR 35%-45%. STD 2.4m LONG SWELLEX ON 1.2m X 1.2m PATTERN.

BRK STRENGTH=11tonne, BOND STRENGTH = 8t/m.





ARCHED	FLAT BACK
FS=7.0 (5m SPAN)	FS=2.0 (5m SPAN)
FS=6.6 (6m SPAN)	FS=1.1 (6m SPAN)

CLASS 2 RMR 35%-45%. STD 2.4m LONG SWELLEX ON 1.2m X 1.8m PATTERN. BOND STRENGTH=8t/m

FS=4.7 (5m SPAN)	FS=1.4 (5m SPAN)
FS=4.4 (6m SPAN)	FS=0.75 (6m SPAN) + SPLITS FS ADD ~0.1FS

CONSIDER 1.2m X 1.2m PATTERN OF SWELLEX ONLY NO SPLITSETS.

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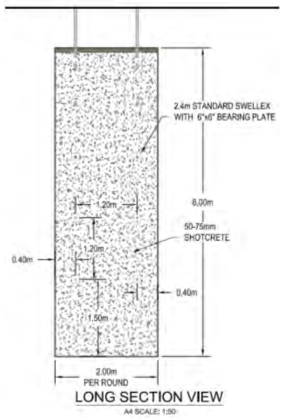


RIB

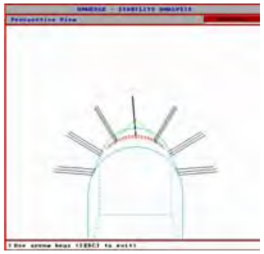
BACK

CLASS 3 GROUND SUPPORT

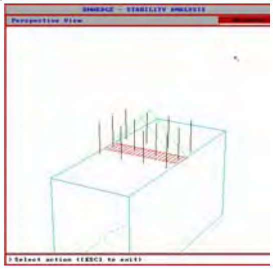
CLASS 3: 25% - 35% CLASS 1 SUPPORT +50mm-75mm SHOTCRETE. BOLT THROUGH SHOTCRETE. ADVANCE LIMITED TO 2m SHORT ROUNDS. BACK SHOULD BE ARCHED IN THIS GROUND (~1m). FIBRE SHOTCRETE OR SHOTCRETE + SCREEN. BOLT THROUGH SHOTCRETE.

CLASS 3 RMR 25%-35%. STD 2.4m LONG SWELLEX ON 1.2m X 1.2m PATTERN.

BRK STRENGTH=11tonne, BOND STRENGTH = 8t/m.

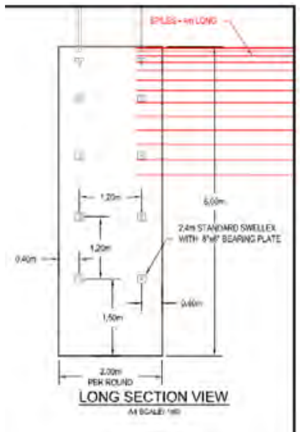


ARCHED
FS=7.0 (5m SPAN)
FS=6.6 (6m SPAN)



FLAT BACK
FS=2.0 (5m SPAN)
FS=1.1 (6m SPAN)

NOTE AFFECT OF SHOTCRETE STRENGTH IS NOT EMPLOYED AS PURPOSE OF SHOTCRETE IS TO CONFINE ROCK MASS INTO SINGLE BLOCK THAT CAN BE SUPPORTED BY THE BOLTS IN PLACE WHICH PROVIDE THE FS ANALYSIS. NOTE BOLT THROUGH THE SHOTCRETE.



RIB

BACK

CLASS 4 GROUND SUPPORT

CLASS 4: <25%: SPILING + CLASS 1. SPILES 4m WITH 2m ADVANCE. SPILES NO FURTHER THAN 0.3m APART. (32mm DIAM). BACK SHOULD BE ARCHED IN THIS GROUND. NOTE SHOULD SHOTCRETE BACK TO CONFINE POTENTIAL FALLOUT BETWEEN SPILES. APPLY 35mm FLASHCOAT, MESH, BOLT THEN APPLY FURTHER 75mm OF SHOTCRETE AND BOLT THROUGH THE SHOTCRETE. SPILE NEXT PRODUCTION ROUND IF REQUIRED. BOLT THRU SHOTCRETE.

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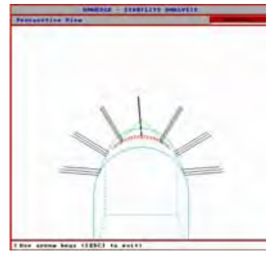
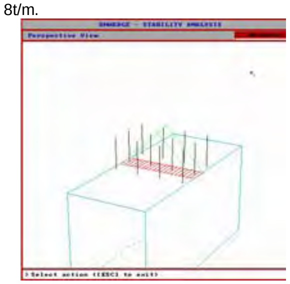
CLASS 4 RMR <25%. STD 2.4m LONG SWELLEX ON 1.2m X 1.2m PATTERN.

BRK STRENGTH=11tonne, BOND STRENGTH = 8t/m.



ARCHED
FS=7.0 (5m SPAN)
FS=6.6 (6m SPAN)



FLAT BACK
FS=2.0 (5m SPAN)
FS=1.1 (6m SPAN)

NOTE AFFECT OF SHOTCRETE STRENGTH IS NOT EMPLOYED AS PURPOSE OF SHOTCRETE IS TO CONFINE ROCK MASS INTO SINGLE BLOCK THAT CAN BE SUPPORTED BY THE BOLTS IN PLACE WHICH PROVIDE THE FS ANALYSIS. NOTE BOLT THROUGH THE SHOTCRETE. THE SPILES PROVIDE A MEANS TO ENABLE THE SUPPORT TO BE PLACED. NOTE MAY REQUIRE 2X BOLTING SO FS IS 2X IF SAME LENGTHS EMPLOYED OF SWELLEX. FLASH COAT + MESH CAN BE CONFINED BY SMALLER LENGTH BOLTS.

NOTE SPILES WILL PROVIDE A FS IN EXCESS OF 5.0.

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QUALITY CONTROL/ASSURANCE – TESTING ULTIMATE LAB

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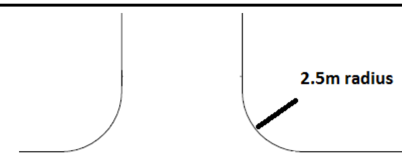

5.0m wide decline and x-cut chamfered corner starts 2.5m from projected intersection corner, i.e. intersection of straight walls.

For of equipment of comparable size to what we will be using, I have previously applied a 2.5m radius to reflect the chamfering of corners at the intersection. Attached is a rough sketch of how the intersection corners will look after mining based on this assumption.

This chamfering of the corners increases the overall span at the intersection and as Don mentioned we would like this to be considered in the design of the support.

On the longer swellex bolts, the supplier actually does make the 4m long bolt however there is a delay in the supply of these and we have ordered a stock of 200 x 3.6m long bolts thus our interest in the support design using this length of bolt.

Regards

Stuart

NOTE LARGEST SPAN IN 3 WAY IS 7.5m.

• **MINE PASS THE POTENTIAL INTERSECTION AND SUBSEQUENTLY BOLT TO REQUIRED THEN OPEN INTERSECTION TO ULTIMATE DIMENSIONS.**

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TAHOE RESOURCES INC. – ESCOBAL PROJECT

FIGURE 1: SUPPORT – INTERSECTION

PAKALNIS & ASSOCIATES

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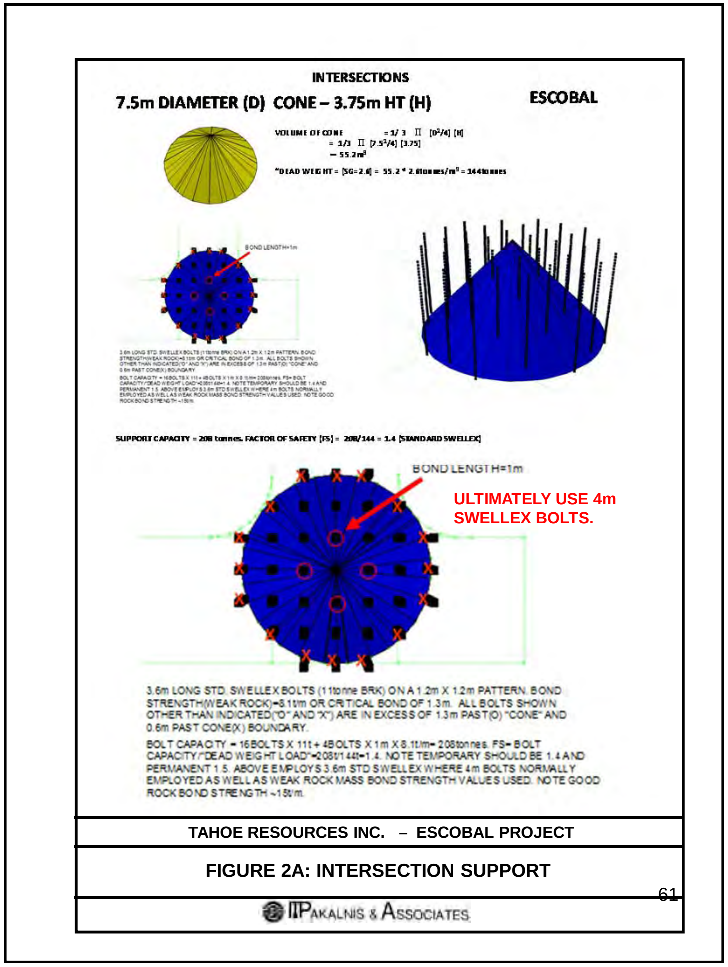
INTERSECTIONS
7.5m DIAMETER (D) CONE – 3.75m HT (H)
ESCOBAL
VOLUME OF CONE = 1/ 3 Π [D²/4] [H]
= 1/3 Π [7.5²/4] [3.75]
= 55.2m³
"DEAD WEIGHT = [SG=2.6] = 55.2 * 2.6tonnes/m³ = 144tonnes
BOND LENGTH=1m
SUPPORT CAPACITY = 208 tonnes. FACTOR OF SAFETY [FS] = 208/144 = 1.4 [STANDARD SWELLEX]
BOND LENGTH=1m
ULTIMATELY USE 4m SWELLEX BOLTS.
3.6m LONG STD. SWELLEX BOLTS (11tonne BRK) ON A 1.2m X 1.2m PATTERN. BOND STRENGTH(WEAK ROCK)=8.1t/m OR CRITICAL BOND OF 1.3m. ALL BOLTS SHOWN OTHER THAN INDICATED("O" AND "X") ARE IN EXCESS OF 1.3m PAST(O) "CONE" AND 0.6m PAST CONE(X) BOUNDARY.
BOLT CAPACITY = 16BOLTS X 11t + 4BOLTS X 1m X 8.1t/m= 208tonnes. FS= BOLT CAPACITY/"DEAD WEIGHT LOAD"=208t/144t=1.4. NOTE TEMPORARY SHOULD BE 1.4 AND PERMANENT 1.5. ABOVE EMPLOYS 3.6m STD SWELLEX WHERE 4m BOLTS NORMALLY EMPLOYED AS WELL AS WEAK ROCK MASS BOND STRENGTH VALUES USED. NOTE GOOD ROCK BOND STRENGTH ~15t/m.
TAHOE RESOURCES INC. – ESCOBAL PROJECT
FIGURE 2A: INTERSECTION SUPPORT
PAKALNIS & ASSOCIATES
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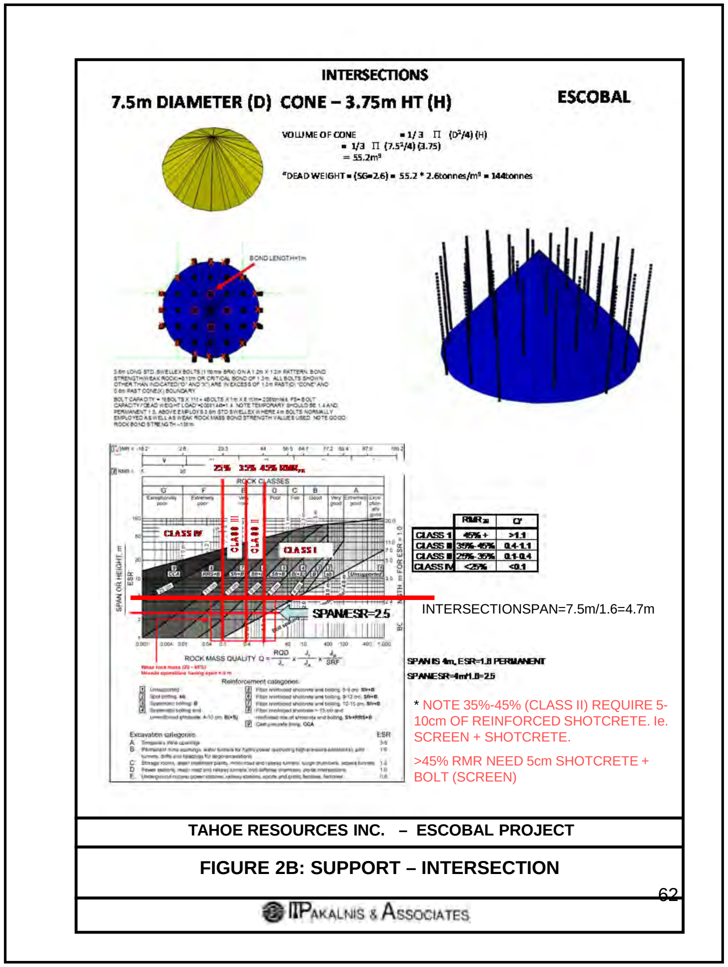
INTERSECTIONS
7.5m DIAMETER (D) CONE – 3.75m HT (H)
ESCOBAL
VOLUME OF CONE = 1/3 Π (D²/4) (H)
= 1/3 Π (7.5²/4) (3.75)
= 55.2m³
"DEAD WEIGHT = (SG=2.6) = 55.2 * 2.6tonnes/m³ = 144tonnes
BOND LENGTH=1m
ROCK CLASSES
CLASS IV
CLASS III
CLASS II
CLASS I
SPAN/ESR=2.5
ROCK MASS QUALITY
	RMR	Q'
CLASS 1	45% +	>1.1
CLASS II	35%-45%	0.4-1.1
CLASS III	25%-35%	0.1-0.4
CLASS IV	<25%	<0.1
INTERSECTIONSPAN=7.5m/1.6=4.7m
SPAN IS 4m, ESR=1.6 PERMANENT
SPAN/ESR=4m/1.6=2.5
* NOTE 35%-45% (CLASS II) REQUIRE 5-10cm OF REINFORCED SHOTCRETE. Ie. SCREEN + SHOTCRETE.
>45% RMR NEED 5cm SHOTCRETE + BOLT (SCREEN)
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FIGURE 2B: SUPPORT – INTERSECTION
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TRAINING

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Barrenos W10-12 y W10-13

WEST PORTAL

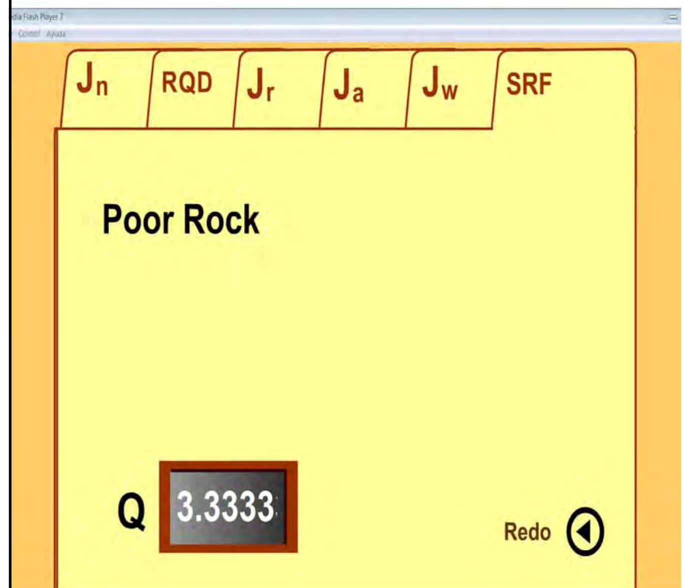
WEDGE IN PORTAL OBSERVED.

RECOGNITION OF GROUND HAZARDS - MAPPING

65

Calculo del Q´ en el Portal Este

SWITCH TO RMR!!

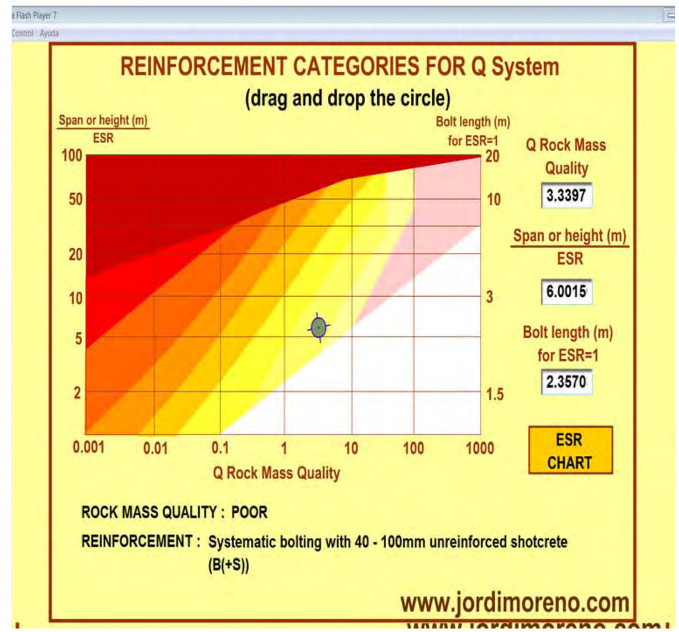


66

Calculo del RMR en el Portal Este

67

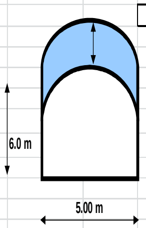
Clasificación de la Roca y tipo de Soporte Para el Portal Este



68

Calculo Longitud de Anclaje y Arco de Carga del Portal Este por Método Noruego y Barton

Calculo de la longitud de anclaje y Arco de CargaPortal Este



2.50 ARCO DE CARGA								
Formula 1	1.40 +0.184 *CLARO (5) →				Longitud de Anclaje		2.32	Int Noruego
Formula 2	2+(0.15 H/ESR)				Longitud de Anclaje		2.13	Barton
Arco de carga								
formula 2	100 - RMR / 100	*B claro Obr	100 / 100	50	50 / 100	0.5	2.50	
	Peso del Arco de Carga							
	b*h/2	5	2.5	2	6.25			
					15.625	TON		

"DEAD WEIGHT WEDGE" CALCULATION

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WASTE DEVELOPMENT – PERMANENT RAMP SUPPORT – 5m(W) X 6m(H) ARCHED TO 0.5 X SPAN. UPDATED AUGUST 2011.

CLASS 0: 55% + RMR. REQUIRE 2.4m LONG STD SWELLEX ON 1.2m X 1.2m PATTERN + SPOTBOLTS ON RIBS. NO SCREEN/SHOTCRETE REQUIRED. MUST ENSURE SCALED/RAMP INSPECTION PERIODICALLY TO REMOVE LOOSE/POTENTIAL SCALE THAT WILL RESULT DUE TO FURTHER MINING/ACTIVITY.

CLASS 1: 45% - 55% RMR. REQUIRE 2.4m LONG STD SWELLEX ON 1.2m X 1.2m PATTERN + SPOTBOLTS ON RIBS. NO SCREEN/SHOTCRETE REQUIRED. MUST ENSURE SCALED/RAMP INSPECTION PERIODICALLY TO REMOVE LOOSE/POTENTIAL SCALE THAT WILL RESULT DUE TO FURTHER MINING/ACTIVITY. SHOULD CONSIDER SCREEN.

CLASS 2: 35% - 45% RMR. REQUIRE 2.4m LONG STD SWELLEX ON 1.2m BETWEEN BOLTS X 1.8m BETWEEN ROWS +1.8m SPLITSETS (5-3-5 PATTERN) + MESH IN BACK/WALLS. RIBS BOLTED/MESHED TO 1.5m OF FLOOR. 1.8m SPLITS IN BACK. 2.4m SWELLEX 1.5m OFF FLOOR IN RIB AUGMENTED BY SPLITS. FIBRE SHOTCRETE(50mm-75mm) OR SHOTCRETE + SCREEN.

CLASS 3: 25% - 35% CLASS 1 SUPPORT +50mm-75mm SHOTCRETE. BOLT THROUGH SHOTCRETE. ADVANCE LIMITED TO 2m SHORT ROUNDS. BACK SHOULD BE ARCHED IN THIS GROUND (~1m). FIBRE SHOTCRETE OR SHOTCRETE + SCREEN.

CLASS 4: <25%: SPILING + CLASS 1. SPILES 4m WITH 2m ADVANCE. SPILES NO FURTHER THAN 0.3m APART. (32mm DIAM). BACK SHOULD BE ARCHED IN THIS GROUND. NOTE SHOULD SHOTCRETE BACK TO CONFINE POTENTIAL FALLOUT BETWEEN SPILES. APPLY 35mm FLASHCOAT, MESH, BOLT THEN APPLY FURTHER 75mm OF SHOTCRETE AND BOLT THROUGH THE SHOTCRETE. SPILE NEXT PRODUCTION ROUND IF REQUIRED.

	RMR_{76}	Q'
CLASS 0	55% +	>3.4
CLASS 1	45%-55%	1.1-3.4
CLASS 2	35%-45%	0.4-1.1
CLASS 3	25%-35%	0.1-0.4
CLASS 4	<25%	<0.1

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FIGURE : SUPPORT – UPDATED AUGUST 2011 (DISCUSSION)

PAKALNIS & ASSOCIATES

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RECOMMENDATIONS/OBSERVATIONS-SUPPORT

• EAST RAMP IS PREDOMINANTLY TYPE 1 AND 0 GROUND. SHOTCRETE/SCREEN SHOULD BE EMPLOYED IN TYPE 1 AND CONSIDERED FOR TYPE 0 OTHERWISE MUST ENSURE RAMP INSPECTIONS/SCALE THROUGH LIFE OF MINE.

• EAST RAMP HAS WEDGES IN BACK. PATTERN BOLTS 2.4m LONG WILL CONFINE WEDGE IN AN ARCHED BACK. THIS IS CRITICAL AS "DEAD WEIGHT" WEDGES IDENTIFIED THROUGHOUT. NOTE THE SPAN MUST BE KEPT DOWN TO 5m AS THE ~6m SPANS MUST BE CONFINED BY 2.4m LONG BOLTS WHICH ARE MARGINAL IF ARCH IS LOST

• BOLTS DRILLED NORMAL TO WALL. PLATES CONTACT TO WALL.

• REQUIRE SHOTCRETE TO BE SPRAYED TO CONSISTENT 50mm-100mm THICK HAVING A STRENGTH IN EXCESS OF 1MPa PRIOR TO ENTRANCE.

• CLASS 4 GROUND CAN BE ACCESSED BY 100mm SHOTCRETE, ARCHING, BOLTING + SPILING IF NECESSARY. NOTE STEEL ARCHES NOT REQUIRED AND IN PLACES SHOTCRETE ARCHES CAN BE CONSIDERED AS ALTERNATIVE ie. GROUND THAT IS 15% RMR (FLOWING/WET).

	RMR_{76}	Q'
CLASS 0	55% +	>3.4
CLASS 1	45%-55%	1.1-3.4
CLASS 2	35%-45%	0.4-1.1
CLASS 3	25%-35%	0.1-0.4
CLASS 4	<25%	<0.1

TAHOE RESOURCES INC. – ESCOBAL PROJECT

FIGURE : SUPPORT – RECOMMENDATIONS/OBSERVATIONS

PAKALNIS & ASSOCIATES

RECOMMENDATIONS/OBSERVATIONS-SUPPORT

• NO MAJOR FLAWS/CONCERNS IN SUPPORT IN PLACE AS EAST RAMP IS LARGELY CONTROLLED BY BOLTS AND SCREEN.

• SHOTCRETE IS CONCERN BUT HAS BEEN AUGMENTED BY STEEL SETS IN WEST RAMP AS ONE CAN NOT RELY ON SUPPORT IN PLACE DUE TO THE SHOTCRETE AS LARGELY UNDER 3mm IN THICKNESS IN MANY PLACES.

• SHOULD BOLT THROUGH SHOTCRETE

• MAINTAIN ARCH – SHOULD LOWER SPRINGLINE AND HAVE RIB BOLTS ON EITHER SIDE OF SPRINGLINE.

• RMR/STRUCTURE MAPPING PLACED ON DRIVE PRINTS BY VICTOR ISLA TO FOREWARN OF CHANGED GROUND OR POTENTIAL WEDGES IN BACK.

	RMR_{76}	Q'
CLASS 0	55% +	>3.4
CLASS 1	45%-55%	1.1-3.4
CLASS 2	35%-45%	0.4-1.1
CLASS 3	25%-35%	0.1-0.4
CLASS 4	<25%	<0.1

TAHOE RESOURCES INC. – ESCOBAL PROJECT

FIGURE : SUPPORT – RECOMMENDATIONS/OBSERVATIONS

PAKALNIS & ASSOCIATES

CLOSE OUT SESSION – OPERATIONS
WEDNESDAY AUGUST 24, 2011

- RON CLAYTON VP – COO
- CHARLIE MUERHOFF – TECHNICAL SERVICES DIRECTOR
- STUART O'BRIEN – OPERATIONS MANAGER
- LOUIS MANCHA – MINE MANAGER
- SAM TELLEZ – MINE TRAINER

EXIT REPORT

MINE PLAN/METHOD

Tahoe RESOURCES INC

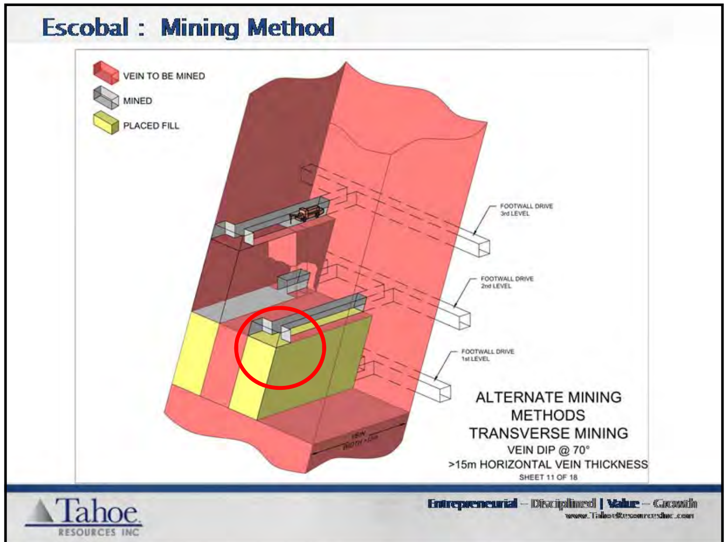
ESCOBAL MINE SITE VISIT

AUGUST 20-25, 2011

RON CLAYTON – CEO

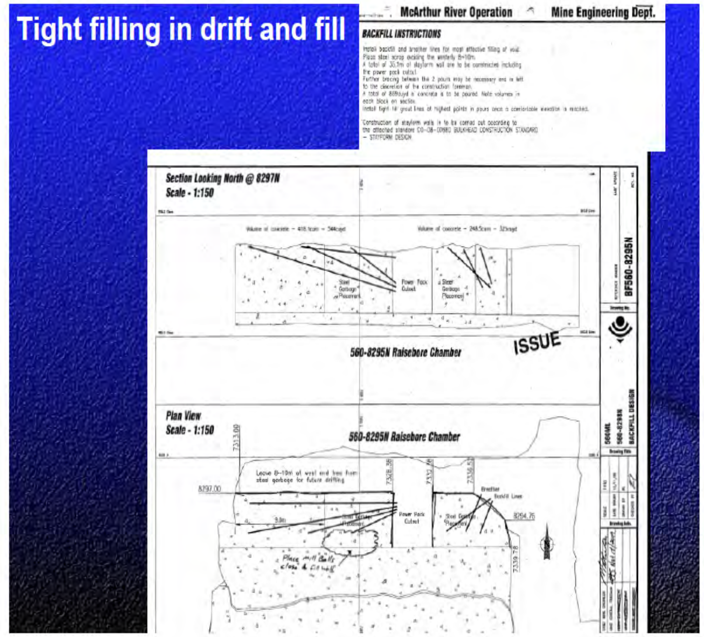
CHARLIE MUERHOFF – TECHNICAL SERVICES DIRECTOR

Minera San Rafael
GUATEMALA



FILL TIGHT SO AS TO ENSURE DRAW LEVEL WILL NOT EXCEED 5-6m IN SPAN WHEN OPERATOR WITHIN.

75



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Tight filling in drift and fill

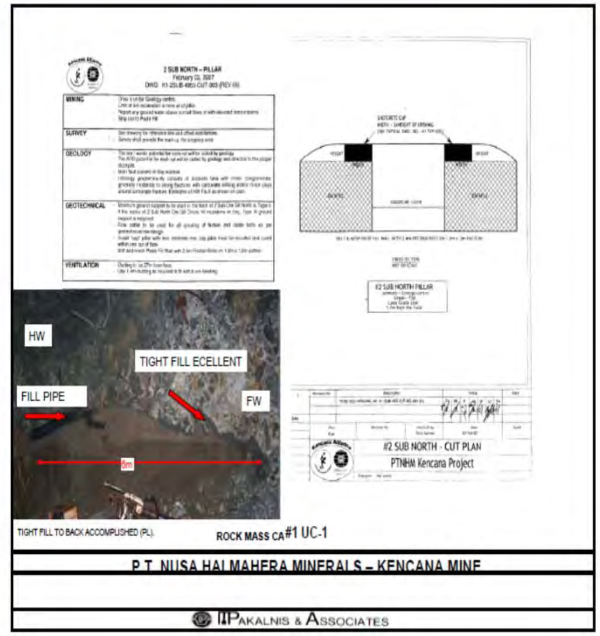
DRIFT AND FILL TIGHT TO BACK FOR PILLAR ~15m W X 20m L. EXCELLENT FORMED CONCRETE WALLS/BACK NO CONCERN OF TIGHT FILL. BEST TIGHT FILL EVER OBSERVED FOR MINE OPERATIONS!!

LOOKING AT BACK OF DRIFT AND FILL PILLAR. PILLAR ~15m X 20m AND TIGHT FILLED. SOME SPALL IN BACK DUE TO OVERLYING ~35m SPAN WITH 20m PILLAR SEPARATING TO ACCESSES OF ~7.5m EACH.

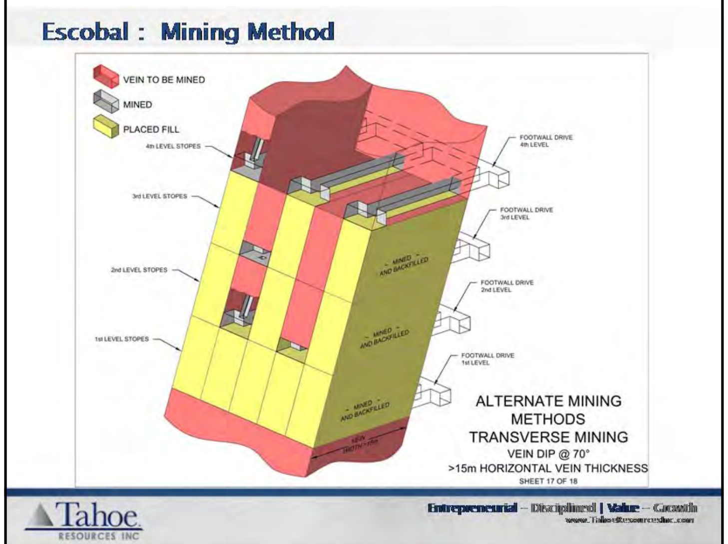
STOP #6: 560L, CHAMBER 8234N R/B

STOP #2: 530L, ZONE 2 PANEL 3 BAY 7 530L-640L

EXCELLENT TIGHT FILL TO BACK AS IS CRITICAL FOR SPAN MITIGATION!!



POTENTIAL SLOUGHING INTO SECONDARIES



STRENGTH OF PASTE AND EXPOSURE OF VERTICAL RIB EXTENDS 25m VERTICAL X 30m FW-HW.

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HAUNCHES BELOW DRILL LEVEL WILL MOST LIKELY HAVE TO BE CABLED TO MINIMIZE FAILURE.

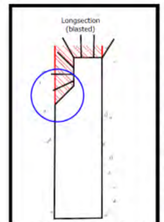

FIGURE 3: HANGING PILLAR – VERT SECTION

The success of the mining methods proposed by Snowden is largely with maintaining the stability of the "hanging pillar" as shown in Figure 3. This support within the exposed back and the support of the hanging pillar is based upon "dead weight". The methodology is sound and it is critical that the base of the pillar be confined through the placement of the swellex bolts and/or filled tight to the back with shotcrete/crf to prevent potential unravelling once the stope is being filled. Pull tests of cables/swellex support should be determined in the field within CRF/Ore/Waste so as to determine design bond strengths. Conservative estimates have been employed, however, they must be verified within the field. The hanging pillar has been employed at other mines such as the Musslewhite Mine of Placer Dome (Bob MacDonald/Chief Engineer) where support when in place was successful in limiting the unravelling as shown in Figure 4.

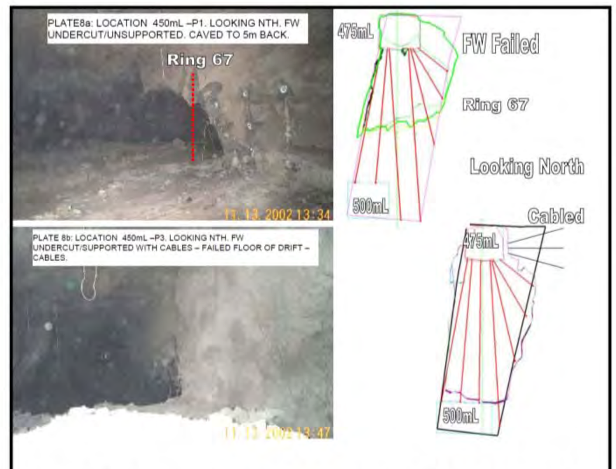

FIGURE 4: FAILED HANGING PILLAR (UNSUPPORTED) AND STAB LE HANGING PILLAR (SUPPORTED) AT MUSSELWHTIE MINE/PLACER DOME.

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2.0 MINE DESIGN

This section largely refers to Section 3 (Existing Operations) and Section 4 (New Mine Plan for the Lower Rodeo) of the SMIC report.

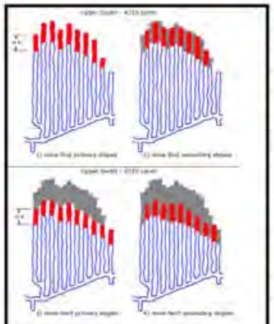


FIGURE 5: END SLICING - FLEXABILITY

The "end slicing" method as proposed by Snowden provides flexibility in enabling one to modify the mining method as a limited strike length of the primary is excavated prior to mining of the secondary. This enables modifications to the method to be made as ground behaviour is assessed. In addition, by mining the secondaries from the primary access reduces the potential for "nose pillars" which require increased support due to the small dimensions/time exposed.

The concern with respect to fan drilling is that one results in "butting" the blast holes at an angle to the final wall. This will invariably result in increased damage as compared to drilling vertical holes parallel to the final wall. This, however, can be mitigated by offsetting the drill holes from the final wall which is determined through design and field trials. The benefits in reduced drift dimensions/dilution/cost have been shown by Snowden to be significant.

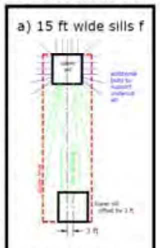


FIGURE 6: RING DRILLING CONCERN

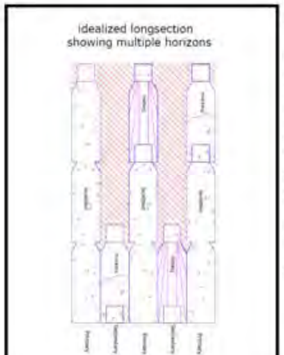


FIGURE 7: HANGING PILLAR/SEQUENCE

The methods as outlined by Snowden which involve a) Base Case, b) Transverse Wide Primary and c) Tertiary Method have similar geotechnical implications particularly with the "hanging pillar" and stope dimensioning as discussed previously and shown in Figure 7.

It is critical that the back span exposed be limited to the immediate ore/waste and the adjacent panels tight filled or mined and filled to above the panel as this would

5

Figure 1: Simplified Depiction of Historic Practices

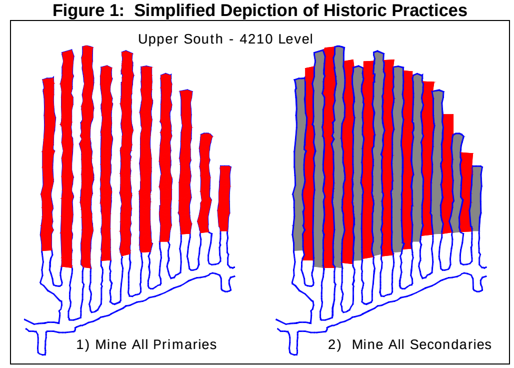
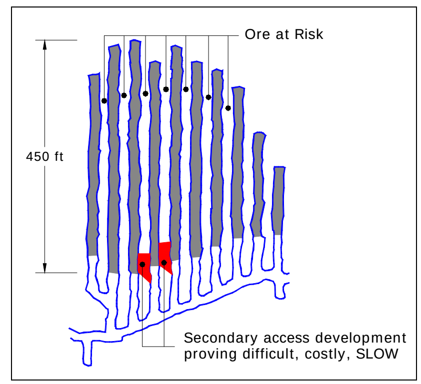


Figure 2: Depiction of Problems With Current Method



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1.Mining Between Backfill Walls

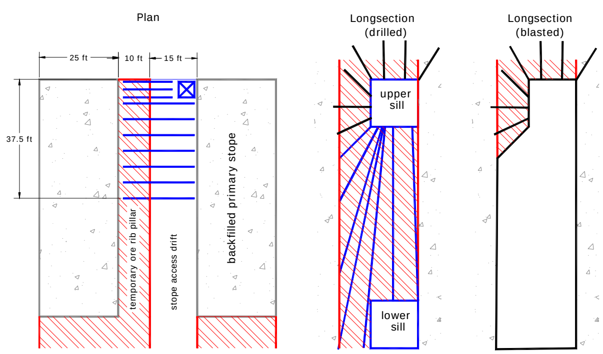
The most difficult and costly mining will be in the stopes that are between backfill walls: the secondary panel stopes of the Base Case and TWP methods and the tertiary stopes of the Tertiary Method. These have all been shown at 75 ft lengths in the depictions of the three mining methods, the same as the primary panel stopes. This length may not be possible, however, and these may have to be extracted in two shorter stopes of 37.5 ft lengths.

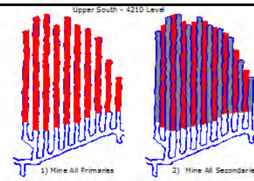
Ore cantilevers will be created by the upper and lower sills of the secondary panels when mining between backfill. These may require temporary rib pillars to be left for stability on upper and lower sills. Such pillars would be drilled by jumbo and blasted with a production blast.

The ground above temporary pillars would not be bolted, and some subsequent caving could result. This would be mitigated by bolting the ground as well as possible using longer inclined bolts.

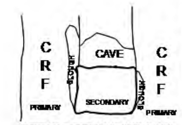
These concepts are depicted in Figure 10.

Figure 10: Section through Stope Bounded by Backfill





SCHEMATIC PLAN OF PRIMARY/SECONDARY EXTRACTION

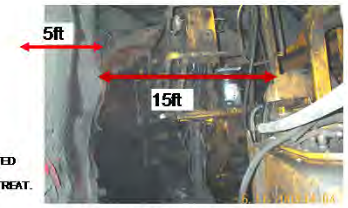


VERTICAL SECTION LOOKING DOWN STRIKE. FAILURE MECHANISM OBSERVED (L.CLARK) IN SECONDARIES. ESTIMATED COHESION IN CRF IS 25psi (L.C.)

PRIMARIES EXCAVATED FULL FACE – 20ft (6.1m) IN ORE.

SECONDARIES EXCAVATED 15ft ON ADVANCE AND SLASHED TO 20ft ON RETREAT.

VERTICAL SECTION LOOKING DOWN STRIKE



SECONDARY ACCESSED OFF PRIMARY. HAVE 5ft OF ROCK ON LEFT WALL. SPAN EXPOSED IS 15ft. RMR OF WALL/BACK IS 30%-35%. CUT HEIGHT IS 15ft.

STOP #6: MINING OF SECONDARIES. GENERALLY MINE FULL FACE IN PRIMARIES (20ft) AND 15ft WIDE IN SECONDARIES WITH SUBSEQUENT 5ft SLASH ON RETREAT. POTENTIAL FAILURE MECHANISM OBSERVED IN SECONDARIES (L. CLARK).

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2.1 Other

The longitudinal mining method for remnants is one practised at Carlin East (Newmont) where the stope height range from 15ft to 60ft+ over lengths 150ft. The concern as would be the case for the Lower Rodeo would be ensuring tight fill in adjacent panels as mining above the level is negated due to the limited vertical height.

The narrow longitudinal long hole stopes which largely parallel to strike of the orebody and in turn the major faults will result in potential back instability as is the case with existing FW drives throughout the Upper Rodeo. A possible method to minimize the volume of wedges that result would be to have the drill drives trending $\pm 20^{\circ}$ off the trend of the faults and still mining the stopes by fan drilling to final ore boundaries. Minimizing hanging wall slough would be thru reducing dimensions/support etc.

3.0 SUMMARY AND CONCLUSIONS

The methodology presented by Snowden in my professional opinion is geotechnically sound and adheres to standard industry practise. It is innovative in terms of addressing large stoping dimensions in a weak rock mass which must be calibrated/assessed and modified as mining continues. A trial stope should be considered to assess the overall assumptions and behaviour of the mining method that is selected. Instrumentation of the back/walls and pillars should be implemented to determine the overall ground behaviour particularly with the undercutting of the hanging pillar. Field testing of support and thereby quantifying its effect, minimizing blast damage to final walls along with the placement of fill as tight to the back and thereby reducing the potential for increased spans largely dictate the success of the overall methods proposed.

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OTHER INFORMATION

WEST PORTAL 1390, EAST AT 1400elev

BOTH GO TO 1190mL

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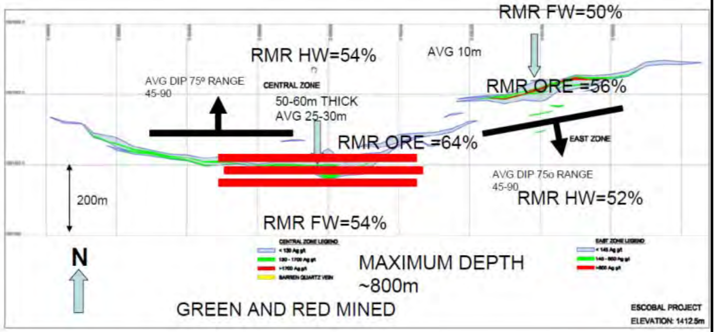

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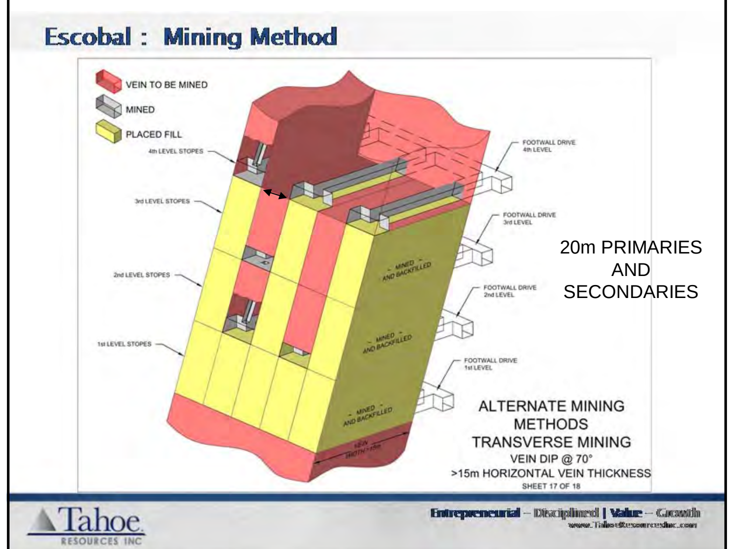
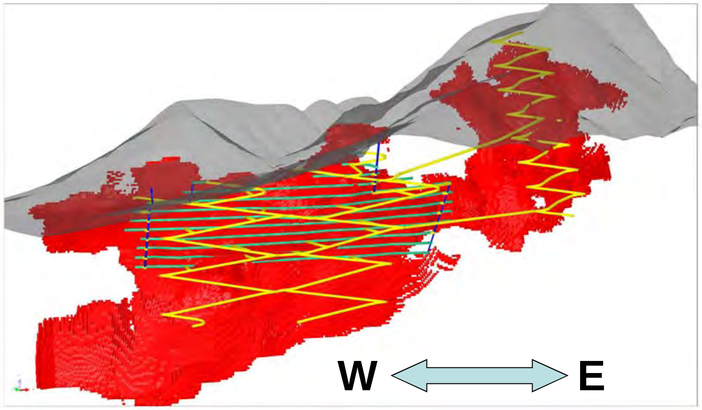


TRANSVERSE AHEAD OF LONGITUDINALS

30m TO RAMP FROM ORE. 15m TO FW DRIVE FROM RAMP. 15m NOSE PILLARS AS IS 20m PRIMARIES SECONDARIES AND 5M ADVANCE.

88



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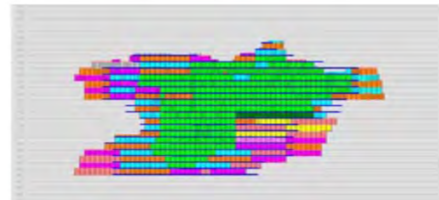
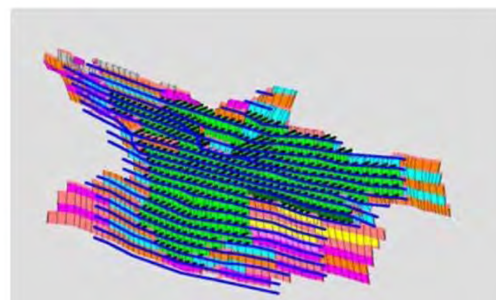
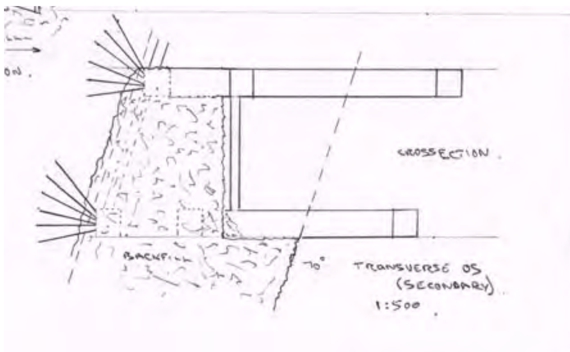
Below are some screen dumps showing the Transverse stopes (in green coloured stopes), the Longitudinal 'stoping units' (all the other coloured stopes), the Footwall Drifts (in blue), and the Transverse Crosscuts (in black).



600m DEPTH MAX. TO 900m TO EAST BECAUSE OF TOPO.



90



NEED SIX HOLES SINGLE CABLE OR THREE HOLES 2X CABLE. 5/8" CABLE. CABLES 6m LONG.

HOLE SIZE SHOULD BE MINIMUM OF 57mm FOR DOWNHOLES AND 51mm FOR UPHOLES. EMPLOY 64mm DRILL HOLE DIAM WITH SPACING OF CABLE RINGS EVERY 2.4m. CABLES PLATED AND BULGED PROVIDING 50tonne CAPACITY. NOTE PLATE ONE CABLE ONLY.

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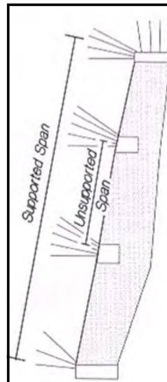
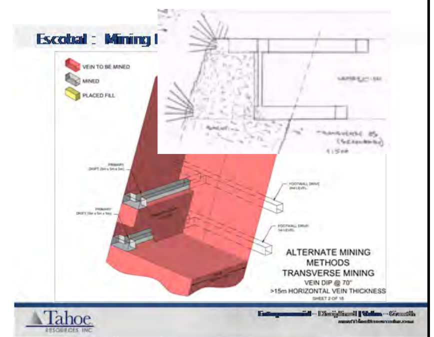
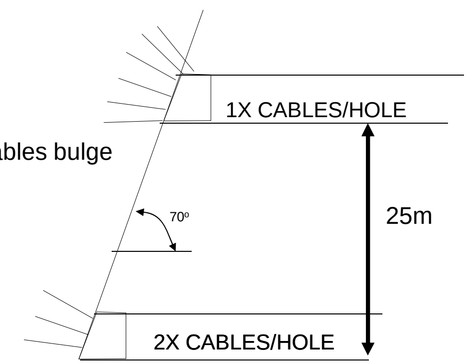




NOTE ENTIRE HW DRILL DRIVE IS ACCESSIBLE.

OPTION SINGLE OR 2X CABLE PER HOLE



TO SCALE

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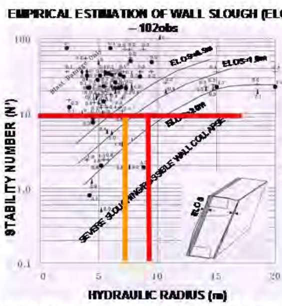
20m STRIKE

HT=70o DIP AT 30m ~32m , HR=32 X 20/(64+40)=6.2m



ORE RMR 60%

HW IS 45m STRIKE X 30m HR= 45 X 30/(90+60) =9m

30m VERTICAL

~1.5m ON EACH WALL (45m)

HW IS 30m STRIKE X 30m HR= 30 X 30/(60+60) =7.5

30m VERTICAL

~1.m ON EACH WALL (30m)

RMR = 60% , Q'=5.9, DIP OF 70º C=5.9

N = 5.9 X 1 X 0.2 X 8=9.4

HW IS 15m STRIKE X 32m HR= 15 X 32/(30+64) =5.1m

30m VERTICALLY AT 70o

30/SIN 70=32m

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ACTUAL IS FLOOR TO FLOOR AS UPPER DRILL DRIVE IS SUPPORTED SO HT= 70o DIP AT 25m ~27m, HR=27 X 20/(54+40)=5.7m

RIB EXPOSED HR=30m X 25m /(60+50)=6.8m

ORE RMR= 52% Q=2.4 N=2.4 X 1 X 0.2 X 8= 3.8

N=2.4 X 1 X1 X 8 AS STRUCTURE PARALLEL TO ORE. N=19

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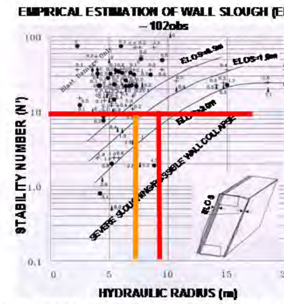

ORE RMR 60%

HW IS 45m STRIKE X 30m HR= 45 X 30/(90+60) =9m

30m VERTICAL

~1.5m ON EACH WALL (45m)

HW IS 30m STRIKE X 30m HR= 30 X 30/(60+60) =7.5

30m VERTICAL

~1.m ON EACH WALL (30m)

RMR = 60% , Q'=5.9, DIP OF 90º C=8

N = 5.9 X 1 X 0.2 X 8=9.4

HW IS 15m STRIKE X 32m HR= 15 X 32/(30+64) =5.1m

30m VERTICALLY AT 70o

30/SIN 70=32m

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WEST PORTAL 1390, EAST AT 1400elev

BOTH GO TO 1190mL

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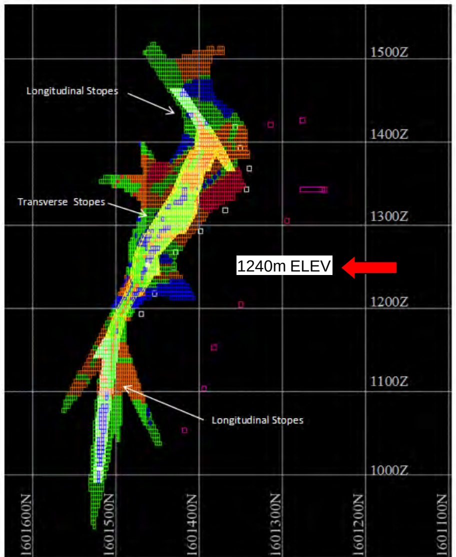
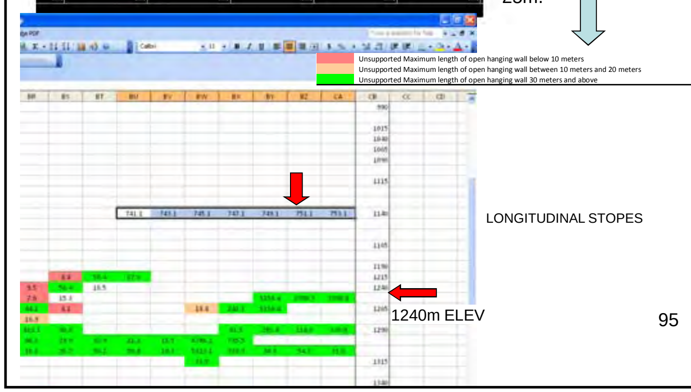


X SECTION
806750E

RMR Block Model
Cross Section 806750 E - Looking East

- **20 - 40 RMR**
- **40 - 60 RMR**
- **60 - 80 RMR**
- **80 - 100 RMR**

RMR MODEL RELATED TO EXPOSED STRIKE LENGTH POSSIBLE FOR ELOS OF 1.0m AND HT OF STOPE 25m.



LONGITUDINAL STOPES

1240m ELEV

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OBSERVATIONS/CONCLUSIONS – MINE PLAN

- TIGHT FILL OF THE TRANSVERSE STOPES IN MY PROFESSIONAL OPINION IS ACHIEVABLE. HANGING PILLAR/HAUNCHES MAY REQUIRE SUPPORT
- DESIGN EMPLOYING HW EXPOSURE AND HR/RMR/GEOMETRY EMPLOYS DATA DERIVED FROM CORE LOGS/PHOTOS AND I BELIEVE IS CONSERVATIVE IN THAT THE CORE MOST LIKELY EMPLOYS “OPEN JOINTS/SLT OPN” WHICH IN FIELD WILL BE TIGHT
- THE RIB WILL BE FW-HW ~30m X 25m HT RESULTING IN HR OF 6.8m COUPLED WITH RMR ORE AVG OF 50%-60% AND VERTICAL AND DRIVEN NORMAL TO JOINTING (A=1, B=1,C=8). THIS SHOULD BE ASSESSED BUT IS GENERALLY FAVOURABLE
- LONGITUDINAL STOPES SHOULD BE DESIGNED SIMILAR TO HR DESIGN OF TRANSVERSE STOPES. THE RELATIONSHIP BETWEEN THE RMR MODEL (C. MUERHOFF) AND THE DESIGN MODEL (R. CLAYTON) AND INTERACTION ENABLES ONE TO ARRIVE AT RECOMMENDED SPANS FOR VARIABLE INPUTS. THE ,LONGITUDINAL AND TRANSVERSE MODELS HAVE BEEN REVIEWED AND FORM A STRONG DESIGN TOOL (R. CLAYTON)
- CABLE SUPPORT OF HW SHOULD CONSIST OF 6m LONG 2X CABLES PER HOLES (BULB) THAT ARE PLATED ON A RING SPACING OF 2.4m (MAX). ALTERNATIVELY 6m LONG SINGLE CABLES (BULB) CAN BE USED. A TOTAL OF SIX CABLES ARE TO BE EMPLOYED AS A MINIMUM STD.
- THE ADDITION OF CABLES WILL ENABLE DOUBLE LIFT HEIGHT STOPES TO BE MINED OFF ONE DRAW LEVEL, THE CABLES PROVIDE COVERAGE OVER THE ENTIRE STRIKE OF THE HW AT THE DRILL AND DRAW LEVEL AND WILL REDUCE THE VERTICAL HT OF STOPE FROM 30m TO 25m AND A FURTHER 20m IF THE DRAW LEVEL HW BEAM IS NOT BROKEN. THE FLOOR TO FLOOR HEIGHT IS 25m.

TAHOE RESOURCES INC. – ESCOBAL PROJECT

FIGURE : MINE PLAN – OBSERVATIONS/CONCLUSIONS

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PAKALNIS & ASSOCIATES