Technical Report on the Prefeasibility Study of the Tepal Project Michoacán, Mexico

2,116,257mN and 717,161mE, Zone 13Q (UTM - NAD 83) - Centre of Project

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Contents

Tables

Figures

Appendices

NOTICE

This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Geologix Explorations Inc. The quality of information, conclusions and estimates contained herein is based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions and qualifications set forth in this report.

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1 Executive Summary

1.1 Introduction JDS

JDS Energy and Mining Inc. (JDS) was commissioned by Geologix Explorations Inc. (Geologix) to prepare an independent Canadian National Instrument 43-101 (NI 43-101) compliant technical report for the Tepal Project (Tepal or Project) located in Michoacán state, Mexico.

JDS was assisted in the compilation of this report by Micon International Limited (Micon), Allnorth Consultants Ltd. (Allnorth) and Knight Piésold Ltd. (KP).

This report details the results of the pre-feasibility study (PFS). The report includes Mineral Reserves estimates and represents an economically viable, technically credible and environmentally sound development plan for the Project.

1.2 Property Description and Ownership

The Tepal property is located in the municipality of Tepalcatepec, Michoacán State in southwestern Mexico as shown on Figure 1-1. The property is centered at 19° 7' 40" Latitude and 102° 56' 8" Longitude or 2,116,257mN and 717,161mE , Zone 13Q (UTM - NAD 83). The average elevation is 550m. The climate is hot and relatively dry.

Figure 1-1: Location Map of the Tepal Property

The Tepal property consists of seven contiguous concessions totalling 17,237.2ha. The property has been explored by several exploration companies over the past 30 years. Geologix owns 100% of the concession with small underlying royalties.

1.3 Geology and Mineralization

The property is located within the Coastal Range of south-western Mexico south of the Neogene Trans-Mexican Volcanic Belt. Basement rocks consist of Cretaceous to early Tertiary intermediate plutons, stocks and plugs intruding weakly metamorphosed sedimentary and volcanic rocks of probable Jurassic to Cretaceous age.

Three mineralized tonalite stocks have been identified on the property. The mineralization is characteristic of porphyry copper-gold deposits consisting of disseminated copper sulphides in structurally controlled, multi-phase intrusive zones. The North and South Zones have a gold enriched core with a copper dominant periphery and then to barren pyritic halos. There is a distinct oxide zone in the three deposits but the majority (85 to 90%) of the mineralization is sulphides.

1.4 History and Exploration

Geologix completed over 40,000m of infill diamond drilling in 2011, after the last mineral resource estimate was completed. This new drilling combined with the historic drilling was the basis of the latest mineral resource technical report (Makepeace, 2012). This infill drill program upgraded much of the previous Inferred Mineral Resource into higher classifications for use in the preliminary feasibility study.

1.5 Mineral Processing and Metallurgical Testing

There are three sources of gross metal value (GMV) from the Tepal resources. They are chalcopyrite (copper sulphide with interstitial gold and silver) in a quartz matrix, an iron pyrite (iron sulphide with interstitial gold and silver) encased in a secondary quartz/gangue matrix, and a surface oxide layer containing copper minerals (in decreasing amounts; tenorite, malachite, azurite and covellite) which also contain gold and silver values.

Sulphide ore hardness is variable in the three pits, with Tepal North being the moderately hard and Tizate being hard. Over 42 variability tests were completed with Bond Work index hardnesses ranging from a low of 10.1kWh/tonne to a high of 18.4kWh/tonnes. Due to this variation, the milling circuit is designed to process 40,00tpd of Tepal North ore and 35,000tpd of Tepal South and Tizate ore. The oxide ore is soft from all three areas resulting in a design capacity of 56,000tpd through the same milling circuit.

The saleable products for this PFS are a copper concentrate with gold and silver values obtained from a sulphide flotation, and a gold/silver doré bar from an on-site refinery.

The pyrite contains approximately another 30% of the sulphide's gold which is to be processed for this PFS using a pyrite float followed by a carbon-in-leach (CIL) circuit, carbon plant and refinery. The surface oxides contain copper, gold and silver values; however, only the gold and silver is designed to be recovered for this PFS in a CIL circuit, carbon plant and refinery.

The metallurgical results from the 2009 NZ/SZ float and leach tests used in the 2011 preliminary assessment were added to the Tizate locked cycle and leach tests performed in 2012 and are summarized in Table 1-1, Table 1-2 and Table 1-3. These results were used as the design criteria for the PFS.

The copper concentrate is unusually clean owing to the quartz matrix containing the chalcopyrite. No fatal flaws or deleterious elements were found in the metallurgical tests reviewed. There is good separation between chalcopyrite and pyrite due to the faster chalcopyrite flotation kinetics. Fortunately there is little contamination of pyrite in the copper concentrate, which should make the concentrate easy to market.

Table 1-1: Metallurgical Design Criteria Summary

Product	Unit	Flotation
Resource Grade		
Tepal Grade		
Copper	%	0.22
Gold	g/t	0.37
Silver	g/t	1.02
Tizate Grade		
Copper	%	0.17
Gold	g/t	0.19
Silver	g/t	2.23
Recovery		
Tepal Recovery		
Copper	%	88.2
Gold	%	62.4
Silver	%	27.4
Tizate Recovery		
Copper	%	85.9
Gold	%	58.0
Silver	%	59.6
Concentrate Grade		
Concentrate Grade - Tepal		
Copper	%	25.7
Gold	g/t	32.8
Silver	g/t	42.9
Concentrate Grade - Tizate		
Copper	%	26.9
Gold	g/t	15.0
Silver	g/t	267.6

Table 1-2: Pyrite Flotation and Leach Predictions

Product	Unit	Recovery
Pyrite Conc. Leach		
Tepal		
Copper	%	1.0
Gold	%	10.7
Silver	%	6.1
Tizate		
Copper	%	4.0
Gold	%	15.5
Silver	%	7.8
Cu Cleaner Tails Leach		
Tepal		
Copper	%	0.8
Gold	%	6.5
Silver	%	7.5
Tizate		
Copper	%	0.5
Gold	%	5.0
Silver	%	4.3

Sulphide cyanide and lime consumption is expected to average 2.5kg/t and 1.4kg/t for Tepal and 2.8kg/t and 1.5kg/t for Tizate.

Table 1-3: Oxide Leach Predictions

Product	Unit	Recovery
Tepal		
Gold	%	83.2
Silver	%	63.3
Tizate		
Gold	%	75.2
Silver	%	55.9

Oxide cyanide and lime consumption is expected to average 1.4kg/t and 2.4kg/t for Tepal and 0.4kg/t and 3.6kg/t for Tizate.

1.6 Mineral Resource Estimate

A new mineral resource estimate was calculated on March 29, 2012, using the Ordinary Kriging method. The three deposits were defined by mineralogical models which where based on grade and geological boundaries. The interpolation was further constrained by potentially economic pit shells. The following table documents the Measured and Indicated Mineral Resources of the three deposits at US $5/t equivalent value NSR cut-off.

Table 1-4: Measured and Indicated Mineral Resources at US $5/t Equivalent Value Cut-Off

Deposit	**Resource**	**Tonnage**	**In Situ Average Grade**				**Contained Metal**	
	Category	**(kt)**	**Au (g/t)**	**Cu (%)**	**Ag (g/t)**	**Mo (%)**	**Au (koz)**	**Cu (Mlb)**
Tepal North	Measured	14,067	0.50	0.29	0.78	0.002	228	89
	Indicated	55,320	0.30	0.21	1.01	0.002	533	252
	M + I	69,387	0.34	0.22	0.96	0.002	761	341
Tepal South	Measured	20,011	0.47	0.22	1.07	0.002	300	96
	Indicated	20,993	0.45	0.2	1.17	0.002	305	91
	M + I	41,005	0.46	0.21	1.12	0.002	605	187
Tizate	Measured	-	-	-	-	-	-	-
	Indicated	77,375	0.18	0.17	2.29	0.006	438	285
	M + I	77,375	0.18	0.17	2.29	0.006	438	285
Total	Measured	34,078	0.48	0.25	0.95	0.002	528	185
	Indicated	153,688	0.26	0.19	1.67	0.004	1,276	628
	M + I	187,766	0.30	0.20	1.54	0.004	1,804	813

*Assumptions used to calculate soft pit constraint: Au Price US$ 1300/oz, Cu Price US$ 3.30/lb

Tizate Oxide Au Recovery - 68.8%, Cu Recovery - 6.8%

Tizate Sulphide Au Recovery - 66.2%, Cu Recovery - 85.3%

Tepal Oxide Au Recovery - 78.4%, Cu Recovery - 14.3%

Tepal Sulphide Au Recovery - 60.7%, Cu Recovery - 87.4%

*Mineral resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource would be converted into Mineral Reserves.

The following table documents the Inferred Mineral Resources of the three deposits at the same US $5/t equivalent value NSR cut-off.

Table 1-5: Inferred Mineral Resources at US $5/t Equivalent Value Cut-Off

Deposit	Resource	Tonnage	In Situ Average Grade				Contained Metal	
	Category	(kt)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Au (koz)	Cu (Mlb)
Tepal North	Inferred	906	0.22	0.21	1.21	0.003	6.5	4.2
Tepal South	Inferred	412	0.40	0.16	0.95	0.002	5.3	1.5
Tizate	Inferred	34,426	0.15	0.15	1.70	0.007	169.8	114.8
Total	Inferred	35,743	0.16	0.15	1.68	0.006	181.7	120.4

*Assumptions used to calculate soft pit constraint: Au Price US$ 1300/oz, Cu Price US$ 3.30/lb

Tizate Oxide Au Recovery - 68.8%, Cu Recovery - 6.8%

Tizate Sulphide Au Recovery - 66.2%, Cu Recovery - 85.3%

Tepal Oxide Au Recovery - 78.4%, Cu Recovery - 14.3%

Tepal Sulphide Au Recovery - 60.7%, Cu Recovery - 87.4%

1.7 Mineral Reserve Estimate

The estimate of Mineral Reserve as of March 19, 2013 is reported in Table 1-6. Mineral Reserves are a subset of the Mineral Resource. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Mineral Reserves identified in Table 1-6 comply with CIM definitions and standards for a National Instrument NI43-101 technical report. Detailed information on mining, processing, metallurgical, and other relevant factors are contained within this report and demonstrate, at the time of this report, that economic extraction is justified.

This study did not identify any mining, metallurgical, infrastructure or other relevant factors that may materially affect the estimate of the Mineral Reserves or potential production.

Table 1-6: Mineral Reserves

Proven and Probable Reserves									
		Diluted Grade			Contained Metal			Equivalent Metal	
Oxide Ore	Tonnes (Mt)	Au g/t	Ag g/t	Cu%	Au koz.	Ag koz.	Cu Mlbs.	AuEq koz.[1]	CuEq Mlbs.[1]
Proven	3.8	0.56	0.91	0.28	68	111	23.7	129	52.2
Probable	8.0	0.36	1.41	0.18	93	363	32.3	179	72.4
Proven and Probable	11.8	0.42	1.25	0.22	161	474	56.0	308	124.6
Sulphide Ore	Tonnes (Mt)	Au g/t	Ag g/t	Cu%	Au koz.	Ag koz.	Cu Mlbs.	AuEq koz.[1]	CuEq Mlbs.[1]
Proven	28.3	0.48	0.97	0.24	439	885	151.3	830	335.3
Probable	109.5	0.25	1.63	0.19	894	5,741	447.3	2,108	851.9
Proven and Probable	137.8	0.30	1.50	0.20	1,333	6.625	598.6	2,938	1,187.2
Oxide+Sulphide Ore	Tonnes (Mt)	Au g/t	Ag g/t	Cu%	Au koz.	Ag koz.	Cu Mlbs.	AuEq koz.[1]	CuEq Mlbs.[1]
Proven and Probable	149.6	0.31	1.48	0.20	1,494	7,099	654.6	3,247	1,311.8

Notes:

1) Uses Uses PFS Base Case Four-Year Trailing Average Metal Prices: Au US$1389.95/oz, Cu US$3.44/lb and Ag US$26.03/oz.

AuEq = Au oz + (Ag oz * $26.03/$1389.95) + (Cu lbs * $3.44/$1389.95); CuEq = Cu lbs + (Au oz * $1389.95/$3.44) + (Ag oz * $26.03/$3.44)

Au = gold, Cu = copper, Ag = silver, g/t = grams per tonne, % = percent, koz. = thousand ounces, Mlbs. = millions of pounds.

The Reserves stated in the table above conform to CIM guidelines. Resources are not to be confused as reserves.

Reserve numbers are rounded to the nearest 100,000 tonnes, 1,000 oz Au, 1,000 oz Ag, 100,000 lbs Cu, 1,000 oz. AuEq and 100,000 lbs CuEq.

1.8 Mining Methods

Mining of the Tepal deposit is planned as a conventional open pit operation.

Three pits are proposed: North, South and Tizate. Pit slope angles are based on geotechnical studies completed by Knight Piésold Ltd. in 2012. Pit designs are double-benched. Haul roads and in-pit ramps are designed at 10% gradient and 30m width. 30m is sufficient for two-lane CAT 789D traffic. The ramp is narrowed to 23m (single-lane CAT 789D traffic) within 60m of the pit bottom to reduce waste stripping.

The pits are planned to be mined in sequence, targeting the highest value ore early in the mine life to reduce the capital payback period and improve overall project economics.

Two years were allocated for construction of both the mill and site infrastructure. Mining during that period would be focused on supplying non-acid generating waste to construct the starter tailings dam and preparing the pit for full-scale operation. Oxide milling is scheduled to commence in the latter half of the second construction year. Commissioning of the sulphide circuit at design capacity is planned to be completed at the end of the second construction year. Production would begin immediately afterwards, and continue for 11 years.

A total of 11.8Mt of oxide ore, 137.8Mt of sulphide ore and 267.6Mt of waste would be mined at an average daily rate of 88,000tpd. Life of mine stripping ratio is 1.8:1 waste to ore. Tepal has two waste types: 97.7Mt of non-acid generating (NAG) waste and 169.9Mt of potentially acid generating (PAG) waste. 68.7Mt of NAG is planned to be used for the construction of the tailings storage facility. The remaining NAG and PAG would be stored in engineered dumps located adjacent to the open pits.

The mine plan uses a fleet of diesel equipment supplied by Caterpillar (Tracsa), Mexico. The fleet includes: 6050 hydraulic shovels, a 994H and 992K wheel loaders, 789D trucks, and MD6540 rotary drills as the primary mining equipment. The primary mining equipment would be supported by a fleet of track dozers, motor graders, a rubber tire dozer and water truck. A contract waste stripping fleet would be used to supplement the mine fleet in Years 6 through 10.

1.9 Recovery Methods

Two types of ore are planned to be processed: oxide ore from the top layer and sulphide ore from deposit under the oxide layer. The overall process flowsheets can be found in Figures 1-2 and 1-3. Oxide ore would be processed for 4 days of a 32-day cycle at a rate of 56,000tpd and the sulphide ore would be processed for 28 days at a rate of 40,000tpd for Tepal North and 35,000tpd for Tepal South and Tizate in a common crushing and grinding circuit.

The oxide ore is planned to be transported by haul truck and processed using a conventional gyratory crusher, SAG & ball mill grinding circuit at 56,000tpd followed by settled storage in a pond. A dredge would be used to recover this material at 6850tpd for all 32 days, thickened to 50% solids, and pumped to an oxide CIL circuit.

Processing of the sulphide material is planned through the same grinding circuit as the oxide at a rate of 40,000tpd for Tepal North and 35,000tpd for Tepal South and Tizate. Ground material would be fed to a copper flotation circuit. The copper rougher/scavenger concentrates would be reground and cleaned to a final commercial concentrate grade, and then dewatered to 8% moisture. This concentrate is planned to be trucked off site to a smelter. A pyrite flotation concentrate made from copper rougher flotation tailings would be combined with the first copper cleaner tailings and fed to a sulphide CIL circuit. Carbon from both CIL circuits is planned to be sent to a common 5-tonne carbon plant for washing, stripping and regeneration. Stripped solution from the loaded carbon would be processed using electrowinning and smelting to produce doré bars.

The pyrite flotation tailings are proposed to be pumped to the Tailings Storage Facility (TSF) for disposal. A reclaim barge would transfer water from the TSF.

Figure 1-2: Oxide Overall Process Flowsheet

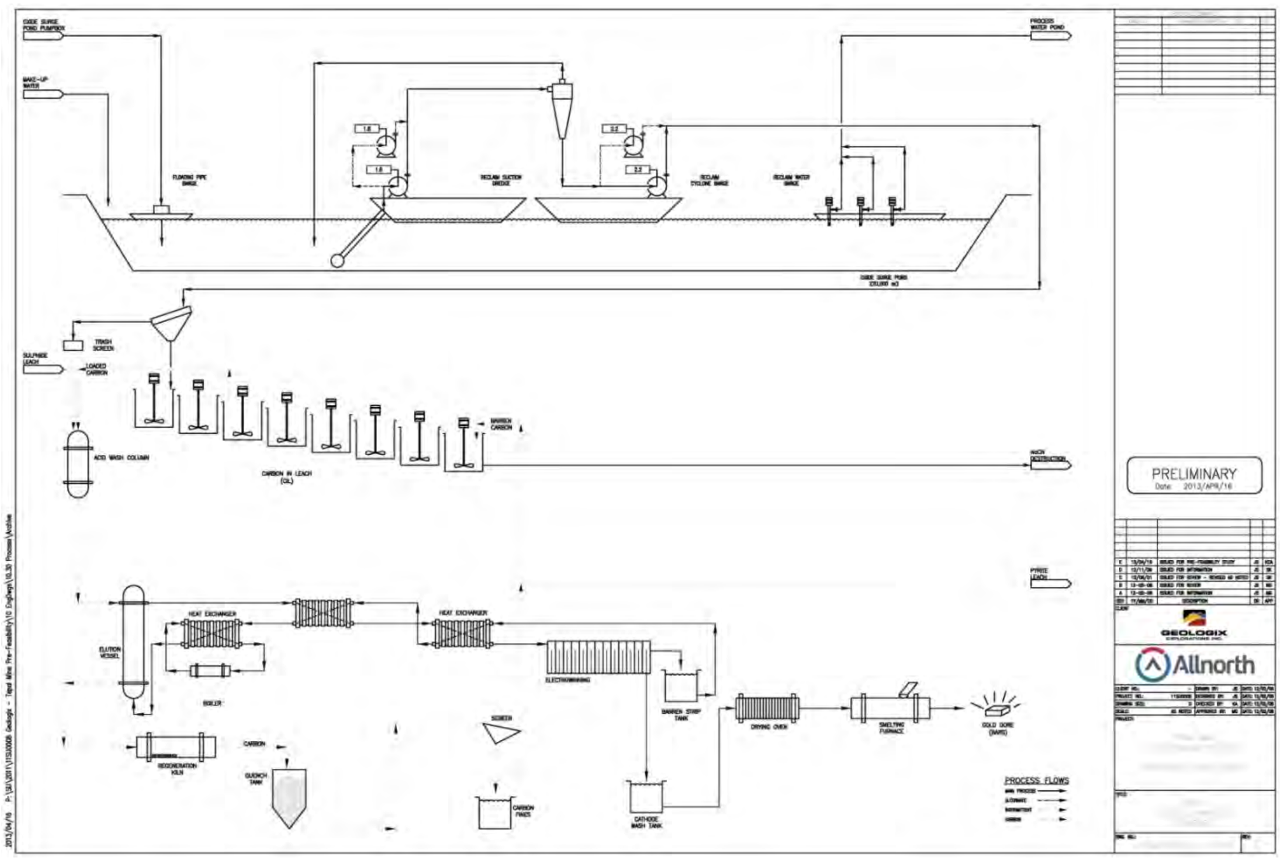
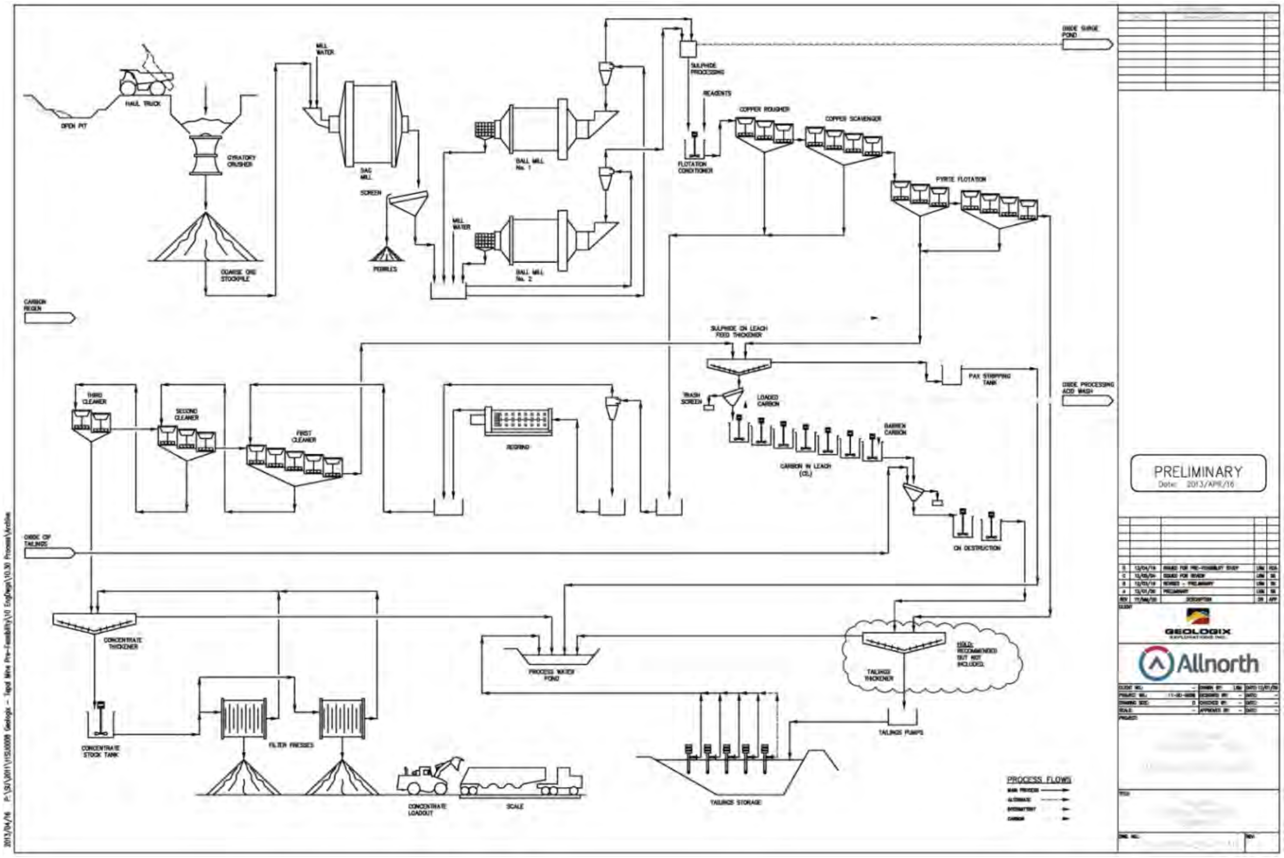


Figure 1-3: Sulphide Overall Process Flowsheet



1.10 Project Infrastructure

The project site is currently vacant or agricultural land, and has very little infrastructure in place. The services and ancillary facilities that would be required for the project include the following:

- Plant site access road
- Haul roads
- Waste rock dumps
- TSF
- Truck shop
- Service roads
- Power supply from the Comisión Federal de Elecricado grid, transmission to site, and project site distribution
- Oxide surge pond
- Process plant
- Assay laboratory

- CIL, carbon plant and refinery facilities
- Fuel storage and dispensing
- Security, scale house, administration and first aid facilities
- Fresh water supply, fire/fresh water storage and distribution, sewage collection and treatment, drainage and runoff settling ponds, and process water pond
- 350 person Temporary housing facilities for construction personnel
- 120 person Permanent accommodation complex
- Laydown areas and parking
- 750m long airstrip.

The majority of these facilities are planned to be constructed in the two year construction period prior to mining or mineral processing taking place. Commissioning would occur in the last 6 months of the 24 month construction period.

1.11 Environment and Permitting

Environmental baseline studies have been carried out for Geologix by Clifton Associates Ltd. out of Guadalajara, Jalisco, México. There are a number of protected species in the area; however the project is not in a protected area and a flora and fauna rescue and protection management plan is a normal requirement during Mexican permitting to manage protected species for mining projects.

Waste characterization studies were carried out by pHase Geochemistry Inc., Vancouver, British Columbia. Primary sulphide mineralization consists of chalcopyrite and pyrite with minor pyrrhotite, bornite, sphalerite, molybdenite and galena. The waste rock static test program on drill core was represented by 300 samples with 100 samples collected from each of the three deposits. With respect to rock type, a large proportion of tonalite (73% of samples tested) at Tepal North classified as potentially acid generating (PAG) compared to Tepal South (58% of samples) and Tizate (48% of samples). For all three deposits, >75% of late dyke and overburden samples typically classify as non-potentially acid generating (NAG). The altered volcanic samples at Tepal North consistently classified as PAG, whereas the unaltered volcanics at Tepal South predominantly classified as NAG. In relation to the in-situ oxidation state, the majority of oxide samples at Tepal South and Tizate classified as NAG. Further planning would be required for appropriate waste rock facility design and closure to manage PAG and metal leaching material, as well as a long-term monitoring protocol.

Water management requirements for the site would include groundwater wells to augment the water from other sources (i.e. pit seepage, tailings pond reclaim, waste rock retention) for use in the processing plant. There would be no discharge of process water to the environment during operation. All potentially acid generating waste dumps would be capped and revegetated at closure. Seepage during closure is planned to be collected, analysed, recycled or treated to ensure it meets standards for release to the environment.

Development of a number of social and environmental management plans would be important for this project including waste, water, air (dust), hazardous materials, public consultation and security plans.

There are a number of permits identified that would be required for the project under the General Law of Ecological Equilibrium and Environmental Protection. An Environmental Impact Manifest (MIA-P) and a Change of Land Use Authorization are the two key items that would be required to advance the project. Once the government would approve the MIA-P and Change of Land Use permit, additional detailed permits would be required for construction and operations.

Although the project is located adjacent to several small communities and the larger community of Tepalcatepec, the skilled workforce is limited. A technical institute, sponsored by the company, would assist with local capacity building for various positions in the mine. The majority of workers would likely come from other areas in Mexico and the plan is to house them at the camp. There are a number of different unions in Mexico that would influence construction and operations and would need to be considered in plans in the feasibility phase of the project.

1.12 Capital and Operating Costs

1.12.1 Capital Costs Estimates

The capital cost estimate was prepared using first principles, applying direct project experience and avoiding the use of general industry factors. The estimate is derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in the study. Costs are expressed in US dollars with no escalation. The target accuracy of the estimate is ±25%.

Total life of mine capital costs are estimated to be $397M. Pre-production capital costs amount to $354M. Capital costs during production years total $44M.These costs are summarized in Table 1-7. The capital costs do not include mining fleet as it is accounted for in operating costs through leasing. Contingency for the project totals $39M. Individual contingency rates were applied to each of the capital cost categories, with most rates being 15-20%. Some of the capital costs did not have any contingency applied as direct quotes were obtained from suppliers. This resulted in a blended contingency rate of 8.7%. Figures 1-4 and 1-5 show the breakdown of capital by pre-production and production period.

Table 1-7: Capital Cost Estimate Summary ($M)

Category	Pre-Production	Production	Total Capital Costs	% of Total
Capitalized Pre-Stripping	21.5	0.0	21.5	5.4
Support Equipment	3.3	1.1	4.4	1.1
Tailings	34.7	42.0	76.7	19.3
Process Plant	229.6	0.0	229.6	57.8
Indirects	20.1	0.0	20.1	5.0
Owner's Costs	13.4	0.0	13.4	3.4
Salvage Value	0.0	-34.4	-34.4	-8.6
Closure	0.0	27.3	27.3	6.9
Contingency (8.7%)	31.3	7.6	38.9	9.8
Total Capital Costs	**353.8**	**43.6**	**397.4**	**100.0**

Figure 1-4: Breakdown of Pre-Production Capital Costs

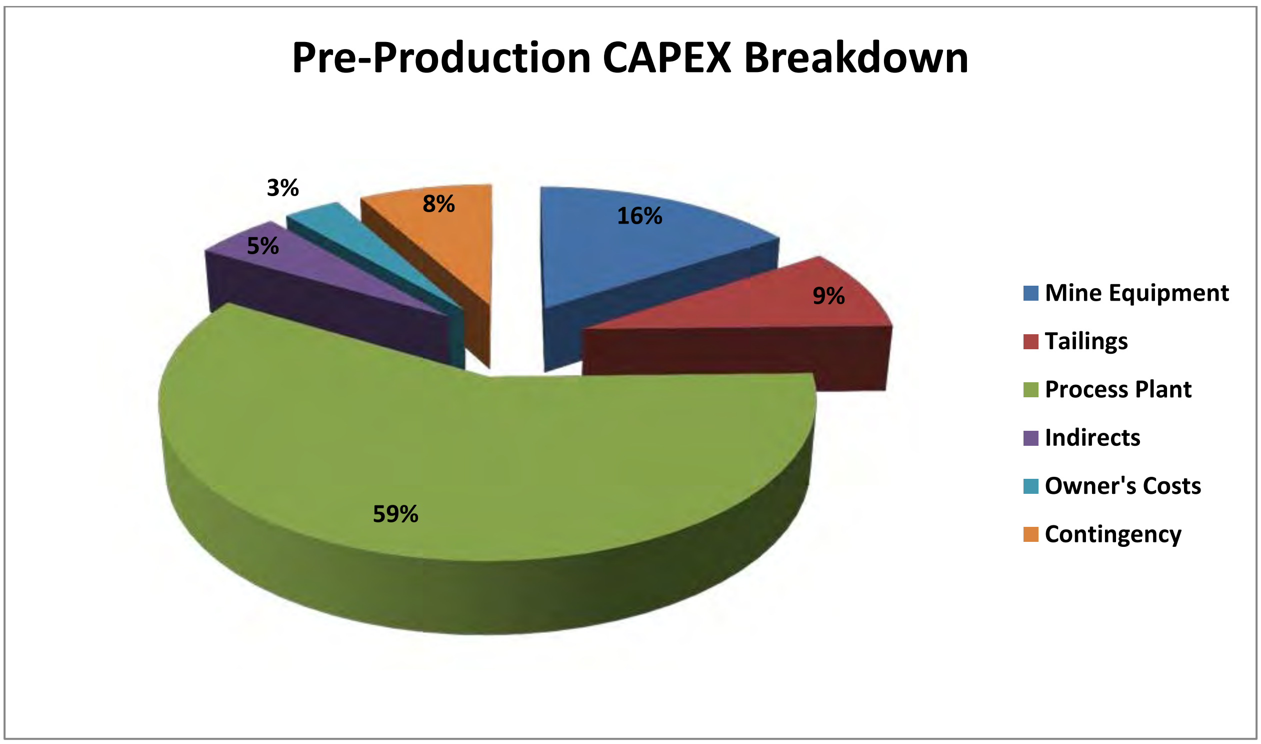
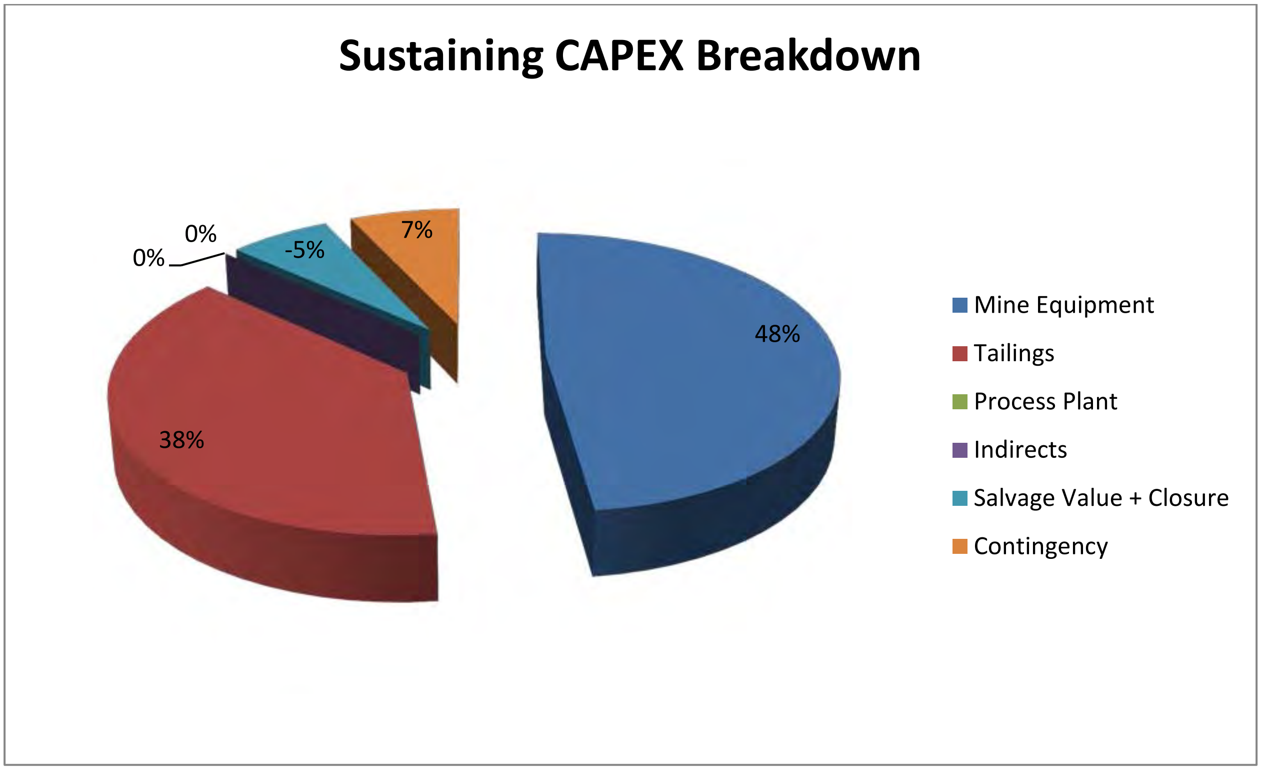


Figure 1-5: Breakdown of Sustaining Capita (including Closure and Salvage Value)



1.12.2 Reclamation/Closure & Salvage Cost Estimate

Closure cost for the project is estimated to be $27M. Of this cost, $25M accounted for the closure and reclamation of the TSF and the waste rock dumps. An additional $2M was allocated for the closure and demolition of mill facility foundations. Closure costs are set to occur in Year 12, one year after the end of production. Salvage value is accounted for in 2027 amounting to $34M. This amounts to 10% of the mine equipment and process plant capital costs.

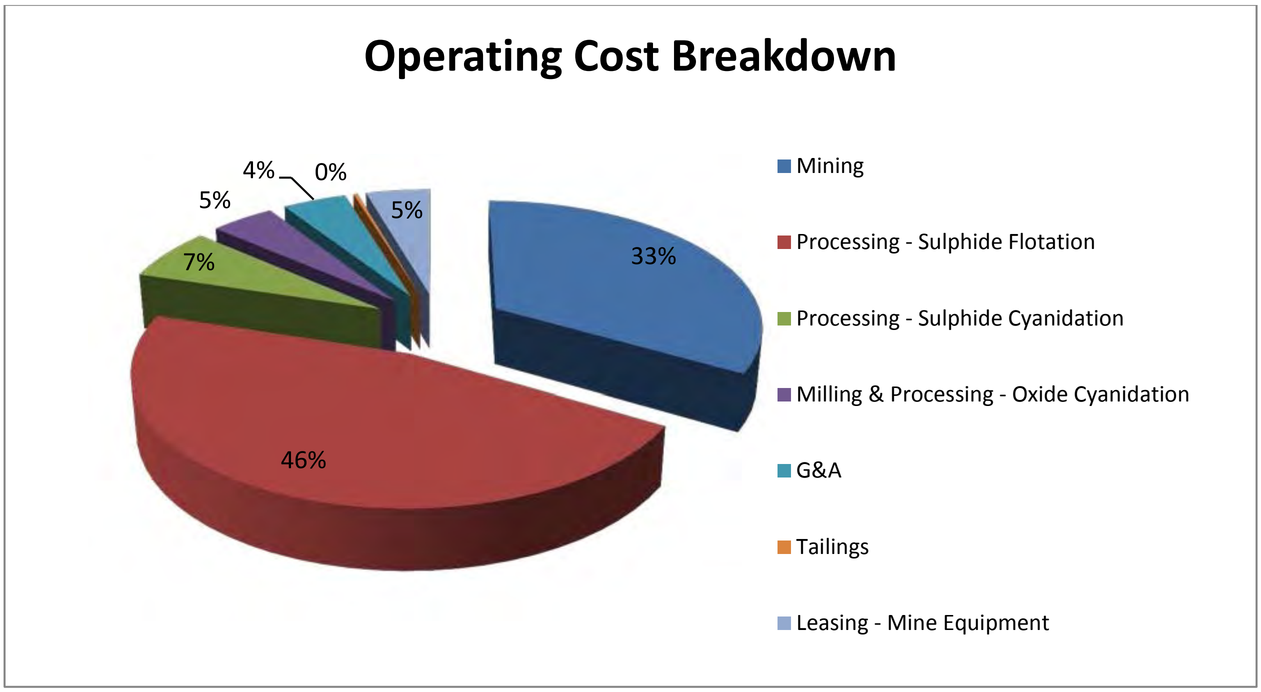
1.12.3 Operating Cost Estimates

Operating cost estimate in this section of the report include mining, processing, tailings, and administration up to the production of concentrate from the site. Mining costs incurred during the construction phase (pre-production Years -2 and -1) are capitalized and form part of the capital cost estimate. Concentrate transportation, treatment and refining charges, and royalties are discussed in Economic Analysis section of this report. Average annual `operating costs over the life of mine are $163M and are summarized in Table 1-8. Figure 1-6 demonstrates the distribution of operating costs.

Table 1-8: Average Annual Operating Cost Estimate

Category	$M	%
Mining	54.9	33.6
Processing - Sulphide Flotation	76.3	46.8
Processing - Sulphide Cyanidation	10.8	6.6
Milling & Processing - Oxide Cyanidation	6.0	3.7
G&A	7.3	4.5
Tailings	0.5	0.3
Mine Equipment Leasing	7.4	4.5
Total Average Annual Operating Costs	**163.2**	**100**

Figure 1-6: Breakdown of Operating Costs



1.13 Economic Analysis

All operating scenarios were modeled to estimate the value that each could potentially realize. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to be more indicative of the true investment value. Sensitivity analyses were performed for variation in metal price, head grades, operating costs, and capital costs to determine their relative importance as project value drivers. The economic analysis presented includes the leasing of the mine equipment fleet. A discount rate of 7% was used for net present value (NPV) calculations.

This technical report contains forward-looking information regarding projected mine production rates and forecast of resulting cash flows as part of this study. The grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment of skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

1.13.1 Metal Price Scenarios

Table 1-9 outlines the metal prices scenarios that were used in the economic analysis.

Table 1-9: Metal Prices and Exchange Rates by Scenario

Parameter	Units	Three-Year Trailing Average as of February 28, 2013	PFS Base Case Four-Year Trailing Average as of February 28, 2013	Five-Year Trailing Average as of February 28, 2013	Whittle Parameter Pricing
Copper Price	USD $/lb	3.71	3.44	3.32	3.15
Gold Price	USD $/oz	1,518	1,390	1,286	1,400
Silver Price	USD $/oz	29.58	26.03	23.68	26.00
Exchange Rate	MEX:USD	13:1	13:1	13:1	13:1
Exchange Rate	CDN:USD	1.00	1.00	1.00	1.00

The reserve estimates used in the economic analysis are outlined in the Section 1.7 of the Executive Summary and were held constant for all scenarios.

1.13.2 Copper, Gold, Silver Production

Recovered metals (for all four scenarios evaluated) are shown in Table 1-10. The amount of concentrate produced during the mine life is estimated at 908kdmt from 2016 to 2026. Figures 1-7 and 1-8 demonstrate the life of mine NSR breakdown as well as the payable metal by year.

Table 1-10: LOM Payable Metal

Category	Unit	Value
Payable Cu	LOM M lbs	503.1
Payable Au	LOM k oz	1,164.3
Payable Ag	LOM k oz	2,952.1

Figure 1-7: Life of Mine NSR Breakdown by Metal

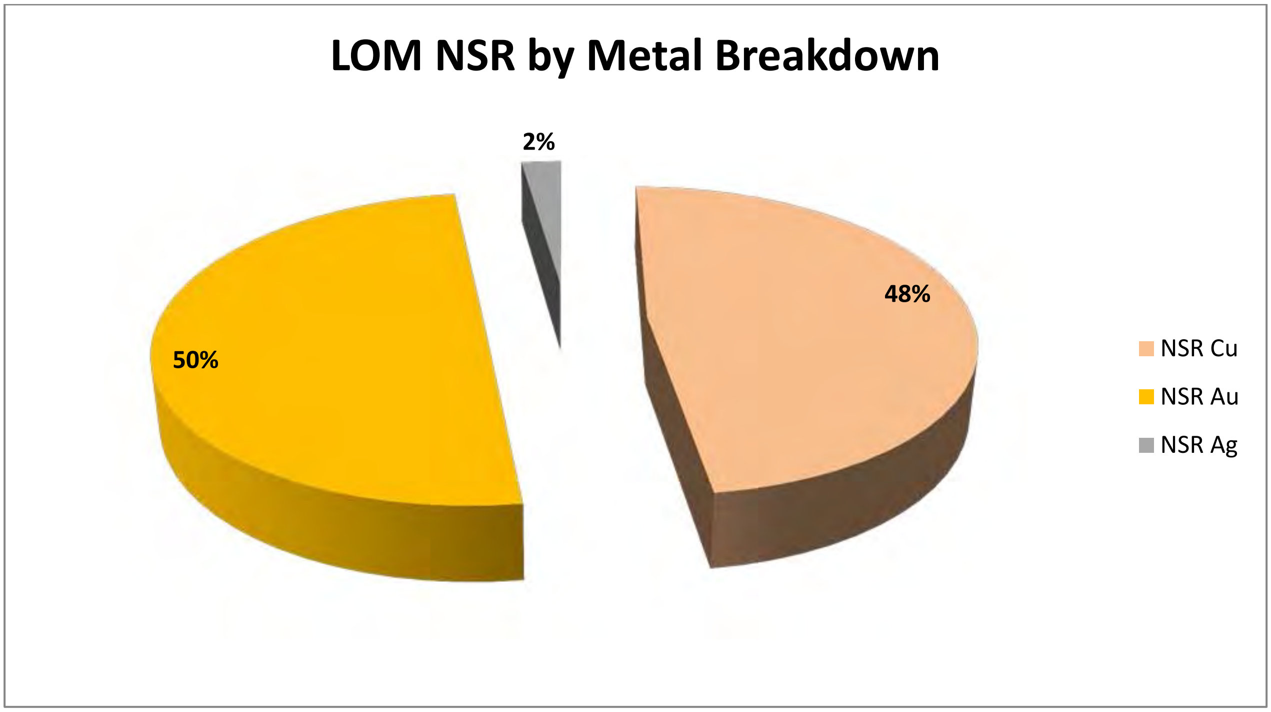
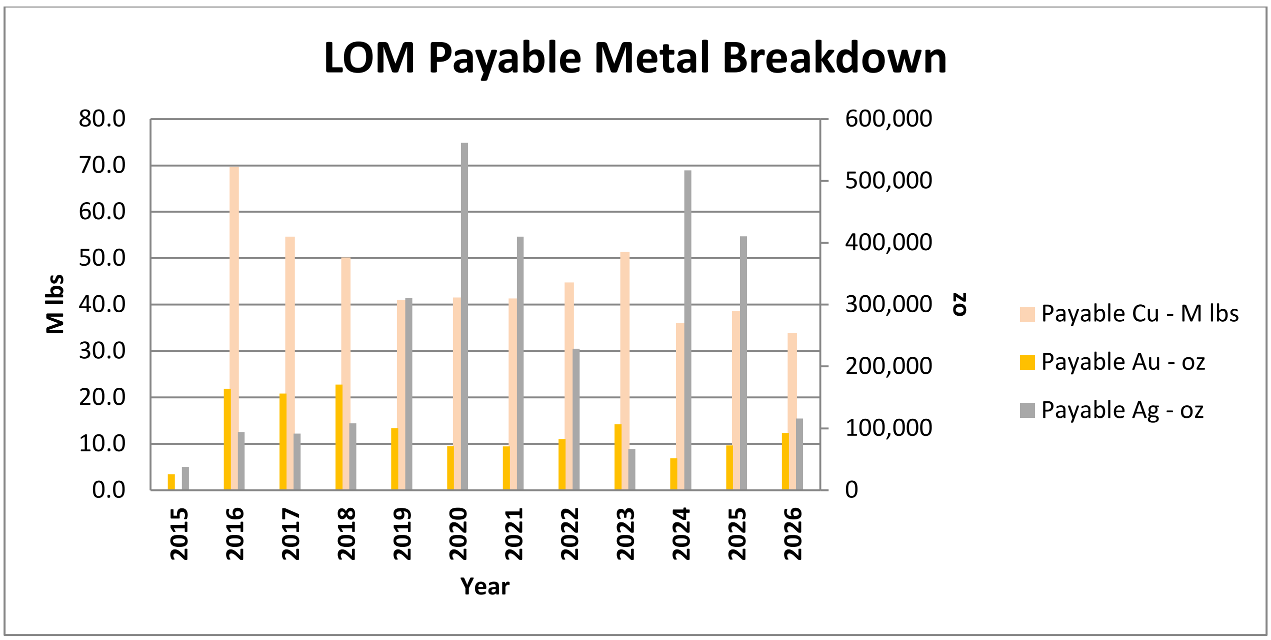


Figure 1-8: Payable Metal by Year



1.13.3 Taxes

The project has been evaluated on an after-tax basis in order to reflect a more indicative value of the project. Geologix commissioned PricewaterhouseCoopers LLP (PwC) in Vancouver, BC to prepare a tax model for the post tax economic evaluation of the project with the inclusion of applicable Mexican income taxes. These tax calculations have been used in the economic analysis presented in this report. The tax calculation uses an inflation factor of 3.5% per year, a 5% employee profit sharing, and a 28% Mexican corporate tax rate. Total taxes for the life of the project amount to $234M.

1.13.4 Financial Performance

Pre-tax and after-tax financial performance for each of the four scenarios is summarized in Tables 1-11 through 1-14.

Table 1-11: Summary of Base Case Economic Results (Four-Year Trailing Average Metal Prices)

Category	Unit	Value
Mine Life	Years	11.5
Average Plant Throughput	M tpa	13.0
Payable Cu LOM	LOM M lbs	503.1
Average Payable Cu (Year 1-7)	M lbs/yr	49.0
Payable Au LOM	LOM k oz	1,164
Average Payable Au (Year 1-7)	k oz/yr	116.6
Payable Ag LOM	LOM k oz	2,952
Average Payable Ag (Year 1-7)	k oz/yr	257.8
Cash Cost (Net of By-Product Credits)	$/Payable Cu lb	0.62
	$/Payable Au oz	170
Cash Cost (Net of By-Product Credits incl. of Sustaining Capital)	$/Payable Cu lb	0.81
	$/Payable Au oz	251
Cash Cost (Net of By-Product Credits incl. Total Capital)	$/Payable Cu lb	1.58
	$/Payable Au oz	587
Unit OPEX (Offsite Costs + Operating Costs)	$/tonne ore	13.40
Avg. Annual Cashflow during production	$ M	86.8
Pre-Production Capital with Leased Equipment	$ M	353.8
Sustaining & Closure Capital	$ M	43.6
Total Capital + Contingency	$ M	397.4
Pre-Tax $NPV_{7\%}$	$ M	495.1
Pre-Tax IRR	%	35.9
Pre-Tax Payback Period	Years	2.7
After-Tax $NPV_{7\%}$	$ M	344.8
After-Tax IRR	%	27.7
After-Tax Payback Period	Years	3.2

Table 1-12: Summary of Results using Three-Year Trailing Average Metal Prices

Category	Unit	Value
Mine Life	Years	11.5
Average Plant Throughput	M tpa	13.0
Payable Cu LOM	LOM M lbs	503.1
Average Payable Cu (Year 1-7)	M lbs/yr	49.0
Payable Au LOM	LOM k oz	1,164
Average Payable Au (Year 1-7)	k oz/yr	116.6
Payable Ag LOM	LOM k oz	2,952
Average Payable Ag (Year 1-7)	k oz/yr	257.8
Cash Cost (Net of By-Product Credits)	$/Payable Cu lb	0.31
	$/Payable Au oz	50
Cash Cost (Net of By-Product Credits incl. of Sustaining Capital)	$/Payable Cu lb	0.50
	$/Payable Au oz	132
Cash Cost (Net of By-Product Credits incl. Total Capital)	$/Payable Cu lb	1.28
	$/Payable Au oz	468
Unit OPEX (Offsite Costs + Operating Costs)	$/tonne ore	13.45
Avg. Annual Cashflow during production	$ M	103.8
Pre-Production Capital with Leased Equipment	$ M	353.8
Sustaining & Closure Capital	$ M	43.6
Total Capital + Contingency	$ M	397.4
Pre-Tax $NPV_{7\%}$	$ M	675.2
Pre-Tax IRR	%	44.2
Pre-Tax Payback Period	Years	2.4
After-Tax $NPV_{7\%}$	$ M	474.5
After-Tax IRR	%	34.1
After-Tax Payback Period	Years	2.9

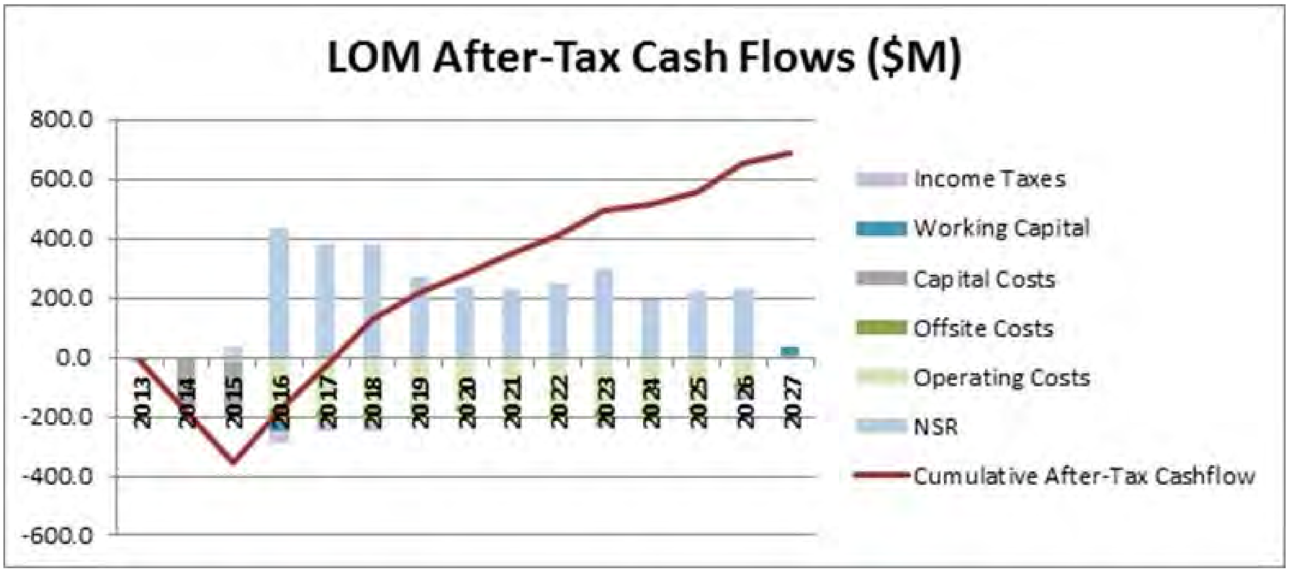
Table 1-13: Summary of Results using Five-Year Trailing Average Metal Prices

Category	Unit	Value
Mine Life	Years	11.5
Average Plant Throughput	M tpa	13.0
Payable Cu LOM	LOM M lbs	503.1
Average Payable Cu (Year 1-7)	M lbs/yr	49.0
Payable Au LOM	LOM k oz	1,164
Average Payable Au (Year 1-7)	k oz/yr	116.6
Payable Ag LOM	LOM k oz	2,952
Average Payable Ag (Year 1-7)	k oz/yr	257.8
Cash Cost (Net of By-Product Credits)	$/Payable Cu lb	0.86
	$/Payable Au oz	224
Cash Cost (Net of By-Product Credits incl. of Sustaining Capital)	$/Payable Cu lb	1.05
	$/Payable Au oz	305
Cash Cost (Net of By-Product Credits incl. Total Capital)	$/Payable Cu lb	1.83
	$/Payable Au oz	641
Unit OPEX (Offsite Costs + Operating Costs)	$/tonne ore	13.37
Avg. Annual Cashflow during production	$ M	76.1
Pre-Production Capital with Leased Equipment	$ M	353.8
Sustaining & Closure Capital	$ M	43.6
Total Capital + Contingency	$ M	397.4
Pre-Tax $NPV_{7\%}$	$ M	379.7
Pre-Tax IRR	%	30.1
Pre-Tax Payback Period	Years	3.0
After-Tax $NPV_{7\%}$	$ M	261.5
After-Tax IRR	%	23.2
After-Tax Payback Period	Years	3.5

Table 1-14: Summary of Results using Whittle Parameter Pricing

Category	Unit	Value
Mine Life	Years	11.5
Average Plant Throughput	M tpa	13.0
Payable Cu LOM	LOM M lbs	503.1
Average Payable Cu (Year 1-7)	M lbs/yr	49.0
Payable Au LOM	LOM k oz	1,164
Average Payable Au (Year 1-7)	k oz/yr	116.6
Payable Ag LOM	LOM k oz	2,952
Average Payable Ag (Year 1-7)	k oz/yr	257.8
Cash Cost (Net of By-Product Credits)	$/Payable Cu lb	0.59
	$/Payable Au oz	292
Cash Cost (Net of By-Product Credits incl. of Sustaining Capital)	$/Payable Cu lb	0.78
	$/Payable Au oz	374
Cash Cost (Net of By-Product Credits incl. Total Capital)	$/Payable Cu lb	1.55
	$/Payable Au oz	709
Unit OPEX (Offsite Costs + Operating Costs)	$/tonne ore	13.38
Avg. Annual Cashflow during production	$ M	79.0
Pre-Production Capital with Leased Equipment	$ M	353.8
Sustaining & Closure Capital	$ M	43.6
Total Capital + Contingency	$ M	397.4
Pre-Tax $NPV_{7\%}$	$ M	414.6
Pre-Tax IRR	%	32.2
Pre-Tax Payback Period	Years	2.9
After-Tax $NPV_{7\%}$	$ M	286.7
After-Tax IRR	%	24.8
After-Tax Payback Period	Years	3.4

Figure 1-9: After-Tax Cash Flows for Base Case



1.13.5 Sensitivity Analyses

Sensitivity Analyses were conducted on pre-tax and after-tax project NPV values for individual parameters including metal prices, head grades, operating costs, and capital costs. The results show that the project is most sensitive to metal price and head grade and least sensitive to changes in capital costs in all four scenarios.

The Base Case was evaluated at different discount rates to determine the effect on the project NPV. Project NPV declined as the discount rate increased. Table 1-15 demonstrates the summary of the discount rate sensitivity results on all three cases evaluated.

Table 1-15: NPV for Various Discount Rates using Four-Year Trailing Average Metal Prices

Discount Rate Sensitivity	Pre-Tax $NPV_{x\%}$ ($M)	After-Tax $NPV_{x\%}$ ($M)
0%	$924.6	$690.1
5%	$590.3	$421.2
7%	**$495.1**	**$344.8**
8%	$453.6	$311.6
10%	$380.6	$253.3

1.14 Conclusions and Recommendations

The financial analysis of the prefeasibility study demonstrates that the project has positive economics and warrants consideration for advancement to feasibility level engineering by Geologix.

Standard industry practices, equipment and processes were used in this study. The Qualified Persons for this report are not aware of any unusual significant risks or uncertainties that could affect the reliability or confidence in the project based on the data and information available to date.

1.14.1 Estimated Cost of Recommended Work Programs

The estimated cost of the next stage of work is presented in Table 1-16.

Table 1-16: Summary of Estimated Costs of Recommended Work Programs

Item	Cost in US$
Mining Methods	**515,000**
Geotechnical and Hydrogeology Study	300,000
Geotechnical Evaluation for High wall Stability	100,000
Blast Pattern Design	15,000
Tailings Dam Design	100,000
Processing and Metallurgy	**480,000**
Testing for ADR and/or Merrill Crowe	5,000
Sulphide Process Testing	100,000
Pilot Plant	250,000
Oxide Mineralization Testing	50,000
Miscellaneous	75,000
Environment and Social	**2,125,000**
Additional Testwork	100,000
Security Risk Assessment	25,000
Environmental Studies, consultation, land acquisition	2,000,000
Feasibility Study	**1,000,000**
TOTAL	**4,120,000**

2 Introduction and Terms of Reference

2.1 Basis of Technical Report

This Technical Report was compiled by JDS Energy & Mining Inc. (JDS) for Geologix Explorations Inc. (Geologix).

This document has been prepared to provide a technical evaluation in compliance with the disclosure requirements of National Instrument 43-101 (NI 43-101). This study is based on the NI 43-101 report prepared by Micon International Limited (Micon) in March 2012, which provided a mineral resource update for the Tepal project.

Several sections of this report are taken from the technical report written by Micon, titled *"Technical Report on the Mineral Resources of the Tepal Gold-Copper Project, Michoacán State, Mexico"*, dated March 29, 2012 (2012 Resource Report).

2.2 Scope of Work

This report is the work carried out by several consulting companies, all of which are independent of Geologix. The scope of work for each company is listed below.

JDS Energy & Mining Inc.'s (JDS) scope of work included:

- Compile a technical report that includes the data and information provided by other consulting companies.
- Conduct optimal pit designs and production schedule.
- Select mining equipment.
- Estimate capital and operating costs for mining.
- Summarize capital and operating costs.
- Prepare a financial model and conduct an economic evaluation including sensitivity and project risk analysis.
- Make recommendations to improve value, reduce risks and move the project toward a feasibility-level of confidence.

Allnorth Consultants Limited (Allnorth) scope of work included:

- Develop a conceptual flowsheet with material balance, specifications and the selection of main process equipment.
- Design required plant infrastructure.
- Estimate power requirements.
- Identify proper sites, plant facilities and other ancillary facilities.
- Estimate all initial and sustaining capital expenditures requirements and operating costs for processing.

Knight Piésold Ltd. (KP) scope of work included:

- Review available geological, structural and geotechnical logging data and establish pit slope angles.
- Identify proper sites for waste storage, tailings disposal and water management facilities
- Estimate all initial and sustaining capital expenditures requirements and operating costs for waste storage, tailings disposal and water storage.
- Estimate water balance.
- Estimate closure costs.

Micon International Limited (Micon) scope of work included:

- Establish resources included in the mining plan using indicated and inferred resources.
- Assess acid rock drainage (ARD) potential.
- Review environmental and other permit requirements.
- Summarize results of an environmental baseline study conducted by Clifton Associates Ltd. out of Guadalajara, Jalisco, Mexico.
- Implement and supervise a metallurgical testing program.
- Establish recovery values based on metallurgical testing results.

2.3 Qualifications & Responsibilities

Table 2-1 list the qualifications of each author, as well as the section(s) of the report for which they are responsible.

Table 2-1: Tepal Project PEA Author Responsibility

Author	Company	Report Section(s) of Responsibility
Mr. Matt R. Bender, P.E	JDS	1,2,3,4,13.4,13.5,19,21,22,24,25,26,27
Mr. Michael E.Makarenko, P.Eng.	JDS	15,16 (exclusive of 16.2 and 16.3)
Mr. Mark Dobbs, P.Eng.	Allnorth	17,18 (exclusive of 18.3 and 18.4)
Mr. Bruno Borntraeger, P.Eng.	KP	16.2,16.3,18.3,18.4
Mr. David K. Makepeace, P.Eng.	Micon	5,6,7,8,9,10,11,12,14,20,23,25.1,25.7,26.3 shared responsibility for 25.11 and 25.12
Mr. Michael Godard, P.Eng.	Micon	13 (exclusive of 13.4 and 13.5)

2.4 Site Visits

Mr. Mike Makarenko of JDS visited the Tepal project site September 4-6, 2012.

Mr. David Makepeace of Micon visited the Tepal site from January 8 to 12, 2012.

Mr. Mike Godard of Micon visited the Tepal site from January 8 to 13, 2012.

Mr. Bruno Borntraeger of KP visited the site September 18 to 22, 2011.

Mr. Mark Dobbs of Allnorth visited the site from January 9 to 13, 2012.

2.5 Currency

Unless otherwise specified, all costs in this report are presented in US Dollars (US$).

2.6 Units of Measure & Abbreviations

All units in this report are based on the International System of Units (SI), except industry standard units, such as troy ounces for the mass of precious metals and pounds for the mass of base metals.

A list of main abbreviations and terms used throughout this report is presented in Table 2.2.

Table 2-2: Units of Measure & Abbreviations

Units of Measure

'	Foot
"	Inch
µm	Micron (micrometre)
Amp	Ampere
Ac	Acre
Ag	Silver
Au	Gold
Cfm	Cubic feet per minute
cm	Centimetre
Cu	Copper
dpa	Days per annum
dmt	Dry metric tonne
ft	Foot
ft^3	Cubic foot
g	Gram
hr	Hour
ha	Hectare
hp	Horsepower
In	Inch
kg	Kilogram
km	Kilometre
km^2	Square kilometer
KPa	Kilopascal
kt	Thousand tonnes
Kw	Kilowatt
KWh	Kilowatt hour
L	Litre
lb or lbs	Pound(s)
m	Metre
M	Million

m^2	Square metre
m^3	Cubic metre
mi	Mile
min	Minute
mm	Millimetre
Mpa	Mega Pascal
mph	Miles per hour
Mtpa	Million tonnes per annum
Mt	Million tonnes
MXP	Mexican pesos
ºC	Degree Celsius
oz	Troy ounce
ppb	Parts per billion
ppm	Parts per million
s	Second
t	Metric tonne
tpd	Tonnes per day
tph	Tonnes per hour
US$	US dollars
V	Volt
W	Watt
wmt	Wet metric tonne

Abbreviations & Acronyms

% or pct	Percent
AAS	Atomic absorption spectrometer
ABA	Acid base accounting
ADIS	Automated Digital Imaging System
Allnorth	Allnorth Consultants Limited
Amsl	Above mean sea level
ANFO	Ammonium Nitrate/Fuel Oil
AP	Acid potential
ARD	Acid rock drainage
BC	British Columbia
BIF	Banded iron formation
BLS	Barren leach solution
Btu	British Thermal Unit
BWI	Ball work index
$CaCO_3$	Calcium carbonate
CAPEX	Capital costs
CAT	Caterpillar
CFE	Comision Federal de Electricidad
CIL	Carbon-in-leach
CIM	Canadian Institute of Mining

CLU	Change of land-use authorization
CPM	Critical path method
CRM	Certified reference material
Cu eq	Copper equivalent
CV	Coefficient of variation
DO	Dissolved oxygen
Elev	Elevation above sea level
ESIA	Environmental-Social Impact Assessment
ETF	Exchange traded fund
FA/grav	Fire assay with gravimetric finish
FEL	Front-end loader
FLOT	Flotation
FS	Feasibility Study
Geologix	Geologix Exploration Inc.
GMV	Gross metal value
GPS	Global positioning system
H:V	Horizontal to vertical
HDPE	High Density polyEthylene
HVAC	Heating, ventilation and air conditioning
ICP-MS	Inductively coupled plasma mass spectrometry
ICSA	Ingeniera de Ciudades South America
ID2	Inverse distance square
IMSS	Social security
IRA	Inter-ramp angles
IRR	Internal rate of return
ISN	Payroll tax
ISRMR	In-situ rock mass rating
ISSSTE	Instituto de Securidad Social al Servicio de Trabajadores del Estado
JDS	JDS Energy & Mining Inc.
KP	Knight Piésold Ltd.
LGEEPA	General Law of Equilibrium and Environmental Protection
LGPGIR	General Law for Prevention and Integral Management of Waste
LOM	Life of mine
MARC	Maintenance and repair contract
MIA-P	Environmental impact manifest
MIBC	Methyl isobutyl carbinol
Micon	Micon International Limited
ML/ARD	Metal leaching/acid rock drainage
MSE	Mechanically stabilized earth
MSS	Instituto Mexicanop del Seguro Social
N,S,E,W	North, South, East, West
NI 43-101	National Instrument 43-101
NN	Nearest neighbour
NAG	Non potentially acid generating
NP	Neutralization potential

NPV	Net present value
NSOX	North, South Oxide
NSR	Net Smelter Return
NZ	North Zone
Ø	Diametre
OEM	Original equipment manufacturer
OK	Ordinary Kriging
OPEX	Operating costs
PA	Preliminary Assessment
PAG	Potentially acid generating
PAX	Potassium Amyl Xanthate
PFS	Prefeasibility Study
PLS	Pregnant leach solution
PM	Project management
POX	Pressure oxidation
PPM	Project procedures manual
PwC	PricewaterhouseCoopers LLP
QA/QC	Quality Assurance/Quality Control
QMS	Quality Management System
RC	Reverse circulation
RFS	Rock Storage Facility
ROM	Run-of-the-mill
RQD	Rock quality designation
SEMARNAT	Secretaria de medio ambiente y recursos naturales
S.G.	Specific gravity
SAG	Semi-autogenous grinding
SRK	SRK Consulting Inc.
STP	Sewage treatment plant
SZ	South Zone
TOX	Tizate Oxide
TSF	Tailings storage facility
UPS	Uninterrupted power system
UTM	Universal Transverse Mercator
Vulcan	Maptek Vulcan TM
Whittle	Gemcom Whittle- Strategic Mine Planning TM
X,Y,Z	Cartesian Coordinates, also Easting, Northing and Elevation

3 Reliance on Other Experts

Preparation of this report is based upon public and private information provided by Geologix and information provided in various previous Technical Reports listed in Section 27 of this report.

The authors have carried out due diligence reviews of the information provided to them by Geologix and others for preparation of this report and are satisfied that the information was accurate at the time of the report and that the interpretations and opinions expressed in them were reasonable and based on current understanding of mining and processing techniques and costs, economics, mineralization processes and the host geologic setting. The authors have made reasonable efforts to verify the accuracy of the data relied on in this report.

The results and opinions expressed in this report are conditional upon the aforementioned information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions made herein the authors reserve the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to the authors subsequent to the date of this report.

Neither JDS nor the authors of this technical report are qualified to provide extensive comment on legal issues associated with the Tepal property. As such, portions of Section 4 dealing with the types and numbers of mineral tenures and licenses, the nature and extent of Geologix's title and interest in the Tepal property, the terms of any royalties, back-in rights, payments or other agreements and encumbrances to which the property is subject are descriptive in nature and are provided exclusive of a legal opinion.

JDS has relied on PwC concerning tax matters relevant to this report. The reliance is based on a letter to Geologix titled "Assistance with insert and review of the Mexican income tax portions of the economic analysis prepared by JDS Energy & Mining Inc. in connection with the Prefeasibility Study Report on Geologix Explorations Inc.'s Tepal Project" dated March 14, 2013.

4 Property Description and Location

The information presented in this section has been adapted from the March, 2012 Technical report by Micon and was updated based on information provided by Geologix in April 2013.

4.1 Property Description and Location

The Tepal Property is located in the municipality of Tepalcatepec, Michoacán State in southwestern Mexico. The property is centered at 19° 7' 40" Latitude and 102° 56' 8" Longitude or 2,116,257mN and 717,161mE , Zone 13Q (UTM - NAD 83). The average elevation is 550m. Figure 4-1 illustrates the location and the infrastructure surrounding the Tepal Property.

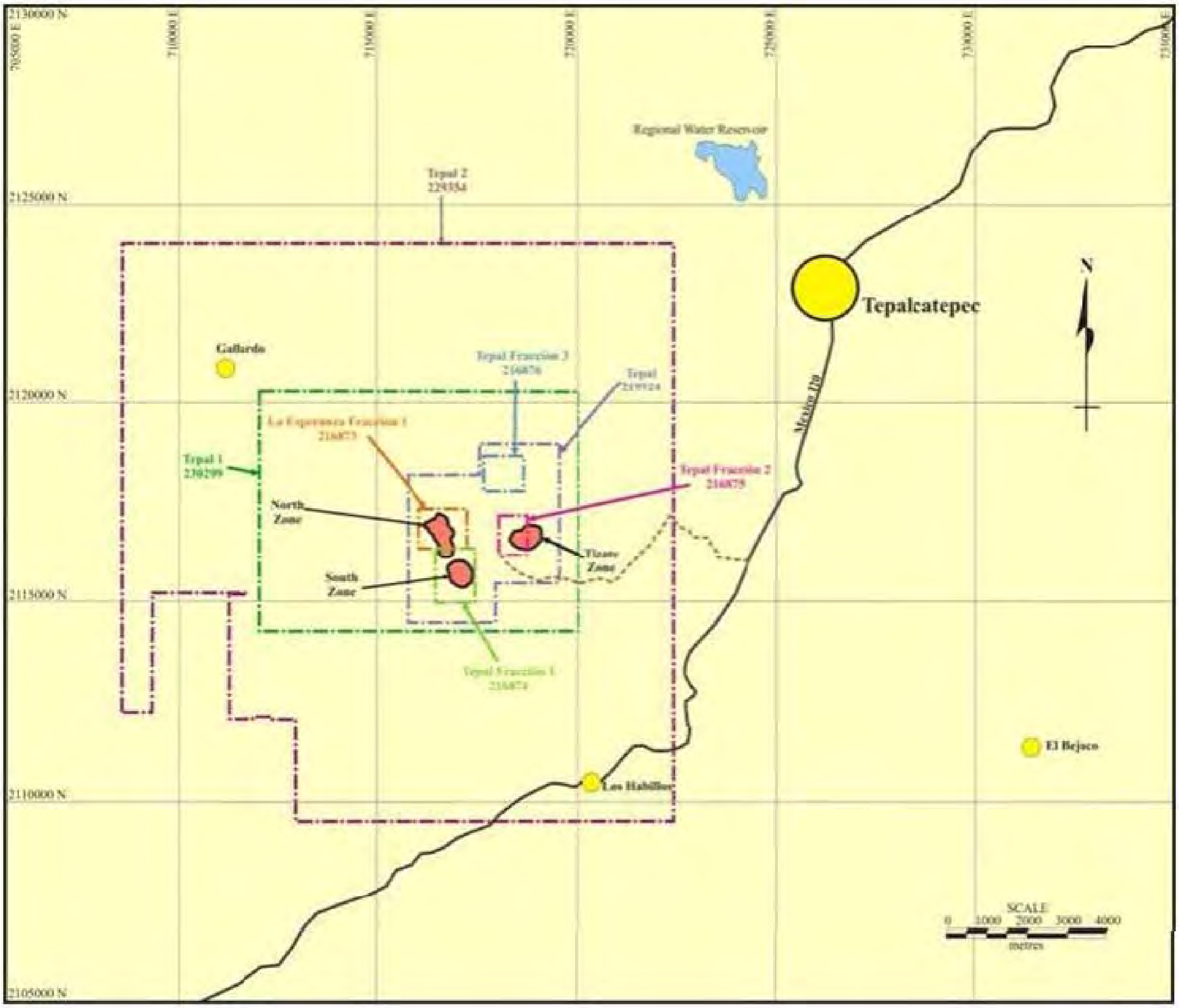
Figure 4-1: Location Map of the Tepal Property, (Micon, 2012)

The Tepal Property consists of seven contiguous concessions totalling 17,237.2ha (Figure 4-2, Table 4-1).

Table 4-1: Concession Titles, Tepal Project

Concession	Title No.	Area(ha)	Date of Title	Expiration Date	Owner
La Esperanza Fr. 1	216873	120.00	June 5,2002	April 18,2044	Geologix Explorations Mexico S.A. de C.V.
Tepal Fr. 1	216874	140.00	June 5,2002	August 17, 2050	Geologix Explorations Mexico S.A. de C.V.
Tepal Fr. 2	216875	70.00	June 5,2002	August 17, 2050	Geologix Explorations Mexico S.A. de C.V.
Tepal Fr. 3	216876	90.00	June 5,2002	August 17, 2050	Geologix Explorations Mexico S.A. de C.V.
Tepal	219924	986.00	May 7,2003	May 6 ,2053	Geologix Explorations Mexico S.A. de C.V.
Div. Tepal 1	230299	3,394.00	August 3,2007	June 27,2055	Minera Tepal S.A. de C.V.
Tepal 2	229354	12,437.20	April 12,2007	April 11,2057	Geologix Explorations Mexico S.A. de C.V.
Total		**17,237.20**			

Figure 4-2: Tepal Property Concession, (Micon, 2012)



The concessions were surveyed in order for the titles to be issued, as required under Mexican law. Lawyers from Mexican company “Sanchez Mejorada, Velasco y Ribe” provided a title opinion for the properties in 2012 (Sanchez Mejorada, Velasco y Ribe, 2012).

Arian Silver de Mexico S.A. de C.V. (Arian) originally optioned the internal concessions (La Esperanza Fracción 1, Tepal, Tepal Fracción 1, Tepal Fracción 2, Tepal Fracción 3) from Minera Tepal S.A. de C.V. (Minera Tepal) for US$5,000,000 to gain 100% interest in the property, subject to a 2.5% net smelter return (NSR).

In 2007, Minera Tepal acquired the Tepal 1 concession (3,394ha) that surrounds the internal concessions. Also in 2007, Arian acquired the Tepal 2 concession (12,437.2ha) which is over free ground and completely surrounding the internal concessions. Tepal 2 is subject to a 2.5% NSR with Minera Tepal.

As of April 4, 2011, Geologix has completed the purchase of the internal concessions and Tepal 2 from Arian and Arian's obligations to Minera Tepal, subject to the 2.5% NSR. There is a first-right-of-refusal on the Minera Tepal NSR royalty should Minera Tepal elects to sell the royalty.

Geologix is presently acquiring 100% interest of Tepal 1 from Minera Tepal. The payments are listed in the following table.

Table 4-2: Tepal Payment Schedule

Amount (US$)	Due Date	
57,000	On signing	Paid
57,000	01-Jun-11	Paid
115,000	01-Dec-11	Paid
172,000	01-Jun-12	Paid
287,500	01-Dec-12	Paid
862,500	01-Dec-13	
1,437,500	01-Dec-14	

Payments are subject to Mexican Value Added Tax (16%) which would be paid by Geologix and applied for reimbursement. A 2% NSR based on the sale of minerals is payable to Minera Tepal. There is a first-right-of-refusal on the Minera Tepal NSR royalty should Minera Tepal elects to sell the royalty. Geologix may purchase at any time all or part of the Tepal 1 NSR for US$1,100,000 plus Value Added Tax for every 1% of the royalty.

The majority of surface rights for the property are owned by three individuals. Some of the peripheral areas of the concession are owned by several parcelised land owners. Geologix has negotiated an agreement for an extended period of time with the main private owner.

Mining taxes for mining concessions, in Mexico are based on the amount of time elapsed from the date the concession title was issued and the number of hectares covered by the concessions. These taxes are paid twice per year and the resulting tax payments for the Tepal Property total approximately US$ 143,103 (US$71,552 paid) for 2013.

Assessment work is calculated on the same basis as property taxes. The assessment work commitment for the property has been met for 2010, 2011 and 2012 and sufficient assessment work credits are available to meet the requirements for 2013.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The following section is taken from Micon Technical report, March 29, 2012 which was modified from Murphy et al, 2011.

5.1 Accessibility

The property can be accessed year round by paved highway Mexico 120 which traverses the southeastern portion of the property. The last 7.5km to the centre of the property is on unimproved dirt roads.

A series of all-weathered roads and the Morelia-Lazaro Cárdenas Autopista (tollway) can be used to reach the capital of Michoacán State, Morelia or Mexico's main west coast port of Lazaro Cárdenas within 3.5 hours.

Two international airports service the area. The General Francisco J. Mujica International Airport (Morelia) is approximately 4.5 hours drive northeast of the property while the Ixtapa Zihuatanejo International Airport is approximately 5 hours south of the property. The closest domestic airport to the property is the Pablo L. Sidar Airport in Apatzingán which is approximately 1 hour drive southeast of the property.

5.2 Climate

The rainy season is usually from June to October while the dry season extends from late November to May. Heavy rains during the rainy season can prevent easy access to the property by turning the dirt roads into mud and/or producing wash outs in places.

Average annual precipitation ranges from 500mm to 700mm (Murphy et al, 2011). The daytime temperatures range from 27°C to 40°C with an average annual temperature between 28ºC to 30ºC.

5.3 Physiography

The property lies within rugged terrain, part of the northeast side of the Mexican Coastal Range as shown on Figure 5-1. The elevation on the property ranges from 500m to 700m. The elevation immediately around the deposit ranges from 550m to 650m. There are large flat areas immediately south and northeast of the property that can be used for mine related infrastructure. A small relatively flat area between the three deposits is acceptable for establishing the mill site.

Vegetation consists of thorny brush, small trees and occasional cactus.

Figure 5-1: North Zone Pil looking South

Figure 5-2: Mill Area looking north to south pit (flat area) and North pit (hill behind pickup)

5.4 Infrastructure

Tepalcatepec is the town nearest the property. It has a population of approximately 30,000. Services available in Tepalcatepec include lodging, a number of small restaurants, gasoline stations, a variety of small hardware, grocery, and retail stores, and an open air market. Geologix has established an exploration compound on the western edge of Tepalcatepec. It also has a secure warehouse for core and rejects sample storage near the exploration compound.

Apatzingán, located approximately 55km southeast of Tepalcatepec, has a population of approximately 90,000. It is the closest town with scheduled domestic air service (Pablo L. Sidar Airport). Daily commuter flights are made to Guadalajara.

Morelia is the capital of Michoacán State and has a population of approximately 550,000. All the regional government and utility offices are located in Morelia. Morelia has an international airport with daily connections to Mexico City and the United States. Morelia is connected to the autopista highway system. Both Guadalajara and Mexico City can be reached within half a day's drive.

There is a three phase power line located 7km east of the deposits. A major power substation is located 2km east of the town of Tepalcatepec. The Comisión Federal de Electricidad (CFE), the federal power authority in Mexico has indicated that sufficient power is available to meet the needs of the project and a power line between the substation and the project could be constructed and power provided from the local electrical grid. Presently there is no power on the property.

There are a series of aqueducts and canals that provide irrigation water to the farms around Tepalcatepec. These aqueducts are feed by several reservoirs in the region. Water for the mine may be available from this reservoir system, however, the property water table appears to be shallow, based on the property wide drill hole information and, therefore, make-up water for the plant is envisioned to come from new water wells and run-off collection ponds. Also several wells in the area of the project indicate that the water table is generally located approximately 3m below the surface.

The dominant land use centred around the three deposits is non-agricultural due to the steep terrain and thick brush. Some of the peripheral land however is used for grazing cattle and goats. In the most arable land at the edges of the property sorghum and corn are grown.

6 History

The following section is taken from Micon Technical report, March 29, 2012 which was modified from Murphy et al, 2011.

The presence of a few small surface workings and several old generations of punto de partido, or concession survey monuments (beacons) in the area of the North and South Zones provide evidence of past exploration on the property. However, there is no anecdotal or written evidence of any production and nothing is known of this early period.

In 1972, the International Nickel Company of Canada, Ltd (INCO) identified the Tepal and the Tizate gossans and associated copper mineralization (Copper Cliff, 1974). INCO worked through its Mexican subsidiary DRACO although the sole surviving report from this time period was prepared by Copper Cliff. Limited data remains from the INCO period.

INCO explored the property during the period 1972 to 1974 by means of surface geochemistry, IP geophysics and drilling. INCO developed a historic (non-NI 43-101 compliant) resource estimate of 27Mt averaging 0.33% Cu and 0.65g/t Au. It is unknown the methodology used to develop the estimate. This estimate was used to attract future companies to the property. Unfortunately INCO abandoned the property. INCO however stressed that more drilling was required to further define the width of the mineralised zones.

The historical estimate prepared by INCO is believed reliable and a good approximation of the amount and grade of mineralization found on the property at the time the estimate was prepared. The historical estimate is no longer relevant as it precedes the estimates presented in this report.

Teck Resources Inc. (Teck) acquired the property in late 1992. Work completed by Teck include geologic mapping, the collection of over 200 rock samples for multi-element analysis, the construction of more than 60 km of grid line, the collection of 1,268 soil samples and 50 rock chip samples from the grid, the construction of 15km of access road and the completion of 50 reverse-circulation holes totalling 8,168m in four phases of work. Teck also undertook some metallurgical testing.

In 1994, Teck completed an historic resource estimate (non-NI 43-101 compliant). The resource estimate was a polygonal block estimate based on the manual definition of polygonal blocks on computer drafted drill sections using manual composited intercept intervals. The total for all categories was 78.8Mt grading 0.40g/t Au and 0.25% Cu with drill indicated resources totalling 55.8Mt grading 0.51g/t Au and 0.26% Cu. The South Zone had a drill indicated resource of 24.3Mt averaging 0.55g/t Au and 0.25% Cu.The North Zone had a drill indicated resource of 31.6Mt averaging 0.49g/t Au and 0.27% Cu. It should be noted that the resource categories defined by Teck were drill indicated, drill inferred and projected and do not directly correspond to the categories of mineral resources prescribed in NI 43-101 but are broadly correlative with Indicated and Inferred resource categories as defined in CIM Definition Standards on Mineral Resources and Reserves (Canadian Institute of Mining, Metallurgy and Petroleum, 2010).

The historical estimate is believed reliable and a good approximation of the amount and grade of mineralization found on the property at the time the estimate was prepared. The historical estimate is no longer relevant as it precedes the estimates presented in this report.

In late 1996, Minera Hecla S.A. de C.V. (Hecla) visited the property and initiated a work program in the spring of 1997. Hecla's expenditures on the property are unknown however Hecla's primary focus on the property was to define a large tonnage, low-grade gold target.

Work by Hecla included the creation of a 1:2,000 scale topographic map from aerial photographs, a geologic mapping program, the collection of nearly 900 rock chip samples on a 50m by 50m grid, the re-analysis of 298 pulps from the Teck reverse-circulation drilling program, the completion of 17 reverse-circulation drill holes totalling 1,506m and the completion of a historic resource estimate (Gómez-Tagle, 1997 and 1998). Although all samples were analyzed for copper and gold, Hecla did not include copper in its resource estimate. The resource estimate was a polygonal block estimate based on manual definition of polygonal blocks on computer drafted drill sections using manual composited intercept intervals. The total resource for oxide and sulphide material in the North and South Zones was 9.06Mt averaging 0.90g/t Au and containing 262,359oz of gold.

The historical estimate prepared by Hecla is believed reliable and a good approximation of the amount and grade of mineralization found on the property at the time the estimate was prepared. The historical estimate is no longer relevant as it precedes the estimates presented in this report.

In 2007, Arian Silver de Mexico S.A. de C.V. (Arian) undertook a diamond drill program consisting of 42 holes totalling 7,180m. In April 2008, ACA Howe did a mineral resource estimate using an inverse weighted method to the third power (ID^3). The constrained +0.18g/t Au mineralised zones at Tepal were interpolated to have a total Inferred Mineral Resource of 78.8Mt grading 0.47g/t Au and 0.24% Cu at a zero cut-off grade for approximately 1.18Moz Au and 421.5Mlbs Cu.

In September, 2008, ACA Howe International Limited undertook a second NI 43-101Technical Report which included a mineral resource estimate. A block model was created and constrained by interpreted geological wireframe solids of the North and South Zones. The blocks were interpolated using an ID^3. The North and South Zones were estimated to contain an Indicated Mineral Resource of 25.0Mt grading 0.54 g/t Au and 0.27% Cu and an Inferred Mineral Resource of 55.0Mt grading 0.41g/t Au and 0.22% Cu, constrained by a 0.18ppm Au envelope that honoured geology. This resource did not include the Tizate Zone.

Micromine software was used to generate a wireframe restricted, linear block model resource estimate of contained gold and copper over the project using ID^3.

In 2010, Geologix completed a 42-hole diamond drill program totalling 10,656m. There were 26 holes that defined the North and South Zone deposits and 14 holes that targeted the Tizate Zone. Two additional holes were completed between the North/South Zones and the Tizate Zone. SRK completed a Preliminary Economic Assessment Technical Report (PEA) in October 8, 2010 and a Preliminary Assessment Technical Report (PA) in April 29, 2011. A new mineral resources estimate was completed as part of the PA Technical Report.

A new mineral resource was completed as part of the 2011 Preliminary Assessment technical report (Murphy et. al., 2011). This estimate included the North, South and Tizate Zones. There was a re-examination of all domains in the three deposits. New drilling up to 2010 was included into the drill database.

New models were constructed by Geologix using envelopes that utilized an US$8.70 equivalent cut-off based on a price of US$900/oz for gold and US$2.75/lb for copper. The cut-off used in the models corresponded closely with the primary economic limits of the mineralization and was based on geological observations on the type and intensity of alteration, veining and sulphide or oxide mineralization.

A digital terrain model (DTM) was created for each deposit to represent the base of the oxide zone which usually corresponded to the base of the hematite mineralization. There is a transition zone in the deposits but is generally narrow (i.e. 1 to 2m) so a separate domain was not created for this zone.

Minimal top cuts were made for copper and gold after an outlier review was made of the data. The cumulative frequency inflection point method was used to determine the capping level.

A two metre composite was chosen as the optimum length for the drill hole data. Variography was used to define the directions of grade anisotropy and spatial continuity of gold and copper grades. This data was used as input parameters for grade interpolation. There was insufficient data to generate correlograms for silver and molybdenum therefore range and orientation parameters were taken from the corresponding copper correlograms.

Two block models were generated for Tepal (North and South Zones) and Tizate. A block size of 10m x 10m x 5m was selected. There was no sub-blocking in the models. Gold and copper grades were interpolated on respective domains for Tepal and Tizate deposits using the Ordinary Kriging interpolation method. Silver and molybdenum grades were only generated for the Tizate deposit. These grades were interpolated using the inverse distance squared (ID^2) method.

In order to determine the quantities of material offering “reasonable prospects for economic extraction” (CIM definition) from an open pit, SRK used the Whittle pit optimizer to evaluate the profitability of each resource block based on certain optimization parameters selected from comparable projects. The optimization parameters include: waste mining costs of US$1.00/t; mining and processing costs of US$5.60/t milled; overall pit slope angles of 45°; metallurgical recoveries of 60% and 78% were applied for gold in sulphide and oxide respectively and recoveries of 87% and 14% were applied for copper in sulphide and oxide. Appropriate dilution and offsite costs and royalties were also considered and applied where appropriate. A gold price of US$1,200/oz and a copper price of US$3.00/lb were used. (Murphy et. al. 2011).

Based on the above, SRK estimated that the Tepal and Tizate deposits contained 57.8Mt of Indicated mineral resources grading 0.42g/t Au and 0.24% Cu at a cut-off grade of US$5.00 equivalent value. The deposits contained an additional 93.2Mt grading 0.28g/t Au and 0.20% Cu classified as Inferred mineral resource at a cut-off grade of US$5.00 equivalent value (Murphy et. al. 2011).

7 Geological Setting and Mineralization

The following section is taken from Micon Technical report, March 29, 2012 which was modified from excerpted Priesmeyer, 2007 and 2013 refined interpretations from Geologix's geological staff.

7.1 Regional Geology

The property is located within the Costal Ranges of south-western Mexico south of the Neogene Trans-Mexican Volcanic Belt. Basement rocks consist of Cretaceous to early Tertiary (?) intermediate intrusions (plutons, stocks and plugs) intruding weakly metamorphosed sedimentary and volcanic rocks of Cretaceous to Early Tertiary age. The Jurassic to Cretaceous sedimentary and volcanic rocks are part of an accreted Mesozoic island arc volcanosedimentary assemblage. At least some of the intrusive rocks are probably coeval with the volcanic units. Neogene basalts locally overly basement rocks and represent outliers of the Trans-Mexican Volcanic Belt.

The property lies just south of the Huacana Batholith (Figure 7-1), a Cretaceous to Early Tertiary batholith that ranges from quartz diorite to tonalite and granodiorite in composition.

The mineralised hypabyssal intrusions at the Tepal prospect are thought to be marginal phases of this batholith (Shonk, 1994).

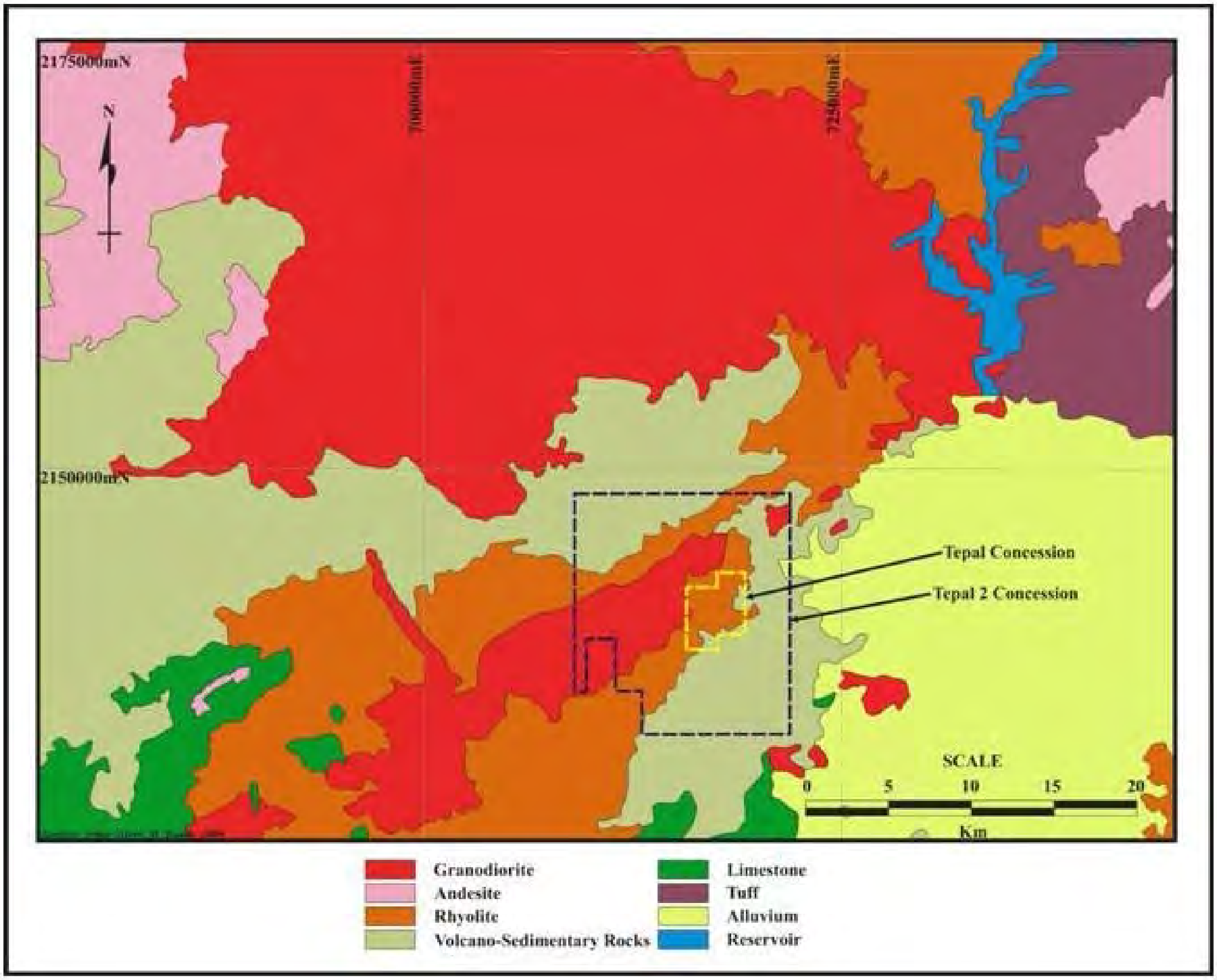
7.1 Property Geology

The geology in the immediate vicinity of the North and South Zones was mapped by Teck geologists in the early 1990's. Geologic mapping of the current property was carried out by Geologix in 2011.

Much of the property is underlain by early to middle Tertiary intrusive rocks. These include granodiorite and, in the core of the property, tonalites. Shonk (1994) noted that the tonalites display a wide variation in texture and phenocrysts abundance indicating diverse cooling histories and suggesting multiple intrusive events with relatively high levels of emplacement. His observations of local tonalite intrusion breccias showing chilled porphyritic to glassy porphyritic textures suggest the same. Limited analysis of rock geochemistry by Geologix in 2011 from tonalities associated with all mineralized zones supports this and indicates a tonalite intrusive complex comprised of several chemically distinct but related phases. At present the extents of different tonalite phases has not been mapped in the field.

The intrusive rocks were emplaced into a lower Cretaceous volcano-sedimentary sequence. In the area of Tepalcatepec, this sequence is formed of thick sections of interbedded limestones and shales, alternating with thick layers of andesitic tuffs and volcanic breccias. These sequences have been mapped as a homoclinal, south-dipping sequence on the southern portion of the Tepal property with the andesitic tuffs and volcanic breccias being encountered in some South Zone drill holes further north. The volcanics have also been mapped to the west of the North Zone. Post-mineral and post-alteration andesite dykes are present and noted to cut the tonalities.

Figure 7-1: Geological Map of the Tepal Property



7.2 Structure

Structurally two main fault trends are present on the property dividing it into parallelogram like blocks. These include an east-northeast trend (N70°E) and a north-northwest trend (N20°W). The east-northeast trend has been mapped at surface and intersected in drill core at the southern edges of the South and Tizate Zones dividing the property into a predominantly tonalite domain to the north and a volcano-sedimentary domain to the south. Other parallel east-northeast structures have been inferred from topography further north. One of these inferred faults lies between the North and South Zones and extending northeast along the north edge of the Tizate zone. Another inferred fault lies to the north of the North Zone. On the one positively identified structure, drill intersections show that it dips 45° to the southeast. Two strong north-northwesterly structures have been inferred from topography and geophysics. One lies to the immediate east of the North and South Zones, while the other is to the east of the Tizate Zone.

Both of these sets of faults appear to have juxtaposed different erosional levels. Rocks to the south of the identified east-northeast fault are mainly those belonging to the volcano-sedimentary package which shows virtually no alteration (minor skarn development is noted locally in the limestones) and have undergone normal fault displacement against the tonalites to the north. To the north of this fault, two blocks are formed by the north-northwest faults. The western block, which contains the North and South Zones, is mainly composed of porphyritic tonalite with minor volcanics, while the eastern fault block which contains the Tizate Zone is comprised mainly of medium grained equigranular tonalite. Shonk (1994) suggested that the western block was from a higher level based on deeper drilling that showed a transition in this area from tonalite porphyry and intrusion breccia near the surface to equigranular, medium grained tonalites at depth, similar to those in the eastern fault block.

7.3 Mineralization

Mineralization on the property consists of structurally controlled zones of stockwork and disseminated sulphide mineralization that are hosted entirely within a multi-phase tonalite intrusive complex. These sulphide bearing zones contain significant concentrations of copper and gold with lesser silver and molybdenum values. . The current resources are hosted in three distinct zones: the relatively high-grade North and South Zones and the lower grade Tizate Zone.

Morphologically, two of the zones, the North and Tizate zones, are crudely tabular with shallow to moderate dips. Both have rough dimensions of approximately 1,100m by 600m and thicknesses of up to 200m. The South zone has a smaller footprint, 600m by 500m, but a greater vertical extent of up to 400m, although this is possibly the result of faulting.

In the North and South Zones some generations of veins within the structural deposits display a prominent 325° - 350° orientation parallel to the north-northwest fault trend. Dips are generally vertical to steep either east or west. Other prominent orientations are also present including a set with a near east-west orientation and moderate southerly dip. The attitudes of vein sets in the Tizate Zone has not yet been accurately determined, however, consistent core to vein angles in drill holes suggest several persistent orientations. The strong preferred orientation of these veins and evidence of shearing suggests development of the zones was during late magmatic stages (Shonk 1994).

There is an oxide horizon and a narrow transition layer present in the deposits on the Tepal property above the sulphide mineralization. The depth of oxidation ranges from 20m to 40m on the hilltops and 0m to 20m in the drainages. Minerals in the oxidized zone include malachite, chalcocite, minor azurite, tenorite and minor chrysocolla. Shonk (1994) indicated that thin supergene-enriched layer exist locally at the base of the oxide horizon and consists of chalcocite and covellite coatings on sulphide grains and local areas of poddy, massive chalcocite. While minor chalcocite has been noted in drill core, drill hole assays do not indicate any leaching of copper from the oxide horizon and no local copper enrichment zones at the oxide-sulphide interface. The transition zone may be up to 15 m thick, however, it is usually significantly less than this and in some cases is absent altogether. The transition is identified by the overlapping presence of iron oxides and sulphide mineralization.

Primary sulphide mineralization consists dominantly of disseminated and stockwork-controlled chalcopyrite and pyrite with minor, locally significant pyrrhotite, bornite, sphalerite, molybdenite and galena. The highest grade mineralization is associated with low total sulphide contents and low pyrite: chalcopyrite ratios. Micron-sized native gold is usually associated with the chalcopyrite either as grains attached to the surface or fracture fillings within copper sulphides (Duesing, 1973) although free grains can also occur. Hypogene sulphide mineralization typically occurs as irregular individual sulphide grains or interstitial patches of pyrite-chalcopyrite-bornite within the granular, altered tonalite porphyry groundmass, often associated with growth of granular quartz in the groundmass, as chalcopyrite-pyrite veinlets and as quartz-hydrobiotite/Fe-chlorite-pyrite-chalcopyrite veinlets associated with sericite-hydrobiotite/Fe-chlorite-pyrite-quartz alteration (Shonk, 1994).

Several different generations of quartz veining, quartz replacement, and silicification are prominently associated with copper-gold mineralization. Quartz vein types include early granular quartz veins with no alteration envelope consisting of quartz-sulphide-biotite of probable late magmatic age. Locally late magmatic veining is so closely spaced that vein material comprises the majority of the rock. Chlorite-quartz-sulphide-calcite and prismatic to comb quartz-sulphide veins are interpreted to be a later stage event.

Granoblastic growth of granular subhedral to euhedral quartz in the groundmass and patchy, finer grained, blue-gray quartz flooding of the groundmass (colour due to very fine grained disseminated sulphides) are often associated with granular quartz veins and are also inferred to be of late magmatic age. This quartz is typically associated with disseminated chalcopyrite and bornite (Shonk, 1994).

Intensity of mineralization is strongly related to the presence of late magmatic quartz and the density of late magmatic veining (Shonk 1994). Both the North and South Zones have a crude zonation with a gold-rich core associated with the highest gold and copper values and highest Au : Cu ratios to a copper dominant periphery with lower Au : Cu ratios and then to a barren pyritic halo (Shonk, 1994). Silver and Mo values are also somewhat elevated in the core areas but distribution is more erratic and is not always coincident with Au or Cu values. In particular Mo often seems to occur with elevated values in the North and South Zones over short drill hole assay intervals, perhaps due to specific structural controls.

In the Tizate Zone, copper values are on average slightly lower than the North and South Zone averages and gold grades are significantly lower. Grade distribution however is very even and the very high grade cores and lower grade fringes seen at the other deposits are not seen here. Both the Ag and Mo values are significantly higher than in the other deposits and they show greater coincidence with Au and Cu, particularly with respect to Mo.

Mineralization on the property is characterized by strongly anomalous Cu, Au, Ag, Zn, and Mo and more erratic and weakly anomalous Pb, Mn, Bi, and As. Unfortunately, inter-element relationships have not been systematically analyzed over the mineralized zones because the Teck soils, which are over the core of the property, and most Teck drill core samples were only analyzed for Cu and Au. Anomalous levels of As, Pb and Zn have been encountered in recent drilling which have full ICP data. In most cases, elevated levels of these elements occur erratically in veins and mineralized structures or areas outside of the deposits.

7.4 Alteration

Alteration in and around the deposits shows alteration features that are typically associated with Cu-Au porphyry systems. Prograde alteration facies consist of a potassic core grading out through an inner propylitic zone to a peripheral or outer propylitic halo. Retrograde alteration facies consist of phyllic and argillic alteration. The type and intensity of these alteration facies varies between the deposits, likely due to a function of depth in the mineralizing system. The overall geometries and thicknesses of these alteration zones are not well defined.

Potassic alteration is only weakly developed in the cores of the North and South Zones but more extensively developed in the Tizate Zone. Hydrothermal potassium feldspar is locally present but uncommon to rare. Instead potassic alteration is characterized by biotite replacement of hornblende phenocrysts and more diffuse felted replacement in the groundmass which imparts a distinct brownish tinge to the rock. The biotite is associated with strong silicification, granular quartz veining and, locally, disseminated magnetite. It also occurs in hydrothermal quartz-biotite-sulphide-magnetite veins (Shonk, 1994). It is most often mineralized, carrying Au, Cu, Ag and Mo values, however, un-mineralized examples do exist, mainly at depth in the Tizate Zone.

An inner propylitic zone is strongly developed and hosts the bulk of the mineralization in the North and South Zones, particularly in the high grade cores, and it may be transitional from the potassic zone. This facies is less well developed in the Tizate Zone. It is characterized by coincident chlorite-sericite-pyrite-quartz alteration, granular quartz flooding of the groundmass and quartz-Fe-chlorite-sulphide veining are also closely associated with copper-gold mineralization. The Fe-rich chlorites have been interpreted as indicating formation temperatures just below the stability limit of biotite, so that Fe-rich chlorites form contemporaneously with the hydrothermal biotite (Shonk, 1994). Other alteration minerals sporadically associated with these assemblages include albite, calcite, epidote, clinozoisite, leucoxene, hematite, tourmaline, apatite, rutile and gypsum after anhydrite (Shonk, 1994).

There is a rapid transition from the inner propylitic to the outer propylitic zone, which is the classic peripheral alteration facies. Alteration consists of weak to moderate chlorite alteration with epidote, weak disseminated pyrite and carbonate as fine veinlets. Quartz veinlets are absent.

Phyllic alteration appears to be retrograde at Tepal, locally overprinting mineralization and the inner propylitic zone in the North and South Zones, and quite extensively overprinting mineralization and potassic alteration in the Tizate Zone. This mineral assemblage consists of sericite, pyrite, quartz (flooding and veinlets), carbonate and clay. Anomalous to lower grade gold and copper values are often associated with this type of alteration but higher grade mineralization is absent unless it is noticeably overprinting earlier mineralized alteration facies. In addition there are examples of phyllic altered tonalite that are barren.

Peripheral to the three deposits and in all cases to the west of them are areas of argillic alteration. Largely defined by outcrop exposures, this alteration type is characterized by sparsely vegetated, red-brown to red colour exposures of argillized rock. This is as a consequence of supergene argillization due to oxidation of the 3-15% disseminated pyrite. Supergene minerals include kaolinite, illite, diaspore, pyrophyllite, and silica (Shonk, 1994). To the west of the North and South Zones this alteration is developed in a thin sliver of Cretaceous volcanics and may also be a contact alteration feature.

8 Deposit Types

The following section is taken from Micon Technical report; March 29, 2012 which was modified from excerpted from Priesmeyer, 2007 and 2013 refined interpretations from Geologix's geological staff.

Mineralization on the property is characteristic of porphyry copper-gold mineralization. Porphyry-type deposits in Mexico occur in a northwest trending belt 2,800km long on the west side of the country, following the Pacific continental margin (Sillitoe, 1976). The belt is located in the Sonoran Basin and Range, Sierra Madre Occidental and Sierra Madre del Sur covering the states of Sonora, Sinaloa, Chihuahua, Durango and Michoacán.

Panteleyev (1995) characterizes porphyries as large masses of hydrothermally altered rock containing quartz veins and stockworks, including sulphide-bearing veinlets and dissemination, covering areas up to 10km^2 in size. These altered zones are commonly coincident with shallow intrusives and/or dike swarms and hydrothermal or intrusion breccias. Deposit boundaries are determined by economic factors, which outline ore zones within larger areas of low-grade concentrically zoned mineralization.

Important geological controls on porphyry mineralization include igneous contacts, cupolas and the uppermost, bifurcating parts of stocks and dike swarms. Intrusive and hydrothermal breccias and zones of intensely developed fracturing due to coincident or intersecting multiple mineralized fracture sets that commonly coincide with the highest metal concentrations.

Surface oxidation commonly modifies the distribution of mineralization in weathered environments.

Normally acidic meteoric waters generated by the oxidation of pyrite leach copper from soluble copper minerals and re-deposit it as secondary chalcocite and covellite immediately below the water table in tabular zones of supergene enrichment. This has never been observed at the Tepal property. The Tepal property exhibits a copper-poor leached cap and a thicker zone of lower grade primary hypogene mineralization at depth.

Copper-gold porphyries differ slightly from copper ± molybdenum porphyries in the following ways:

- They can be associated with alkaline intrusive suites.
- Copper-gold porphyries do not typically contain economically recoverable Mo. They typically contain < 100ppm Mo, but do contain elevated gold (> 0.3g/t) and silver (>2g/t).
- They are commonly associated with abundant hydrothermal magnetite, which is commonly associated with higher gold grades.
- Copper and gold may or may not be associated with zones of quartz veining (depending on degree of silica saturation), in contrast to most "normal" porphyry systems where quartz veining is the norm.
- Supergene enrichment can be restricted due to the general sulphide-poor nature of the alteration and they often lack an extensive peripheral hypogene alteration "footprint".

Porphyry copper-gold deposits range from very large, low-grade deposits such as Bingham Canyon in the United States which contains 3,228 Mt averaging 0.88% Cu and 0.50g/t Au (Cooke and others, 2004) to small high-grade deposits such as Ridgeway in Australia which contains 54Mt averaging 0.77% Cu and 2.5g/t Au (Wilson and others, 2003). The average of 112 deposits from around the world is 200 Mt averaging 0.44% Cu, 0.4g/t Au, 0.002% Mo and 1.4g/t Ag (Singer and et al, 2005).

It should be noted that mineralization on these or any other properties in this class of deposit around the world is not necessarily indicative of the mineralization on the Tepal Property.

9 Exploration

The following section is taken from Micon Technical report, March 29, 2012 which was modified from excerpted from Priesmeyer, 2007 and modified excerpt from Murphy et. al. (2011).

9.1 Inco

In 1972 the International Nickel Company of Canada, Ltd (INCO) recognized the Tepal and the Tizate gossans (Tizate is located approximately 1,400m east of the North Zone) and associated copper mineralization (Copper Cliff, 1974).

The Tepal and Tizate gossans were originally considered as separate entities but were eventually evaluated by a single soil grid. Soil samples were analyzed for Cu, Mo, Zn and Au and anomalous copper zones were identified. In early 1973 six diamond drill holes (57001 –57006) were drilled in the Tepal gossan. Geologic mapping and an Induced Polarization (IP) survey were completed during the winter of 1973-74. IP anomalies were found to be generally confined to geochemically anomalous copper zones. According to Shonk (1994) both a summary map showing extent and strength of interpreted anomalous IP response along each line in conjunction with molybdenum in soil anomalies and drill hole locations and photocopies of contoured IP sections were available. The summary map indicated a strong to moderate IP response over and peripheral to the North Zone, a moderate IP response just South of the South Zone, and a number of lines with weak to strong IP anomalies coinciding with the broad zone of soil geochemical anomalies on the east side of the property. At the time Shonk (1994) prepared his report, many of the IP anomalies had not been drilled.

9.2 Teck

Teck Resources Inc. (Teck) acquired the property in late 1992. Work completed by Teck included geologic mapping, the collection of over 200 rock samples for multi-element analysis, the construction of more than 60km of grid line, the collection of 1,268 soil samples and 50 rock chip samples from the grid, the construction of 15km of access road and the completion of 50 reverse-circulation holes totalling 8,168 m in four phases. Total expenditure by Teck was approximately $875,000 (Shonk, 1994). Teck also completed metallurgical testing.

Only very limited data remains from the Teck period on the property. There is one report, a variety of hand-drafted maps, drill logs and sample pulps from the drilling program. No duplicate samples or coarse rejects are available for review or analysis and there are no original assay certificates for data verification purposes.

Initial mapping on the property was conducted by Richard L. Nielsen, a Denver-based consultant. Nielsen mapped the property at a scale of 1:5,000 and collected 165 samples for multi-element analysis. The west side and portions of the east side of the property we subsequently remapped by another consultant at scales of 1:2,000 and 1:1,000 on a grid base.

The early grid covered the western part of the mineralised area and part of the eastern half with a line spacing of 100m and a station spacing of 50m over areas of known mineralization and alteration and a station spacing of 100m outside areas of known mineralization and alteration.

In late 1993 and early 1994 Tech completed a soil sampling program. Grid lines were spaced 200m apart and sample spacing was 100m and over anomalous areas, line spacing was reduced to 100m and sample spacing to 50m. A total of 1,268 soil samples and 50 rock chip samples were collected from all phases of soil sampling. Soil samples were analyzed for Cu and Au and most rock chip samples were analyzed using multi-element Inductively-Coupled Plasma (ICP). According to Shonk (1994), values from both soil and rock samples showed a strong positive correlation.

While the North Zone was known from previous INCO drilling, soil geochemistry as well as geologic mapping by Teck delineated the South Zone as a new target. Both the North and South Zones occurs as well defined coherent anomalies. A broad zone of less coherent anomalous Cu values covers a 1.5 x 2.0km area on the east side of the property with three poorly defined highs. Au values show the same general pattern though anomalies are more subdued on the east side of the sampling grid.

There is no surviving contoured soil geochemistry maps of the property based on the Teck data. There is a map prepared by Hecla showing the Teck soil sample locations and values in conjunction with their own but the Teck data had not been contoured.

9.3 Hecla

In late 1996 Minera Hecla S.A. de C.V. (Hecla) obtained the property and initiated a work program in the spring of 1997. Work by Hecla included the creation of a 1:2,000 scale topographic map from aerial photographs, a geologic mapping program, the collection of nearly 900 rock chip samples on a 50m by 50m grid, the re-analysis of 298 pulps from the Teck reverse-circulation drilling program, the completion of 17 reverse-circulation drill holes totalling 1,506m and the completion of a resource estimate (Gómez-Tagle, 1997 and 1998).

Hecla's expenditures on the property are unknown.

The work completed by Hecla is the best documented of all the previous work. There are two reports prepared by the project geologist, assay data in digital form and limited documentation for the resource estimate. Hand-written drill logs are also available. Most of the maps generated by Hecla remain, at least in electronic form. Sample splits and chip trays remain from the Hecla drilling. Four of the sample splits were re-sampled by ACA Howe for grade verification purposes.

Hecla mapped the property at a scale of 1:2,000. Mapping was intended to define lithologic units and the type, intensity and extent of mineralization and hydrothermal alteration. There is no mention in the Hecla reports as to whether geologic mapping was done on the rock chip sampling grid. Roads were located using a compass and tape.

In 1997, Hecla collected 895 rock chip samples from trenches, road cuts and constructed a north-south grid on the property. The grid covered an area measuring approximately 1,000 m in a north-south direction and 750 m in an east-west direction. Grid lines were spaced 50 m apart.

Hecla defined a large copper anomaly with the concave portion of the anomaly open to the southwest. The anomaly was defined by copper values in excess of 301ppm copper in rock.

This anomaly measured approximately 1,100m in length and 125m in width and was open to the northeast and the south. Within this large anomaly were three strongly anomalous areas defined by copper values exceeding 1,000ppm. The largest of these strong anomalies measured approximately 300m by 230m and generally defined the North Zone.

The gold anomaly defined by Hecla was more restricted in aerial extent. The anomaly was defined by gold values in excess of 200ppb or 0.2g/t Au in rock and was open to the south and southeast. The anomaly trended 320° and measured approximately 700m by 215m.

Within this anomaly was a smaller, very strong anomaly in which all values exceed 910ppb or 0.91g/t Au. This anomaly measured approximately 230m by 80m and generally corresponded to the North Zone.

In order to confirm the analytical results from the Teck drilling, Hecla re-analyzed 298 pulps from some of the Teck diamond drill holes (i.e. T-9, T-13, T-23, T-24, T-25 and T-30). Results of the Hecla re-analysis indicated that the values obtained by Hecla were 7% higher than those obtained by Teck. Since Hecla's primary focus was gold, ACA Howe presumed that this difference was for gold values only.

9.4 Arian

Exploration by Arian was initiated in April 2007. Exploration consisted of a Tepal Phase 1 diamond drill program.

The following sub-section is a modified excerpt from Murphy et. al. (2011).

9.5 Geologix

During the due diligence period commencing in the 4th quarter of 2009 and continuing into the 1st quarter of 2010 the Company initiated additional metallurgical test work utilizing core from historical drill core, an induced polarization (IP) survey over the core mineral concessions covering 1,526ha, geological test work including geology, mineralization and alteration studies and preliminary economic studies as they pertain to the viability of the Tepal project.

By the end of the 1st quarter of 2010 the geophysical survey had been completed with a total of 78.4 line-km of surveying.

On June 16, 2010, an extensive diamond drill testing program was initiated on the Tepal project. The drill program was geared to evaluate the "near resource" potential of additional mineralization being located near the Arian Silver/ACA Howe resource outlines and to test for additional mineralization on the remainder of the property. Targets on the remainder of the property were defined by geological, geochemical and geophysical anomalies as outlined in historic surveys as well as the geophysical survey completed by the Company in 2010. By the end of 2010 a total of 10,656m of drilling in 42 holes had been completed by two drilling rigs including 26 holes around the resource area at Tepal (North and South Zones), 14 holes in the Tizate zone where no previous resources had been outlined, and two other exploration targets on the property.

Drilling continued with seven drill rigs in 2011. In addition, the Company initiated detailed property geological mapping, prospecting, a soil geochemical grid survey, a silt sampling programs and an airborne geophysics survey which included magnetics, radiometrics and EM to cover the entire 172 km^2 land package. A total of 1,551 line-km were flown with 1,421 line-km flown at a flight line spacing of 150m over the entire concession. A more detailed survey over 19km^2 (130 line-km) was flown over the known deposit area at 75m spacings.

Exploration activities in 2012 concentrated on the seven anomalous areas outlined by the 2011 airborne geophysical survey. All seven anomalies received additional mapping, trenching, continuous chip sampling as well as soil sampling in areas devoid of outcrop. A total of 1,064 soil samples and 1,263 rock chip samples were collected, resulting in the prioritization of five geophysical anomalies to a drill testing stage. To test these, GIX drilled a total of 34 Reverse Circulation (RC) drill holes totaling 4,906 metres. None of this drilling was carried out on the known mineralized zones.

10 Drilling

The following section is taken from Micon Technical report, March 29, 2012 which was modified from Murphy et al, 2011.

10.1 Inco

Between 1973 and 1974, INCO drilled at least 21 diamond drill holes utilizing a Longyear 38 core rig from Boyles Brothers. Holes were collared with NX (core - 54.7 mm) and reduced to BX (42.0mm). Sample intervals ranged from 0.2 to 3.0m and averaged 2.0m. INCO drill the North and Tizate Zones since the South Zone had not been identified. The total number of drill holes is unknown, as is the grand total length of the drill program due to incomplete documentation.

A more detailed description of this drill program is available from Murphy et. al. (2011).

10.2 Teck

In 1994, Teck drilled 50 reverse-circulation (RC) drill holes totalling 8,168.8m. The drilling contractor employed by Teck is unknown as are the drilling procedures.

The majority of Teck's drill holes were drilled in the North and South Zones although a few holes were drilled in the Tizate area. A differential GPS survey was conducted in late January, 1994 to locate the INCO holes and the first 24 Teck holes as well as roads, key grid points, concession monuments and planned drill holes. Compass and tape surveys were used to establish coordinates of later drill holes and map access roads constructed after the survey.

Samples were collected every 2.03m (3 per 20-foot drill rod) for the first 24 holes and every 1.52m (5 ft intervals) for holes T-25 through T-50.

A duplicate analytical sample was collected every tenth sample interval. All drill samples were analyzed for Cu and Au at Chemex (now ALS Chemex). An additional 123 samples from selected intervals were analyzed for Ag, Co, Cu, Fe, Mn, Mo, Ni, Pb, and Zn using a multi-element ICP procedure.

Drilling at Tepal generally indicated that the best values were present within 150m of the surface. Significant intercepts at greater depths were confined to the cores of the North and South Zone resource areas.

Preliminary metallurgical tests were also conducted on a few selected intervals of mineralized intercepts from drill hole IN57002.

A more detailed description of this drill program is available from Murphy et. al. (2011).

10.3 Hecla

In late 1997, Hecla conducted a 17-hole reverse-circulation (RC) drilling program totalling 1,506m.

All but three of the Hecla holes were drilled in the North Zone. The remaining three were drilled in the South Zone. Sample interval for the Hecla reverse-circulation drilling program was 1.0m.

A more detailed description of this drill program is available from Murphy et. al. (2011).

10.4 Arian

The Phase 1 diamond drilling campaign was completed in June 2008, consisting of 42 holes totalling 7,180m. Drilling has been carried out using two Boart Longyear 38 drill rigs owned and operated by GICSA (Geotechnica, Igenieria y Construction, S.A. de C.V.), of Paseos de Taxquena, Mexico, D.F.

The majority of the initial diamond drilling was carried out using HQ drill steel (core - 63.5mm) and reduced if required to NQ (core - 47.6mm). Drill core was not oriented for the Phase 1 program.

A more detailed description of this drill program is available from Murphy et. al. (2011).

10.5 Geologix 2010

Geologix carried out a diamond drilling program in 2010. There were a total of 42 drill holes totalling 10,656m completed on the Tepal property. The drill program utilized two diamond drilling machines. The purpose of the drill program was to evaluate the “near resource” potential for additional mineralization located near the Arian Silver/ACA Howe resource outlines and test for additional mineralization on the remainder of the property. No drilling was completed within the resource limits.

Geologix drilled 26 core holes which targeted the peripheral area of the Tepal (North and South Zone) and 15 holes that targeted the Tizate zone. Two holes tested exploration targets in the area between Tepal and Tizate.

A more detailed description of this drill program is available from Murphy et. al. (2011).

10.6 Geologix 2011

Geologix continued to drill the Tepal (North and South Zones) and the Tizate Zones throughout 2011. There were 202 diamond drill holes in the totalling 41,247.5m. The drill program utilized seven diamond drilling machines from Major Drilling International Inc. and Intercore Perforaciones S. De R.L. de C.V. to complete the program within 2011 time frame.The focus of this diamond drill program was to infill the three deposits thereby upgrading the mineral resource categories for use in a PFS.

The Table 10-1 shows the number of holes and the total length drilled for the Tepal and Tizate.

Table 10-1: Geologix 2011 Drill statistics

Deposit	Holes	Length
Tepal	132	23,074.3
Tizate	70	18,173.2
Total	202	41,247.5

In addition to the infill drill holes there were a series of wide-spaced condemnation and geotechnical holes that were completed on the property. There were 7 in-pit geotechnical drill holes totalling 1,353.6m and a total of 6 condemnation holes totalling 297.5m.

The following table documents some of the significant mineralized intervals obtained in the 2011 drill program.

Table 10-2: Geologix 2011 Significant Assay Results

Hole No.	Zone	From (m)	To (m)	Length (m)	Cu (%)	Au (g/t)	Ag (g/t)
TEP-11-010	South	0.00	64.05	64.05	0.30	0.67	0.8
TEP-11-012	South	146.50	425.90	279.45	0.26	0.54	1.3
	including	301.40	403.85	102.45	0.38	0.86	0.9
	including	303.40	370.95	67.55	0.42	1.01	1
TEP-11-015	South	0.00	91.10	91.10	0.25	0.67	1
TEP-11-016	South	6.20	86.10	79.90	0.26	0.88	1.4
TEP-11-018	South	0.00	140.00	140.00	0.27	0.59	1.4
TEP-11-020	South	0.00	213.40	213.40	0.21	0.39	0.5
TEP-11-026	South	309.20	498.00	188.80	0.40	1.04	2.7
	including	317.20	422.00	104.80	0.44	1.45	1.3
TEP-11-033	North	0.00	41.90	41.90	0.58	0.29	5.9
TEP-11-043	South	152.00	294.55	142.55	0.35	0.91	1.3
	including	162.00	274.00	112.00	0.38	1.04	1.2
TEP-11-060	North	0.00	96.00	96.00	0.26	0.43	2.3
TEP-11-063	North	4.00	67.40	63.40	0.26	0.36	1
TEP-11-064	North	0.00	54.50	54.50	0.29	0.43	2.1
TEP-11-065	North	0.00	29.95	29.95	0.39	0.41	0.5
	and	54.40	77.25	22.85	0.42	0.43	0.8
TEP-11-068	North	52.50	93.50	41.00	0.37	0.74	1.1
TEP-11-072	North	0.00	76.00	76.00	0.59	0.77	1
TEP-11-075	North	0.00	140.70	140.70	0.36	0.87	1.4
	and	162.75	188.90	26.15	0.23	0.53	0.8
TEP-11-084	North	0.00	31.50	31.50	0.30	0.14	0.7
TEP-11-089	North	0.00	41.00	41.00	0.78	0.45	1.8

Hole No.	Zone	From (m)	To (m)	Length (m)	Cu (%)	Au (g/t)	Ag (g/t)
TEP-11-093	North	0.00	67.95	67.95	0.64	0.67	0.9
TEP-11-094	North	18.65	224.70	206.05	0.19	0.42	0.6
TEP-11-102	North	0.00	137.00	137.00	0.23	0.47	0.7
TEP-11-110	North	0.00	78.00	78.00	0.32	0.30	1.4
TEP-11-113	North	0.00	179.35	179.35	0.24	0.54	1.1
TEP-11-115	North	0.00	54.45	54.45	0.32	0.73	1.3
TEP-11-120	North	0.00	119.60	119.60	0.19	0.30	1.2
TEP-11-125	North	0.00	122.05	122.05	0.25	0.60	0.9
TEP-11-128	South	316.00	437.40	121.40	0.18	0.72	2.1
	including	318.00	401.00	83.00	0.20	0.89	2.3
TEP-11-130	South	149.75	253.70	103.95	0.12	0.22	2.5
	and	284.25	439.20	154.95	0.24	0.41	1.2
TIZ-11-003	Tizate	25.90	154.00	128.10	0.20	0.13	3.2
TIZ-11-006	Tizate	182.00	255.00	73.00	0.20	0.13	2.9
TIZ-11-007	Tizate	0.00	41.00	41.00	0.15	0.08	3.3
TIZ-11-011	Tizate	5.25	100.95	95.70	0.13	0.21	1.4
TIZ-11-013	Tizate	76.80	173.40	96.60	0.16	0.13	2.4
	and	218.00	320.00	102.00	0.22	0.14	4
TIZ-11-017	Tizate	60.40	301.04	240.65	0.20	0.18	2.3
TIZ-11-019	Tizate	87.00	148.55	61.55	0.18	0.15	1.3
TIZ-11-021	Tizate	123.90	229.00	105.10	0.20	0.16	1.5
TIZ-11-023	Tizate	0.00	97.75	97.75	0.20	0.17	1.4
TIZ-11-025	Tizate	6.00	106.80	100.80	0.19	0.08	1.2
TIZ-11-027	Tizate	0.00	42.00	42.00	0.16	0.15	1.4
TIZ-11-035	Tizate	0.00	63.00	63.00	0.24	0.27	5.1
TIZ-11-037	Tizate	0.00	63.10	63.10	0.20	0.23	3.9
TIZ-11-050	Tizate	0.00	85.00	85.00	0.18	0.34	1.7
TIZ-11-056	Tizate	0.00	92.15	92.15	0.31	0.21	1.8
TIZ-11-057	Tizate	0.00	107.90	107.90	0.17	0.21	2.5
TIZ-11-061	Tizate	0.00	140.65	140.65	0.19	0.26	1.9
TIZ-11-062	Tizate	4.00	230.05	226.05	0.15	0.32	1
TIZ-11-063	Tizate	52.20	193.60	141.40	0.21	0.19	2
TIZ-11-065	Tizate	5.15	238.00	232.85	0.14	0.32	1.2

Source: Geologix 2011 and 2012 news releases

There has been no additional drilling undertaken on the deposits (North Zone, South Zone and Tizate Zone) since 2011.

11 Sample Preparation, Analyses and Security

The following section is taken from Micon Technical report, March 29, 2012 which was modified from Murphy et al, 2011. A detailed sampling methodology and approach is documented in Murphy et. al. (2011).

11.1 Inco

Nothing is known of the sample preparation, analysis and security methods employed by INCO nor is it known whether INCO employed a quality control/quality assurance program.

11.2 Teck

Nothing is known of the security employed by Teck nor is it known whether Teck employed a full quality control/quality assurance program. Shonk (1994) indicates that every tenth sample submitted for analysis by Teck was a duplicate.

All samples collected by Teck were analyzed by ALS Chemex (ALS) in Vancouver. The analytical methods utilized by Teck for gold consisted of a standard fire assay followed by an atomic absorption finish. The method requires that a sample weighing about 30g weighed be mixed in a crucible with lead oxide, a reducing agent and fluxes. The sample is then fired in a furnace. In the furnace the complete content of the crucible is melted. After cooling, the metallic lead button" at the bottom of the mold is separated from the glassy slag which is discarded.

The metallic lead button is placed into a cupel and placed into a cupelling furnace. In the "cupelling" process lead metal turns back into oxide which volatilizes away from the precious metals and soaks into the bone ash cupel, leaving the minute amount of precious metals as a metallic speck of metal called a "bead" on the bottom of the cupel.

The bead of precious metals that is recovered in the cupel after the lead has been removed is dissolved in aqua regia. The resulting solution is then analyzed by atomic absorption spectrometry, allowing the grade of gold and silver in the original sample to be back calculated. High grade samples were re-analyzed using fire assay with a gravimetric finish.

Teck assayed all samples for copper using an aqua regia digestion followed by ICP analysis. Samples collected from the oxide were analyzed for non-sulphide copper minerals by digestion in dilute sulfuric acid and AA finish.

Micon is not aware of the certification ALS had in the mid-1990. Currently, ALS laboratories in North America are certified with ISO 9001:2000 for the "provision of assay and geochemical analytical services" by QMI Quality Registrars. In addition to ISO 9001:2000 registration, the ALS Vancouver laboratory has received ISO 17025 accreditation from the Standards Council of Canada under CAN-P-1579 "Guidelines for Accreditation of Mineral Analysis Testing Laboratories". They also have CAN-P-1579 which is the Amplification and Interpretation of CAN-P-4D "General Requirements for the Accreditation of Calibration and Testing Laboratories" (Standards Council of Canada ISO/IEC 17025). "Geologix carried out a limited check program of the Teck drill core in

2010. A total of 234 pulps were re-assayed at ALS in Vancouver. The re-assay program results corroborate with the original assay results.

11.3 Hecla

Nothing is known of the sample preparation, analysis and security methods employed by Hecla nor is it known whether Hecla employed a quality control/quality assurance program.

All samples were analyzed by ALS Vancouver. Gold content was determined by fire assay with an atomic adsorption finish following similar procedures to the Teck analyses discussed above. Copper and 30 other elements were determined by ICP.

11.4 Arian

Arian geologists typically used 2m sample intervals within the mineralized zones apart from where broken ground and/or specific geological conditions determine otherwise.

Sampling intervals ranged from 0.25m to 5.95m (which represents an inter zone waste composite sample), with most intervals in the 1.5m to 2m range.

Core was transported from site to the processing facility, in Tepalcatapec, 15 km northeast of the Tepal Project. In the warehouse, the areas of core that had been marked for sampling were cut in half using a diamond-bladed core-saw. One half of the core was replaced into the core-box, and the other half was bagged. Inside the bags were placed sample tickets with a unique sample ID number, and the same sample number was written on the outside of the plastic bag with permanent markers. The bag was then sealed on site.

After the core has been logged and photographed, all information was entered into an Access Database (Booth, 2007b). The samples (in groups of 10 samples) are placed inside nylon rice-bags and sealed with a cable-tie to prevent access. There were 3,532 samples of NQ size. Samples were sent to Inspectorate Labratories in Durango, Durango State, Mexico for sample preparation and the pulps were then shipped to Inspectorate Labratories in Reno Nevada USA for analysis.

Sampling issues were identified by ACA Howe. CRMs that were assayed at Inspectorate Labs using the 3 acid digestion and ICP finish method returned copper results that were generally erratic and higher than expected.

To remedy this, a full review of Inspectorate analytical techniques was undertaken. It was recognized through this review that sample preparation for the 3 acid digestion and ICP finish method was inadequate. Based on these findings it was agreed that re-analysis for copper and gold for all Phase 1 holes must be undertaken, using the more reliable method of Aqua Regia digest with Atomic Adsorption finish.

Once re-analysis was complete, the CRM and duplicate results were greatly improved for gold and were presented in the April 2008 report. It was found that the gold re-assay results undertaken at Inspectorate were sufficient to be, on the whole, suitable for confident use in resource estimation.

Copper control results remained poor and it was agreed that all Phase 1 assays would have to be re-analyzed by ALS Chemex Laboratories Canada. To ensure an adequate level of confidence in assay results for use in resource estimation the majority of samples beyond Sample 143422, hole AS-07-023, were sent to ALS Chemex for gold and copper analysis in place of Inspectorate Labs. The sampling preparation and analytical methods employed by each lab are presented in the following sections.

11.4.1 Inspectorate Labs

Samples sent to Inspectorate Labs for analysis, were collected from Arian's warehouse every two weeks by Inspectorate personnel, who transported the samples to their preparation facility in Durango, Durango State, Mexico.

The entire half-core was crushed to 75% passing 2mm followed by the pulverization of a 150g split in a chromium steel crusher to 85% passing 75 microns. The pulp samples were then air freighted to Inspectorate's analytical laboratories in Reno, Nevada, for analysis.

Gold analysis for samples below 3ppm Au used an Aqua Regia digestion with an AAS finish (Detection range was 0.005 to 10ppm Au). Samples over 3ppm Au used the fire assay method with a gravimetric finish (Detection range was 0.005 to 100ppm Au).

Copper analysis used an Aqua Regia digestion with an AAS finish (Detection range was 0.2 to 10,000ppm Cu).

11.4.2 ALS Chemex Labs

Samples analyzed by ALS were collected from Arian's warehouse and transported the samples to ALS's sample preparation facility in Guadalajara, Jalisco State, Mexico. It is uncertain whether ALS personnel collected the samples at Arian's warehouse or whether the samples were couriered via a private company.

Once the samples were received by ALS, the entire half-core was crushed and pulverized to 85% passing 75 microns. The pulps are then air freighted to the ALS analytical laboratories in Vancouver, Canada, for analysis.

Gold analysis for samples below 3ppm Au used an Aqua Regia digestion with an AAS finish (Detection range was 0.005 to 10ppm Au). Samples over 3ppm Au used the fire assay method with a gravimetric finish (Detection range was 0.005 to 100ppm Au).

Copper analysis for samples below 10,000ppm Cu used a 3 acid digestion with an ICP analysis (Detection range was 0.2 to 10,000ppm Cu). Samples over 10,000ppm Cu used an Aqua Regia digestion with an AAS finish (Detection range was 0.01 to 3% Cu).

Results were received from the labs via email and hardcopy certificate. For each laboratory used, the sample dispatch routines, security, preparation and analysis are considered consistent with satisfactory working practices for this type of deposit and type of exploration work.

Micon believes that the appropriate steps were taken to identify and re-assay the samples. Micon feels that the resulting Arian assays presented by Geologix are appropriate for use in a mineral resource estimate.

11.5 Geologix

Geologix geologists typically used 2m sample intervals within the mineralized zones apart from where broken ground and/or specific geological conditions determine otherwise. Sampling intervals ranged from 0.25m to 5.95m (which represents an inter zone waste composite sample), with most intervals in the 1.5m to 2m range.

In 2010, core was transported from site to the processing facility, housed in the grounds of the house that the company currently occupies in Tepalcatapec, 15kms northeast of the Tepal Project. In the warehouse, the areas of core that had been marked for sampling were cut in half using a diamond-bladed core-saw. One half of the core was replaced into the core-box, and the other half was bagged. Inside the bags were placed sample tickets with a unique sample number and the same sample number was written on the outside of the respective bag. Each bag was then sealed on site. The sample bags in groups of ten were placed inside nylon rice-bags and sealed with a cable-tie to prevent access.

In 2011, Geologix built a new covered core logging facility and secure storage area within the new exploration camp facilities on the Tepal property, south of the South Zone. The identical sample procedure was used at this new facility as the old one. The facility is surrounded by a high wire mesh fence which is locked and secure. The rock saws have been moved from town and are housed beside the logging facility.

A QA/QC program was implemented to ensure all core and sample handling procedures were in accordance with the best possible practices. The assay protocol included the insertion of standards, blanks and duplicates into the sample stream on an average basis of one standard, one blank, and one duplicate sample for every 30 samples. At no time after this the rice bags were seal, were the samples handled by Geologix personnel or contractors working for Geologix.

After the core has been logged and photographed, all information was entered into a Microsoft Access Database.

Samples were analyzed by ALS Chemex. They were collected from Geologix's warehouse and transported to ALS Chemex's sample preparation facility in Guadalajara, Jalisco State. The analytical work was completed at ALS Chemex's laboratory facilities in North Vancouver, B.C.

All samples were assayed for gold by Aqua Regia digest with AAS finish on a 30g sample and by ICP-AES for 33 elements, including copper, using a four acid "near total" digestion. High grade gold (>10.0g/t) samples were re-analyzed using fire assay with a gravimetric finish. High grade (>10,000ppm) copper samples were re-analyzed on a single element basis using an ore grade 4 acid digestion with ICP-AES finish.

Results were received from the lab via email along with hardcopy certificates.

ALS Chemex (ALS Minerals) is an ISO 9001 and ISO 17025:2005 accredited facility. Micon believes that the sampling, transportation, preparation and analysis are considered consistent with exploration best practices for this type of deposit and is acceptable for use mineral resource estimation.

12 Data Verification

The following section is taken from Micon Technical report, March 29, 2012 which was modified from Murphy et al, 2011. It is unknown what data verification was undertaken with INCO, Teck and Hecla sample results.

12.1 Arian

A quality assurance and quality control (QA/QC) program was implemented during the 2007 and 2008 drilling campaign at Tepal, in an attempt to provide adequate confidence that sample and assay data could be used in resource estimation.

An assessment of QA/QC samples submitted to Inspectorate laboratories was completed (White, 2008, 2009). Inspectorate gold results were sufficient to be, on the whole, confident in assay precision and accuracy.

The review of sampling and assaying procedures indicates that an adequate system was in place to maximize the quality of drill hole samples and to assess the reliability, accuracy and precision of subsequent assay data for use in resource estimation.

The QA/QC program consisted of:

- The inclusion of Certified Reference Material standards (CRM's) in sample batches sent to both Inspectorate and ALS laboratories, to assess analytical accuracy (4 per 100 samples).
- The inclusion of field blanks and pulp blanks to assess laboratory sample preparation and analytical accuracy (3 per 100 samples).
- The inclusion of field duplicates and externally assayed pulp duplicates to assess sample preparation and precision (3 per 100 samples).

12.1.1 Certified Reference Material

Certified Reference Material (CRM) samples were prepared from mineral matrices that contain gold and copper values similar to the grade of the Tepal deposit, which are uniformly distributed throughout the pulverized rock. CRM samples were routinely submitted for assaying with core at a ratio of up to 1:60, totalling 2% of all samples. Three CRM's were used CU139 (low grade) and CU150 and OX14 (higher grades) (see Table 12-1). The CRM's were prepared by WCM Minerals, Burnaby, British Columbia and Rock Labs, New Zealand.

Table 12-1: Arian CRM Statistics

CRM	Recommended Values		Standard Deviation	
	Au (ppm)	**Cu (%)**	**Au (ppm)**	**Cu (%)**
CU139	0.55	0.43	0.031	0.007
CU150	0.79	0.59	0.033	0.012
Ox14	1.22	NA	0.057	NA

A detail of Arian's CRM plots is available from Murphy et. al. (2011) for gold and copper.

Field blanks were prepared from samples of un-mineralized Tonalite taken from a quarry near Arian's San Jose property and submitted along with the core samples. All Pulp Blanks were prepared from the un-mineralized Tonalite at the Inspectorate Laboratories sample preparation facility.

12.1.2 Blanks

Blanks were typically inserted at the end of an expected high grade run, after vein intersections that contained significant sulphides. Blanks were inserted with core samples at a ratio of 1:54 and totalled 2% of all samples. A total of 144 blanks were submitted including 33 Field Blanks and 33 Pulp Blanks.

Gold grades in Field Blanks submitted to ALS showed that only 3 results returned values marginally greater than the lower limit of detection 0.5ppm Au and were well within tolerance limits, returning values of up to 0.009ppm Au. Copper grades in Field Blanks were on the whole acceptable with 67% returning values below 1 standard deviation of 0.002% Cu based on all samples. There were two copper outliers of 0.007% and 0.008% however these were considered insignificant and within tolerance limits.

As part of the Phase 1 quality control sample resubmission 33 pulp blanks, prepared by Inspectorate, were submitted for reanalysis. Gold grades for Pulp Blanks showed that 67% of returned grades were below the limit of detection. Of the remaining samples 8 returned values greater than 0.01ppm Au, including one outlier, sample 145521 at 0.08ppm Au. Copper values were much more variable with only 52% returning values below 1 standard deviation of 0.007% Cu based on all samples, with the majority of samples returning grades of 0.009% Cu. There was one outlier, again sample 145521, which returned a grade of 0.04% which is considered beyond acceptable limits.

On the whole the results of Blank Sample Analysis are acceptable; however there were some anomalous assays for both field and pulp Blanks. Field Blanks were acceptable indicating that there were no significant contamination issues in field sample preparation. Pulp samples demonstrate limited but significant values over acceptable limits for gold and copper, indicating a potential error in the numbering of sample 145521 or contamination during sample preparation. This anomalous value should be investigated.

12.1.3 Duplicates

Sixty-nine (69) duplicate samples were re-analyzed and compared, accounting for 2% of all samples.

Duplicates were either obtained from a Coarse Reject sample comprising a 1 kg or 25% split taken from a randomly selected coarse reject sample that had been returned from Inspectorate or from a Pulp Reject sample comprising a 100 gram sample taken from a randomly selected pulp reject sample that had been returned from Inspectorate after analysis.

There was a good correlation for pulp and coarse reject duplicates for gold, indicated by the correlation coefficients of 0.9319 and 0.9717 respectively. There is good level of precision between original assays and duplicate assays. 44% of gold duplicate assays were within 10% of the original assay value.

A lesser level of precision between original and duplicate assays was shown for the copper analysis. There appears to be some significant overestimating of coarse duplicates particularly at higher grades with one anomaly indicating a 102% difference in copper grade. The sample has been flagged for reassessment. Correlation coefficients of 0.8112 and 0.867 indicate a reasonable level of precision.

12.1.4 Historic Duplicates

Arian undertook a program of historical pulp duplicate re-analysis on available pulp samples to verify historical drill sample assay results. Pulps were available for a number of Teck and Hecla drill holes.

Pulp duplicate assessment shows repeatability of historical Au assay data is reasonable with correlation coefficients of 0.94 and 0.91 for Teck and Hecla samples respectively. Pulp duplicate assessment of Cu values returned equally satisfactory correlation coefficient values of 0.93 and 0.98 respectively.

As part of the Phase 1 diamond drill program Arian also twinned a number of historical drill holes for data verification purposes. Identification of twin holes by Arian was done by reference to historical collar co-ordinates in the historical database.

Arian was unable to locate evidence on the ground to confirm the accurate location of all but one of the INCO drill holes (IN-57002). Lack of evidence for the INCO drilling on the ground suggests co-ordinates for the INCO drilling listed in the historical database are incorrect. Due to the inability to accurately locate and verify the INCO hole data, these have been removed from the data verification assessment and subsequent resource study.

Arian geologists indicated poor correlation between Arian diamond drill hole results and historical Hecla RC drill grades. The 'average' difference for Au was 19% and for copper 16% (with maximums of 72% and 142% respectively). For this reason, the historic assay results provided by Hecla were deemed inaccurate and therefore removed from the Tepal database.

12.2 Geologix

Geologix established a QA/QC program for all of its drilling at Tepal and Tizate in an attempt to provide adequate confidence that sample and assay data could be used in resource estimation. Procedural documentation pertaining to sample collection, field preparation, sample dispatch, assay lab sample preparation, sample analysis and collation of assay results was presented and reviewed prior to resource estimation.

The review of sampling and assaying procedures indicates that an adequate system is in place to maximize the quality of drill hole samples and to assess the reliability, accuracy and precision of subsequent assay data for use in resource estimation.

The QA/QC program consisted of:

- The inclusion of Certified Reference Material standards (CRM's) in sample batches sent to ALS to assess analytical accuracy (1 per 30 samples).
- The inclusion of field blanks and pulp blanks to assess laboratory sample preparation and analytical accuracy (1 per 30 samples).
- The inclusion of field duplicates and externally assayed pulp duplicates to assess sample preparation and precision (1 per 30 samples).

Approximately 20% of all samples submitted to the laboratory were quality control samples.

12.2.1 CRM

Certified Reference Material samples were prepared from mineral matrices that contain gold and copper values similar to the grade of the Tepal deposit, which are uniformly distributed throughout the pulverized rock. Standard statistical techniques were used to assign a recommended assay value with associated 95% confidence interval (Table 12-2). CRM's were prepared by CND Laboratories Langley, British Columbia and Ore Research and Exploration Pty Ltd. of Australia.

Table 12-2: Geologix CRM Statistics

CRM	Recommended Values		3 Standard Deviations		Failures	
	Au (ppm)	Cu (%)	Au (ppm)	Cu (%)	Au	Cu
CDNCGS-21	0.99	1.3	0.265	0.252	2	0
CDNCGS-23	0.218	0.182	0.108	0.03	3	3
Oreas 50Pb	0.841	0.744	0.19	0.126	1	3
Oreas 52Pb	0.307	0.334	0.104	0.046	0	2
Oreas 53Pb	0.623	0.546	0.128	0.081	2	6
Oreas 52c	0.346	0.344	0.100	0.057	2	7
Oreas 151a	0.043	0.166	0.014	0.031	2	5
Oreas 152a	0.116	0.385	0.03	0.057	5	15
Oreas 153a	0.311	0.712	0.069	0.151	2	1

CRM samples were routinely submitted for assaying with core at a ratio of up to 1:30, totalling 4% of all samples. Initial drilling utilized CDNCGS-21,CDNCGS-23, 50pb and 52pb while the 2011 used 52c, 151a, 152a and 153a. Error plots for each CRM for gold and copper are presented in the following pages (Figures 12-1 to 12-18). Failures are identified as yellow squares in each plot.

Figure 12-1: CRM - CDN-CGS-21 - Au Values

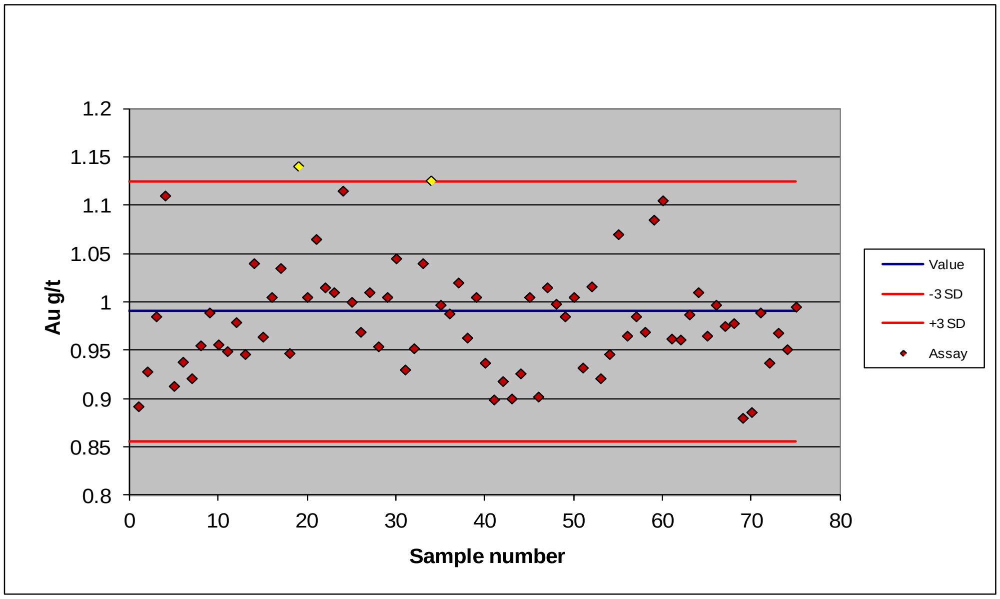


Figure 12-2: CRM - CDN-CGS-21 - Cu Values

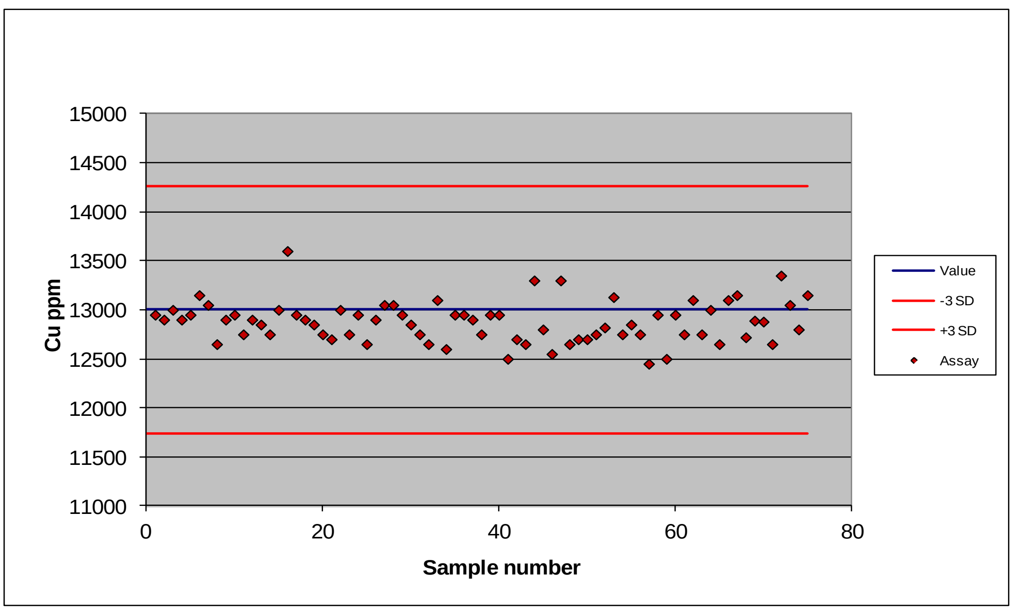


Figure 12-3: CRM - CDN-CGS-23 - Au Values

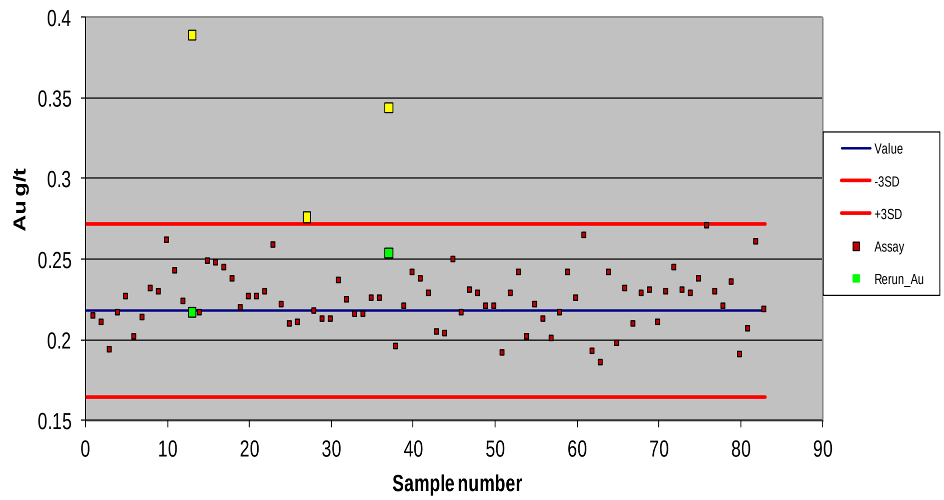


Figure 12-4: CRM - CDN-CGS-23 - Cu Values

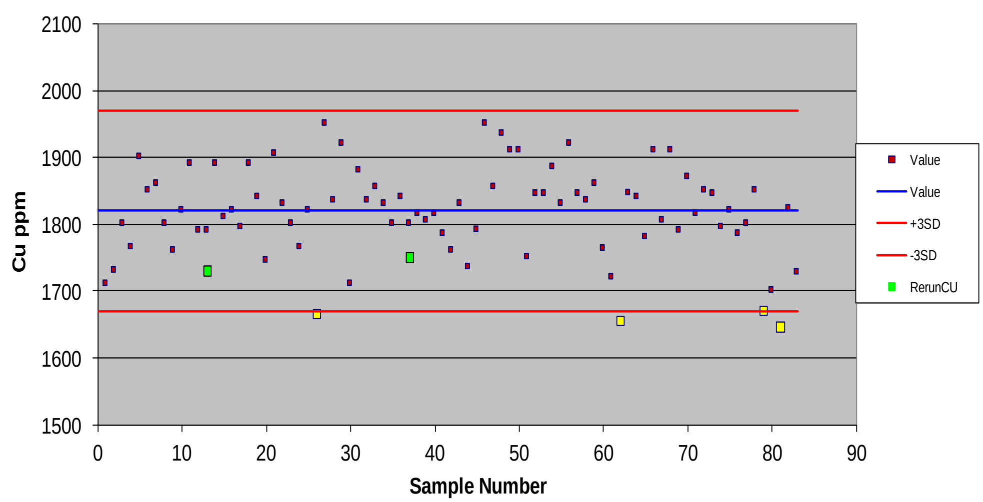


Figure 12-5: CRM - Oreas-50Pb - Au Values

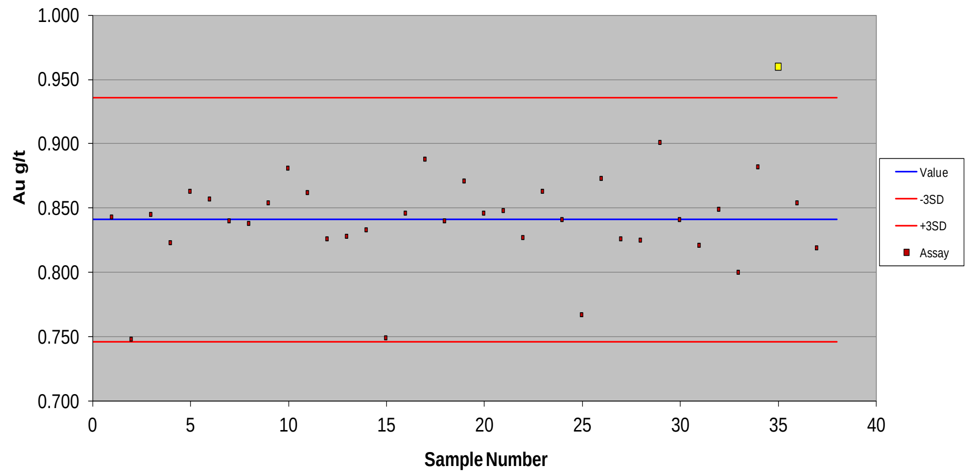


Figure 12-6: CRM - Oreas-50Pb - Cu Values

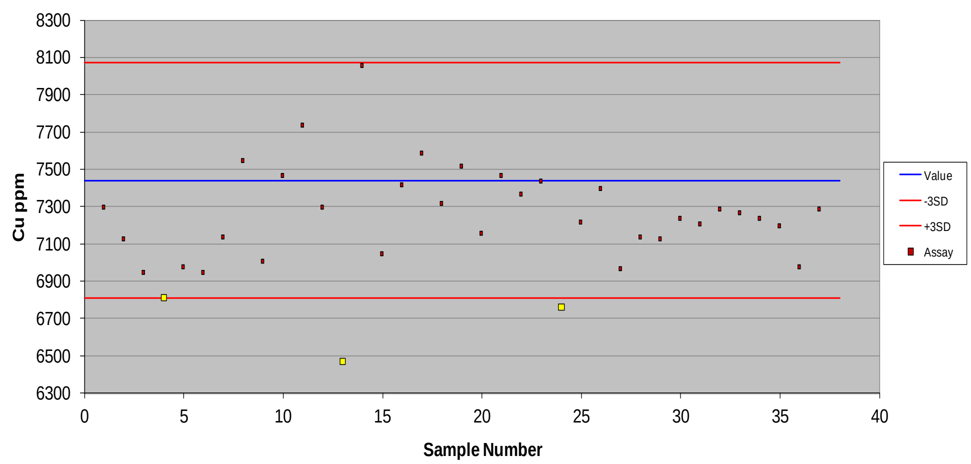


Figure 12-7: CRM - Oreas-52Pb - Au Values

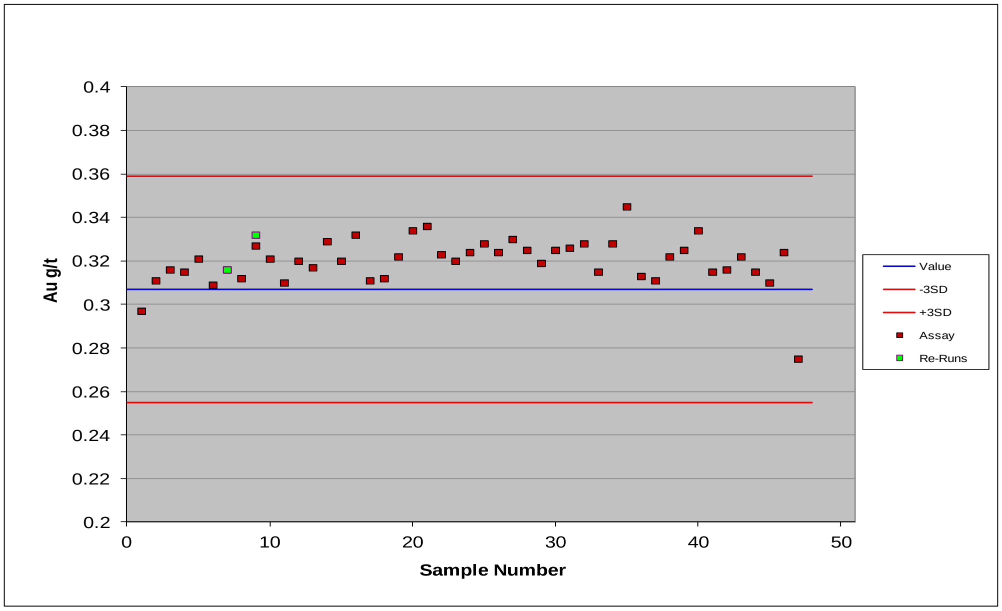


Figure 12-8: CRM - Oreas-52Pb - Cu Values

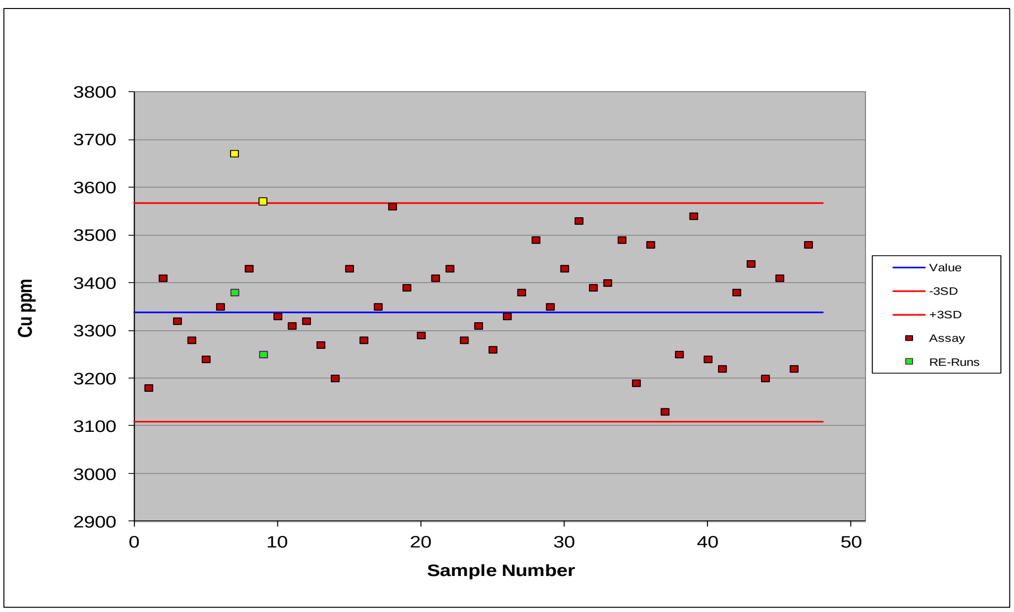


Figure 12-9: CRM - Oreas-53Pb - Au Values

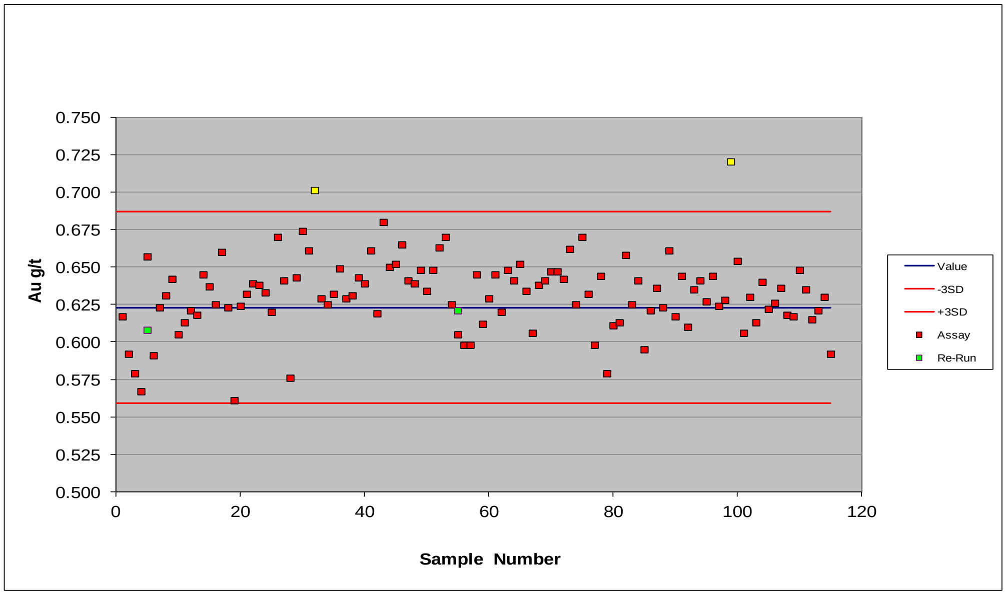


Figure 12-10: CRM - Oreas-53Pb - Cu Values

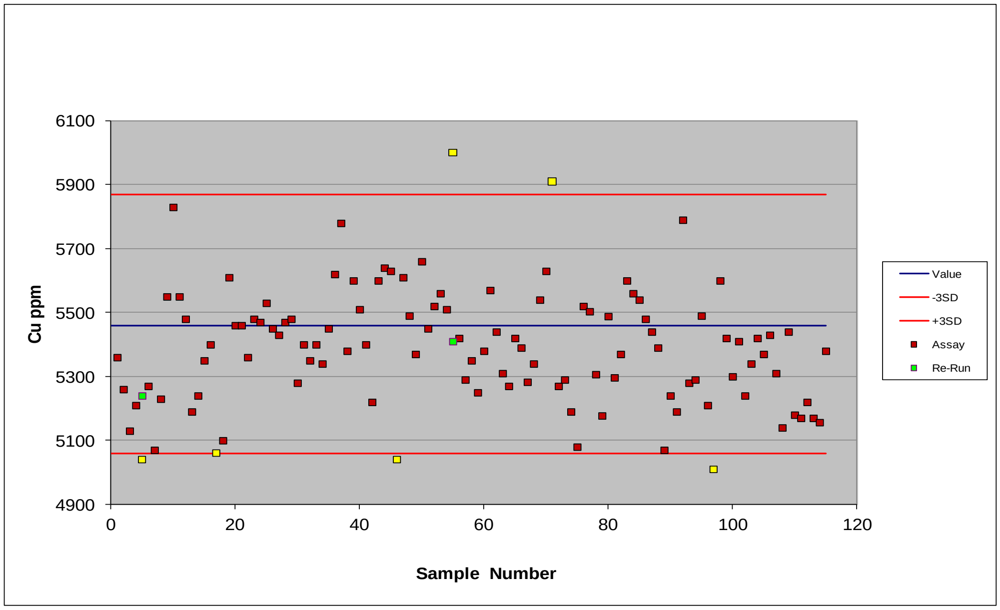


Figure 12-11: CRM - Oreas-52c - Au Values

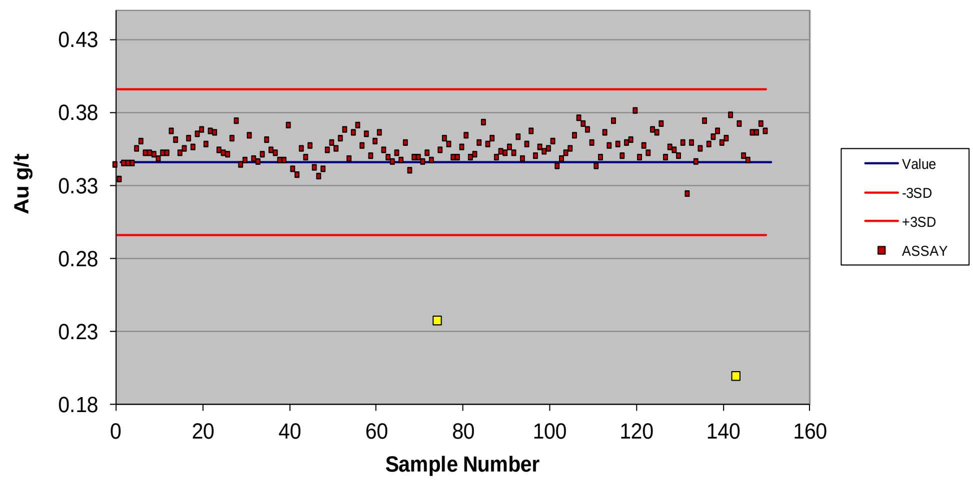


Figure 12-12: CRM - Oreas-52c - Cu Values

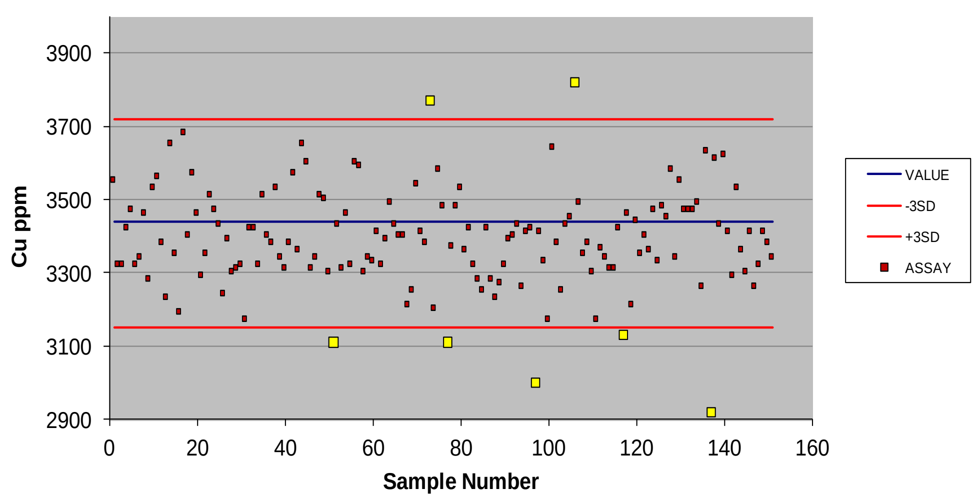


Figure 12-13: CRM - Oreas-151a - Au Values

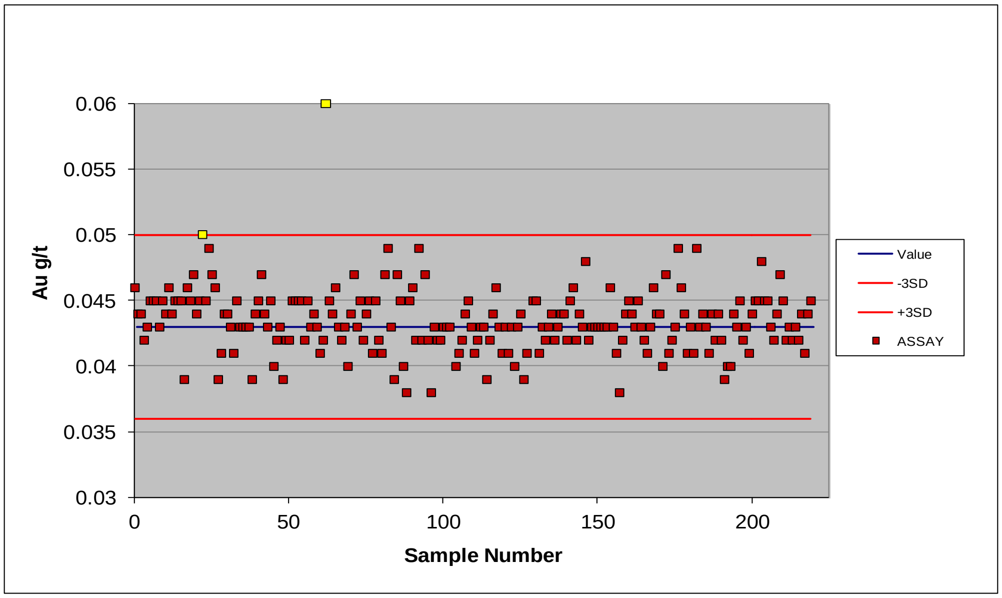


Figure 12-14: CRM - Oreas-151a - Cu Values

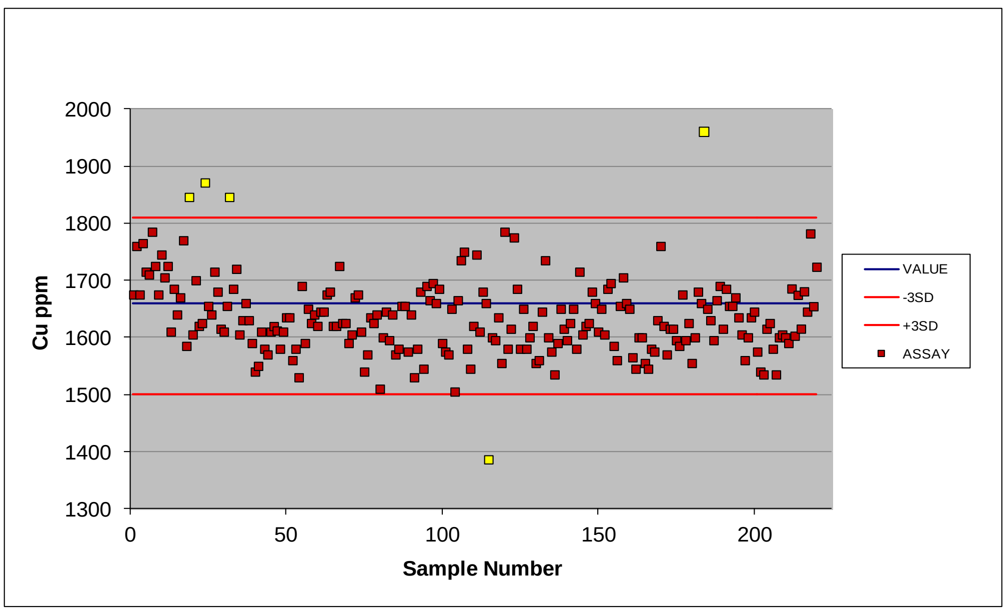


Figure 12-15: CRM - Oreas-152a - Cu Values

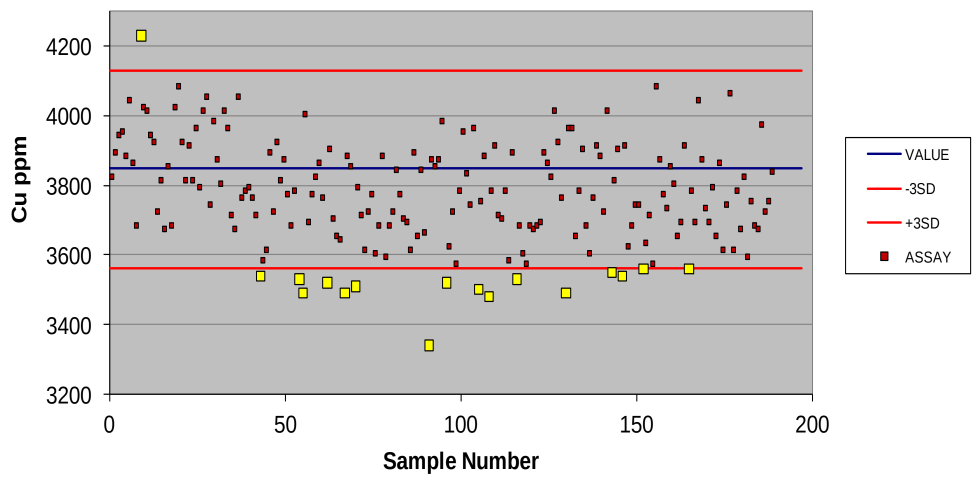


Figure 12-16: CRM - Oreas-152a - Au Values

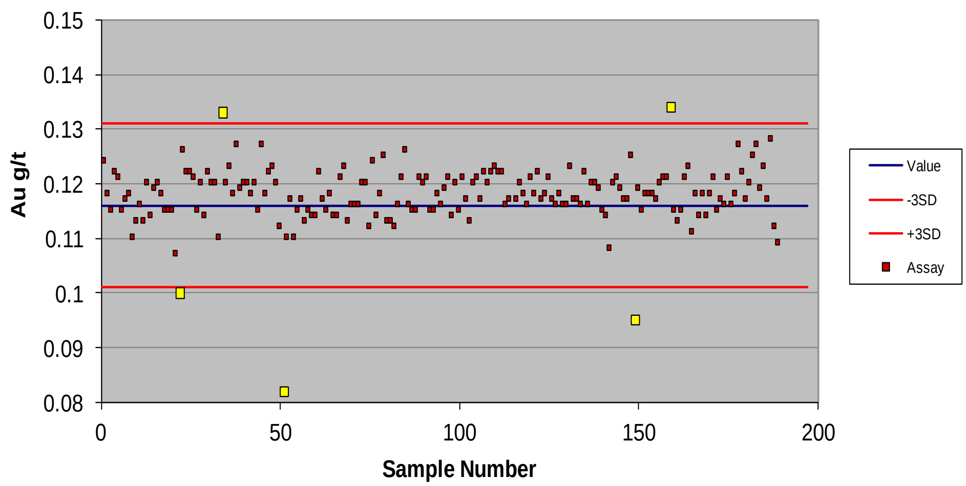


Figure 12-17: CRM - Oreas-153a - Au Values

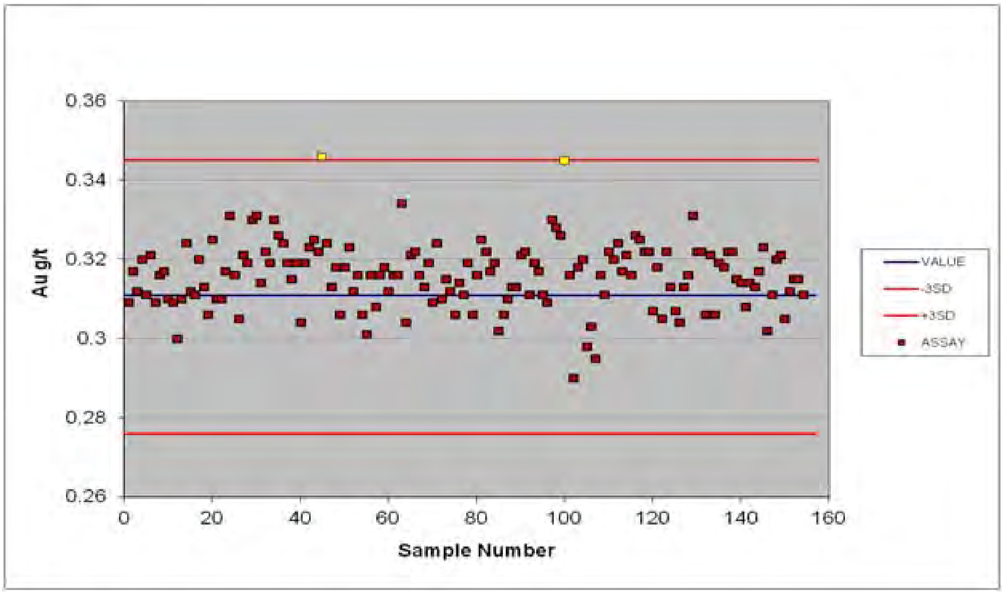
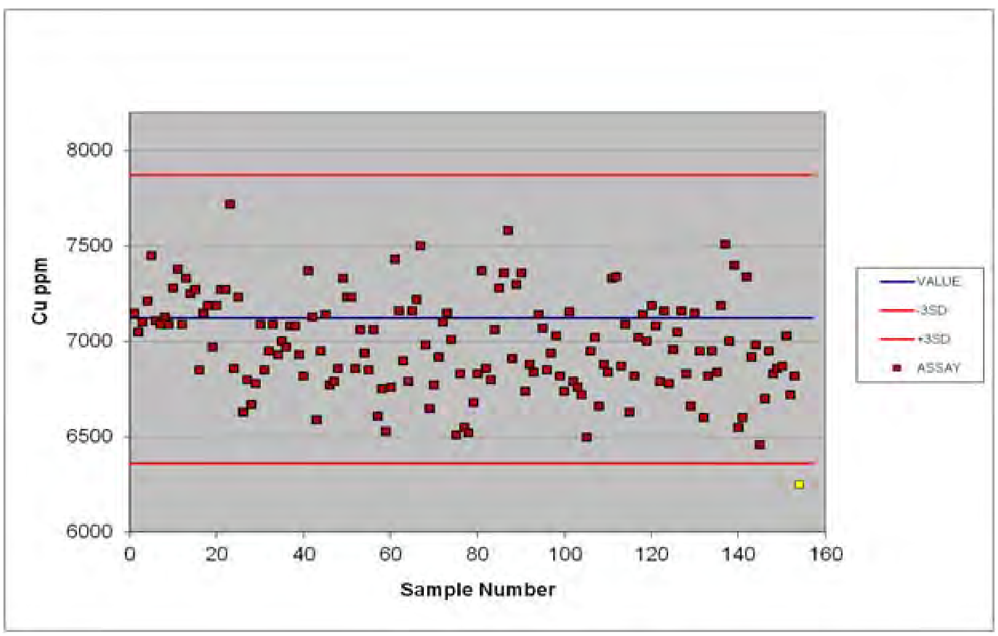


Figure 12-18: CRM - Oreas-153a - Cu Values



Most of the CRM for both gold and copper fall well within the ±2 Std.Dev. of the expected value. Of the failed CRMs (±3 Std.Dev.), there were a total of 733 samples that were associated with the failed

CRMs.Out of that total, there were 377 samples within the mineralized zones and 356 samples considered waste. These samples have been sent for re-assay. Assay results from roughly two-thirds of the samples have shown little change in their respective original assays. The re-assay data were entered in the database.

In general, submitted standard samples showed good repeatability for both copper and gold at both low and high grades. Standards CGS-23, 52Pb, 53Pb, 52c, 152a and 153a seem to consistently report above the expected value for gold but well within the accepted value for each of the standards. Standard CGS-23 also seems to consistently report above the expected value for copper. Standards 52c and 153a seem to have a very narrow range for gold while CGS-21 to have a very narrow range for copper but well within the accepted value for each of the standards.

New or fresh CRMs may alleviate the random but minor failed CRM assays. Micon believes that the procedures in place for CRM are to industry standards and that the resultant assays reflect the mineralization within the deposits.

12.3 Blanks

Blanks monitor the calibration of analytical equipment and potential sample contamination during sample handling and preparation. Blanks were inserted with core samples at a ratio of approximately 1:30.

Blanks were obtained from two locations within the concessions but away from the known deposits (Location 1: 720954 E, 2115284 N and Location 2: 719423 E, 2115012 N). The blanks were identified as non-mineralized porphyritic andesite and non-mineralized granodiorite.

Figure 12-19 : Blank – Analyses Au (g/t)

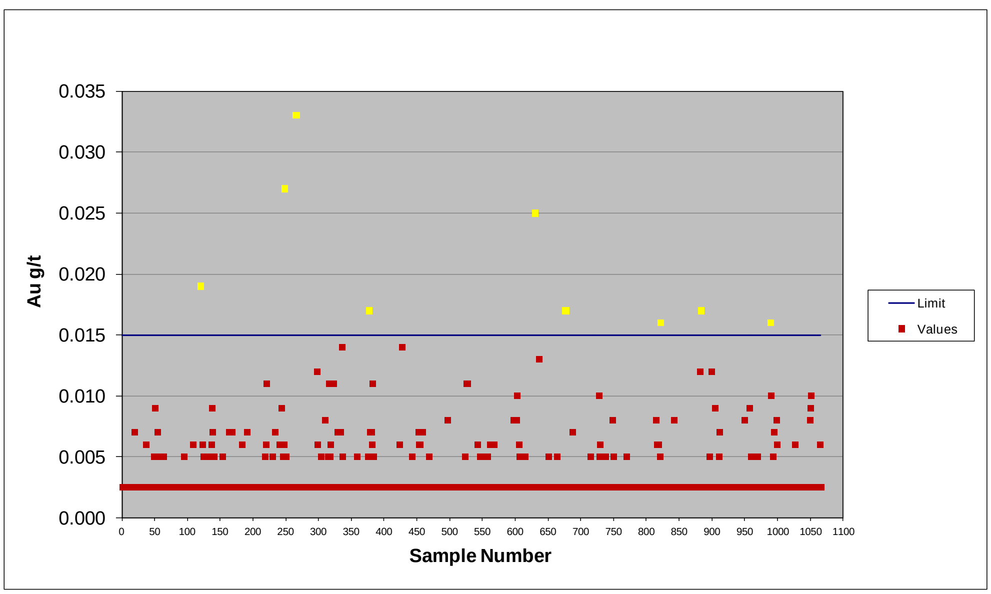
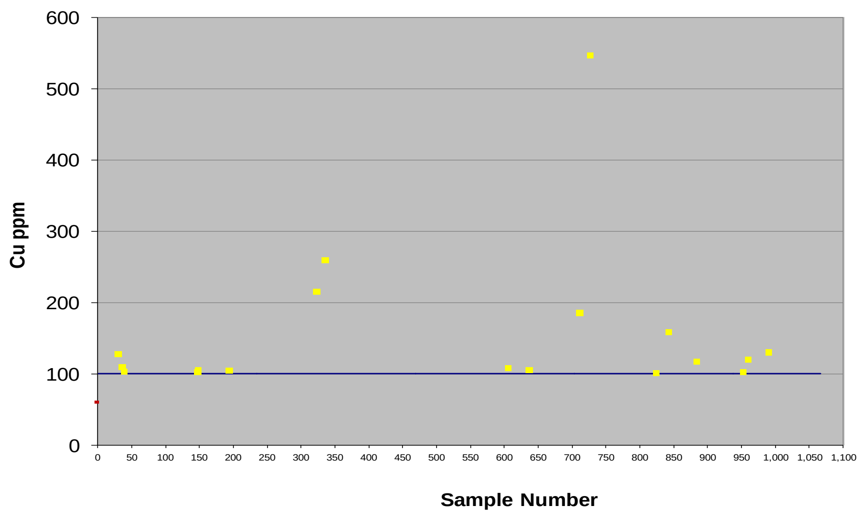


Figure 12-20: Blank - Analyses Cu (ppm)



There were 1067 blank samples inserted into the sample stream. The following figures illustrate the results for gold and copper. Table 12-3 documents the outliers with respect to gold and copper.

Table 12-3: Blank Failures

Outliers	Percentage (%)
11	1.03
18	1.69

Micon believes that in general the results of Blank Sample Analysis are acceptable indicating that there are no significant contamination issues in field sample preparation. However, Micon believes that a certified blank should be used to detect sample preparation cross- contamination. The use of local lithologies for a source of blanks can be misleading if the material is at all mineralized. Local material should initially be thoroughly analyzed before being used as a blank.

12.4 Duplicates

There were 1048 duplicate core samples assayed in the sample stream. Duplicates samples were prepared by sawing the core in half and sending both halves of the core for assay. Assays were part of the ALS sample stream. There is a very good correlation for both gold and copper for the duplicate assays from coarse reject (Figure 12-21 and Figure 12-22). There is good level of precision between original assays and duplicate.

Figure 12-21 : Tepal Core Duplicates - Au

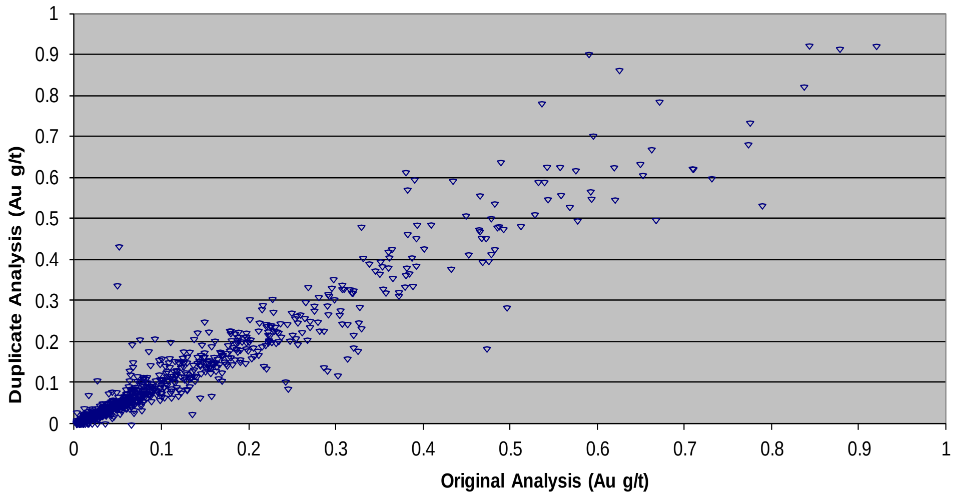
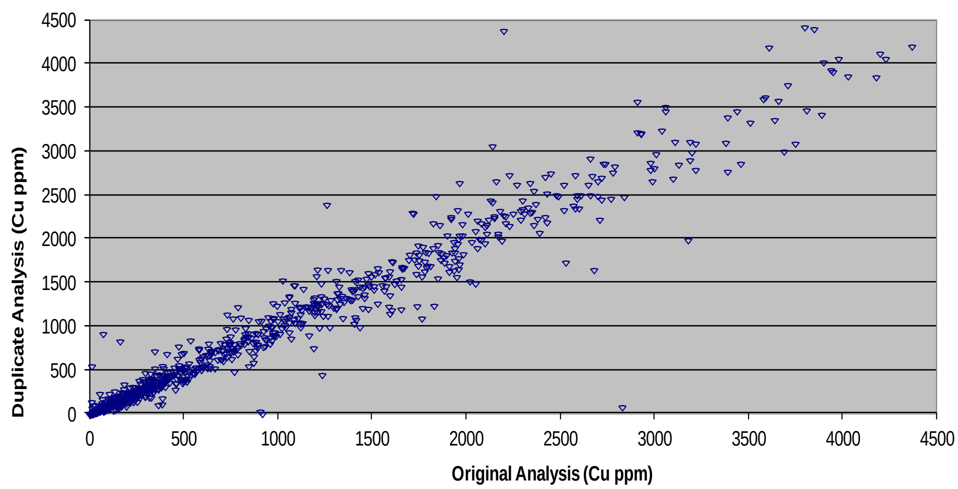


Figure 12-22: Tepal Core Duplicates - Cu



12.5 Check Assays

Geologix selected 603 samples for re-assay to Acme Analytical Laboratories as a check on the primary laboratory. Samples were selected from pulp rejects from ALS and forwarded to ACME for re-assay.

ACME is a well-recognised laboratory based in Vancouver. The laboratory maintains ISO 9001:2000 and has been approved for ISO/IEC 17025:2005 accreditation.

The results from the pulp re-assay program for gold, copper, silver and molybdenum are illustrated in Figures 12-23 to 12-26 respectively. The results seem to indicate that ALS is reporting slightly higher than ACME for silver. Values for gold, copper and molybdenum appear to correlate very well between the original lab and Acme labs.

Figure 12-23: Gold Check Assays

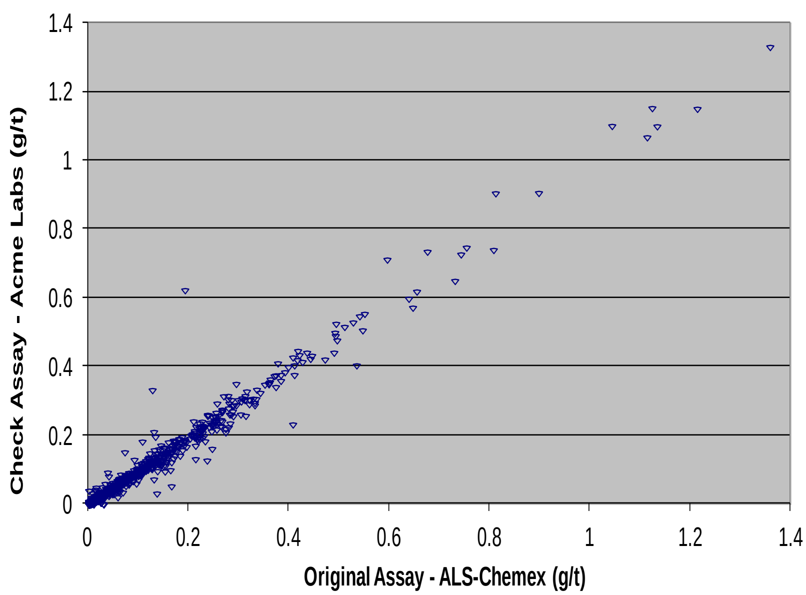


Figure 12-24: Copper Check Assays

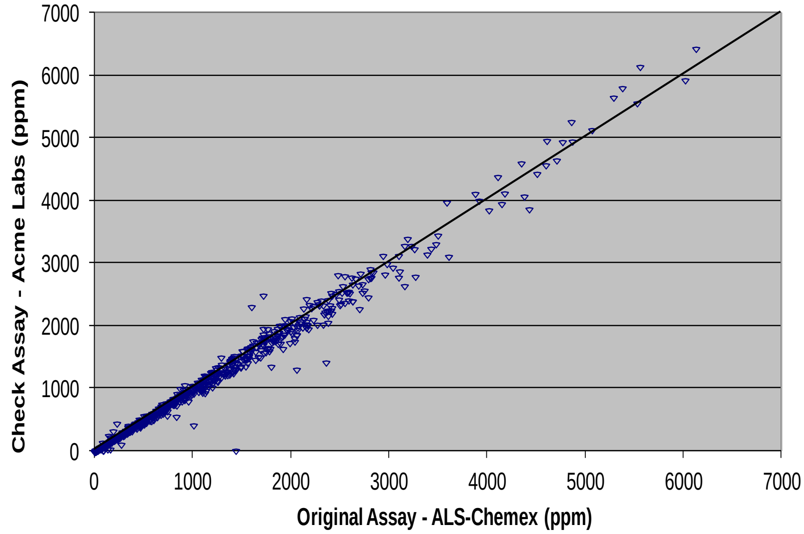


Figure 12-25: Silver Check Assays

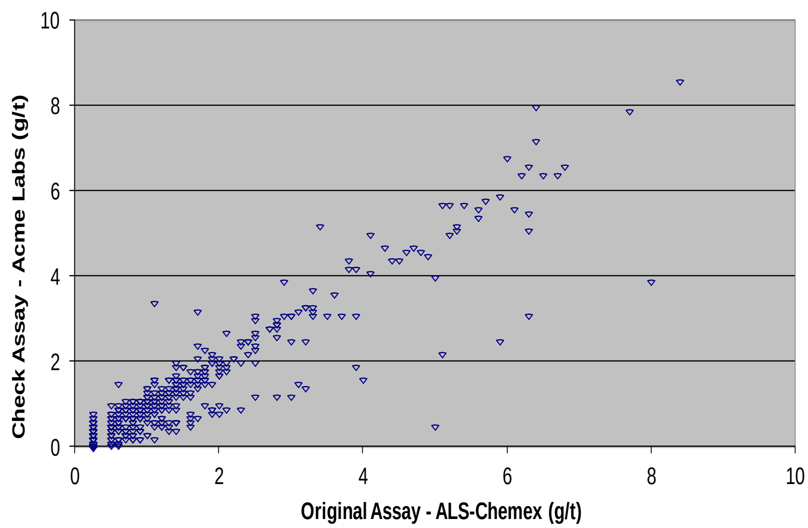
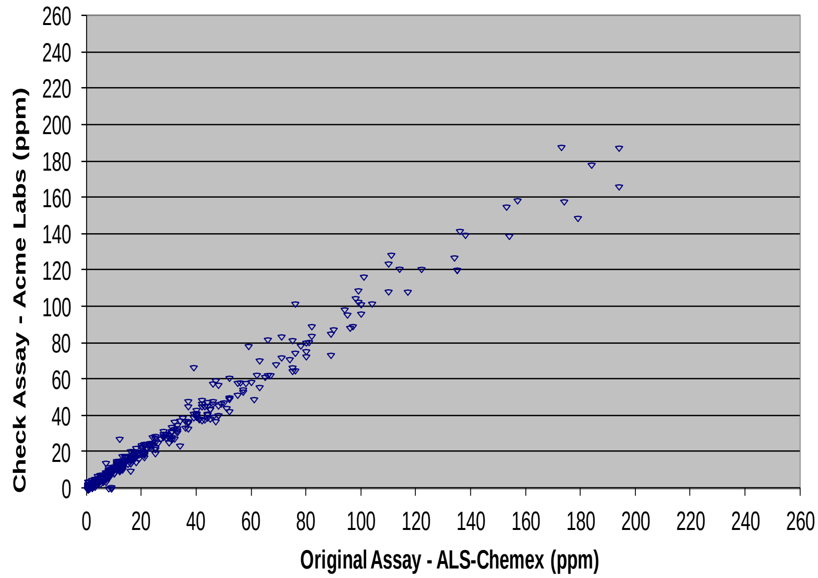


Figure 12-26: Molybdenum Check Assays



12.6 Historic Check Assays

Geologix undertook a program of historical pulp duplicate re-analysis on available pulp samples to verify historical drill sample assay results. A total of 103 Hecla pulps were selected and sent for re-assay. The Hecla pulp re-assays were carried by ACME laboratory. Figure 12-27 and 12-28 illustrate the comparison of the Hecla check assays.

There were 1,688 Teck pulps that were selected and sent for re-assay. The Teck re assays were carried out by ALS laboratories. Figure 12-29 and 12-30 illustrate the comparison of the Teck check assays.

Results of the re-assay program returned very similar results to the original data entered in the database for the historical drill holes in most cases. There was a wider scatter of Teck gold values than Teck copper values. As the grades increased especially for gold there was some scatter of data, but this is to be expected due to possible nugget effect.

Figure 12-27: Historic Hecla Gold Check Assays

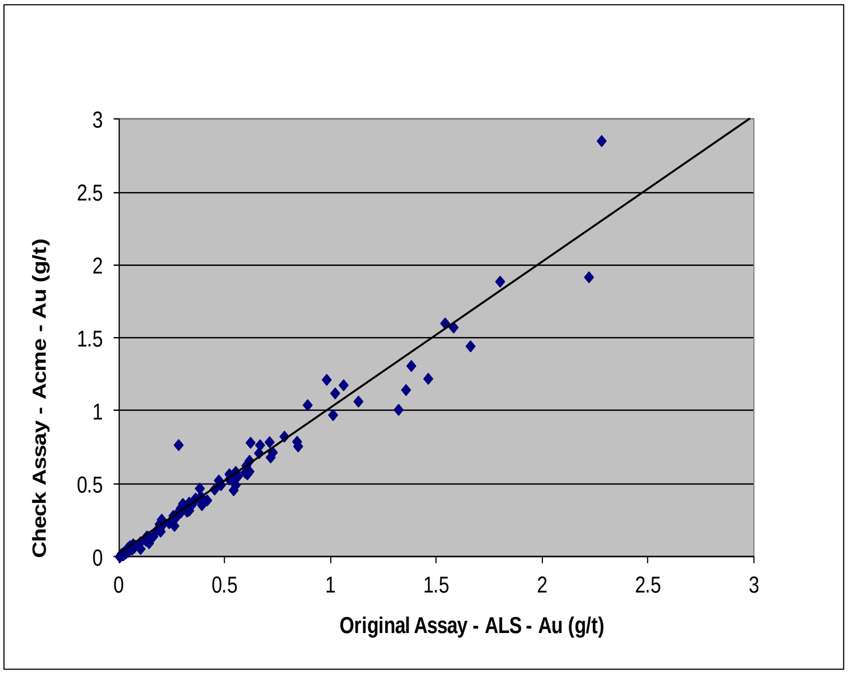


Figure 12-28: Historic Hecla Copper Check Assays

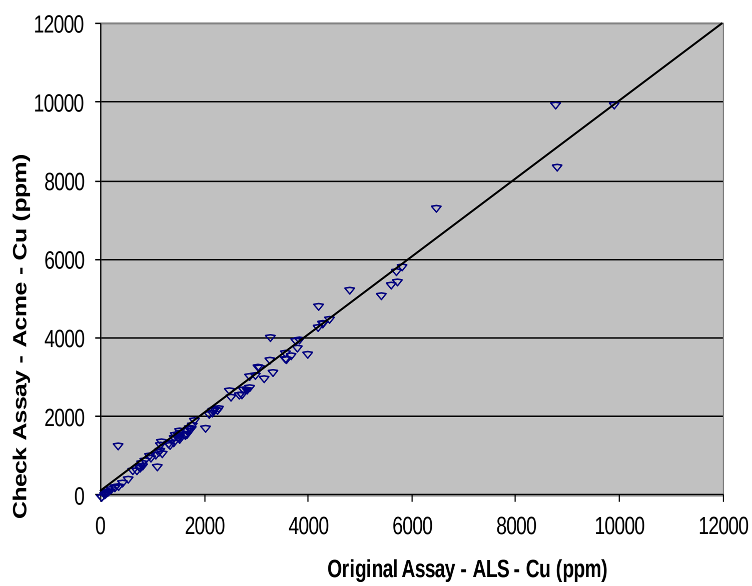


Figure 12-29: Historic Teck Gold Check Assays

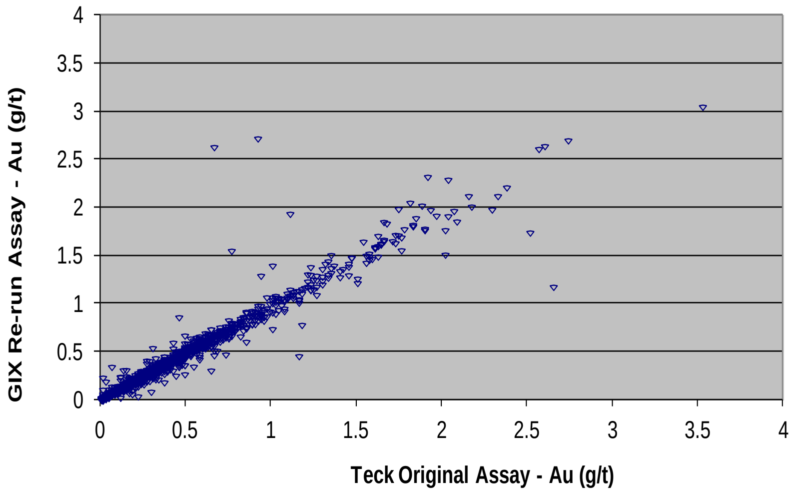
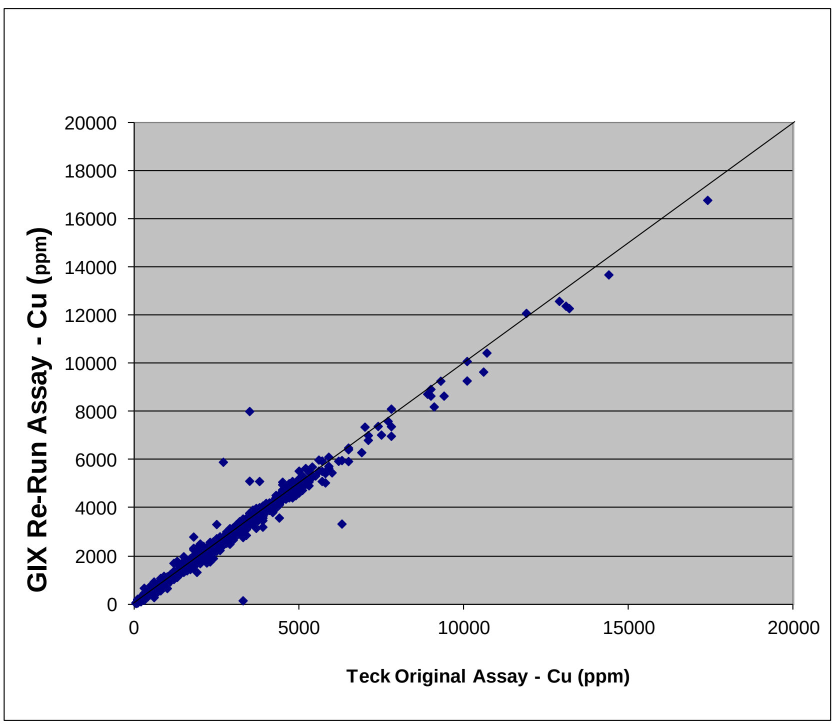


Figure 12-30: Historic Teck Copper Check Assays



12.7 Historic Drill Holes

Only INCO drill hole IN-57002 has been located by Arian and Geologix. Lack of evidence for the INCO drilling on the ground suggests co-ordinates for the INCO drilling listed in the historical database are incorrect. Due to the inability to accurately locate and verify the INCO hole data, these holes have been removed from the data verification assessment and subsequent resource study.

The geology in the Hecla drill-holes indicated a good correlation with Arian's drill-holes. There was an excellent correlation between the original Hecla assays and the Geologix re- assay program. Therefore Micon has included the Hecla drill holes in the drill hole database and mineral resource estimate.

12.8 Micon Database Validation

Micon obtained the Adobe Acrobat assay certificates of the drill hole assay database. Approximately 5% of the drill hole assays were examined and compared to the digital database for validation of the database. There were only minor errors in transferring some of the peripheral multi-element ICP data to the database. This was transmitted to Geologix and the database was amended. None of the main elements reported in the mineral resource were affected by these minor errors. Micon believes that the present digital database is clean of errors and is acceptable for use in the mineral resource.

Micon located several drill hole collars from each of the deposits as a check on the drill database. A Garmin GPS 60Csx was used to obtain the coordinates of these holes. Table 12-4 compares the database collar coordinates with Micon's coordinates.

Table 12-4: Drill Collar Coordinate Comparison

Zone	Hole	Geologix			Micon			Difference		
	No.	N	E	El.	N	E	El.	N	E	El.
		(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)	(m)
North	TEP-11-116	2116249	716715	535	2116251	716721	543	-2	-6	-8
	TEP-11-127	2116548	716528	569	2116552	716527	577	-4	1	-8
	TEP-11-039	2117256	716472	580	2117260	716471	594	-4	1	-14
South	TEP-11-128	2115699	717316	489	2115703	717315	495	-4	1	-6
	TEP-11-013	2115551	717105	511	2115557	717105	516	-6	0	-5
Tizate	TIZ -11-070	2116630	718474	502	2116626	718447	490	4	27	12
	TIZ-11-059	2116558	718460	498	2116560	718443	489	-2	17	9
	TIZ-11-004	2116712	718974	431	2116713	718972	438	-1	2	-7

Elevations tend to be less accurate than northings and eastings depending on the number of satellites available and the time allotted to a reading, especially a non-differential GPS unit. Two of the Tizate holes have a large difference in the Easting which could be due to the limited time taken to obtain those readings. Most of the northing and easting readings are approximately within the tolerance of the GPS used. Micon is confident that the locations documented for the drilling are accurate.

12.9 Validation Summary

Results of the QA/QC work indicate that the analytical techniques employed by the laboratories are generally reliable and repeatable. There is a good level of accuracy and precision. CRM and duplicate analysis indicate that there are no significant biases to over or under-reporting of assay results.

It is Micon's opinion that of the QA/QC protocol used by Geologix is in keeping with best industry practices and sufficient for the estimation of mineral resources.

13 Mineral Processing and Metallurgical Testing

13.1 Background

There are three sources of gross metal value (GMV) from the Tepal resources. They are chalcopyrite (copper sulphide with interstitial gold and silver) in a quartz matrix, an iron pyrite (iron sulphide with interstitial gold and silver) encased in a secondary quartz/gangue matrix, and a surface oxide layer containing copper minerals (in decreasing amounts; tenorite, malachite, azurite and covellite) which also contain gold and silver values.

Currently three pits are planned to be mined that include the North Zone (NZ), South Zone (SZ) and Tizate Zone. Metallurgically, the NZ and SZ can be considered similar since they contain similar copper and gold grades (see variability results in Table 4) and are of similar rock hardness. Tizate should be considered slightly different since it is of lower grade and is harder than the NZ/SZ ore.

Sulphide ore hardness is variable in the three pits, with NZ being the moderately hard and Tizate being hard. Over 42 variability tests were completed with Bond Work index hardnesses ranging from a low of 11.0kWh/tonne to a high of 20.0kWh/tonnes, (SRK, 2012, *Grinding and Crushing Circuit Equipment Sizing*). Due to this variation, the milling circuit is designed to process 40,00tpd of NZ ore and 35,00tpd of SZ and Tizate ore. The oxide ore is soft from all three areas resulting in a design capacity of 56,000tpd through the same milling circuit.

The saleable products for this PFS are a copper concentrate with gold, silver and molybdenum values obtained from a sulphide flotation, and a gold/silver doré bar from the site refinery. The chalcopyrite sulphides in the concentrate contain approximately 40% of the total gold and also recoverable silver and molybdenum with slightly higher silver and molybdenum in the Tizate Zone.

The molybdenum currently does not add value to the concentrate. A molybdenum separating flotation step is needed to make a saleable molybdenum concentrate. Additional metallurgical testing is necessary for inclusion of molybdenum in any economic evaluation; therefore, this has been included as a recommendation.

The iron pyrite contains approximately another 30% of the sulphide's gold which is to be processed for this PFS using a pyrite float followed by a CIL circuit, carbon plant and refinery. The surface oxides contain copper, gold and silver values; however, only the gold and silver is designed to be recovered for this PFS in a CIL circuit, carbon plant and refinery.

13.2 Historical Metallurgical Testing

Metallurgical testing was first performed on the NZ and SZ in 1973 by INCO Ltd, and in the mid-1990s by Teck-Cominco Corporation. Further tests were performed in 2009 and 2010 to support an NI 43-101 compliant Preliminary Assessment (PA) done by SRK Consulting on October 8, 2010 which was updated in another NI report on April 29, 2011 with an increase from 25,000 to 35,000tpd.

Data from the locked cycle flotation tests performed in 2010 and used in the preliminary assessment reports were used in this PFS. Only the NZ and SZ oxide ore had cyanide leach column tests performed for the April, 2011 PA report. Additional tests carried out since April 2011 included a leach column test on the Tizate oxide ore and variability sampling and flotation tests on the NZ, SZ, and Tizate sulphides which are included in this PFS.

From the review of all test results, covellite was the only detrimental mineral found in the oxide ore that leaches copper to solution along with the desired gold. This has been identified as an operating cost risk because it consumes large quantities of cyanide and lime within the heap leaching stack. Opportunities to mitigate this are listed in the recommendations section.

13.2.1 Summary of Pre-2009 Tests

Initial metallurgical tests were performed on samples from the Tepal mineral deposits starting in 1973 by INCO Ltd. Minor testing then continued until 2009 when further float and leach tests were commissioned by Geologix Exploration Inc. to support a preliminary assessment.

For a detailed account of the pre-2009 tests, the reader is referred to the *Revised Tepal Project Preliminary Assessment Technical Report* (SRK Consulting, April 29, 2011).

Reports prior to 2009 include:

- Duesing, C., July 3 1973. *Tepalcuatita Copper Prospect*. INCO Memorandum.
- Cruymingin, V., 1973. *Tepalcuatita Copper Prospect, Borehole 57002 Mill Testing*. INCO Memorandum.
- Eliott, M., 1993. *The Extraction of Gold and Copper from the Tepalcatepec Samples*. Teck Corp. Progress Report.
- Shonk, K., 1994. *The Tepal Gold-Copper Property*. Teck Corp Technical Report.

The INCO study consisted of two flotation locked cycle tests on NZ ore, the results are presented in Table 13-1. The concentrate grade was poor but recoveries were reasonable, with recommendations to regrind the concentrate to achieve a saleable copper grade above 25%.

Table 13-1: Summary of 1973 Tepal Average Flotation Results

Product	Unit	Value
Head Assay		
Copper	%	0.43
Gold	g/t	1.30
Silver	g/t	1.25
Concentrate Grade		
Copper	%	12.7
Gold	g/t	41
Silver	g/t	39
Recovery		
Copper	%	74
Gold	%	76
Silver	%	75

The tests conducted by Teck in 1993 focused on fine grind, bottle roll and cyanide leaching of gold opposed to the flotation of copper minerals containing gold as performed in the INCO tests. The work was completed at Lakefield Research in Peterborough, Ontario. Four core samples were used, grading 1.07g/t to 1.36g/t gold. Recoveries were good at 84% gold recovered to solution with a medium cyanide consumption of 0.75kg/t.

These tests concluded that the ore can be processed by either flotation or by a cyanide leach, with recommendations to conduct more tests that could support an economical model to best optimize recovery.

13.3 Metallurgical Tests - 2009 to 2012

Geologix commissioned flotation and leach tests in 2009. G&T Metallurgical Services in Kamloops, Canada conducted the sulphide flotation recovery tests. Oxide ore cyanide leach and column tests were conducted by McClelland Laboratories Inc. in Sparks, USA.

The results used in previous preliminary assessments were from the NZ/SZ and Tizate sulphide flotation locked cycle tests and the NZ/SZ oxide leach column tests. Further tests were conducted on Tizate oxide samples after 2011 to fill in the Tizate float and leach results for this PFS.

The flotation results are summarized in G&T's August 2010 report *Metallurgical Assessment of the Tepal Project*.

The column leach results were summarized in the September 2010 McClelland report entitled *Heap Leach Cyanidation Testing* and updated with the Tizate zone leach results in the 2012 McClelland report entitled *Heap Leach Cyanidation Testing*.

The metallurgical results from the 2009 NZ/SZ float and leach tests used in the 2011 preliminary assessment were added to the Tizate locked cycle and leach tests performed in 2012 and are summarized in Table 13-2. These results were used as the design criteria for the PFS.

The concentrates had minor element assays performed to determine if any deleterious elements would diminish the value when calculating a Net Smelter Return (NSR) for this resource. The results in Table 13-3 are the minor element assays from the two locked cycle tests completed by G&T in 2010 used to calculate the NSR for this PFS.

The copper concentrate is unusually clean owing to the quartz matrix containing the chalcopyrite. There is good separation between chalcopyrite and pyrite due to the faster chalcopyrite flotation kinetics. Fortunately there is little contamination of pyrite in the copper concentrate, which should make the concentrate easy to market.

Further heap leach cyanidization tests were completed in June 2012 by McClelland Laboratories on the Tizate oxide to complete the dataset of column leach tests which already tested the NZ/SZ ore. G&T Metallurgical Services also performed variability tests on 42 core samples from the NZ, SZ and Tizate zones in late 2011. The flotation results are shown in Table 13-4.

Table 13-2: Metallurgical Design Criteria Summary

Product	Unit	Flotation	Column Leach
Resource Grade			
Tepal Grade			
Copper	%	0.22	N/A
Gold	g/t	0.37	0.35
Silver	g/t	1.02	0.92
Tizate Grade			
Copper	%	0.17	N/A
Gold	g/t	0.19	0.22
Silver	g/t	2.23	2.17
Recovery			
Tepal Recovery			
Copper	%	88.2	N/A
Gold	%	62.4	76
Silver	%	27.4	10
Tizate Recovery			
Copper	%	85.9	N/A
Gold	%	58.0	62
Silver	%	59.6	60
Concentrate Grade			
Concentrate Grade - Tepal			
Copper	%	25.7	N/A
Gold	g/t	32.8	as doré
Silver	g/t	42.9	
Concentrate Grade - Tizate			
Copper	%	26.9	N/A
Gold	g/t	15.0	as doré
Silver	g/t	267.6	as doré

Table 13-3: Concentrate Minor Element Assays (G&T, 2010)

Element	Unit	Test 32-South	Test 34-North
Aluminum	%	0.80	0.62
Antimony	g/t	129	33
Arsenic	g/t	238	55
Bismuth	g/t	54	25
Cadmium	g/t	12	<10
Calcium	%	0.34	0.29
Cobalt	g/t	132	80
Copper	%	19.6	27.0
Fluorine	g/t	125	141
Gold	g/t	28.1	33.8
Iron	%	33.7	32.4
Lead	%	0.0	0.0
Magnesium	%	0.23	0.19
Mercury	g/t	<1	<1
Manganese	%	0.01	0.01
Molybdenum	%	0.09	0.06
Nickel	g/t	172	172
Phosphorus	g/t	110	99
Selenium	g/t	89	123
Silicon	%	2	1
Silver	g/t	28	47
Sulphur	%	38.3	34.8
Zinc	%	0.02	0.02

For the variability tests G&T concluded that ”Mineralogically, the remaining sulphides are very similar across the three zones”, and for ore hardness results G&T concluded that the “Bond ball mill work indices also indicate the samples to be hard with an average index of 14.2kWh/t.”

The variability test showed that all resources can be treated with the same flowsheet for all NZ, SZ, and Tizate ore; therefore, a common mill can be designed for all. The harder Tizate ore will have a lower throughput at the given grinding circuit design, and due to its lower head grades, would have a lower concentrate production. For this reason, the Tizate should be mined later on in the mine life to ensure NZ/SZ ore is processed at the highest rates and highest head grades to generate the highest cash flow at the beginning of the mine life.

No fatal flaws or deleterious elements were found in the metallurgical tests reviewed.

Work indices shown in Tables 13-5 to 13-7 varied greatly with the hardest ore appearing deeper in the Tizate zone. The flowsheet treating both NZ/SZ and Tizate is described in Section 17 and consists of conventional crushing, grinding, flotation, cyanidation and dewatering.

Table 13-4: Sulphide Flotation Variability Test Results

Sample	Zone	Concentrate Grade						Percent Distribution					
		Cu (%)	Mo (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)	Cu	Mo	Fe	S	Ag	Au
V17	North	26.37	0.073	28.6	30.91	41	57.96	88.7	35.4	6.4	26.4	29.6	78.0
V18	North	28.06	0.066	28.4	31.9	59	24.51	86.8	33.4	8.2	10.8	37.3	60.3
V19	North	27.38	0.031	27.9	31.05	35	39.43	88.2	29.3	7.8	11.9	27.9	70.0
V20	North	23.20	0.085	28.9	31.60	24	19.53	72.3	10.8	2.6	11.0	9.6	41.5
V21	North	26.97	0.049	27.3	30.20	170	45.36	77.3	18.3	2.6	12.6	40.5	53.4
V22	North	27.42	0.012	28.4	31.56	47	45.31	81.6	9.4	6.6	19.0	13.7	60.4
V26	North	25.64	0.121	27.9	30.8	134	17.36	68.4	11.2	2.3	2.5	22.9	49.9
V27	North	27.80	0.056	31.0	34.80	120	2.00	64.1	4.7	1.0	3.4	9.5	2.6
V28	North	28.73	0.046	29.2	33.0	128	30.07	83.9	15.8	6.1	8.2	43.3	66.3
V29	North	28.30	0.034	33.5	36.8	86	62.70	55.0	7.3	4.0	4.8	18.9	55.2
V30	North	29.76	0.043	29.4	32.78	67	40.16	89.8	21.2	6.6	9.9	28.4	73.3
V31	South	32.30	0.105	28.4	33.6	94	46.03	82.4	22.8	2.2	3.2	24.4	45.2
V32	South	25.60	0.210	24.3	28.30	90	34.10	54.1	14.9	0.7	5.6	8.2	31.0
V33	South	31.20	0.013	28.6	33.40	44	54.59	68.1	2.5	2.0	8.6	11.0	30.7
V34	South	26.58	0.059	25.6	31.59	21	19.40	84.8	23.0	4.6	10.5	14.5	58.0
V35	South	25.98	0.017	25.5	29.28	36	45.78	85.7	16.0	7.0	29.4	31.1	68.7
V36	South	25.64	0.052	28.3	30.9	20	27.01	52.5	14.2	3.4	6.0	9.7	40.2
V37	South	25.80	0.260	34.3	36.2	162	23.12	83.2	28.1	4.7	5.9	27.1	30.6
V38	South	25.40	0.091	34.6	36.9	266	13.11	67.5	7.6	2.6	2.1	24.9	19.7
V39	South	28.29	0.047	26.3	30.23	30	43.62	76.4	18.6	3.3	11.6	13.1	40.4
V40	South	26.38	0.128	24.6	27.53	27	18.71	68.7	25.6	2.7	13.1	11.9	38.0
V41	South	27.30	0.021	28.3	30.10	16	34.25	79.3	9.5	7.0	20.0	15.4	38.4
V42	South	28.38	0.246	30.7	31.34	22	19.70	80.6	40.8	6.5	16.7	17.8	48.7

Sample	Zone	Concentrate Grade						Percent Distribution					
		Cu (%)	Mo (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)	Cu	Mo	Fe	S	Ag	Au
V04	Tizate	27.30	0.228	30.5	34.7	151	49.1	85.2	49.0	4.6	9.2	46.0	69.1
V05	Tizate	27.10	0.174	28.1	32.6	66	31.70	75.9	16.7	2.3	6.6	20.3	43.4
V06	Tizate	25.00	0.180	28.6	32.80	133	0.50	43.8	14.6	0.7	3.3	11.4	0.6
V07	Tizate	25.30	0.034	28.8	31.30	206	39.10	63.7	15.9	3.7	8.3	47.1	39.6
V08	Tizate	32.50	0.631	28.9	33.60	156	29.30	74.9	54.6	2.2	9.7	42.6	46.6
V09	Tizate	26.49	1.141	28.3	31.04	164	16.58	83.2	77.0	4.2	15.6	45.2	50.5
V10	Tizate	31.20	1.123	28.9	33.00	276	11.60	76.6	58.4	2.6	8.9	52.9	28.7
V11	Tizate	26.92	0.335	26.7	29.74	346	10.59	84.1	55.8	3.8	8.8	66.3	52.2
V12	Tizate	30.40	1.352	28.6	33.70	432	10.20	60.6	37.0	2.2	9.3	55.5	36.9
V13	Tizate	29.50	0.086	29.9	33.6	782	1.74	71.3	18.7	5.0	10.2	65.9	30.0
V14	Tizate	30.60	1.002	30.3	33.00	552	8.52	82.6	37.9	5.5	18.4	75.0	39.5
V15	Tizate	28.35	0.997	28.1	31.82	120	18.61	83.2	70.7	4.1	11.8	42.4	56.6
V16	Tizate	27.42	0.598	28.0	32.30	87	7.59	84.3	63.0	3.7	7.1	32.3	36.9
Average	North	27.24	0.06	29.14	32.31	82.82	34.94	77.83	17.89	4.80	14.32	27.65	55.54
Std. Dev.	North	1.74	0.03	1.74	1.96	48.22	18.24	11.53	10.71	2.42	13.40	13.37	20.56
Average	South	27.40	0.10	28.29	31.61	69.00	31.62	73.61	18.63	3.89	11.06	17.43	40.80
Std. Dev.	South	2.27	0.09	3.44	2.92	75.83	13.44	11.47	10.32	2.08	7.88	7.59	13.20
Average	Tizate	28.31	0.61	28.75	32.55	267.00	18.09	74.57	43.79	3.43	9.78	46.38	40.82
Std. Dev.	Tizate	2.34	0.47	1.03	1.32	210.65	14.88	12.21	21.93	1.35	3.83	17.93	16.40

Table 13-5: Sulphide Hardness Variability Test Results, North Zone

Sample Designation	BBWI	SMC Test Data								
	kWh/t	DWi	DWi	Mia	Mih	Mic	A	b	SG	t_a
		kWh/m³	%	kWh/t	kWh/t	kWh/t				
Composite 1	-	8.06	79	22.7	17.4	9	61.9	0.54	2.69	0.32
Composite 17	-	7.53	73	22	16.7	8.6	78.5	0.44	2.62	0.34
Composite 18	-	9	86	24.7	19.3	10	82.1	0.37	2.7	0.29
Composite 19	-	6.61	62	19.4	14.3	7.4	65.3	0.62	2.68	0.39
Composite 20	-	6.65	63	19.4	14.4	7.4	58.2	0.69	2.69	0.39
Composite 21	-	7.99	78	23	17.6	9.1	64.3	0.51	2.63	0.32
Composite 22	-	8.69	84	24.1	18.7	9.7	84.2	0.37	2.69	0.3
Composite 23	13.1	6.14	56	18.4	13.3	6.9	65.7	0.66	2.67	0.42
Composite 26	15.7	3.32	20	11.4	7.3	3.8	60.4	1.32	2.64	0.78
Composite 27	17.2	6.99	67	21.1	15.7	8.1	75.7	0.49	2.58	0.37
Composite 28	11.1	2.86	15	10.3	6.4	3.3	59.9	1.51	2.59	0.91
Composite 29	14.5	5.32	45	16.5	11.7	6	73.7	0.67	2.65	0.49
Composite 30	14.2	5.08	41	15.8	11	5.7	66.9	0.78	2.67	0.51

Table 13-6: Sulphide Hardness Variability Test Results, South Zone

Sample Designation	BBWI	SMC Test Data								
	kWh/t	DWi	DWi	Mia	Mih	Mic	A	b	SG	t_a
		kWh/m³	%	kWh/t	kWh/t	kWh/t				
Composite 24	14.1	5.28	44	16.9	11.9	6.2	63.3	0.77	2.56	0.49
Composite 25	10.3	3.49	21	12	7.8	4	62.9	1.18	2.6	0.74
Composite 31	14.3	7.55	74	16.7	12.6	6.5	75.5	0.61	3.47	0.34
Composite 32	16	6.72	64	19.5	14.4	7.5	62.3	0.65	2.7	0.39
Composite 33	12.2	2.69	14	9.5	5.9	3	57.7	1.71	2.65	0.96
Composite 34	13.6	7.61	74	21.7	16.5	8.5	78.9	0.45	2.68	0.34
Composite 35	14.7	9.71	90	26.5	21.1	10.9	100	0.28	2.67	0.27
Composite 36	18.4	10.59	93	28.1	22.8	11.8	100	0.25	2.69	0.24
Composite 37	13.8	6.29	58	18.1	13.2	6.8	74.2	0.59	2.76	0.41
Composite 38	12.3	4.01	27	12.8	8.6	4.4	57.6	1.18	2.71	0.65
Composite 39	16.5	8.07	79	22.4	17.2	8.9	67.7	0.5	2.73	0.32
Composite 40	16.4	6.99	67	19.9	14.8	7.7	57.9	0.68	2.73	0.37
Composite 41	13.5	8.12	79	23.1	17.8	9.2	63.6	0.51	2.65	0.32
Composite 42	15.4	6.44	60	19.7	14.4	7.5	68.7	0.58	2.59	0.4

Table 13-7: Sulphide Hardness Variability Test Results, Tizate

Sample Designation	BBWI	SMC Test Data								
	kWh/t	DWi	DWi	Mia	Mih	Mic	A	b	SG	t_a
		kWh/m^3	%	kWh/t	kWh/t	kWh/t				
Composite 2	-	7.29	71	21.2	15.9	8.2	71.6	0.51	2.66	0.36
Composite 3	-	6.99	67	20.3	15.1	7.8	60.5	0.63	2.68	0.37
Composite 4	-	9.6	89	25.5	20.3	10.5	100	0.29	2.75	0.27
Composite 5	-	8.54	83	23.4	18.1	9.4	78.3	0.41	2.73	0.3
Composite 6	-	8.54	83	23.4	18.1	9.4	79.7	0.4	2.73	0.3
Composite 7	-	8.55	83	23.3	18.1	9.4	75.6	0.42	2.74	0.3
Composite 8	-	10.02	91	26.2	21	10.9	96.1	0.29	2.77	0.26
Composite 9	-	11.3	95	29.2	24	12.4	100	0.24	2.73	0.23
Composite 10	-	8.25	80	22.7	17.5	9	69.6	0.48	2.74	0.31
Composite 11	-	9.53	89	25.4	20.2	10.4	74.8	0.38	2.74	0.27
Composite 12	-	7.36	72	20.8	15.7	8.1	73	0.51	2.72	0.35
Composite 13	-	8.26	80	23	17.7	9.2	85.6	0.38	2.7	0.31
Composite 14	-	5.54	48	16.9	12	6.2	65.3	0.74	2.68	0.47
Composite 15	-	11.72	96	29.9	24.7	12.8	100	0.23	2.75	0.22
Composite 16	-	9.27	88	24.7	19.5	10.1	69.5	0.43	2.76	0.28

The mineral composition of the sulphide ore in all 42 samples was relatively consistent across all zones as was concluded in G&T's Variability Metallurgical Assessment, February 8, 2012.

The copper concentrate grade produced for all three zones was good and was well above the range for a marketable concentrate. Copper grades averaged 27.24% (1.74% Std. Dev.), 27.40% (2.27% Std. Dev.) and 28.31% (2.34% Std. Dev.) for the NZ, SZ, and Tizate zones, respectively. Copper recovery was variable and driven by the copper feed grades that averaged 0.25%, 0.21% and 0.20% copper for the NZ, SZ, and Tizate zones, respectively. Copper recovery averaged 77.83% (11.53% Std. Dev.), 73.61% (11.47% Std. Dev.) and 74.57% (12.21% Std. Dev.) for the NZ, SZ, and Tizate zones, respectively. These recoveries are lower than the values being used in the design due to high concentrate grades produced in the variability bench tests. Locked cycle tests typically are a better indicator of actual recoveries in the process plant than batched, bench scale tests. Therefore, for design purposes, a 26.5% average copper concentrate at an 87% average recovery is estimated (Table 13-2).

Ball mill hardness indices tests were only completed for the NZ and SZ samples, but crusher hardness was completed for all 42 samples. The average Bond Work indices were 14.3kWh/t (2.1 Std. Dev.) and 14.4kWh/t (2.1 Std. Dev.) for the NZ and SZ, which indicates a fairly hard ore. The crusher work indices were 6.5 kWh/m3 (1.9 Std. Dev.), 6.7kWh/m3 (2.3 Std. Dev.) and 8.7kWh/m3 (1.6 Std. Dev.) for the NZ, SZ, and Tizate zones, respectively. Tizate was found to be about 17%

harder than the NZ/SZ and may need extra SAG capacity in the form of pebble crushers in the later stages of the mine life when the lower benches of the Tizate pit are scheduled to be mined.

13.4 Metallurgical Tests – 2012 to Present

An initial economic study was done using copper, gold and silver recovered to a chalcopyrite concentrate and gold and silver recovered from an oxide heap leach. The study showed the project economics could be improved by including the recovery of the gold and silver occurring with the pyrite which would normally be sent to tails. This would involve a simple pyrite flotation step on the chalcopyrite flotation tails followed by a cyanide leach of the pyrite concentrate to recover gold and silver to a doré bar.

The study also indicated that the oxide heap leach option had high operating and capital costs, so alternative methods of oxide treatment were developed and tested at G&T Labs. The tests were to determine the economics of using these different methods to process the oxides rather than a heap leach:

- Flotation was investigated to recover both copper and gold. The test investigated simple sulphide flotation as well as controlled potential flotation of copper oxide and copper carbonate minerals.
- Gravity concentration was conducted using a Knelson concentrator followed by panning of the gravity concentrate at a primary grind size of 143 and 157um K80.
- Cyanide leaching was conducted on whole ore with two different grind sizes for each composite. The leach time was 48 hours and the pH of the pulp was modulated to pH 11 with lime.
- Two tests were conducted using a sulphuric acid leach to investigate copper leaching. The leach pulp was maintained at pH 2 at each stage with sulphuric acid.

The comparison results for the proposed oxide metallurgical processes are tabulated below in Tables 13-8 to 13-11.

Table 13-8: Oxide Flotation Flowsheet

Test Number	Composite	Mass (percent)	Assay - Percent or g/tonne				Distribution - percent			
			Cu	S	Ag	Au	Cu	S	Ag	Au
1	NSOX	11.4	0.7	2	2	2	37	83	29	59
7	NSOX	10.2	0.74	2.54	3	1.98	36	83	43	60
Average		**10.8**	**0.72**	**2.27**	**2.4**	**1.99**	**37**	**83**	**36**	**59**
2	TOX	11	0.31	0.25	7	1.11	18	45	29	43
8	TOX	7.3	0.32	0.37	10	1.42	12	35	44	44
14	TOX	6.7	0.35	0.48	6	0.95	11	37	17	29
15	TOX	11.4	0.3	0.42	5	0.87	17	43	41	41
Average		**9.1**	**0.32**	**0.38**	**6.9**	**1.09**	**15**	**40**	**33**	**39**

Table 13-9: Gravity Flowsheet

Test Number	Composite	Grind Size um K80	Knelson Concentrate		Pan Concentrate	
			Grade - g/t	Rec - %	Grade - g/t	Rec - %
6	NSOX	157	3	13.2	4.7	6.1
10	NSOX	157	2.4	11	3.3	7.3
Average		**157**	**3**	**13.2**	**4.7**	**6.1**
5	TOX	143	3.1	20.7	4.6	8.6
9	TOX	143	1.4	10.7	4	3.5
Average		**143**	**2.3**	**15.7**	**4.3**	**6.1**

Table 13-10: Cyanide Leach Flowsheet

Test Number	Composite	Grind Size um K80	Extraction - percent		Consumption kg/t	
			Au	Ag	NaCN	Lime
3	NSOX	143	83.8	63.7	1.4	2.4
11	NSOX	89	93.2	83.1	1.6	2.1
18	NSOX	89	85.7	79.6	3.4	2.2
Average		**116**	**88.5**	**73.4**	**1.5**	**2.3**
4	TOX	157	75.7	57.4	0.4	3.6
12	TOX	102	89.3	84.1	0.5	3.3
Average		**130**	**82.5**	**70.7**	**0.5**	**3.5**

Table 13-11: Acid Flowsheet

Test Number	Composite	Grind Size um K80	Liquor Assay - g/tonne			Extraction - percent		
			Cu	Ag	Au	Cu	Ag	Au
16	NSOX	89	366	0.5	0.1	29	63	37
17	TOX	102	189	0.5	0.1	18	41	41

The bulk flotation process resulted in low grade copper and gold concentrates. For the North, South oxide (NSOX) composite, copper and gold in the feed were 37 and 59 percent recovered to the flotation concentrate. The Tizate oxide (TOX) sample results were much poorer, with only 15 and 39 percent of the copper and gold in the feed recovered to the concentrate. Due to the disappointing results, flotation would not be a suitable process based on these samples.

Similarly, gravity recovery of a gold concentrate returned relatively poor results. Gold was only 6 percent recovered into a concentrate grading about 4g/t gold.

The best overall gold and silver extraction performance was achieved by direct cyanidation. The average gold leach performed was about 89 and 83 percent the NSOX and TOX Composites, respectively. The best results were achieved at the finer primary grind size of nominally 95um K80. Average cyanide consumptions levels were 1.5 and 0.5kg/t for the NSOX and TOX samples, respectively.

The acid leach, which investigated copper disassociation, also had relatively poor performance. More mineralogical information on the copper minerals present in the samples would be required to further advance this process.

The optimum flowsheet selected will be a conventional sulphide copper flotation to a sellable chalcopyrite concentrate followed by a pyrite flotation and pyrite cyanide agitated tank leach to recover gold and silver to a doré bar.

For the oxide testwork described above, a new sample composite was made up from metallurgical core at G&T in October 2012. Figure 13-1 below shows a representative sample for this program. Enough sample composites were made in anticipation to run a one ton per day pilot plant. At present the pilot plant has not been run. The head assays closely match the schedule mine grades as compared to the mine production schedule presented in Section 16. A comparison of the head assays with the variability tests performed in 2011 was also done, see Table 13-12.

Table 13-12: Sulfide Head Grade Comparisons - Pilot vs. Variability Composites

Zone	Study	Cu %	Au g/t	S% (total)	ADIS Py%
NZ/SZ	Variability	0.23	0.42	2.2	3.15
NZ/SZ	Pilot Plant	0.22	0.39	2.4	3.87
	% diff V vs PP	*-4.30%*	*-7.10%*	*9.10%*	*22.80%*
NZ/SZ	*Block Model 2012*	*0.22%*	*0.39%*		
	% diff BM vs. PP	*0.00%*	*0.00%*		
Tizate	Variability	0.2	0.24	1.86	2.56
Tizate	Pilot Plant	0.18	0.21	2.13	3.75
	% diff V vs. PP	*-10.00%*	*-12.50%*	*14.50%*	*46.50%*
Tizate	*Block Model 2012*	*0.17%*	*0.17%*		
	% diff BM vs. PP	*-5.60%*	*-19.00%*		

*Variability assays for Cu, Au and S are averages

* S%= Total sulphur by Leco.

Sulphur percent and automated digital imaging system (ADIS) pyrite percent along with assay data were examined. The investigations suggest that there are slightly higher amounts of pyrite in the pilot plant material than in the variability samples but no consistent relationship between gold and pyrite can be established.

Figure 13-1: Tepal Metallurgical Test Core As Received At G&T Lab

13.5 Leach Recovery Predictions

13.5.1 Pyrite Flotation and Leach Results

Table 13-13 below summarizes the recovery of gold and silver as parts of each unit operation and as an overall recovery from the pyrite flotation and leach. The leach feed is the combined pyrite concentrate and the pyrite rejected from the copper cleaner tails stream.

Overall gold recovery to a copper concentrate and to a cyanide solution is 79.7% for the N/S zone and 78.5% for the Tizate zone. Overall silver recovery to a copper concentrate and to a cyanide solution is 41.0% for the N/S zone and 71.8% for the Tizate zone. Cyanide and lime consumption is expected to average 2.5 kg/t and 1.4kg/t for Tepal and 2.8kg/t and 1.5kg/t for Tizate. It is expected that recovery to a doré bar in the CIL circuit from solution losses and carbon fines would decrease the overall recovery by 2%. It is anticipated that the carbon fines would either be treated off-site for precious metal recovery or sold outright.

Table 13-13: Pyrite Flotation and Leach Predictions

Product	Unit	Recovery
Pyrite Conc Leach		
Tepal		
Copper	%	1.0
Gold	%	10.7
Silver	%	6.1
Tizate		
Copper	%	4.0
Gold	%	15.5
Silver	%	7.8
Cu Cleaner Tails Leach		
Tepal		
Copper	%	0.8
Gold	%	6.5
Silver	%	7.5
Tizate		
Copper	%	0.5
Gold	%	5.0
Silver	%	4.3

13.5.2 Oxide Leach Results

Table 13-14 below summarizes the recovery of gold and silver from the oxide leach.

Table 13-14: Oxide Leach Predictions

Product	Unit	Recovery
Tepal		
Gold	%	83.2
Silver	%	63.3
Tizate		
Gold	%	75.2
Silver	%	55.9

Cyanide and lime consumption is expected to average 1.4kg/t and 2.4kg/t for Tepal and 0.4 kg/t and 3.6 kg/t for Tizate. It is expected that recovery to a doré bar in the ADR plant from solution losses and carbon fines would decrease by 2%. It is anticipated that the carbon fines would either be treated off-site for precious metal recovery or sold outright.

14 Mineral Resource Estimates

The following section is taken from Micon Technical report, March 29, 2012.

Three NI 43-101 compliant Mineral Resource estimates have been completed on the Tepal property, details of which can be found in Section 6, History. The mineral resource estimate reported below supersedes these previous estimates.

14.1 Micon Estimates

The Tepal property mineral resource was based on 353 drill hole data. Mineralogical models were generated by Geologix and used to constrain the grade estimation. Datamine Studio V3 mining software data was used to create block models of the three deposits. Grades were interpolated using the ordinary kriging method. The data was converted to Surpac V6.2 mining software to generate a soft pit for each deposit that provided the limit for defining material which offered a reasonable prospect for economic extraction. An NSR cut-off equivalent value of US$5.00/t was used to select a break even mining cost for an open pit type operation of this size. The following table summarizes the Measured and Indicated Tepal Property Mineral Resource estimate.

Table 14-1: Measured & Indicated Mineral Resources at US$5/t Equivalent Value Cut-Off

Deposit	Resource	Tonnage	Average Grade				Contained Metal	
	Category	(kt)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Au (koz)	Cu (Mlb)
Tepal North	Measured	14,067	0.50	0.29	0.78	0.002	228	89
	Indicated	55,320	0.30	0.21	1.01	0.002	533	252
	M + I	69,387	0.34	0.22	0.96	0.002	761	341
Tepal South	Measured	20,011	0.47	0.22	1.07	0.002	300	96
	Indicated	20,993	0.45	0.20	1.17	0.002	305	91
	M + I	41,005	0.46	0.21	1.12	0.002	605	187
Tizate	Measured	-	-	-	-	-	-	-
	Indicated	77,375	0.18	0.17	2.29	0.006	438	285
	M + I	77,375	0.18	0.17	2.29	0.006	438	285
Total	Measured	34,078	0.48	0.25	0.95	0.002	528	185
	Indicated	153,688	0.26	0.19	1.67	0.004	1,276	628
	M + I	187,766	0.30	0.20	1.54	0.004	1,804	813

*Assumptions used to calculate the soft pit constraint: Au Price US$ 1300/oz, Cu Price US$ 3.30/lb

Tizate Oxide Au Recovery - 68.8%, Cu Recovery - 6.8%

Tizate Sulphide Au Recovery - 66.2%, Cu Recovery - 85.3%

Tepal Oxide Au Recovery - 78.4%, Cu Recovery - 14.3%

Tepal Sulphide Au Recovery - 60.7%, Cu Recovery - 87.4%

*Mineral resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into Mineral Reserves.

The Table 14-2 summarizes the Inferred Mineral Resources of the three deposits above the same US$ 5/tonne equivalent value NSR cut-off.

Table 14-2: Inferred Mineral Resources at US$5/t Equivalent Value Cut-Off

Deposit	Resource	Tonnage	Average Grade				Contained Metal	
	Category	(kt)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Au (koz)	Cu (Mlb)
Tepal North	Inferred	906	0.22	0.21	1.21	0.003	6.5	4.2
Tepal South	Inferred	412	0.40	0.16	0.95	0.002	5.3	1.5
Tizate	Inferred	34,426	0.15	0.15	1.70	0.007	169.8	114.8
Total	Inferred	35,743	0.16	0.15	1.68	0.006	181.7	120.4

*Assumptions used to calculate the soft pit constraint: Au Price US$ 1300/oz, Cu Price US$ 3.30/lb

Tizate Oxide Au Recovery - 68.8%, Cu Recovery - 6.8%

Tizate Sulphide Au Recovery - 66.2%, Cu Recovery - 85.3%

Tepal Oxide Au Recovery - 78.4%, Cu Recovery - 14.3%

Tepal Sulphide Au Recovery - 60.7%, Cu Recovery - 87.4%

The following are the parameters and assumptions made to complete this estimate.

14.1.1 Mineralogical Model

Geologix generated new mineralogical model for each of the three deposits. The models were designed to contain all drill hole intervals with a dollar value of greater than US$ 8.70/tonne based on metal prices of US$ 1,000/oz for gold and US$2.75/lb for copper. The envelopes took into consideration all historic and new infill drill holes, geological contacts and updated interpretations of the three deposits. The boundary of the models corresponded to geological observations and the approximate primary economic limits of the mineralization. Geological parameters included the type and intensity of alteration, the type, style and abundance of veinlets and the type, style and abundance of sulphide and oxide mineralization. Minor internal dilution below the US$8.70 limit was included for continuity of the model. Blocks inside the mineralogical models were classified as “Ore” and those outside were classified as “Waste”.

14.1.2 Oxide Zone

A wireframe surface was generated to further divide the models into a near surface oxide domain and a sulphide domain at depth. The surface generated was based on data supplied to Micon by Geologix with the base of the oxide interval usually corresponding to the first appearance of sulphide mineralization.

14.1.3 Drill Data

The digital drill hole database used 353 drill holes from the various drill programs that have been run on the property (Table 14-3).

Table 14-3: Tepal Drill Hole Summary

Company	Holes Drilled	Type	Holes Used	Length (m)
Inco	21	DD	0	0
Teck	50	RC	49	8,169
Hecla	17	RC	17	1,506
Arian	42	DD	42	7,180
Geologix 2010	43	DD	43	10,656
Geologix 2011	215	DD	202	41,248
Total	388		353	68,759

The locations of the Inco holes could not be confirmed so these were removed from the database. In addition, 13 condemnation and geotechnical holes, completed in 2011, were not included in the database.

14.1.4 Composites

The composite length for the interpolations was determined by considering the lengths of all the assay intervals within the mineralized zones. The dominant sample interval length is 2m which has been chosen as the composite length. Therefore the samples were composited to 2m, honouring domain contacts. The minimum composite length was 1m with remnants and less than 1m intervals were added to the previous composite.

Basic statistics were generated for each deposit with respect to oxide and sulphide domains. A comparison of uncapped values to capped values is listed in Tables 14-4 to 14-9.

Table 14-4: Tepal North Zone Sulphide Domain Uncapped and Capped Composite Statistics

Statistics	Gold (g/t)		Copper (%)		Silver (g/t)		Molybdenum (ppm)	
	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
Mean	0.38	0.38	0.25	0.24	1.10	1.00	21.00	20.00
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	7.20	3.00	6.32	2.50	209.00	12.50	569.00	300.00
Median	0.25	0.25	0.20	0.20	0.70	0.70	12.50	12.50
Standard Deviation	0.43	0.40	0.23	0.21	3.73	1.25	29.96	27.45
Coeff. of Variation	1.13	1.05	0.92	0.85	3.46	1.26	1.45	1.34
Number of Samples	4,135	4,135	4,135	4,135	4,135	4,135	4,135	4,135

Table 14-5: Tepal North Zone Oxide Domain Uncapped and Capped Composite Statistics

Statistics	Gold		Copper		Silver		Molybdenum	
	(g/t)		(%)		(g/t)		(ppm)	
	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
Mean	0.39	0.39	0.23	0.23	0.90	0.80	17.00	17.00
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	2.52	2.52	3.23	3.23	35.00	7.00	220.00	200.00
Median	0.25	0.25	0.16	0.16	0.60	0.60	10.00	10.00
Standard Deviation	0.39	0.39	0.26	0.26	1.49	0.94	21.93	21.77
Coeff. of Variation	1.00	1.00	1.12	1.12	1.72	1.14	1.30	1.30
Number of Samples	1,097	1,097	1,097	1,097	1,097	1,097	1,097	1,097

Table 14-6: Tepal South Zone Sulphide Domain Uncapped and Capped Composite Statistics

Statistics	Gold		Copper		Silver		Molybdenum	
	(g/t)		(%)		(g/t)		(ppm)	
	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
Mean	0.48	0.48	0.22	0.22	1.20	1.10	21.00	21.00
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	3.24	2.54	1.72	1.00	84.90	10.00	363.00	363.00
Median	0.38	0.38	0.19	0.19	0.80	0.80	15.00	15.00
Standard Deviation	0.39	0.39	0.14	0.14	3.04	1.30	22.14	22.14
Coeff. of Variation	0.81	0.81	0.63	0.62	2.43	1.14	1.05	1.05
Number of Samples	2,855	2,855	2,855	2,855	2,855	2,855	2,855	2,855

Table 14-7: Tepal South Zone Oxide Domain Uncapped and Capped Composite Statistics

Statistics	Gold		Copper		Silver		Molybdenum	
	(g/t)		(%)		(g/t)		(ppm)	
	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
Mean	0.42	0.41	0.19	0.19	1.30	1.00	15.00	15.00
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	1.37	1.10	0.77	0.77	36.40	6.00	65.00	65.00
Median	0.35	0.35	0.17	0.17	0.70	0.70	11.50	11.50
Standard Deviation	0.28	0.27	0.11	0.11	3.06	1.01	12.02	12.02
Coeff. of Variation	0.67	0.66	0.58	0.58	2.42	1.04	0.80	0.80
Number of Samples	253	253	253	253	253	253	253	253

Table 14-8: Tizate Zone Sulphide Domain Uncapped and Capped Composite Statistics

Statistics	Gold		Copper		Silver		Molybdenum	
	(g/t)		(%)		(g/t)		(ppm)	
	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
Mean	0.18	0.18	0.17	0.17	2.20	2.20	69.00	69.00
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	5.24	1.10	1.30	0.80	44.10	15.00	1,691.00	625.00
Median	0.15	0.15	0.17	0.17	1.66	1.66	53.00	53.00
Standard Deviation	0.16	0.13	0.08	0.08	2.10	1.82	75.06	64.87
Coeff. of Variation	0.90	0.74	0.49	0.48	0.93	0.82	1.08	0.95
Number of Samples	3,932	3,932	3,932	3,932	3,932	3,932	3,932	3,932

Table 14-9: Tizate Zone Oxide Domain Uncapped and Capped Composite Statistics

Statistics	Gold		Copper		Silver		Molybdenum	
	(g/t)		(%)		(g/t)		(ppm)	
	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
Mean	0.19	0.19	0.18	0.18	2.20	2.20	31.00	31.00
Minimum	0.01	0.01	0.01	0.01	0.00	0.00	0.00	0.00
Maximum	1.28	0.60	1.11	0.50	8.40	8.00	144.00	144.00
Median	0.16	0.16	0.16	0.16	1.90	1.90	26.00	26.00
Standard Deviation	0.15	0.13	0.14	0.10	1.55	1.55	23.11	23.11
Coeff. of Variation	0.79	0.68	0.75	0.56	0.70	0.69	0.74	0.74
Number of Samples	255	255	255	255	255	255	255	255

14.1.5 Capping

The coefficient of variation (CV) is an indicator of outliers that may bias the grade generated in the interpolation. This is sometimes referred to as a “nugget effect”. A CV value of over 1.2 is an indication that capping of high-grade composites may be required. The methods used to identify the level of capping were Decile Analysis and Log Probability plots.

The results of the capping for gold, copper, silver and molybdenum are documented in Tables 14-10 to 14-13 with respect to each deposit and the oxide/sulphide domains.

Capping was done after generating the 2m composites so that the capping was less harsh.

Table 14-10: Tepal Property Capping Summary for Gold

Zone	Domain	Threshold	Data Capped		
		Au (g/t)	Number	Proportion (%)	Metal (%)
North	Sulphide	3.00	9	0.22	0.9
	Oxide	-	-	-	-
South	Sulphide	2.54	3	0.11	0.1
	Oxide	1.10	2	0.79	0.3
Tizate	Sulphide	1.10	6	0.15	1.0
	Oxide	0.60	6	2.35	3.0

Table 14-11: Tepal Property Capping Summary for Copper

Zone	Domain	Threshold	Data Capped		
		Cu (%)	Number	Proportion (%)	Metal Loss (%)
North	Sulphide	2.5	2	0.05	0.4
	Oxide	-	-	-	-
South	Sulphide	1.0	1	0.04	0.1
	Oxide	-	-	-	-
Tizate	Sulphide	0.8	4	0.10	0.1
	Oxide	0.5	7	2.75	4.2

Table 14-12: Tepal Property Capping Summary for Silver

Zone	Domain	Threshold	Data Capped		
		Ag (g/t)	Number	Proportion (%)	Metal Loss (5)
North	Sulphide	12.5	11	0.27	7.9
	Oxide	7.0	4	0.36	5.0
South	Sulphide	10.0	17	0.60	8.8
	Oxide	6.0	5	1.98	22.7
Tizate	Sulphide	15.0	8	0.20	1.1
	Oxide	8.0	1	0.39	0.1

Table 14-13: Tepal Property Capping Summary for Molybdenum

Zone	Domain	Threshold	Data Capped		
		Mo (ppm)	Number	Proportion (%)	Metal Loss (%)
North	Sulphide	300	5	0.12	0.9
	Oxide	200	1	0.09	0.1
South	Sulphide	-	-	-	-
	Oxide	-	-	-	-
Tizate	Sulphide	625	7	0.18	1.1
	Oxide	-	-	-	-

*Capping threshold derived by Decile Analysis and Log Probability plots.

14.1.6 Geostatistics

Spatial data analysis was considered prior to block model grade estimation in an attempt to generate a series of variograms and variogram maps that would define the directions of spatial continuity of gold and copper grades. The results of the variograms were used as input parameters for Ordinary Kriging grade estimation.

The drill spacing over the deposits is sufficient sample density to be able to generate variograms for gold and copper, especially in the sulphide zones. Average ranges from gold and copper is used so every block will be estimated with same search distance. Data are insufficient to generate variogram ranges for silver and molybdenum so the search range and orientation parameters for silver and molybdenum were derived from the gold and copper variogram. The following table summarizes the strike orientation and dip orientation of the variograms for each metal, with respect to each deposit and oxide/sulphide domain.

Table 14-14: Variogram Parameters, North Tepal

Zone	Metal	Nugget	Sill	Rotation			Ranges		
		C_0	$C_1/C_2/C_3$	Z	Y	X	X	Y	Z
North Tepal Oxide	Au	0.07	0.63	112.5	0	0	47	32	23
			0.3	112.5	0	0	79	320	42
	Cu	0.08	0.61	112.5	0	0	78	28	16
			0.3	112.5	0	0	109	175	79
	Ag	0.06	0.2	112.5	0	0	8	3	5
			0.57	112.5	0	0	20	12	71
			0.17	112.5	0	0	89	105	117
	Mo	0.05	0.26	112.5	0	0	8	3	5
			0.5	112.5	0	0	20	12	71
			0.19	112.5	0	0	89	105	117
North Tepal Sulphide	Au	0.1	0.3	112.5	0	0	20	8	7
			0.35	112.5	0	0	37	67	52
			0.25	112.5	0	0	152	134	198
	Cu	0.16	0.37	112.5	0	0	6	10	7
			0.25	112.5	0	0	51	29	33
			0.23	112.5	0	0	129	158	127
	Ag	0.07	0.29	112.5	0	0	7	17	13
			0.44	112.5	0	0	84	60	77
			0.2	112.5	0	0	133	119	217
	Mo	0.09	0.26	112.5	0	0	20	12	12
			0.37	112.5	0	0	71	55	59
			0.29	112.5	0	0	124	117	194

Table 14-15: Variogram Parameters, South Tepal

Zone	Metal	Nugget	Sill	Rotation			Ranges		
		C_0	$C_1/C_2/C_3$	Z	Y	X	X	Y	Z
South Tepal Oxide	Au	0.06	0.35	80.25	30	35.25	32	8	7
			0.01	80.25	30	35.25	66	62	32
			0.59	80.25	30	35.25	116	211	84
	Cu	0.19	0.39	80.25	30	35.25	10	10	4
			0.42	80.25	30	35.25	39	47	15
	Ag	0.13	0.25	80.25	30	35.25	6	10	5
			0.56	80.25	30	35.25	32	37	115
			0.06	80.25	30	35.25	83	69	200
	Mo	0.06	0.46	80.25	30	35.25	15	17	6
			0.48	80.25	30	35.25	73	91	71
South Tepal Sulphide	Au	0.08	0.4	80.25	30	35.25	50	12	7
			0.34	80.25	30	35.25	74	83	90
			0.18	80.25	30	35.25	127	510	238
	Cu	0.1	0.5	80.25	30	35.25	54	22	18
			0.28	80.25	30	35.25	77	105	53
			0.12	80.25	30	35.25	123	334	241
	Ag	0.13	0.64	80.25	30	35.25	22	6	29
			0.06	80.25	30	35.25	126	163	117
			0.17	80.25	30	35.25	278	305	191
	Mo	0.13	0.53	80.25	30	35.25	9	8	22
			0.27	80.25	30	35.25	28	153	119
			0.07	80.25	30	35.25	83	284	248

Table 14-16: Variogram Parameters, Tizate

Zone	Metal	Nugget	Sill	Rotation			Ranges		
		C_0	$C_1/C_2/C_3$	Z	Y	X	X	Y	Z
Tizate Oxide	Au	0.14	0.36	-28.68	15.7	42.74	5	5	6
			0.51	-28.68	15.7	42.74	144	200	82
	Cu	0.07	0.49	-28.68	15.7	42.74	19	8	4
			0.45	-28.68	15.7	42.74	141	68	166
	Ag	0.05	0.31	-28.68	15.7	42.74	21	7	7
			0.64	-28.68	15.7	42.74	137	51	117
	Mo	0.15	0.47	-28.68	15.7	42.74	15	12	5
			0.38	-28.68	15.7	42.74	108	75	208
T*izate* Sulphide	Au	0.17	0.29	-28.68	15.7	42.74	38	17	6
			0.41	-28.68	15.7	42.74	81	84	28
			0.12	-28.68	15.7	42.74	167	250	246
	Cu	0.16	0.28	-28.68	15.7	42.74	18	8	8
			0.38	-28.68	15.7	42.74	69	92	27
			0.18	-28.68	15.7	42.74	229	189	372
	Ag	0.09	0.31	-28.68	15.7	42.74	6	8	6
			0.33	-28.68	15.7	42.74	72	34	39
			0.26	-28.68	15.7	42.74	138	360	295
	Mo	0.1	0.3	-28.68	15.7	42.74	28	6	10
			0.37	-28.68	15.7	42.74	91	88	34
			0.23	-28.68	15.7	42.74	297	126	333

14.1.7 Specific Gravity

Specific gravity (SG) samples were collected approximately every 50 metres in the sulphide zone from all available Arian and Geologix core from the three deposits. Samples were taken from mineralized and non-mineralized core (i.e. ore and waste). The oxide samples were collected from as many Arian holes as possible and from the 2010 Geologix core. There were also oxide samples taken from two 2011 Tizate holes (TIZ-11-001 to TIZ-11-037). A total of 1,053 samples have had SG determinations.

SG determination for each sample was performed by ALS, Vancouver, BC. SG measurements were derived by gravimetric methods. Core was covered in a paraffin wax coating and weighed. The sample was then weighed while it was suspended in water and the SG determined by measuring the volumetric displacement of the rock in water and dividing the weight of rock by the volume. The Table 14-17 lists the SG for each zone and domain used in the block model.

Table 14-17: Tepal Property SG Averages

Zone	Domain	Category	Density (t/m^3)	No. Samples
North	Oxide	Ore	2.42	13
	Sulphide	Ore	2.70	86
	Oxide	Waste	2.45	14
	Sulphide	Waste	2.73	229
South	Oxide	Ore	2.46	4
	Sulphide	Ore	2.72	81
	Oxide	Waste	2.45	16
	Sulphide	Waste	2.73	109
Tizate	Oxide	Ore	2.49	4
	Sulphide	Ore	2.74	169
	Oxide	Waste	2.39	10
	Sulphide	Waste	2.73	318
Total				1053

The number of oxide ore sample determinations is low compared to sulphide determinations. Micon recommends that additional oxide ore samples be sent to ALS for SG determination to obtain a more representative average oxide SG in each deposit.

14.1.8 Block Model

Two block models were created. The Tepal block model contains both the North and South Zones. The Tizate block model encompasses the Tizate Zone. The block model extents are documented in Table 14-18 and Table 14-19.

Table 14-18: Tepal (North & South Zones) Block Model Limits (UTM)

Axis	Minimum (m)	Maximum (m)	Block Size (m)	No. of Blocks
X (North)	715,600	718,100	10	250
Y (East)	2,114,800	2,117,800	10	300
Z (Elev.)	-300	1,000	5	260

Table 14-19: Tizate Block Model Limits (UTM)

Axis	Minimum (m)	Maximum (m)	Block Size (m)	No. of Blocks
X (North)	717,500	719,900	10	240
Y (South)	2,115,800	2,117,650	10	185
Z (Elev.)	-100	1,000	5	220

A series of block model codes were developed to identify the zones and domains within the block models. Table 14-20 documents these codes. No sub-blocks were created in the model to facilitate transfer of the block model to other software platforms.

Table 14-20: Tepal Property Block Codes

Code	Description
101	Tepal North Oxide Ore
102	Tepal North Sulphide Ore
129	Tepal North Oxide Waste
130	Tepal North Sulphide Waste
201	Tepal South Oxide Ore
202	Tepal South Sulphide Ore
229	Tepal South Oxide Waste
230	Tepal South Sulphide Waste
301	Tizate Oxide Ore
302	Tizate Sulphide Ore
329	Tizate Oxide Waste
330	Tizate Sulphide Waste

14.1.9 Grade Interpolation

Gold, copper, silver and molybdenum grades were interpolated into both block models. The interpolation for each block model was constrained by block codes and the respective mineralogical model domains. Interpolation only used composite data falling within the constraints. Blocks outside the constraints were also interpolated using the same boundary constraints.

Each block model used the Ordinary Kriging (OK) method to estimate the grades in each block. Interpolation was performed using multiple passes with successively larger search ellipses until all blocks within each domain had received an interpolated grade. The search distances were derived from the ranges derived from the variogram analysis. To ensure that clustered sample groups did not preferentially bias block grades, interpolations included a restriction on the minimum and maximum number of samples used as well as the maximum number of samples used per drill holes. Interpreted search ellipse parameters for each model are documented in Table 14-21.

Table 14-21: Search Parameters

	Metal	Search Pass	Rotation			Range			Composites		Max. per Hole
			Z	Y	X	X	Y	Z	Min	Max	
			(°)	(°)	(°)	(m)	(m)	(m)			
North Tepal	Oxide	1	45	0	0	49	68	23	5	15	4
		2	45	0	0	74	102	34	5	15	4
		3	45	0	0	123	170	57	4	15	4
	Sulphide	1	45	0	0	40	41	41	5	15	4
		2	45	0	0	60	62	62	5	15	4
		3	45	0	0	100	103	103	4	15	4
South Tepal	Oxide	1	45	45	0	**Zone**	63	25	5	15	4
		2	45	45	0	62	94	38	5	15	4
		3	45	45	0	103	157	63	4	15	4
	Sulphide	1	45	45	0	48	53	43	5	15	4
		2	45	45	0	72	80	64	5	15	4
		3	45	45	0	120	133	107	4	15	4
Tizate	Oxide	1	315	45	0	88	82	73	5	15	4
		2	315	45	0	176	164	146	4	15	4
	Sulphide	1	315	45	0	70	79	25	5	15	4
		2	315	45	0	140	158	50	4	15	4

14.1.10 Block Model Validation

Global validation of the block models were undertaken to confirm the OK method was reporting the appropriate results. To validate the block models for global bias, the models were re-estimated by using the Inverse Distance Squared (ID2) and the Nearest Neighbour (NN) methods. The following table documents the metal loss of the two different methods compared to OK for each deposit.

Table 14-22: Metal Loss Comparison between OK and ID2 and NN

Domain	ID2		NN	
	Gold	Copper	Gold	Copper
	Metal Loss (%)	Metal Loss (%)	Metal Loss (%)	Metal Loss (%)
Tepal North	-2.1	-1.2	0.7	2.4
Tepal South	-1.9	-1.3	-0.4	-0.1
Tizate	-1	-0.8	1.4	1.3

Note: Based on US$ 5 equivalent

The Table 14-22 shows that there are small losses and gains of metal compared to OK. These small losses and gains validate that the OK method is not biasing for any of the deposits.

Normally, both methods (ID2 and NN) tend to under-estimate the tonnage and over-estimate the grade compared to the OK method. In general, the NN method tends to over-estimate the grade more than ID2 method. The table illustrates these relationships.

Swath plots were generated on each deposit for gold and copper. The plots include declustered composite sulphide grades compared to OK, ID2 and NN sulphide block grades in west-east, south-north and vertical directions through each deposit.

Figure 14-1: Tepal North Sulphide Gold W-E Swath Plot

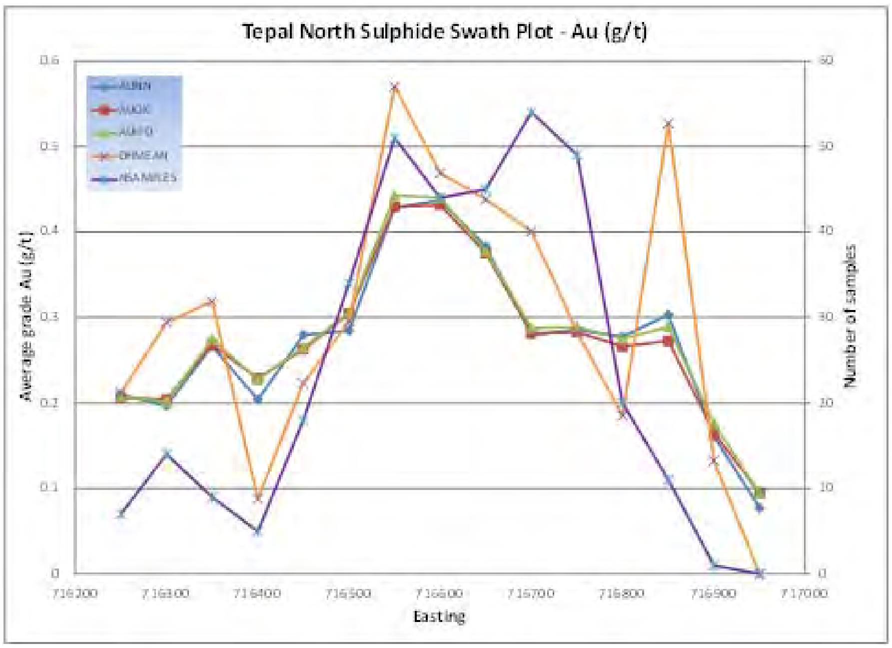


Figure 14-2 and 14-5 illustrate a potential starter pit at approximately 2117000 mN.

Figure 14-2: Tepal North Sulphide Gold S-N Swath Plot

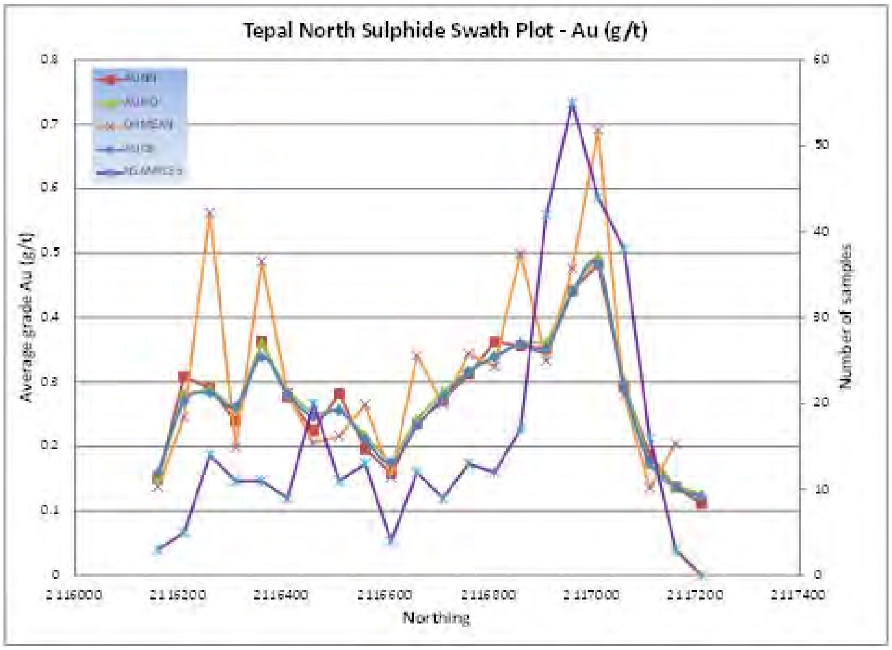


Figure 14-3: Tepal North Sulphide Gold Elevation Swath Plot

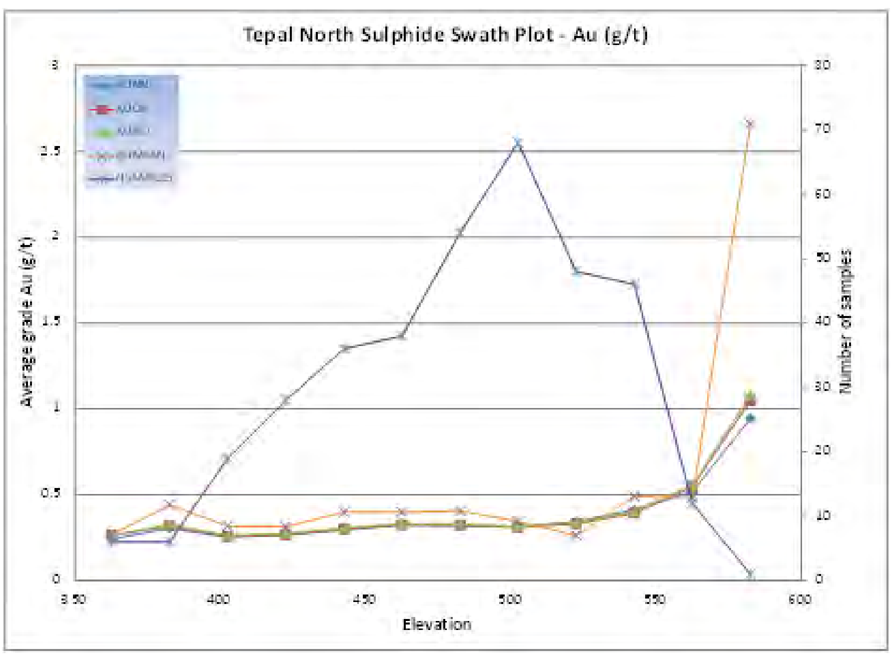


Figure 14-4: Tepal North Sulphide Copper W-E Swath Plot

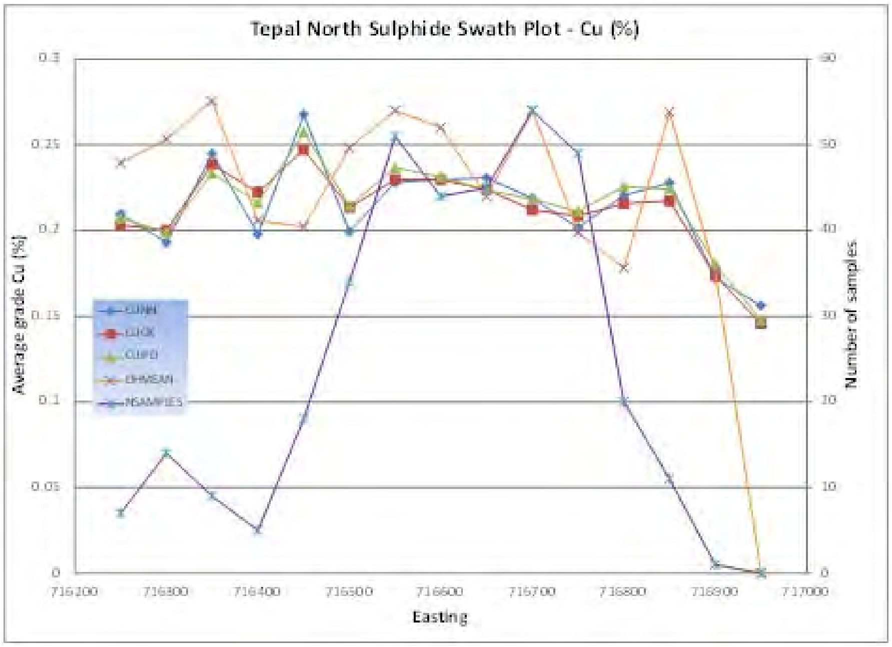


Figure 14-5: Tepal North Sulphide Copper S-N Swath Plot

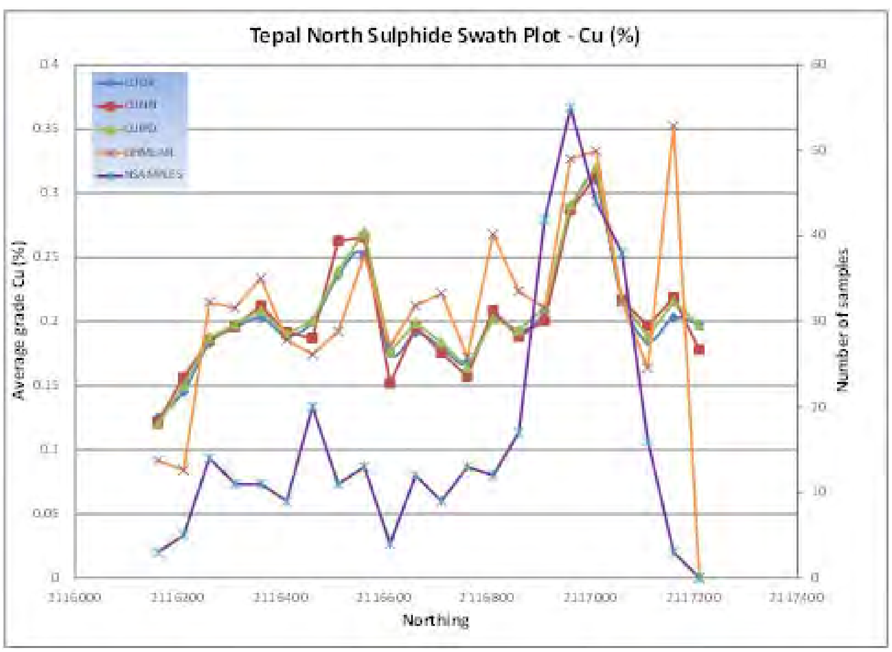


Figure 14-6: Tepal North Sulphide Copper Elevation Swath Plot

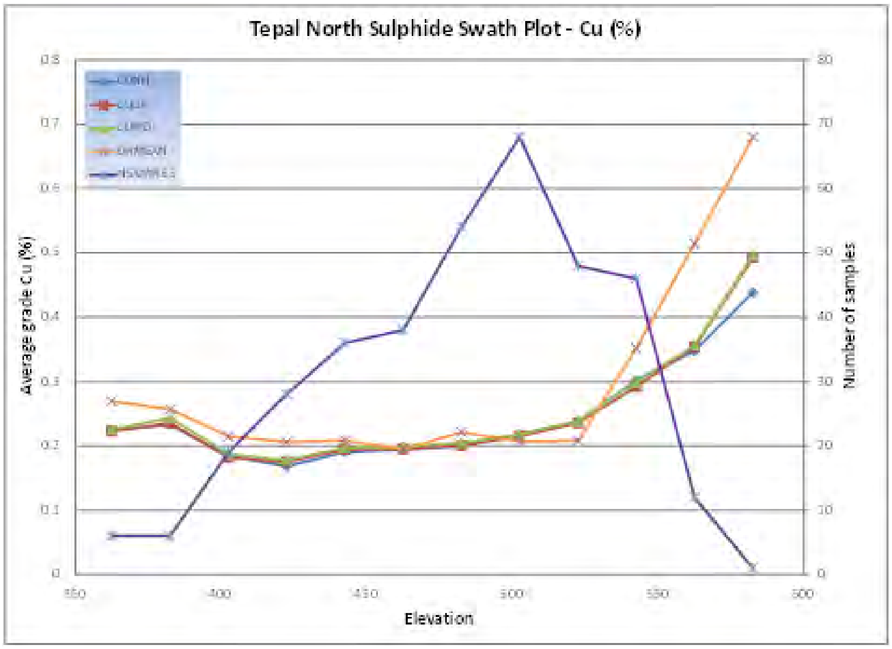


Figure 14-7: Tepal South Sulphide Gold W-E Swath Plot

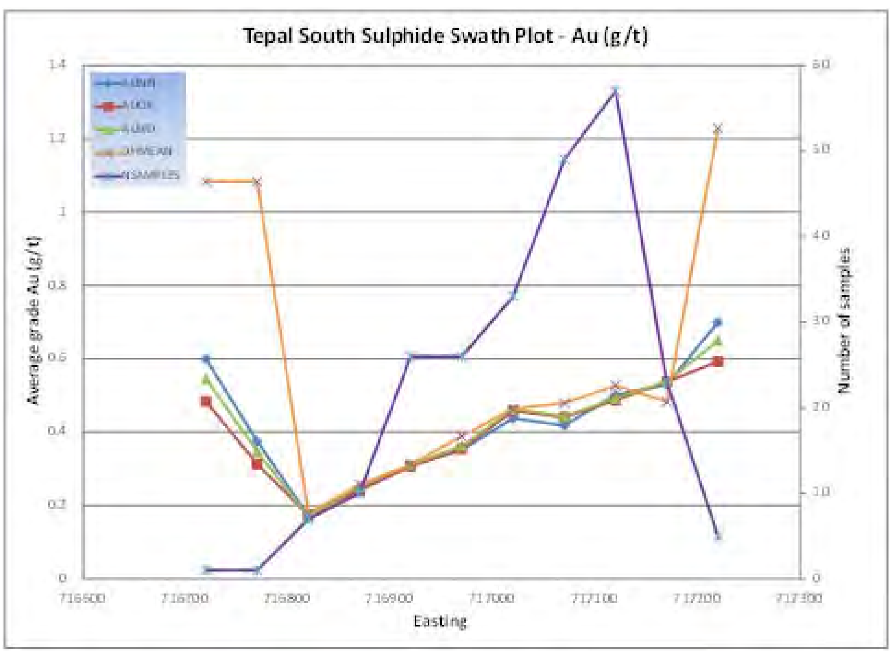


Figure 14-8: Tepal South Sulphide Gold S-N Swath Plot

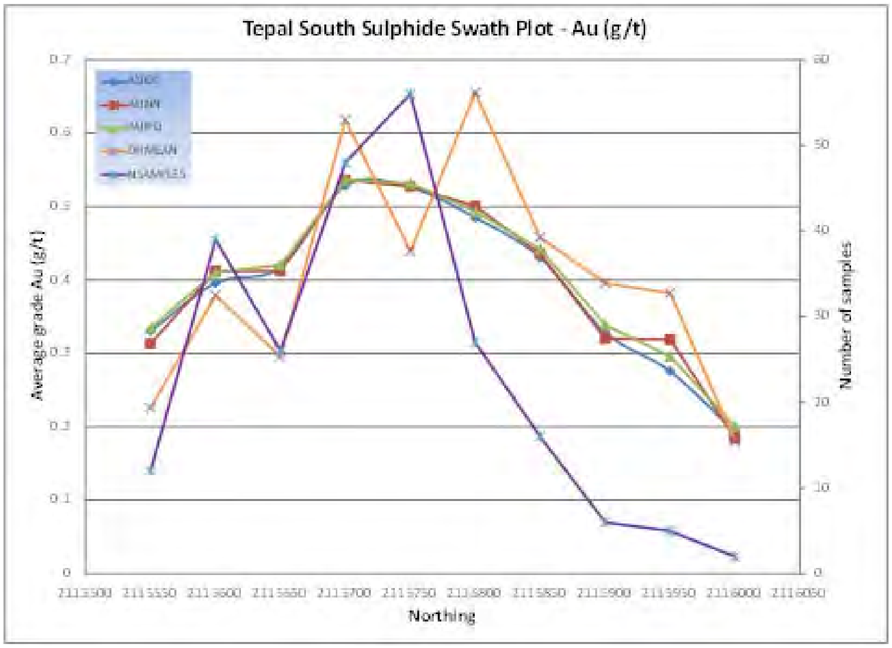


Figure 14-9 and 14-12 illustrate the high grade mineralization below the South Zone optimized soft pit.

Figure 14-9: Tepal South Sulphide Gold Elevation Swath Plot

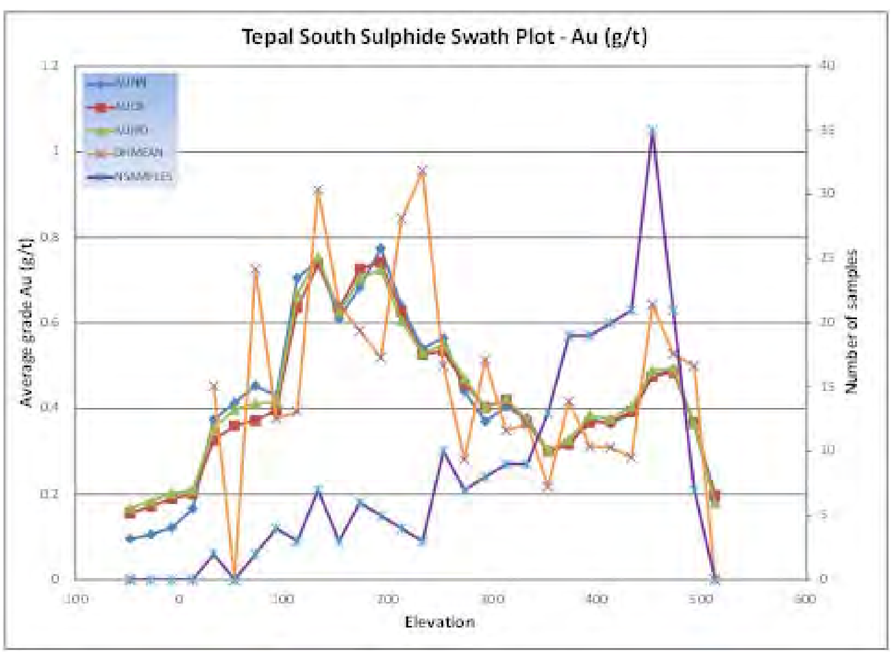


Figure 14-10: Tepal South Sulphide Copper W-E Swath Plot

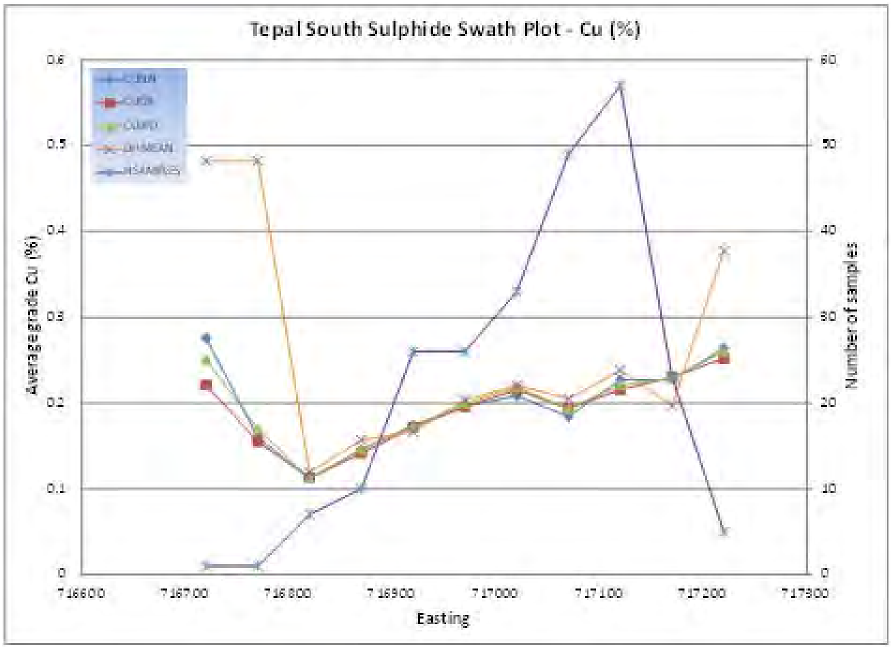


Figure 14-11: Tepal South Sulphide Copper S-N Swath Plot

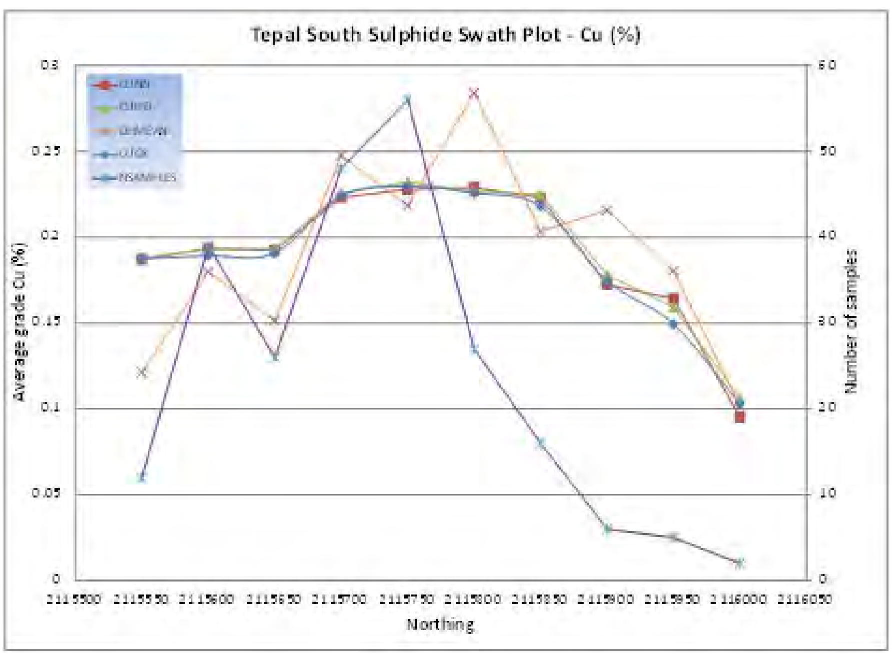


Figure 14-12: Tepal South Sulphide Copper Elevation Swath Plot

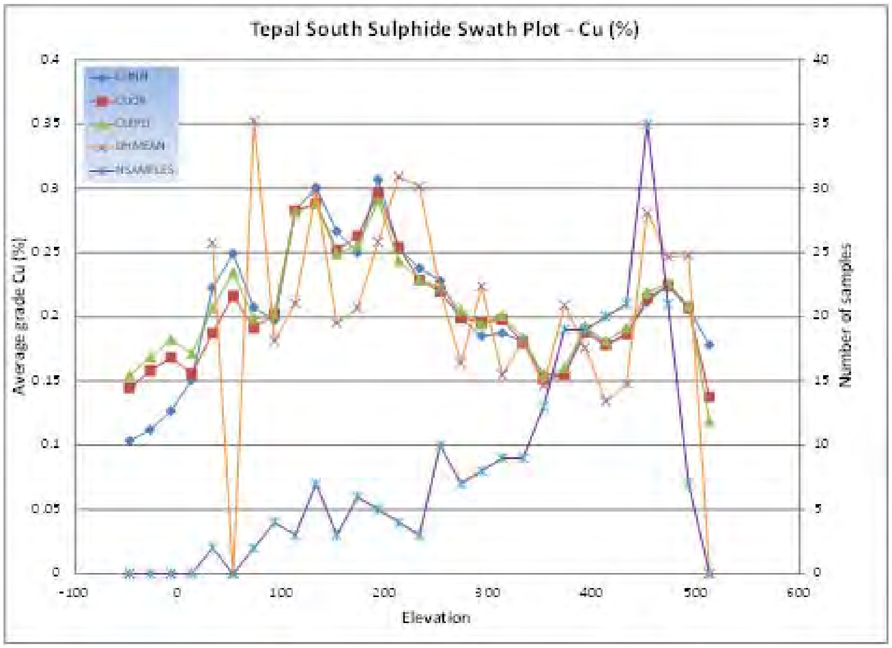


Figure 14-13: Tizate Sulphide Gold W-E Swath Plot

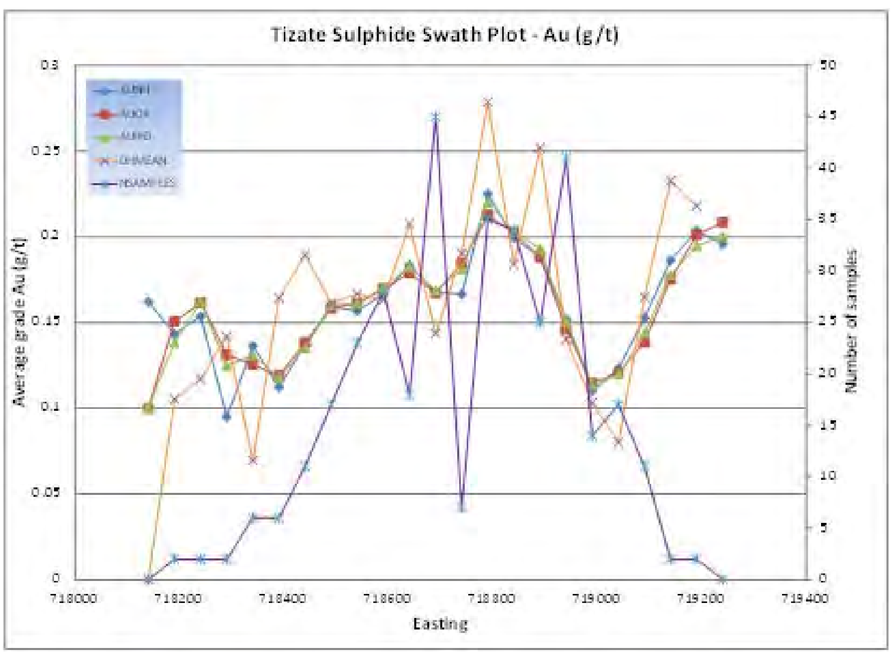


Figure 14-14: Tizate Sulphide Gold S-N Swath Plot

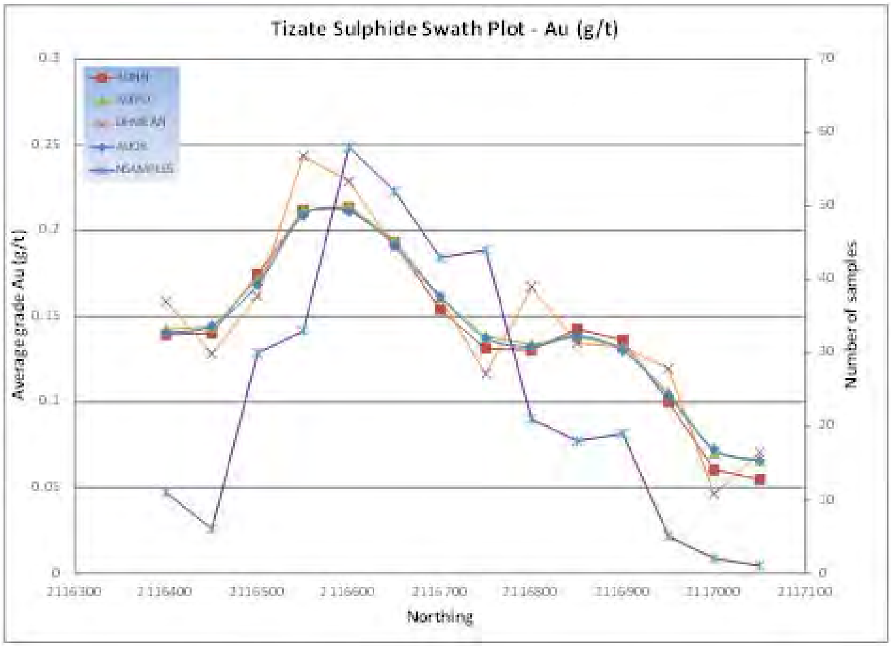


Figure 14-15: Tizate Sulphide Gold Elevation Swath Plot

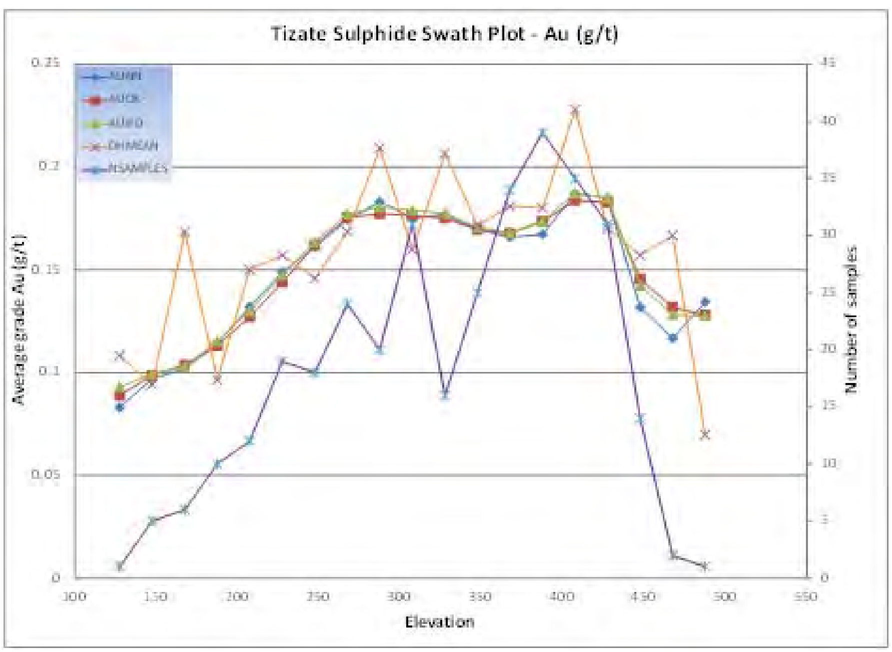


Figure 14-16: Tizate Sulphide Copper W-E Swath Plot

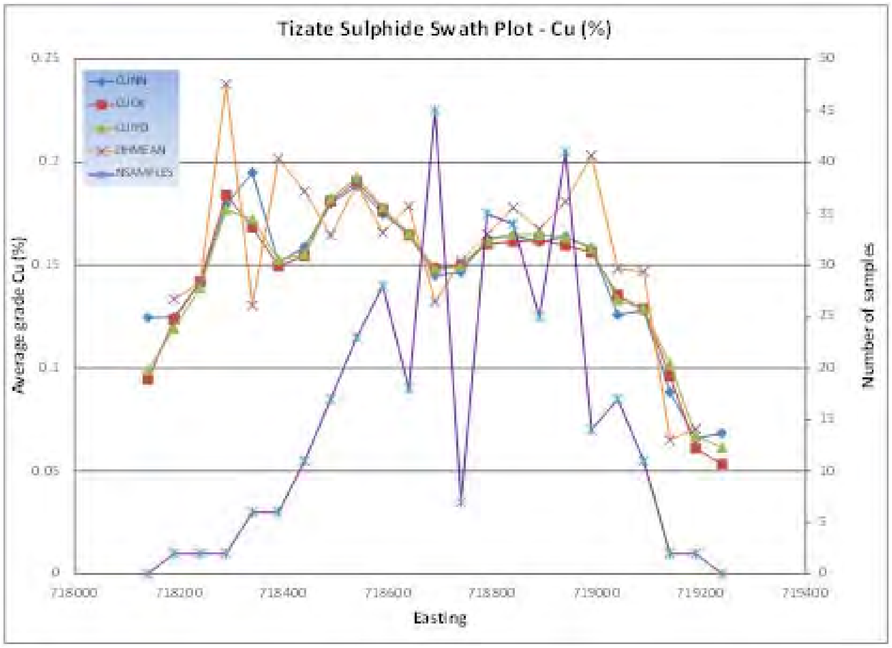


Figure 14-17: Tizate Sulphide Copper S-N Swath Plot

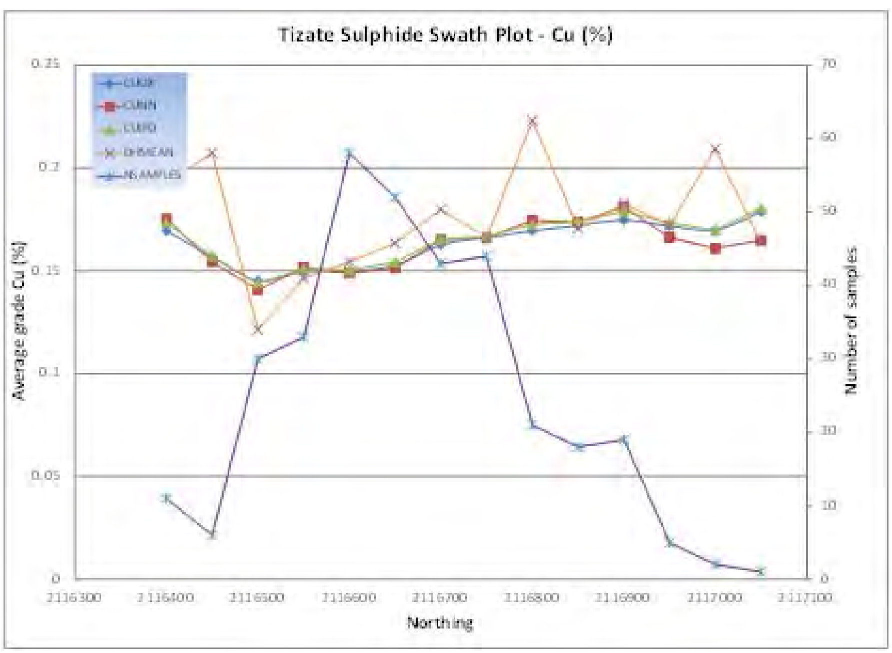
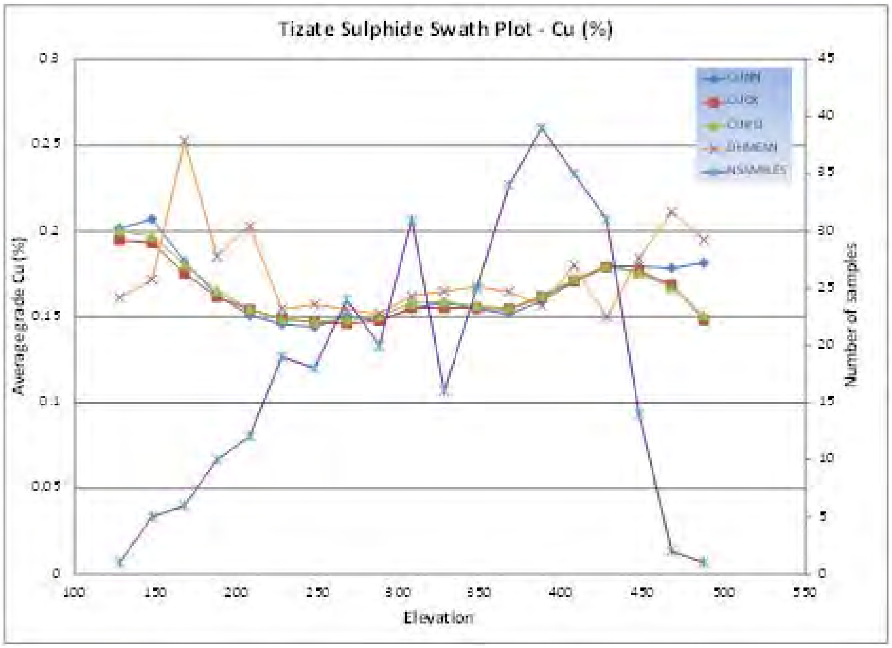


Figure 14-18: Tizate Sulphide Copper Elevation Swath Plot



The swath plots illustrate that all three interpolation method block grades compare well with each other. All three sets of block grades trend well with the composite grades for both metals, in all three axes and for all three deposits. The NN block grades show the most variability especially when there are a small set of samples like near the edges of deposits.

A comparison of the gold and copper composites has been compared to the blocks in the models to assess the potential of over or under estimating during interpolation. Tables 14-23 to 14-28 list the statistics for the various domains in each deposit.

Table 14-23: Tepal North Sulphide Domain Gold & Copper Composite versus BM Statistics

Statistics	Au (g/t)		Cu (%)	
	Composite	Block Model	Composite	Block Model
Mean	0.38	0.33	0.24	0.22
Minimum	0.00	0.00	0.00	0.02
Maximum	3.00	2.65	2.50	1.73
Standard Deviation	0.40	0.23	0.21	0.12
Coeff. of Variation	1.05	0.71	0.85	0.54
Number of Samples	4,135	44,445	4,135	44,445

Table 14-24: Tepal North Oxide Domain Gold & Copper Composite versus BM Statistics

Statistics	Au (g/t)		Cu (%)	
	Composite	Block Model	Composite	Block Model
Mean	0.39	0.35	0.23	0.21
Minimum	0.00	0.00	0.00	0.00
Maximum	2.52	1.91	3.23	1.75
Standard Deviation	0.39	0.24	0.26	0.14
Coeff. of Variation	1.00	0.68	1.12	0.68
Number of Samples	1,097	12,681	1,097	12,681

Table 14-25: Tepal South Sulphide Domain Gold & Copper Composite versus BM Statistics

Statistics	Au (g/t)		Cu (%)	
	Composite	Block Model	Composite	Block Model
Mean	0.48	0.45	0.22	0.21
Minimum	0.00	0.01	0.00	0.00
Maximum	2.54	2.08	1.00	0.69
Standard Deviation	0.39	0.27	0.14	0.09
Coeff. of Variation	0.81	0.60	0.62	0.45
Number of Samples	2,855	35,541	2,855	35,541

Table 14-26: Tepal South Oxide Domain Gold & Copper Composite versus BM Statistics

Statistics	Au (g/t)		Cu (%)	
	Composite	Block Model	Composite	Block Model
Mean	0.41	0.41	0.19	0.18
Minimum	0.00	0.06	0.00	0.04
Maximum	1.10	0.89	0.77	0.43
Standard Deviation	0.27	0.19	0.11	0.06
Coeff. of Variation	0.66	0.45	0.58	0.32
Number of Samples	253	3,227	253	3,227

Table 14-27: Tizate Sulphide Domain Gold & Copper Composite versus BM Statistics

Statistics	Au (g/t)		Cu (%)	
	Composite	Block Model	Composite	Block Model
Mean	0.18	0.17	0.17	0.16
Minimum	0.00	0.02	0.00	0.01
Maximum	1.10	0.76	0.80	0.57
Standard Deviation	0.13	0.08	0.08	0.05
Coeff. of Variation	0.74	0.48	0.48	0.29
Number of Samples	3,932	82,837	3,932	82,837

Table 14-28: Tizate Oxide Domain Gold & Copper Composite versus BM Statistics

Statistics	Au		Cu	
	Composite	Block Model	Composite	Block Model
Mean	0.19	0.17	0.18	0.17
Minimum	0.01	0.03	0.01	0.05
Maximum	0.60	0.48	0.50	0.41
Standard Deviation	0.13	0.08	0.10	0.05
Coeff. of Variation	0.68	0.44	0.56	0.28
Number of Samples	255	7,396	255	7,396

The statistics indicate that the degree of smoothing has been reduced due to the in-fill drilling program. Composites and the blocks correlate well with each other in most domains, even though the composite number of samples is significantly smaller. This indicates that the blocks are being interpolated correctly and without bias, on a statistical basis.

The block models and accompanying drill hole database were compared visually in section (east-west). Visually the blocks and their respective grade attributes corresponded well to both grade and 3D location of the mineralized intervals within the database.

Micon believes that the block model results portray a reliable estimate of the mineralization within each of the deposits, with the available data.

14.1.11 Classification

Mineral resource reporting in Canada follows National Instrument 43-101 and its companion policy 43-101CP and technical report requirements 43-101F1 which have been in place since February 1, 2001. The mineral resource definitions are based on the Canadian Institute of Mining, Metallurgy and Petroleum's (CIM) definitions (CIM Definition Standards – For Mineral Resources and Mineral Reserves, adopted on November 27, 2010).

Under these definitions:

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. (CIM, 2010)

There are three subdivisions within the mineral resource category, which are based on decreasing geological confidence (*Measured, Indicated and Inferred*). The Tepal property has mineral resources in all three categories based on geostatistics. The definitions of the categories are as follows:

14.1.11.1 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.

14.1.11.2 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.

14.1.11.3 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.

In order to determine the resources that offered a reasonable prospect for economic extraction from an open pit, Micon used the Whittle pit mining software package to create soft pits. The software evaluated the profitability of each resource block within each model, based on the parameters listed in Table 14-29.

Table 14-29: Soft Pit Optimization Parameters

Parameters	Units	Oxide	Sulphide	Comment
Mining Cost	US$/t	1.35	1.35	SRK PA report, April 29, 2011
Processing Cost	US$	4.30	4.30	SRK PA report, April 29, 2011
G & A	US$/t	0.68	0.68	SRK PA report, April 29, 2011
Gold Price	US$/oz	1300	1300	Kitco 3 yr trailing avg. 02/2012
Copper Price	US$/lb	3.3	3.3	LME 3 yr trailing avg. 02/2012
Recovery Tizate Au	%	68.8	66.2	SRK PA report, April 29, 2011
Recovery Tizate Cu	%	6.8	85.3	SRK PA report, April 29, 2011
Recovery Tepal Au	%	78.4	60.7	SRK PA report, April 29, 2011
Recovery Tepal Cu	%	14.3	87.4	SRK PA report, April 29, 2011
Pit Slope Angle	°	45	45	SRK PA report, April 29, 2011

Note: The SRK PA values will be updated during the Prefeasibility Study

Using the soft pit and the mineralogical models as constraints on the block model, the following mineral resource estimates were derived using a range of equivalent value cut-offs. The following tables document the different mineral resources at various equivalent cut-off values for the deposits with respect to oxides and sulphides. However Micon believes that US$5.00/t equivalent is an appropriate cut-off value that would represent a break even open pit mining cost operation with a mining rate of approximately 35,000tpd of ore which is anticipated by Geologix.

The mineral resource classification was based on variography and the resulting search passes. For North and South Tepal, search pass 1 represented the measured category, search pass 2 represented the Indicated category and search pass 3 represented the Inferred category. For the Tizate, search pass 1 represented the Indicated category and search pass 2 represented the Inferred category. There are no measured blocks in Tizate.

Both Measured and Indicated categories were forced to look for 2 drill holes (maximum 4 composites per hole) and 5 composites total (Table 14-21). The Inferred category needed 1 drill hole (maximum 4 composites per hole) and 4 composites total (Table 14-21).

Mineral esources are summarized in Tables 14-30 to 14-35.

Table 14-30: Tepal North Zone Oxide Mineral Resources

Resource	Cut-off	Tonnes	Average Grade				Metal	
Class	Eq. V.		Au	Cu	Ag	Mo	Au	Cu
	($/t)	(x1000)	(g/t)	(%)	(g/t)	(%)	(koz)	(Mlb)
Measured	1	3,455	0.50	0.30	0.71	0.001	56	23
Measured	3	3,447	0.50	0.30	0.71	0.001	56	23
Measured	**5**	**3,398**	**0.51**	**0.31**	**0.72**	**0.001**	**56**	**23**
Measured	7	3,085	0.55	0.32	0.75	0.001	54	22
Measured	9	2,761	0.59	0.33	0.77	0.001	52	20

Indicated	1	10,359	0.30	0.18	0.93	0.002	99	42
Indicated	3	10,330	0.30	0.18	0.93	0.002	99	42
Indicated	**5**	**10,050**	**0.30**	**0.19**	**0.94**	**0.002**	**98**	**41**
Indicated	7	8,712	0.33	0.19	0.97	0.002	92	37
Indicated	9	6,402	0.38	0.20	1.02	0.002	78	28

M + I	1	13,814	0.35	0.21	0.87	0.002	155	65
M + I	3	13,776	0.35	0.21	0.88	0.002	155	65
M + I	**5**	**13,448**	**0.36**	**0.22**	**0.88**	**0.002**	**154**	**64**
M + I	7	11,797	0.39	0.23	0.91	0.002	146	59
M + I	9	9,163	0.44	0.24	0.94	0.002	130	48

Inferred	1	30	0.24	0.18	0.77	0.002	0.2	0.1
Inferred	3	28	0.26	0.19	0.82	0.002	0.2	0.1
Inferred	**5**	**24**	**0.29**	**0.21**	**0.86**	**0.002**	**0.2**	**0.1**
Inferred	7	21	0.31	0.22	0.8	0.002	0.2	0.1
Inferred	9	15	0.34	0.26	0.73	0.002	0.2	0.1

Table 14-31: Tepal North Zone Sulphide Mineral Resources

Resource Class	Cut-off Eq. V. ($/t)	Tonnes (x1000)	Average Grade Au (g/t)	Average Grade Cu (%)	Average Grade Ag (g/t)	Average Grade Mo (%)	Metal Au (koz)	Metal Cu (Mlb)
Measured	1	10,670	0.50	0.28	0.81	0.002	172	66
Measured	3	10,670	0.50	0.28	0.81	0.002	172	66
Measured	**5**	**10,669**	**0.50**	**0.28**	**0.81**	**0.002**	**172**	**66**
Measured	7	10,623	0.50	0.28	0.81	0.002	172	66
Measured	9	10,457	0.51	0.28	0.81	0.002	172	66
Indicated	1	45,335	0.30	0.21	1.02	0.002	435	211
Indicated	3	45,325	0.30	0.21	1.02	0.002	435	211
Indicated	**5**	**45,270**	**0.30**	**0.21**	**1.02**	**0.002**	**435**	**211**
Indicated	7	45,016	0.30	0.21	1.03	0.002	434	210
Indicated	9	44,110	0.30	0.21	1.03	0.002	431	209
M + I	1	56,005	0.34	0.22	0.98	0.002	607	277
M + I	3	55,996	0.34	0.22	0.98	0.002	607	277
M + I	**5**	**55,939**	**0.34**	**0.22**	**0.98**	**0.002**	**607**	**277**
M + I	7	55,639	0.34	0.23	0.98	0.002	606	276
M + I	9	54,567	0.34	0.23	0.99	0.002	602	274
Inferred	1	882	0.22	0.21	1.22	0.003	6	4
Inferred	3	882	0.22	0.21	1.22	0.003	6	4
Inferred	**5**	**882**	**0.22**	**0.21**	**1.22**	**0.003**	**6**	**4**
Inferred	7	874	0.22	0.21	1.23	0.003	6	4
Inferred	9	863	0.23	0.21	1.23	0.003	6	4

Table 14-32: Tepal South Zone Oxide Mineral Resource

Resource Class	Cut-off Eq. V. ($/t)	Tonnes (x1000)	Average Grade Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Metal Au (koz)	Cu (Mlb)
Measured	1	2,145	0.46	0.20	1.06	0.001	32	9
Measured	3	2,140	0.46	0.20	1.07	0.001	32	9
Measured	**5**	**2,103**	**0.47**	**0.20**	**1.08**	**0.001**	**32**	**9**
Measured	7	2,035	0.48	0.20	1.09	0.001	31	9
Measured	9	1,917	0.50	0.21	1.11	0.001	31	9
Indicated	1	1,484	0.34	0.17	0.90	0.002	16	5
Indicated	3	1,483	0.34	0.17	0.90	0.002	16	5
Indicated	**5**	**1,380**	**0.36**	**0.17**	**0.94**	**0.002**	**16**	**5**
Indicated	7	1,127	0.41	0.18	1.02	0.001	15	5
Indicated	9	954	0.45	0.19	1.07	0.001	14	4
M + I	1	3,629	0.41	0.18	1.00	0.001	48	15
M + I	3	3,623	0.41	0.18	1.00	0.001	48	15
M + I	**5**	**3,483**	**0.43**	**0.19**	**1.02**	**0.001**	**48**	**14**
M + I	7	3,162	0.45	0.2	1.07	0.001	46	14
M + I	9	2,871	0.48	0.2	1.09	0.001	44	13
Inferred	1	47	0.28	0.13	0.75	0.002	0	0
Inferred	3	47	0.28	0.13	0.75	0.002	0	0
Inferred	**5**	**46**	**0.28**	**0.13**	**0.76**	**0.001**	**0**	**0**
Inferred	7	43	0.29	0.13	0.76	0.002	0	0
Inferred	9	30	0.32	0.14	0.72	0.002	0	0

Table 14-33: Tepal South Zone Sulphide Mineral Resource

Resource	Cut-off	Tonnes	Average Grade				Metal	
Class	Eq. V.		Au	Cu	Ag	Mo	Au	Cu
	($/t)	(x1000)	(g/t)	(%)	(g/t)	(%)	(koz)	(Mlb)
Measured	1	17,908	0.47	0.22	1.07	0.002	268	87
Measured	3	17,908	0.47	0.22	1.07	0.002	268	87
Measured	**5**	**17,908**	**0.47**	**0.22**	**1.07**	**0.002**	**268**	**87**
Measured	7	17,908	0.47	0.22	1.07	0.002	268	87
Measured	9	17,767	0.47	0.22	1.07	0.002	268	86
Indicated	1	19,786	0.45	0.20	1.19	0.002	289	86
Indicated	3	19,734	0.46	0.20	1.19	0.002	289	86
Indicated	**5**	**19,613**	**0.46**	**0.20**	**1.19**	**0.002**	**289**	**86**
Indicated	7	19,281	0.46	0.20	1.19	0.002	288	86
Indicated	9	18,455	0.48	0.21	1.19	0.002	284	85
M + I	1	37,694	0.46	0.21	1.13	0.002	558	173
M + I	3	37,642	0.46	0.21	1.13	0.002	558	173
M + I	**5**	**37,521**	**0.46**	**0.21**	**1.13**	**0.002**	**557**	**173**
M + I	7	37,189	0.47	0.21	1.13	0.002	556	173
M + I	9	36,221	0.47	0.21	1.13	0.002	552	171
Inferred	1	366	0.42	0.17	0.97	0.002	5	1
Inferred	3	366	0.42	0.17	0.97	0.002	5	1
Inferred	**5**	**366**	**0.42**	**0.17**	**0.97**	**0.002**	**5**	**1**
Inferred	7	366	0.42	0.17	0.97	0.002	5	1
Inferred	9	346	0.43	0.17	1	0.002	5	1

Table 14-34: Tizate Zone Oxide Mineral Resources

Resource	Cut-off	Tonnes	Average Grade				Metal	
Class	Eq. V.		Au	Cu	Ag	Mo	Au	Cu
	($/t)	(x1000)	(g/t)	(%)	(g/t)	(%)	(koz)	(Mlb)
Indicated	1	5,997	0.20	0.18	2.45	0.003	38	24
Indicated	3	5,904	0.20	0.18	2.46	0.003	38	23
Indicated	**5**	**4,181**	**0.23**	**0.19**	**2.27**	**0.003**	**31**	**17**
Indicated	7	2,288	0.28	0.19	2.19	0.003	21	10
Indicated	9	954	0.33	0.20	1.79	0.003	10	4
Inferred	1	2,341	0.13	0.14	2.26	0.003	10	7
Inferred	3	2,176	0.13	0.14	2.27	0.003	9	7
Inferred	**5**	**640**	**0.17**	**0.13**	**2.14**	**0.002**	**4**	**2**
Inferred	7	19	0.25	0.19	2.60	0.004	0	0
Inferred	9	5	0.29	0.19	2.22	0.003	0	0

Table 14-35 Tizate Zone Sulphide Mineral Resources

Resource	Cut-off	Tonnes	Average Grade				Metal	
Class	Eq. V.		Au	Cu	Ag	Mo	Au	Cu
	($/t)	(x1000)	(g/t)	(%)	(g/t)	(%)	(koz)	(Mlb)
Indicated	1	73,335	0.17	0.17	2.28	0.007	407	267
Indicated	3	73,334	0.17	0.17	2.28	0.007	407	267
Indicated	**5**	**73,194**	**0.17**	**0.17**	**2.29**	**0.007**	**406**	**267**
Indicated	7	72,516	0.17	0.17	2.3	0.007	405	266
Indicated	9	69,771	0.18	0.17	2.33	0.007	397	261
Inferred	1	33,887	0.15	0.15	1.69	0.007	166	113
Inferred	3	33,872	0.15	0.15	1.69	0.007	166	113
Inferred	**5**	**33,786**	**0.15**	**0.15**	**1.69**	**0.007**	**166**	**113**
Inferred	7	33,343	0.15	0.15	1.70	0.007	165	112
Inferred	9	31,331	0.16	0.16	1.74	0.007	159	108

14.1.12 Cut-off Grade Sensitivity

The following graphs illustrate the Tepal North, Tepal South and Tizate Zones sensitivities of tonnage and grade to cut-off values.

Figure 14-19: Grade/Tonnage Curve for Tepal North Measured & Indicated Sulphide Mineral Resource

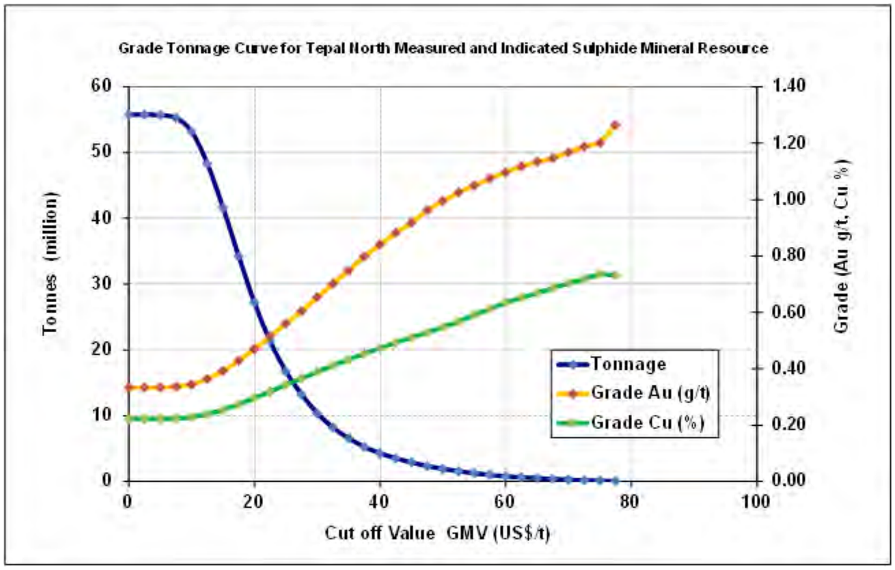


Figure 14-20: Grade/Tonnage Curve for Tepal South Measured & Indicated Sulphide Mineral Resource

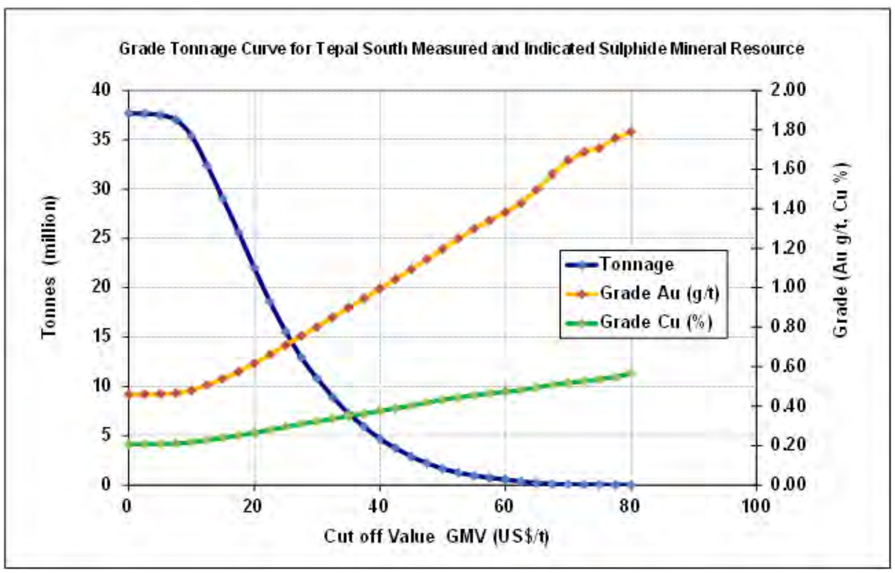
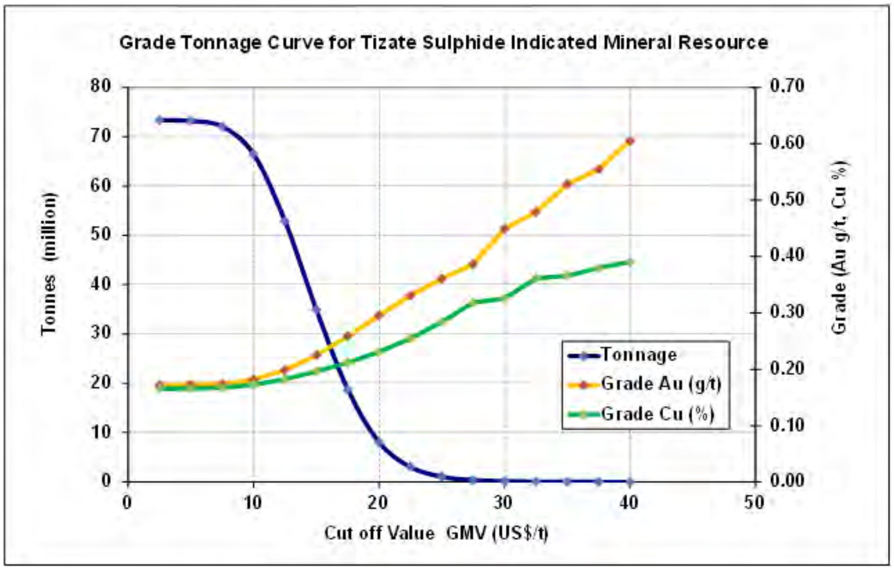


Figure 14-21: Grade/Tonnage Curve for Tizate Indicated Sulphide Mineral Resource



The deposits are very sensitive to cut-off grade. The sharp decline in tonnage at approximately US$10.00/t cut-off in all three deposits is partly due to the mineralogical models developed by Geologix that were based on US$8.70/t (US$1,000/oz for gold and US$2.75/lb for copper). This parameter guarantees that most of the material within the models is at least above a US$8.70/t cut-off. Consequently, there is little variation in tonnage or grade below this cut-off, as illustrated in the charts above.

14.1.13 Deep South Zone Resources

There is deep and relatively high grade mineralization within the South Zone mineralogical model that is immediately below the South Zone soft pit boundary. It has not been included in the mineral resource estimate because it is below the optimized pit limits and as such, is presently uneconomic to extract from the open pit mining method. Although some of the mineralization meets the search pass criteria for Indicated resources, this mineralization is being classified as an Inferred resource in this report due to resource definitions.

This mineralization may have the potential to be mined using underground mining methods, if found to be economic, to extract. A study is needed to determine the economic viability of this mineralization being extracted.

The Table 14-34 lists the tonnage and grade at a variety of cut-off equivalents (US$1,000 Au and US$2.75 Cu). For the purposes of this report, a $20.00/t value has been identified as a preliminary suitable cut-off equivalent value that could potentially give a reasonable prospect for economic extraction using underground mining methods. Further analysis needs to be done to corroborate this cut-off value.

Table 14-36: South Tepal Below-Pit Inferred Resources

Cut-off	Tonnes	Average Grade				Metal	
Eq. V.		Au	Cu	Ag	Mo	Au	Cu
($/t)	(x1000)	(g/t)	(%)	(g/t)	(%)	(koz)	(Mlb)
5	8,331	0.42	0.21	0.89	0.003	114	39
10	8,129	0.43	0.22	0.9	0.003	113	39
12	7,619	0.45	0.23	0.93	0.003	110	38
14	7,228	0.46	0.23	0.94	0.003	107	37
16	6,566	0.48	0.24	0.97	0.003	102	35
18	5,339	0.54	0.26	1.08	0.003	93	30
20	**4,767**	**0.57**	**0.27**	**1.12**	**0.003**	**87**	**28**
22	4,231	0.6	0.28	1.17	0.003	81	26
24	3,604	0.63	0.29	1.23	0.003	74	23

14.1.14 Discussion

The increase in mineral resource tonnage with respect to the previous resource estimate is primarily due to the 2011 drill program. The combination of definition and delineation drilling has not only increased the size of each of the deposits but has upgraded the resource categories within each deposit. The Tizate Zone has benefited the most from this drilling program. The Tizate deposit has expanded approximately 300m to the southwest and 150m to the northeast. In-fill drilling in all three deposits has increased the confidence in the continuity of mineralization and hence the up-grading of resource categories within each deposit.

The drill program has also identified high grade mineralization below the optimized pit limit in the Tepal South Zone. This mineralization although not part of the present mineral resource estimate has been classified as an Inferred resource that could create future opportunities for Geologix, if found to be economic via underground mining methods. Future analysis and further drilling is required.

15 Mineral Reserves Estimate

Mineral Reserves is the economically mineable part of a measured and indicated mineral resource demonstrated by at least a Preliminary Feasibility Study (PFS). This study includes adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

15.1 Mineral Reserves Statement

The estimate of Mineral Reserves as of March 19, 2013 is reported in Table 15-1 below. Mineral Reserves are a subset of the mineral resource. Mineral resources that are not Mineral Reserves do not have demonstrated economic viability.

The mineral resource estimate is described in Section 14 and Technical Report on the Mineral Resources of the Tepal Gold-Copper Project Michoacan State, Mexico (2012 Resource Report) with an effective date of March 29, 2012 and filed on SEDAR.

The resources were used directly for conversion to Proven and Probable reserve. Reserves have utilized resource block grades, adjusted for dilution, and applied economic criteria outlined in this report. Optimized pit shells were created using PFS applied operating costs. Economic cutoffs were applied to ore types. Pit shells were subsequently designed with access ramps and a production schedule applied. The production schedule is discussed in Section 16.4.

Table 15-1: Mineral Reserves

Proven and Probable Reserves									
		Diluted Grade			Contained Metal			Equivalent Metal	
Oxide Ore	Tonnes (Mt)	Au g/t	Ag g/t	Cu%	Au koz.	Ag koz.	Cu Mlbs.	AuEq koz.[1]	CuEq Mlbs.[1]
Proven	3.8	0.56	0.91	0.28	68	111	23.7	129	52.2
Probable	8.0	0.36	1.41	0.18	93	363	32.3	179	72.4
Proven and Probable	11.8	0.42	1.25	0.22	161	474	56.0	308	124.6
Sulphide Ore	Tonnes (Mt)	Au g/t	Ag g/t	Cu%	Au koz.	Ag koz.	Cu Mlbs.	AuEq koz. [1]	CuEq Mlbs.[1]
Proven	28.3	0.48	0.97	0.24	439	885	151.3	830	335.3
Probable	109.5	0.25	1.63	0.19	894	5,741	447.3	2,108	851.9
Proven and Probable	137.8	0.30	1.50	0.20	1,333	6.625	598.6	2,938	1,187.2
Oxide+Sulphide Ore	Tonnes (Mt)	Au g/t	Ag g/t	Cu%	Au koz.	Ag koz.	Cu Mlbs.	AuEq koz. [1]	CuEq Mlbs.[1]
Proven and Probable	149.6	0.31	1.48	0.20	1,494	7,099	654.6	3,247	1,311.8

Notes:

1) Uses Uses PFS Base Case Four-Year Trailing Average Metal Prices: Au US$1389.95/oz, Cu US$3.44/lb and Ag US$26.03/oz.

AuEq = Au oz + (Ag oz * $26.03/$1389.95) + (Cu lbs * $3.44/$1389.95); CuEq = Cu lbs + (Au oz * $1389.95/$3.44) + (Ag oz * $26.03/$3.44)

Au = gold, Cu = copper, Ag = silver, g/t = grams per tonne, % = percent, koz. = thousand ounces, Mlbs. = millions of pounds.

The Reserves stated in the table above conform to CIM guidelines. Resources are not to be confused as reserves.

Reserve numbers are rounded to the nearest 100,000 tonnes, 1,000 oz Au, 1,000 oz Ag, 100,000 lbs Cu, 1,000 oz. AuEq and 100,000 lbs CuEq.

15.2 Reserve Estimation Parameters and Methodology

15.2.1 Overall Pit Slope Angles

Pit slope recommendations were provided by Knight Piésold Ltd. (KP) and have been detailed in Section 16.2. JDS estimated overall pit slope angles to apply to the Whittle pit optimizations using the inter-ramp angles recommended by KP and assuming a maximum pit depth of 300m, 20m double bench heights and allowing for two 23m wide ramp crossings. Proposed overall slope angles are recorded in Table 15-2. The estimated overall slope angles ensure that Whittle optimized shell shapes and economics would closely match the pit designs.

Table 15-2: Overall Pit Slope

Pit	Design Sector (Inter-Ramp Angle)	Face Angle	Catchment Width	Overall Pit Slope
Units		**deg.**	**m**	**deg.**
Tepal North	35°	60	17	32
	45°	65	11	41
Tepal South	42°	65	13	38
	45°	65	11	41
	48°	65	9	44
Tizate	48°	65	9	44
	51°	70	9	46

15.2.2 Block Models

Micon International Ltd. completed an NI 43-101 compliant mineral resource estimate for the Tepal property, discussed in Section 14. The Tepal property was divided into two block models: the Tepal model (containing both the North and South zones) and Tizate. JDS imported these models into Gemcom Whittle-Strategic Mine Planning™ (Whittle) and Maptek Vulcan™ (Vulcan) software to develop optimized pit shells and detailed pit designs from which the Mineral Reserve could be calculated.

15.2.3 Block Value

A script was created in Vulcan to calculate the value of sulphide blocks based on the parameters recorded in Table 15-3. Sulphide block value is based on the Net Smelter Return (NSR) of concentrate produced by flotation and revenue from doré bar produced by sulphide cyanidation. The sulphide block value was stored in the block model and exported to Whittle for open pit optimization.

The value of oxide blocks was calculated within the Whittle software using the parameters recorded in Table 15-3. The value of oxide blocks was based on revenue from doré bar produced by the oxide cyanidation process.

Table 15-3: Block Value Inputs

Parameter	Units	Sulphide Flotation	Sulphide Cyanidation	Oxide Cyanidation
Metal Prices				
Cu	US$/lb	3.15	3.15	3.15
Au	US$/oz	1,400.00	1,400.00	1,400.00
Ag	US$/oz	26.00	26.00	26.00
Tepal Recovery				
Cu	%	88.2	0.0	0.0
Au	%	62.4	17.2	83.2
Ag	%	27.4	13.6	63.3
Solution Losses	%	0.0	2.0	2.0
Tizate Recovery				
Cu	%	85.9	0.0	0.0
Au	%	58.0	20.5	75.2
Ag	%	59.6	12.1	55.9
Solution Losses	%	0.0	2.0	2.0
Cu Concentrate – Tepal				
Cu	%	25.7		
Au[1]	%	variable		
Ag[1]	%	variable		
Moisture	%	8.0		
Cu Concentrate – Tizate				
Cu	%	26.9		
Au[1]	%	variable		
Ag[1]	%	variable		
Moisture	%	8.0		
Royalties	%	2.50	2.50	2.50
Offsite Costs				
Cu Concentrate Transport	US$/dmt	60.00		
Cu Refining	US$/pay. lb	0.06		
Au Refining	US$/pay. oz	5.00	7.50	7.50
Ag Refining	US$/pay. oz	0.50	0.14	0.14
Truck Transportation	US$/wmt	36.73		
Ocean Freight	US$/wmt	60.00		
Representation at Port	US$/wmt	1.00		
Port Charges	US$/wmt	10.50		
Insurances	US$/wmt	0.07% x 110% x NIV		

Parameter	Units	Sulphide Flotation	Sulphide Cyanidation	Oxide Cyanidation
Losses	US$/wmt	0.30% x NIV		
Smelter Payables				
Cu Deduction	Unit	1.00		
Payable Cu	%	96.50		
Payable Au	%	97.00	99.90	99.90
Payable Ag	%	90.00	97.00	97.00
Silver deduction	Unit	30.00		

Notes:

1) Au and Ag contained within Cu concentrate vary due to the head grade and recovery of those metals in the flotation process.

15.2.4 Open Pit Optimization Inputs

Whittle open pit optimization was performed using the operating costs and miscellaneous inputs listed in Table 15-4. Measured and Indicated blocks were treated as ore; Inferred blocks were treated as waste.

Table 15-4: Whittle Input Parameters

Parameter	Units	Sulphide Flotation	Sulphide Cyanidation	Oxide Cyanidation
Operating Costs				
Mining Cost	US$/t mined	1.20		1.20
Incremental Mining Cost	US$/t mined	0.02		0.02
Processing Cost (Tepal/Tizate)	US$/t milled	6.09 / 6.09	7.25 / 7.25	7.37 / 4.82
G&A Cost	US$/t milled	0.24		0.24
Tailings Cost	US$/t milled	0.35		0.35
Misc. Inputs				
Discount Rate	%	8.0	8.0	8.0
Overall Pit Slope	deg	Refer to Section 15.2.		
Dilution	%	5.0	5.0	5.0
Mining Recovery	%	100.0	100.0	100.0
Processing Rate[2]	tpy	Variable	12 % of Sulphide	2,190,000

Notes:

1) Incremental Mining Cost is added to Mining Cost on a cumulative basis per bench below the pit rim: Tepal North & South pit rim is 520m, Tizate is 470m.

2) Sulphide processing rates varies due to ore hardness in the different pits: Tepal North 40,000tpd, Tepal South 35,000tpd and Tizate 35,000tpd.

15.2.5 Open Pit Optimization Results

Whittle revenue factor 1.0 pits were selected as the basis for detailed design. Key results are shown in Table 15-5 below.

Table 15-5: Whittle Revenue Factor 1.0 Pit Shells

Parameter	Units	Tepal North	Tepal South	Tizate	Total
Revenue Factor		1.0	1.0	1.0	
Sulphide Tonnage	M t	52.5	32.3	60.9	145.7
Au[1]	g/t	0.35	0.44	0.19	0.30
Ag[1]	g/t	0.97	1.13	2.22	1.53
Cu[1]	%	0.23	0.20	0.17	0.20
Oxide Tonnage	M t	7.1	2.3	2.5	11.9
Au[1]	g/t	0.47	0.48	0.26	0.43
Ag[1]	g/t	0.96	1.10	2.20	1.25
Cu[1]	%	0.23	0.21	0.19	0.22
NAG Waste	M t	27.2	59.1	10.7	97.1
PAG Waste	M t	57.4	64.5	39.5	161.4
Total Waste	M t	84.6	123.6	50.3	258.5
Stripping Ratio	w:o	1.4	3.6	0.8	1.6

Notes:

1) Undiluted grades for Au, Ag and Cu.

15.2.6 Open Pit Design

The Whittle optimized shells were imported to Vulcan in order to guide the pit design process. Pit designs were developed by JDS using Vulcan. Numerous iterations employing different ramp and bench layouts were completed to maximize ore tonnage and grade while minimizing waste stripping. Pit designs are discussed in Section 16.3. The pit designs were used to calculate the Mineral Reserves.

15.2.7 Economic Cutoff Grade

15.2.7.1 Sulphide Ore Cutoff Value

The calculation for sulphide ore cutoff value is shown in Table 15-6.

Table 15-6: Sulphide Ore Cutoff Value

Heading	Units	Tepal (North & South)	Tizate
Processing Cost[1]	US$/t	6.96	6.96
G&A Cost	US$/t	0.24	0.24
Tailings Cost	US$/t	0.35	0.35
Total Cost	US$/t	7.55	7.55
Sulphide Cutoff Value	**US$/t**	**7.55**	**7.55**

Notes:

1) Processing cost is sulphide flotation cost + 12% of sulphide cyanidation cost (12% of sulphide feed goes to the sulphide cyanidation process).

15.2.7.2 Oxide Ore Cutoff Grade

The calculation for oxide ore cutoff grade is shown in Table 15-7. Note that silver is less than 2% of the value of gold given PFS assumptions for metal price, and is not included in the calculation for oxide ore cutoff grade.

Table 15-7: Oxide Ore Cutoff Grade

Heading	Units	Tepal (North & South)	Tizate
Gold Value[1]	$/g	35.56	32.14
Processing Cost[2]	US$/t	7.37	4.82
G&A Cost	US$/t	0.24	0.24
Tailings Cost	US$/t	0.35	0.35
Rehandle Cost	US$/t	0.47	0.47
Total Cost	US$/t	8.43	5.88
Oxide Cutoff Grade[3]	**g/t**	**0.24**	**0.18**

Notes:

1) $Gold\ Value = \frac{\left(Gold\ Price\frac{US\$}{oz} - Gold\ Refining\ Charge\frac{US\$}{oz}\right) x\ (Recovery\%)\ x\ (PayableMetal\%)\ x\ (1 - Losses\ to\ Solution\%)\ x\ (1 - Royalty\%)}{31.1035\frac{g}{oz}}$.

2) Processing cost is oxide cyanidation cost

3) $Oxide\ Cutoff\ Grade = \frac{Total\ Cost}{Gold\ Value}$.

Oxide ore is planned to be processed through the grinding circuit in a monthly scheduled batch campaign. Oxide ore mined outside of a scheduled batch campaign would be stockpiled and subsequently rehandled to the mill during the campaign. Rehandle Cost covers the cost of stockpiling and rehandling the oxide ore.

15.2.8 External Dilution & Mining Recovery

External dilution for each ore block was estimated based on the number of surrounding waste blocks and the grade of those waste blocks.

A diluting volume of 33.5m^3 per waste edge was calculated based on the working bench height of 10m and a 75 degree working face slope. The diluting volume was divided in half, 16.8m^3, to be applied to the 5m tall blocks in the model.

For each block at a given elevation, the number of waste edges was counted; where the number can vary from zero (block surrounded by ore) to four (block surrounded by waste).

The diluting grade is the average grade of the surrounding waste blocks.

The calculations for diluting mass and diluted grade are shown below:

$$Diluting\ Mass = Number\ of\ Waste\ Edges\ x\ Diluting\ Volume\ x\ Block\ Density$$

$$Diluted\ Block\ Grade = \frac{(Block\ Mass - Diluting\ Mass)\ x\ Block\ Grade + Diluting\ Mass\ x\ Diluting\ Grade}{\text{Block Mass}}$$

This analysis resulted in external dilution as recorded in Table 15-8.

Table 15-8: External Dilution

Category	Mass	Diluting Mass	Undiluted			Diluted		
			Au	Ag	Cu	Au	Ag	Cu
Units	Mt	Mt	g/t	g/t	%	g/t	g/t	%
Oxide Ore								
Proven	3.8	0.0	0.564	0.915	0.285	0.563	0.915	0.285
Probable	8.0	0.2	0.361	1.411	0.183	0.359	1.408	0.183
Total Oxide	11.8	0.2	0.426	1.252	0.216	0.425	1.250	0.216
Difference						-0.002	-0.002	-0.000
Dilution (%)		1.7%				-0.4%	-0.2%	-0.2%
Sulphide Ore								
Proven	28.3	0.2	0.483	0.975	0.244	0.482	0.972	0.243
Probable	109.5	1.3	0.255	1.635	0.186	0.254	1.630	0.185
Total Sulphide	137.8	1.4	0.302	1.499	0.198	0.301	1.495	0.197
Difference						-0.001	-0.004	-0.001
Dilution (%)		1.0%				-0.3%	-0.3%	-0.5%
Total Ore	149.6	1.6	0.311	1.480	0.1995	0.311	1.476	0.1985
Difference						-0.001	-0.004	-0.001
Dilution %		1.1%				-0.3%	-0.3%	-0.5%

Mining recovery is assumed to be 100%.

15.3 Factors That May Affect the Mineral Reserves Estimate

This study did not identify any mining, metallurgical, infrastructure or other relevant factors that may materially affect the estimate of the Mineral Reserves or potential production.

16 Mining Methods

16.1 Summary

- Tepal project is proposed to be a conventional open pit mine, operating 365 days per year, 24 hours per day.
- Mining is planned to be carried out using diesel-powered Caterpillar (CAT) 6050 hydraulic shovels, a 994H and 992K wheel loader, 789D trucks and MD6540 rotary drills. The production equipment would be supported by a fleet of tracked dozers, motor graders, a rubber tire dozer and a water truck.
- It was assumed that the owner would purchase and operate the majority of the earthmoving equipment, and a contract waste stripping fleet would support the owner's fleet in Years 6 through 10.
- The owner's fleet would deliver and place non-acid generating (NAG) waste rock at the tailings storage facility Zones C and E and F. A contractor, using contract equipment, would complete other work identified in the construction plan for the tailings storage facility.
- The owner would employ maintenance personnel with support from major suppliers.
- Two years have been allocated for construction of the mill and other infrastructure. Oxide milling is planned to commence during the latter half of the second construction year. Commissioning of the sulphide circuit at design capacity is scheduled to be completed at the end of the second construction year, with the production to begin immediately afterwards.
- Mill grinding rates, based on ore hardness in each pit are set out in Table 16-1. The grinding rate determines the amount of ore required to feed the mill and is a key component of the mine production schedule.
- North pit is scheduled to be mined out in Year 8, and available for tailings storage in Year 9.

Table 16-1: Mill Grinding Rate

Ore Type and Pit	Daily Grinding Rate
Units	**tpd**
Sulphide Ore – North	40,000
Sulphide Ore – South	35,000
Sulphide Ore – Tizate	35,000
Oxide Ore – All pits	56,000

16.2 Geotechnical Criteria

16.2.1 Pit Geotechnical Characterization

Knight Piésold Ltd. (KP) conducted a geotechnical site investigation program at the Tepal North, Tepal South and Tizate deposits in October 2011 to collect geomechanical and hydrogeological data for the pre-feasibility engineering study. Geomechanical logging of oriented core, in-situ hydrogeological testing, rock core sampling, and laboratory rock strength testing were completed.

Supplementary downhole televiewer surveys were performed in selected geomechanical and exploration drillholes in January 2012.

Four major geotechnical units were defined for the pit slope geotechnical assessment including: Broken/Oxidized Tonalite, Tonalite, Altered Volcanics, and Volcanics. Overburden is not significant at the project site, with a typical thickness of 1 to 2m. Tonalite is the dominant rock type in all three mineralization zones, with the Volcanics present along the south side of the deposit. The Broken/Oxidized Tonalite extends to a depth of approximately 20 to 30m in most areas of the deposit, but reaches a depth of 150m in the west section of the Tepal South Zone. The weaker Altered Volcanics unit extends to a depth of 100m along the west side of the Tepal North Zone.

The intact strength of the tonalite unit was found to be strong with an average laboratory Unconfined Compressive Strength (UCS) of 76MPa. No UCS testing was completed in the volcanic unit, but the geology logs indicated that it is similar to the tonalite. Field UCS estimates for the altered volcanics in the North Zone average 15MPa and the rock mass quality appears to be POOR, with an average Rock Mass Rating (RMR) value of 33. The rock mass quality for the Tonalite unit was characterized as GOOD with an average RMR of 69. The Broken/Oxidized Tonalite unit is generally POOR with a typical RMR value of 36.

The rock mass structural features in the deposit are complex. A northwest/southeast trending, steeply dipping structure was identified in the tonalite unit. The structural orientations of other rock mass units are distorted by various faults, particularly at the Tepal South Zone. The average friction angle of the discontinuities is approximately 28 degrees.

16.2.2 Pit Slope Stability Analyses

The proposed Tepal North Pit is proposed to reach a maximum depth of 200m, the Tepal South Pit 300m, and the Tizate Pit 300m. A series of design sectors were defined for each pit based on wall orientations, projected wall geology, and rock mass characteristics. The proposed pit design sectors along with the projected geotechnical units on the final pit walls are shown on Figure 16-1. It should be noted that the open pit designs shown on Figure 16-1 differ from the final designs. However, the final PFS pit designs have followed the geotechnical pits slope recommendations.

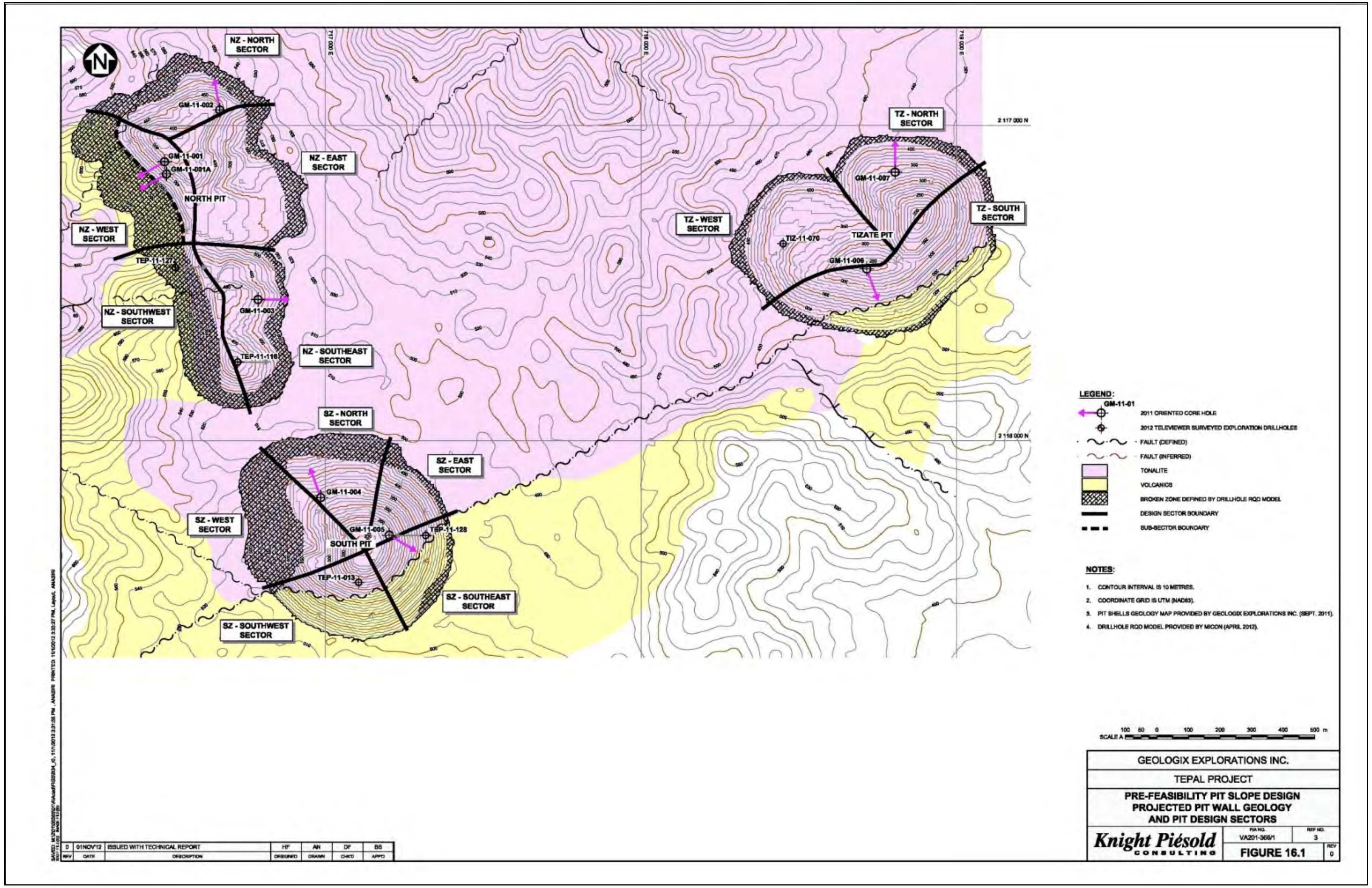
The methods used to analyse appropriate pit slope design angles included kinematic stability assessment and evaluation of the overall rock mass stability. The pit slope geometries for each design sector were determined based on minimum acceptable criteria for each of these methods of analysis.

Stereographic analyses were performed to determine the potential kinematic failure modes in the rock slopes. Rock mass structures throughout the deposit area are variable and the defined faults have little impact on pit wall stability. Some possible kinematic controls on bench face and inter-ramp angles were evaluated for potential wedge and planar failures. The achievable bench face angles for the competent rock slopes are expected to be between 65 and 70 degrees.

A bench height of 10m was assumed for the pit development in accordance with the anticipated mining equipment. A 20m high double benching configuration is recommended for the tonalite and volcanic units with good rock mass quality. The kinematically determined inter-ramp slope angles would range from 45 to 51 degrees given the minimum bench widths. Single benching is recommended for the weaker Altered Volcanics unit exposed along the west portion of the Tepal North Pit and for the broken/oxidized tonalite unit along the uppermost slopes of the pits.

Limit equilibrium analyses were performed to assess the stability of the rock mass slopes. It was determined that slopes excavated in the altered volcanics unit along the upper West Wall of the Tepal North Pit require a flatter inter-ramp slope angle (in the order of 35 degrees). A wider stepout should be designed immediately below the altered volcanics and tonalite contact in this area. The analyses confirm that the kinematically determined inter-ramp slopes in competent rock units would be achievable at a height of less than 300m. Low-damage blasting and slope depressurization are recommended for the final pit wall development. The planned overall pit slope angles are typically 3 to 5 degrees flatter than the inter-ramp slope angles because of the flatter upper slopes, haul ramps, and stepouts that are accounted for in the overall slope design.

Figure 16-1: Projected Pit Wall Geology and Design Sectors



16.2.3 Seismicity

A preliminary assessment of the seismic hazard for the Tepal Project was carried out from a review of probabilistic seismic hazard analysis for others projects in the region of southern Mexico, and by examination of published seismic hazard maps for Mexico (GSHAP, 1999 and Tanner, 2004).

Site-specific ground motion parameters have been estimated for the Tepal project site using information provided by the review of probabilistic seismic hazard information. These values are likely conservative, but are representative of the very high seismic hazard in the region.

A conservative earthquake magnitude of 8 is recommended for geotechnical foundation design and seismic design analyses, including soil liquefaction assessment. A site specific seismic hazard assessment is recommended for the project in the next stage of study.

16.2.4 Recommended Pit Slope Configurations

A total of 13 pit design sectors were defined for the three proposed pits based on geotechnical domains, rock mass structures and orientations of the pit walls as shown on Figure 16-1. The recommended pit slope configurations, including bench face angles, bench heights, bench widths and inter-ramp angles are summarized in Table 16-2. Sections of each pit design sector are shown on Figure 16-2 and 16-3.

Table 16-2: Recommended Pit Slope Configurations

<table>
<tr><th>Pit</th><th>Pit Design Sector</th><th>Geotechnical Unit</th><th>Bench Face Angle (°)</th><th>Bench Height (m)</th><th>Catch Berm Width (m)</th><th>Inter-ramp Angle (°)</th></tr>
<tr><td rowspan="6">Tepal North (250m)</td><td>NZ-North</td><td>Tonalite</td><td>65</td><td>20</td><td>11</td><td>45</td></tr>
<tr><td>NZ-East</td><td>Tonalite</td><td>65</td><td>20</td><td>11</td><td>45</td></tr>
<tr><td>NZ-Southeast</td><td>Tonalite</td><td>65</td><td>20</td><td>11</td><td>45</td></tr>
<tr><td>NZ-Southwest</td><td>Altered Volcanics/ Broken Tonalite</td><td>60</td><td>10</td><td>8.5</td><td>35</td></tr>
<tr><td rowspan="2">NZ-West</td><td>Altered Volcanics</td><td>60</td><td>10</td><td>8.5</td><td>35</td></tr>
<tr><td>Tonalite</td><td>65</td><td>20</td><td>11</td><td>45</td></tr>
<tr><td rowspan="5">Tepal South (300m)</td><td>SZ-North</td><td>Tonalite</td><td>65</td><td>20</td><td>11</td><td>45</td></tr>
<tr><td>SZ-East</td><td>Tonalite</td><td>65</td><td>20</td><td>11</td><td>45</td></tr>
<tr><td>SZ-Southeast</td><td>Volcanics/Tonalite</td><td>65</td><td>20</td><td>9</td><td>48</td></tr>
<tr><td>SZ-Southwest</td><td>Volcanics/Tonalite</td><td>65</td><td>20</td><td>9</td><td>48</td></tr>
<tr><td>SZ-West</td><td>Broken Tonalite/ Tonalite</td><td>65</td><td>20</td><td>13</td><td>42</td></tr>
<tr><td rowspan="3">Tizate (300m)</td><td>TZ-North</td><td>Tonalite</td><td>70</td><td>20</td><td>9</td><td>51</td></tr>
<tr><td>TZ-South</td><td>Volcanics/Tonalite</td><td>70</td><td>20</td><td>9</td><td>51</td></tr>
<tr><td>TZ-West</td><td>Tonalite</td><td>65</td><td>20</td><td>9</td><td>48</td></tr>
</table>

Figure 16-2: Slope Angle Geometries

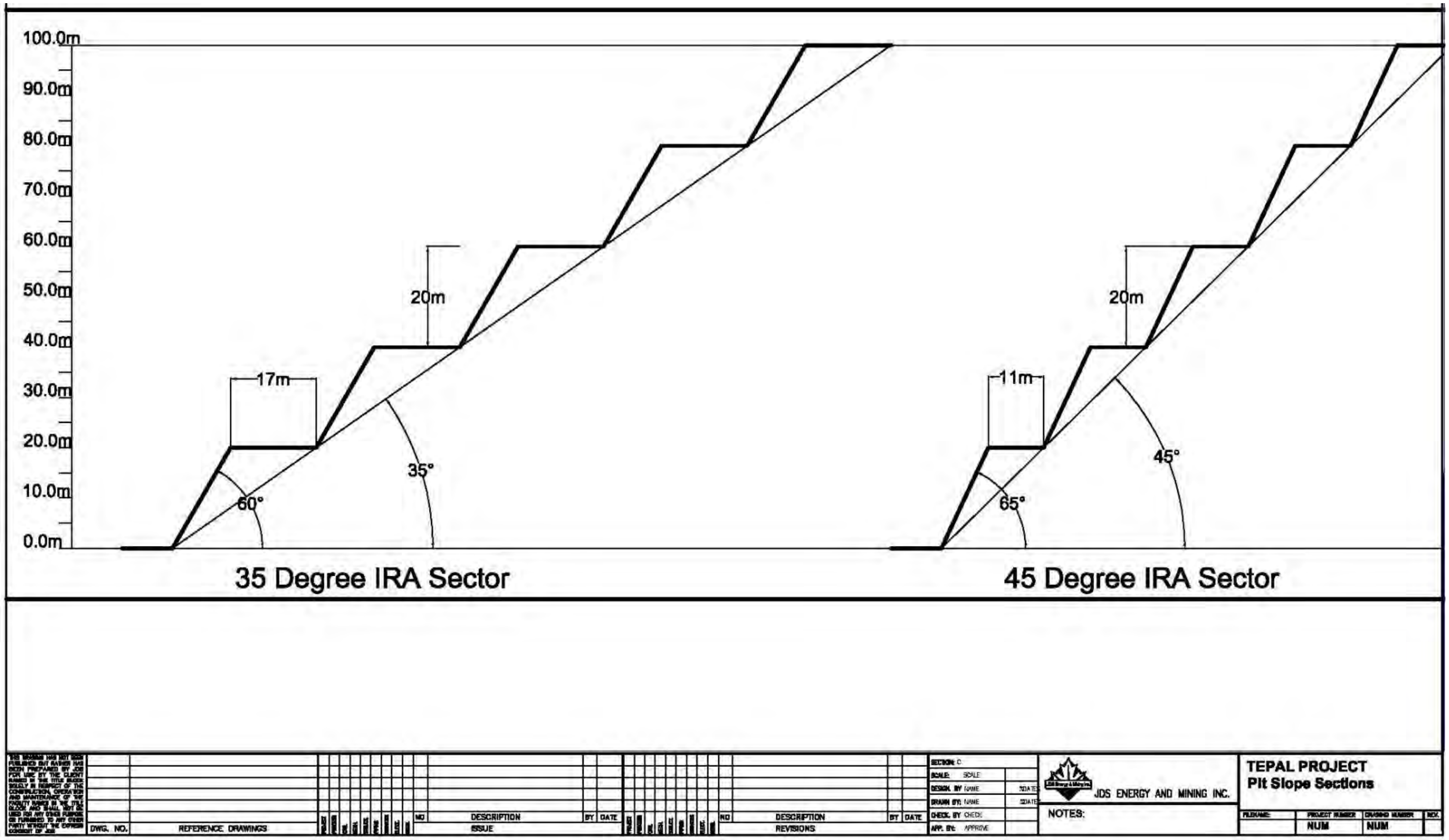
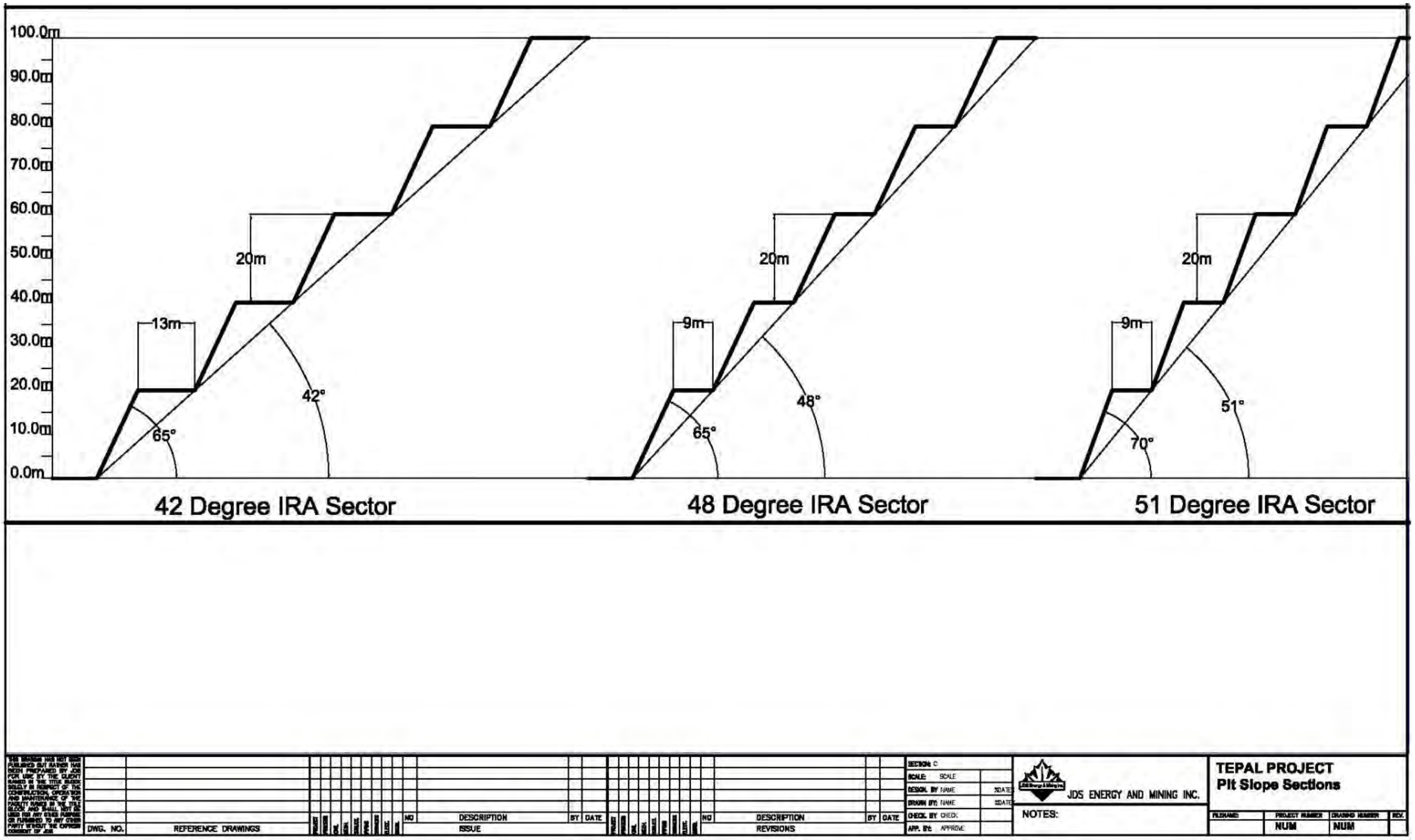


Figure 16-3: Slope Angle Geometries Continued



16.3 Pit Dewatering

Based on the existing information the groundwater table is expected to vary between 10 and 15m below ground surface in the Tepal South and Tizate zones, and between 15 and 45m in the Tepal North zone. The development of the open pits would gradually draw down the existing phreatic surface, and the pits would become groundwater discharge areas.

The majority of the groundwater flows into the pits would likely come from fractured zones intercepted by the excavations. Aggressive slope depressurization would be required for the Altered Volcanics unit to achieve the design slope angles. Slope depressurization would also be required in the deeper broken/oxidized tonalite in the west sector of the Tepal South pit. The potential slope depressurization measures may include perimeter dewatering wells, and/or horizontal drains and a drainage ditch at the toe of the affected slopes. Slope depressurization and groundwater control measures would be investigated in further detail during the next stages of the study when additional hydrogeological information is available.

The pit water would consist of groundwater inflow, and seasonal runoff resulting from direct precipitation. The proposed conceptual pit dewatering system consists of a series of pumps, sumps, and pipelines that would transfer water from the pits to the mill for use in the process. The pit dewatering system is designed to manage the inflow volume resulting from a 1-in-10 year, 24-hour storm event within 72 hours. The proposed pit dewatering system design is conceptual and would be detailed in the feasibility study.

The peak pumping requirements for each pit are summarized in Table 16-3.

Table 16-3: Pumping Design Basis

Pit	Design Peak Flow Rate	Elevation Difference
Units	m^3/h	m
Tepal North (North Bowl)	340	160
Tepal North (South Bowl)	220	160
Tepal South	680	320
Tizate	780	380

16.4 Open Pit Design

Whittle optimized shells were imported to Vulcan in order to guide the pit design process. Pit designs were developed by JDS using Vulcan software. Pit designs are proposed to incorporate slope recommendations provided by KP including 10m working bench height, and haul road widths as discussed below. Final pit walls are planned to be double-bench, with 20m height between berms.

The first step in the pit design process was to define a Phase 1 pit targeting two years of higher value ore. Whittle shells at lower revenue factor were selected and pit designs completed. The second step was to complete the Phase 2 (or ultimate) pit designs based on the revenue factor 1.0

shells. Phase 1 and Phase 2 designs were checked to ensure suitable mining width and access to all areas exists. Mineral Reserves by phase are recorded in Table 16-4 on the following page. The ultimate pit designs for Tepal North, Tepal South and Tizate are shown in Figure 16-4 to Figure 16-9. Typical sections are included as Figure 16-10 through 16-13.

16.4.1 Haul Road Design

Haul roads and in-pit ramps are designed at 10% gradient and 30m width. 30m width is sufficient for two-lane CAT 789D traffic (3 x 7.6m truck width), a safety berm (75% of the height of a 37.00R57 tire) and drainage ditch.

The ramp is planned to be narrowed to 23m width in the bottom 60m of the pit to reduce waste stripping. The 23m width would be sufficient for single-lane 789D traffic (2 x 7.6m truck width), safety berm and drainage ditch.

The width of the road and height of the safety berm height complies with Mexican mining regulations as well as the Canadian mining best practices.

16.4.2 Bench Heights

Production bench heights of 10m were selected to be within the safe and efficient digging envelope of the CAT 6050 diesel hydraulic shovels. The 10m height takes into account swell and the maximum reach of the shovels.

Table 16-4: Mineral Reserves by Phase

Phase	**Sulphide**	**Cu[1]**	**Au[1]**	**Ag[1]**	**Oxide**	**Au**	**Ag**	**Cu**	**NAG**	**PAG**	**S/R**
Units	Mt	%	g/t	g/t	Mt	g/t	g/t	%	Mt	Mt	w:o
Tepal North											
Phase 1	25.1	0.27	0.42	0.83	6.3	0.48	0.95	0.23	11.1	6.9	0.6
Phase 2	25.1	0.19	0.27	1.09	0.7	0.36	1.04	0.21	15.1	54.3	2.7
Subtotal	50.2	0.23	0.35	0.96	7.0	0.47	0.96	0.23	26.1	61.2	1.5
Tepal South											
Phase 1	19.4	0.21	0.44	1.17	2.3	0.48	1.10	0.21	24.2	27.1	2.4
Phase 2	11.5	0.19	0.42	1.05	-	-	-	-	37.5	39.7	6.7
Subtotal	31	0.20	0.43	1.12	2.3	0.48	1.10	0.21	61.6	66.8	3.9
Tizate											
Phase 1	32.2	0.18	0.20	2.17	2.5	0.26	2.20	0.18	8.10	9.7	0.5
Phase 2	24.5	0.16	0.17	2.18	-	-	-	-	3.10	32.3	1.4
Subtotal	56.7	0.17	0.19	2.17	2.5	0.26	2.20	0.18	11.02	42	0.9
Total	**137.8**	**0.20**	**0.30**	**1.50**	**11.8**	**0.42**	**1.25**	**0.22**	**98.9**	**169.9**	**1.8**

Note: 1-Dilted grades

Figure 16-4: Tepal North Phase 1 Pit Design

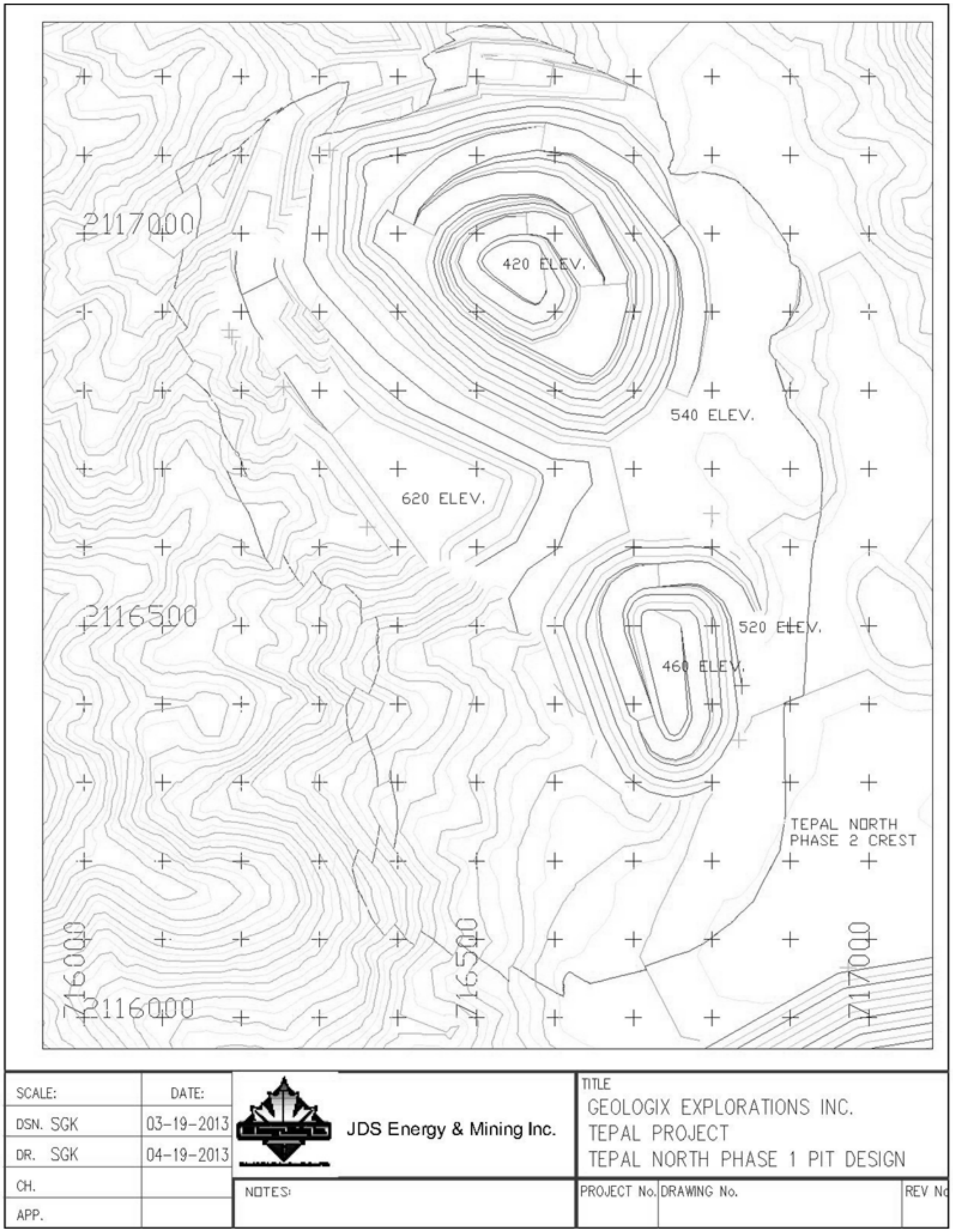


Figure 16-5: Tepal North Face 2 Pit Design

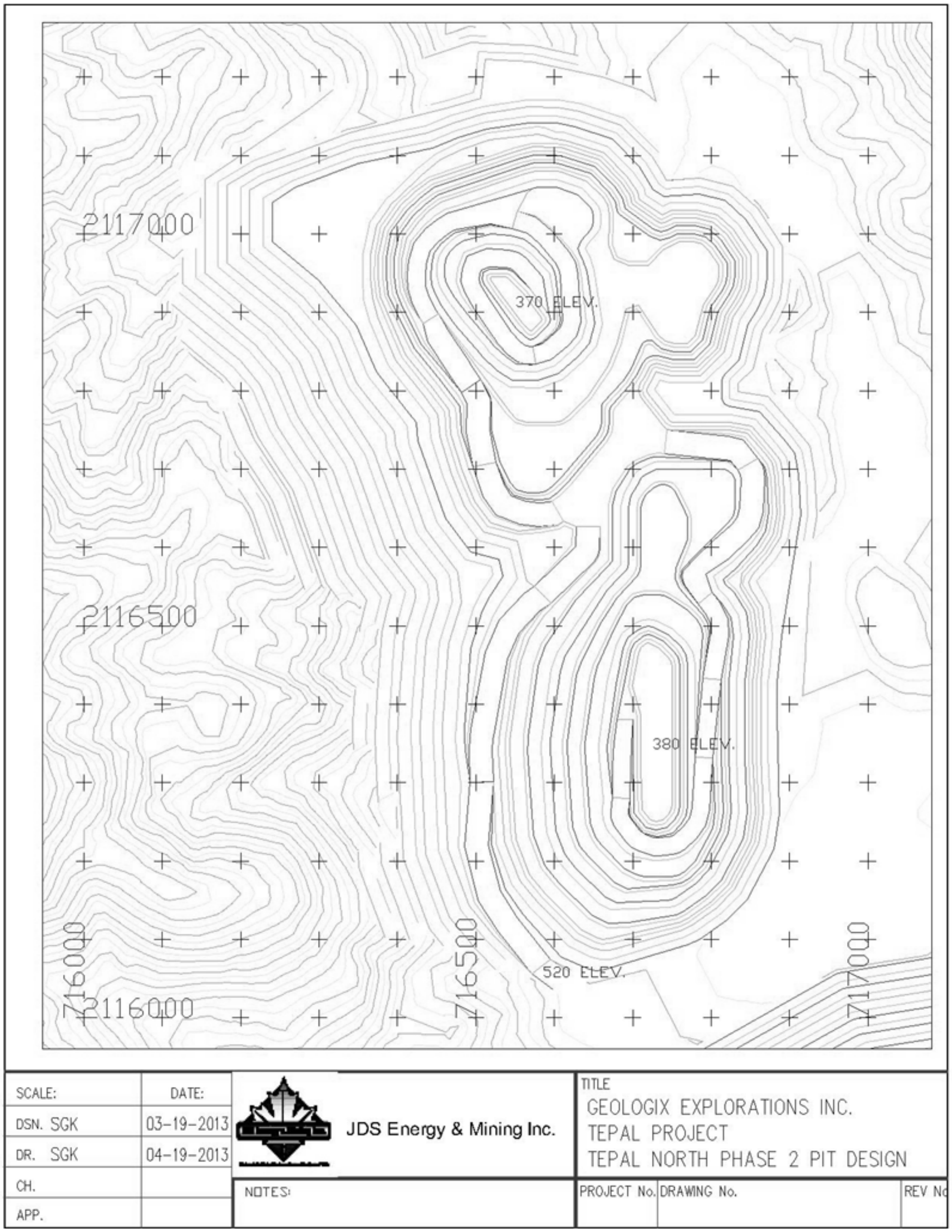
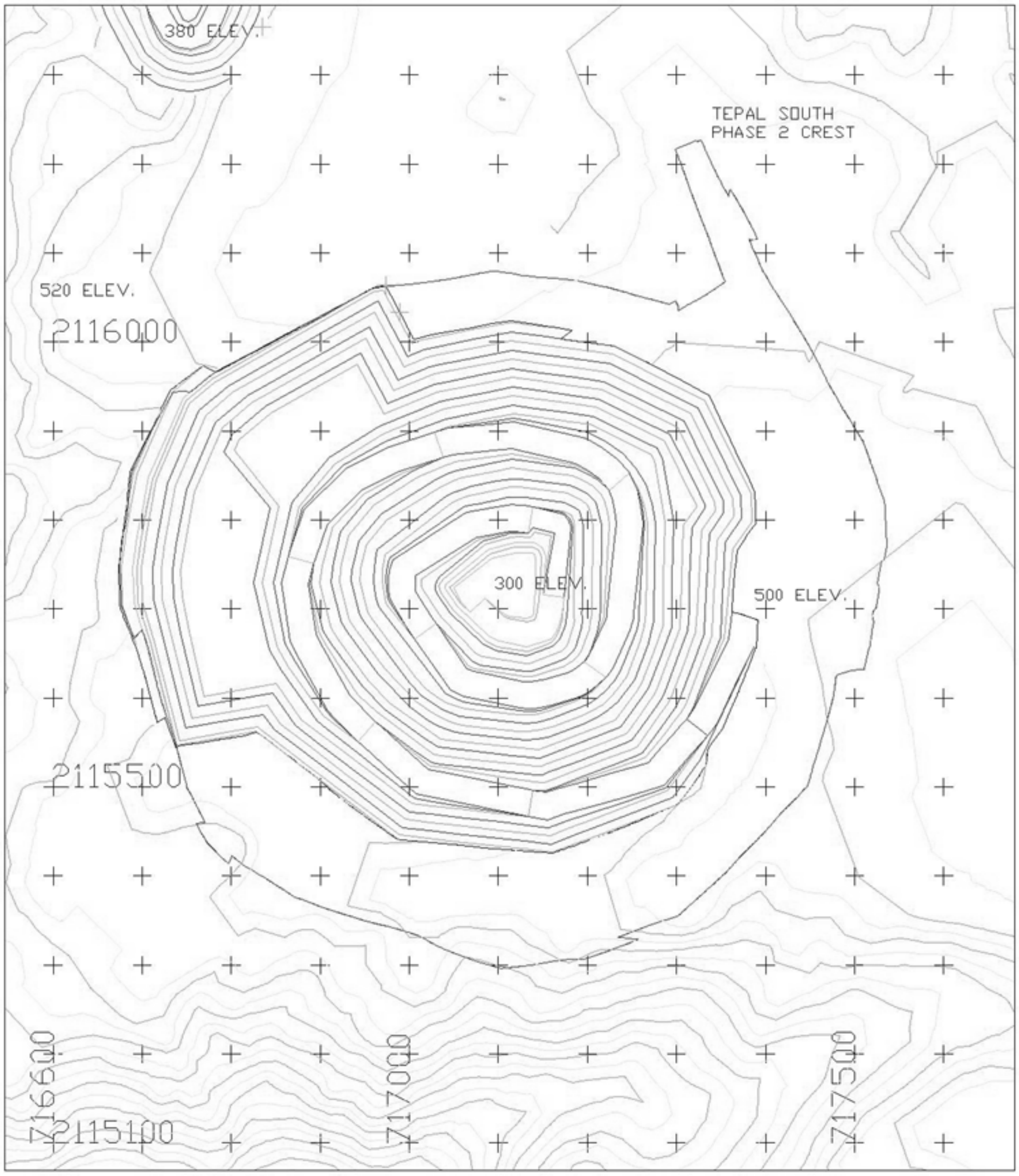


Figure 16-6: Tepal South Phase 1 Pit Design



SCALE:	DATE:		TITLE
DSN. SGK	03-19-2013	JDS Energy & Mining Inc.	GEOLOGIX EXPLORATIONS INC.
DR. SGK	04-19-2013		TEPAL PROJECT
			TEPAL SOUTH PHASE 1 PIT DESIGN
CH.		NOTES:	PROJECT No. DRAWING No. REV No
APP.			

Figure 16-7: Tepal South Phase 2 Pit Design

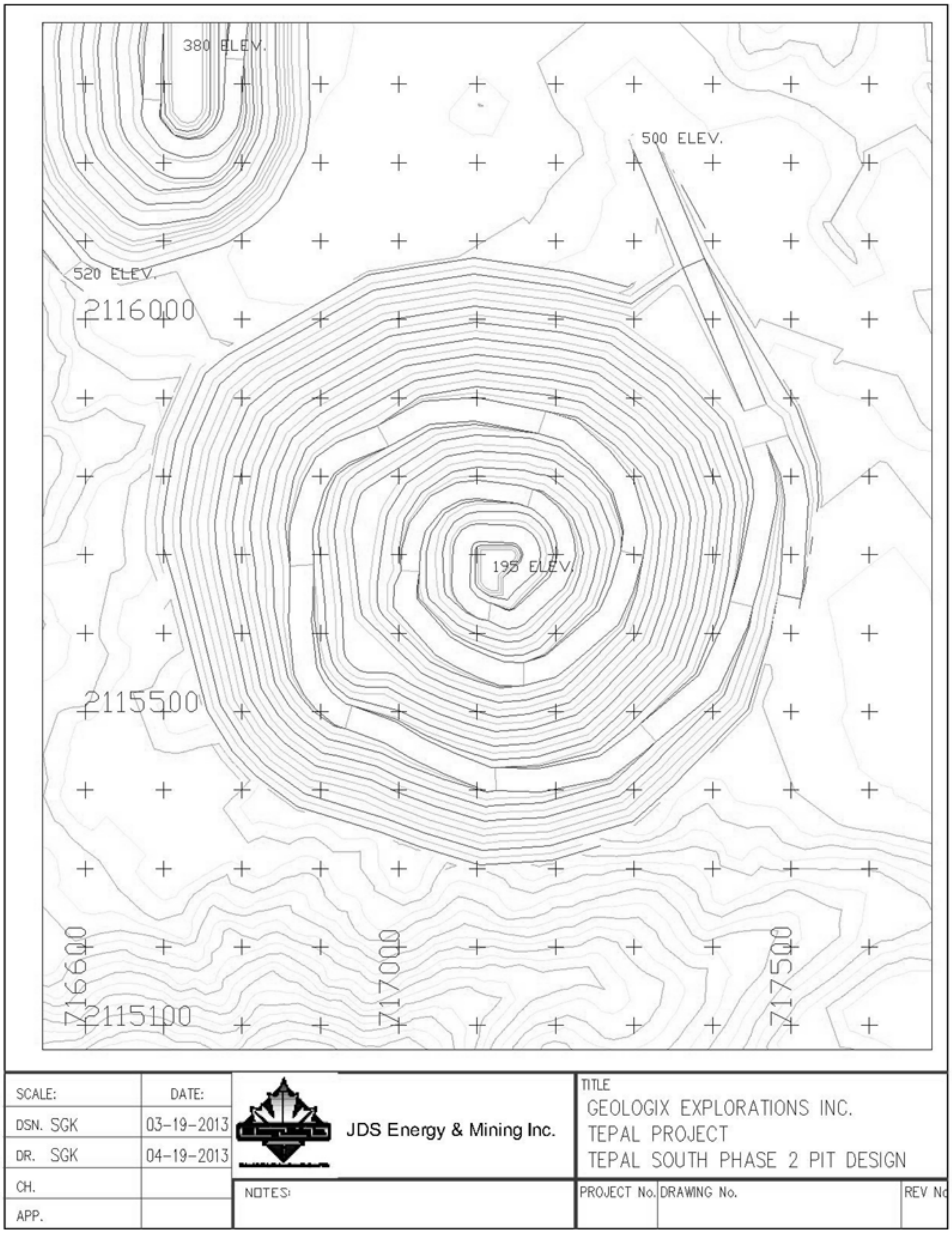


Figure 16-8: Tizate Phase 1 Pit Design

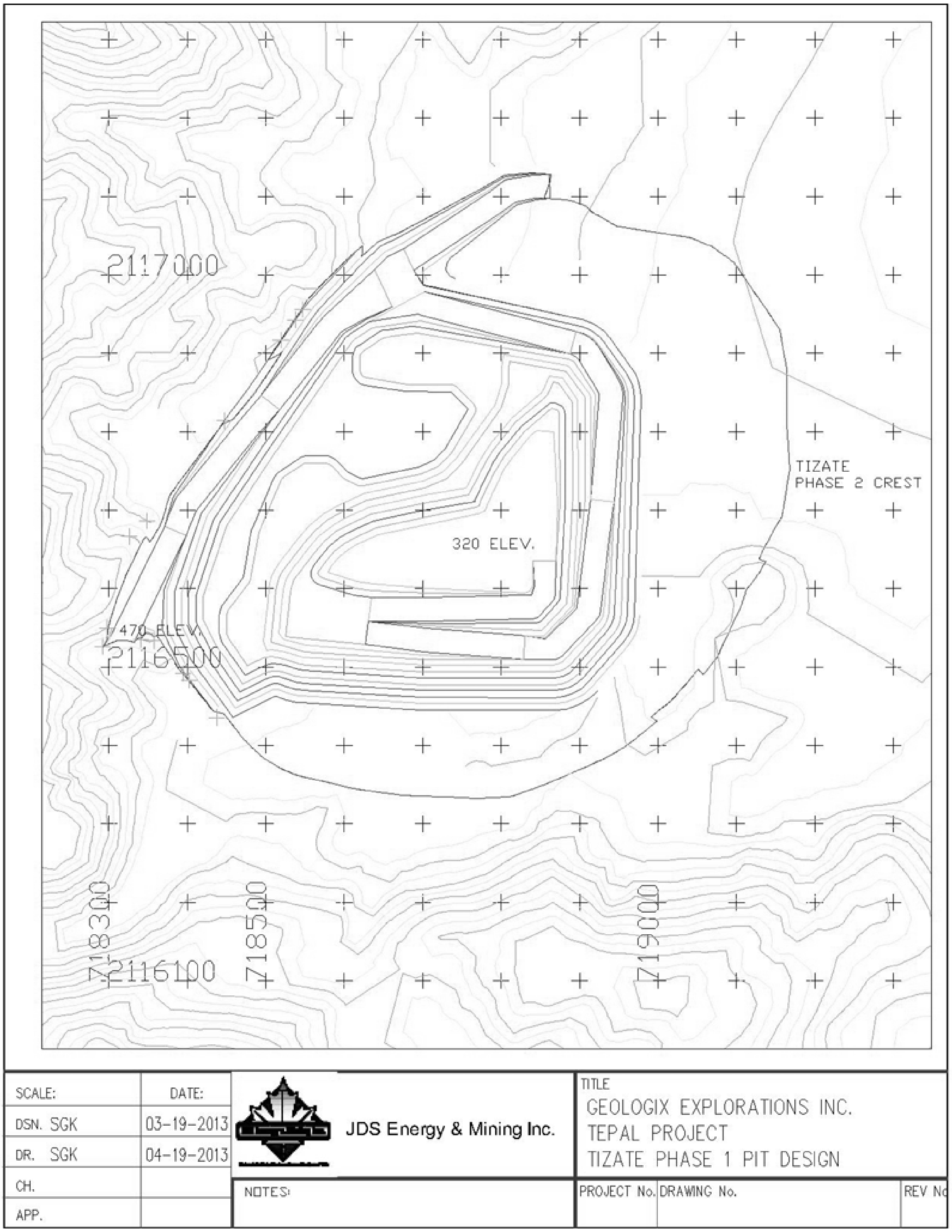


Figure 16-9: Tizate Phase 2 Pit Design

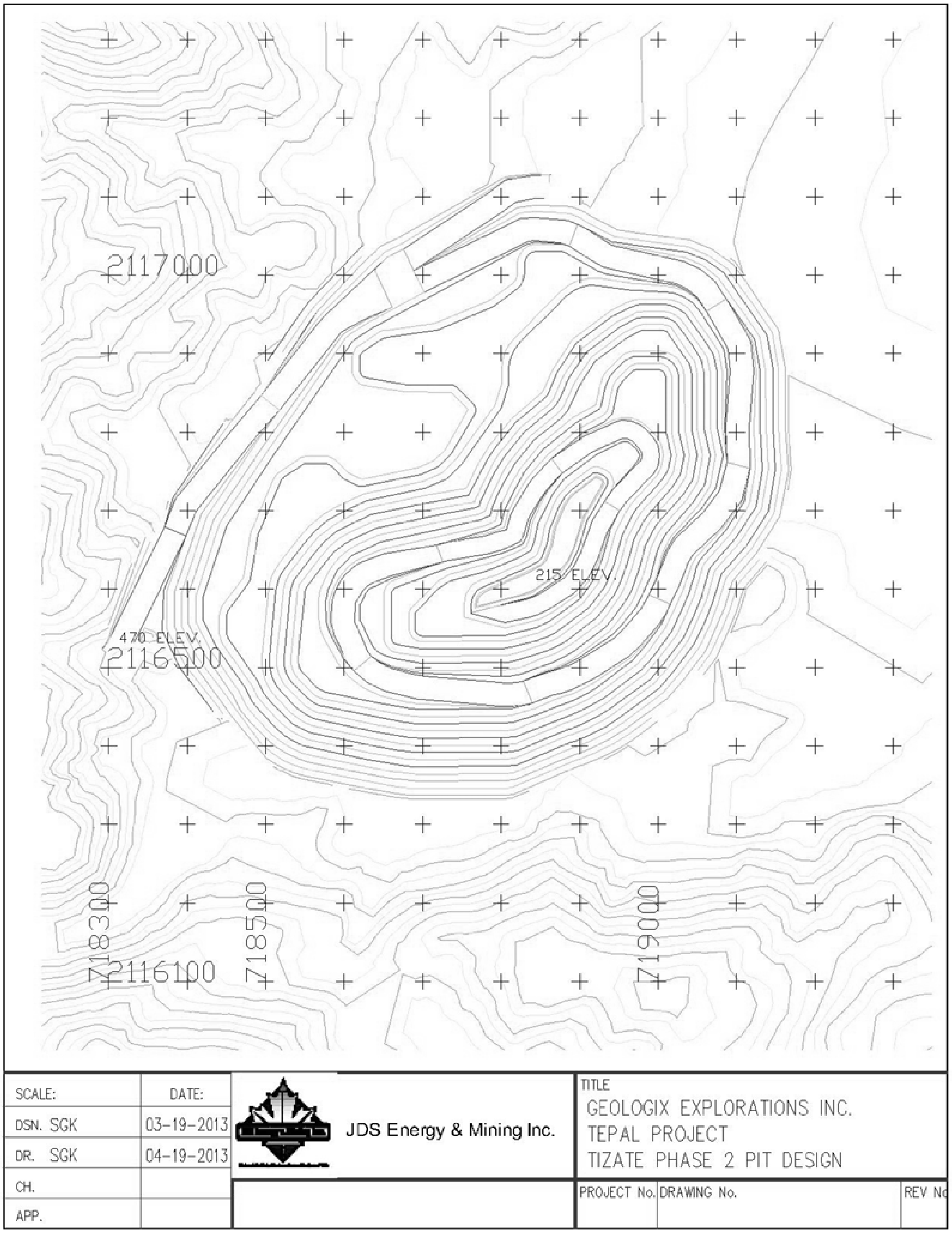


Figure 16-10: Tepal North Pit (Phase 1 & 2), Typical Section 2116900N

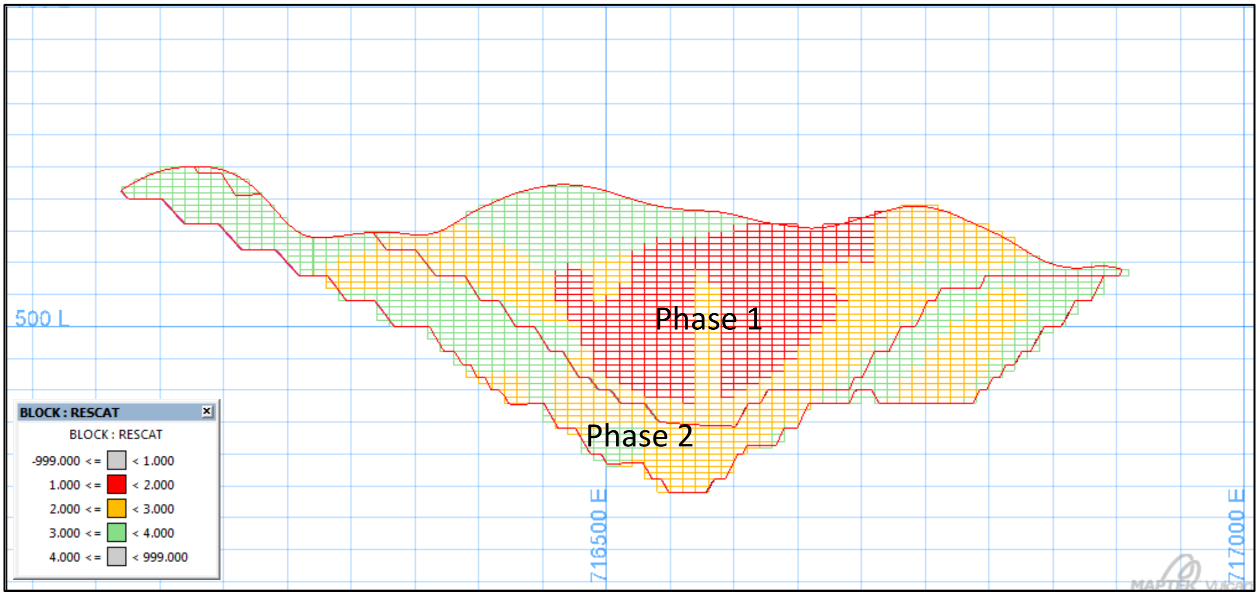


Figure 16-11: Tepal North Pit (Phase 1 & 2), Typical Long-Section 716600E

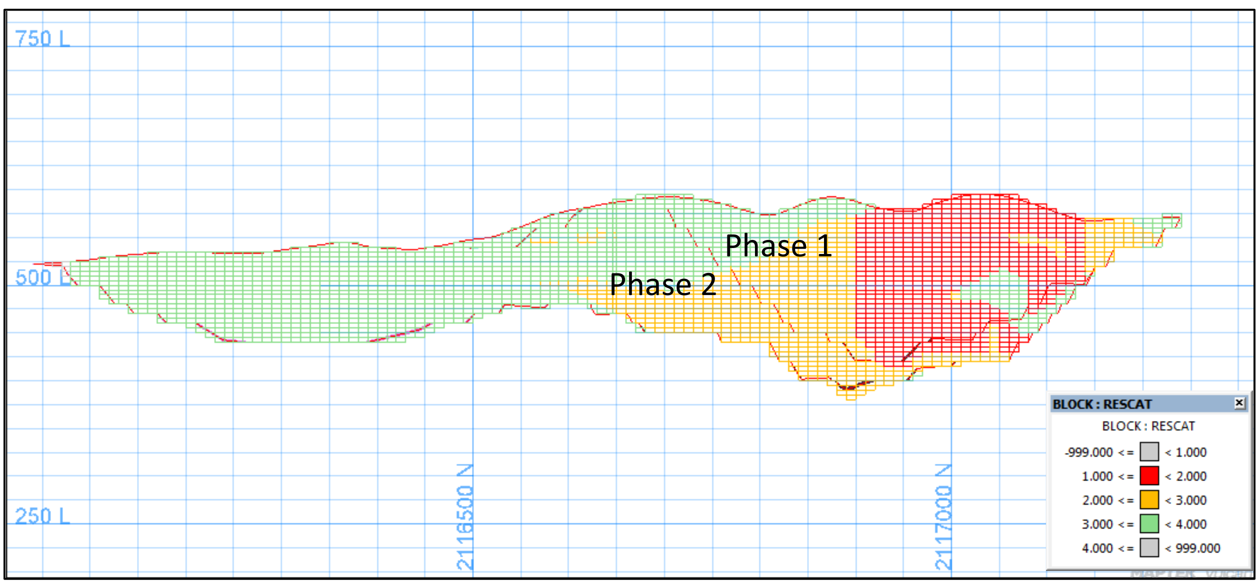


Figure 16-12: Tepal South Pit (Phase 1 & 2), Typical Section 2115700N

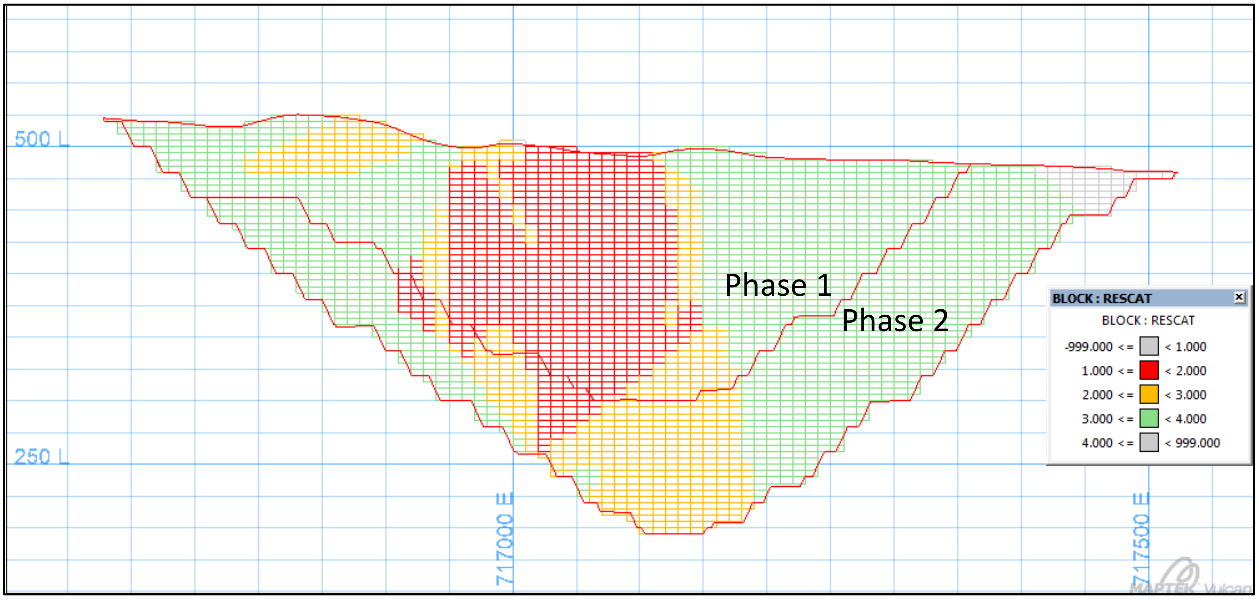
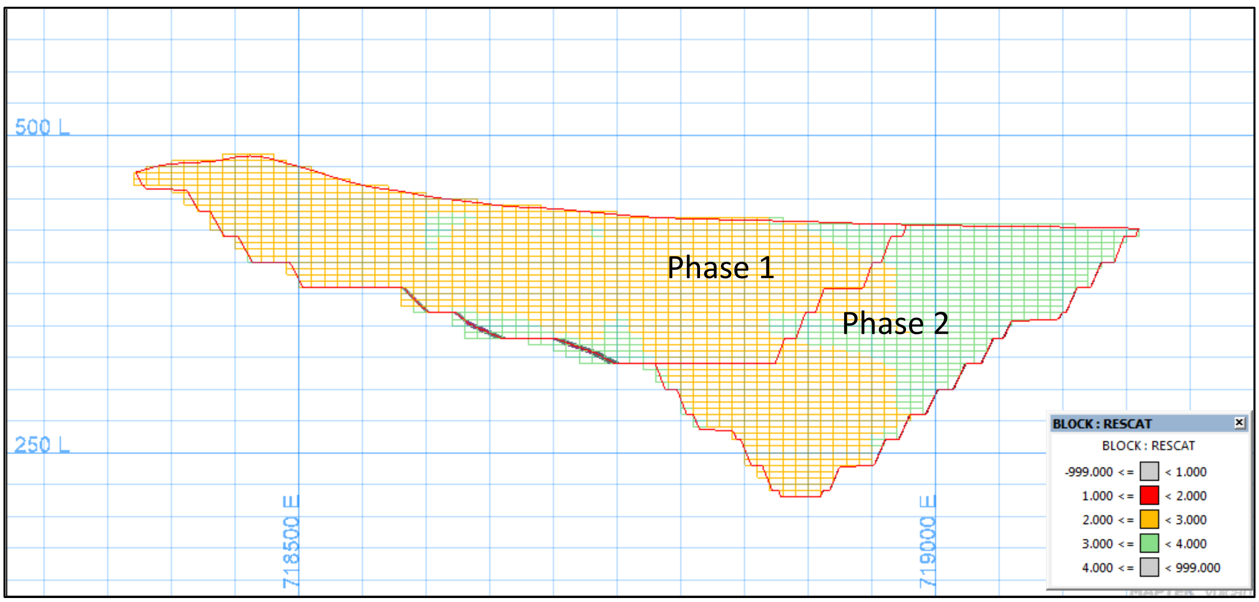


Figure 16-13: Tizate Pit (Phase 1 & 2), Typical Section 2116700N



16.5 Comparison Whittle Optimized Shells to Pit Designs

The pit designs compare well to the Whittle optimized shells as shown in Table 16-5. Ore and waste contained within the pit design are within -5% and +5% respectively compared to the optimized shells.

Table 16-5: Comparison Whittle Optimized Shells to Pit Designs

Pit	Oxide Ore	Sulphide Ore	NAG Waste	PAG Waste	Total Ore + Waste
Units	Mt	Mt	Mt	Mt	Mt
Whittle Shell					
Tepal North	7.1	52.5	27.2	57.4	144.2
Tepal South	2.3	32.3	59.1	64.5	158.1
Tizate	2.5	60.9	10.7	39.5	113.7
Total	11.8	145.6	97.1	161.4	416.0
Pit Design					
Tepal North	7.0	50.2	26.1	61.2	144.5
Tepal South	2.3	31.0	61.6	66.8	161.6
Tizate	2.5	56.7	11.2	42.0	112.3
Total	11.8	137.8	98.9	169.9	418.5
Difference					
Tepal North	-0.1	-2.3	-1.1	3.8	0.3
Tepal South	0.0	-1.3	2.5	2.3	3.5
Tizate	0.0	-4.2	0.5	2.5	-1.4
Total	0.0	-7.8	1.8	8.5	2.5
% Difference	0%	-5%	2%	5%	1%

16.6 Mine Production Schedule

Two years have been allocated for construction of the mill and other infrastructure. Mine production in those years would be focused on supplying non-acid generating waste to construct the starter dam and preparing the pit for full-scale operation. Oxide milling is proposed to commence during the latter half of the second construction year. Commissioning of the sulphide circuit at design capacity would be completed at the end of the second construction year. Production is scheduled to begin immediately afterwards, and continue for 11 years.

A total of 11.8Mt of oxide ore, 137.8Mt of sulphide ore, and 267.6Mt of waste is planned to be mined at an average daily mining rate of 88,000tpd. The life of mine stripping ratio is estimated at 1.8 : 1 waste to ore.

Each pit is planned to be mined in two phases targeting the highest value ore earlier in the mine life to reduce the capital payback period and improve overall Project economics. The pits are scheduled to be mined in the order of: Tepal North 1, Tepal South 1, Tizate 1, Tepal North 2, Tizate 2 and Tepal South 2. Higher stripping ratio associated with Tepal South Phase 2 would be deferred until Year 6. At that time, a waste stripping contractor would be used to supplement the owner's fleet.

The mine production schedule is summarized in Figure 16-14, Figure 16-15 and Table 16-6.

16.6.1 Oxide Ore Mining

A total of 11.8Mt of oxide ore is scheduled to be delivered to the mill from Tepal North, Tepal South and Tizate pits. Annual oxide ore head grades are shown in Figure 16-16 and average: 0.42g/t Au, 1.25g/t Ag and 0.22% Cu. Increased Ag production in Year 4 and 5 is due to feed from the Tizate pit during that period. The supply of oxide ore would be exhausted in Year 7.

Oxide ore would be fed through the crushing and grinding circuit in a monthly scheduled batch campaign. The anticipated grinding rate for oxide is proposed to be 56,000tpd, and approximately 4 days of oxide grinding would be required each month. Oxide ore encountered outside the scheduled batch campaign would be stockpiled, and subsequently rehandled to the crusher. In the same way, sulphide ore encountered while grinding oxide would be stockpiled and rehandled.

16.6.2 Sulphide Ore Mining

Full sulphide ore production is proposed to be achieved in Year 1. A total of 137.8Mt of sulphide ore would be fed to the concentrator at average daily grinding rates as set out in Table 16-1. Assumed average head grades are 0.20% Cu, 0.30g/t Au and 1.50g/t Ag. Annual sulphide ore head grades are shown in Figure 16-17.

16.6.3 Waste Mining

The planned mine would have two defined waste types: non-acid generating (NAG) and potentially acid generating (PAG). A total of 97.7Mt of NAG and 169.9Mt of PAG would be mined over the life of mine. 68.7Mt of NAG material would be used in the construction of the tailings storage facility. The remaining NAG and PAG material is planned to be stored in engineered dumps located adjacent to the open pits. Organic soils encountered during stripping would be stockpiled for future reclamation. Annual waste mined and estimated strip ratio is shown in Figure 16-18.

Figure 16-14: Annual Material Mined

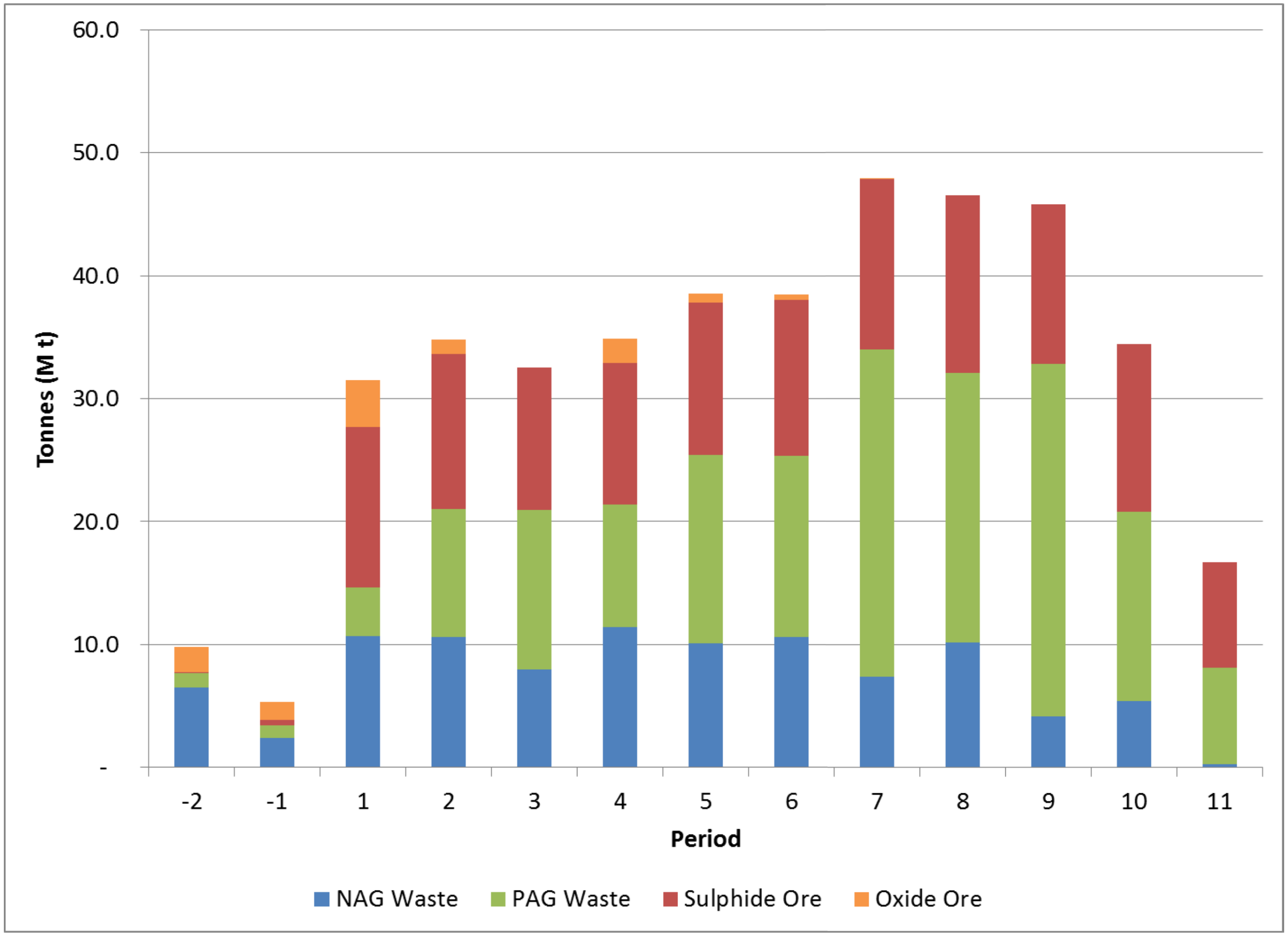


Figure 16-15: Annual Total Mined by Pit

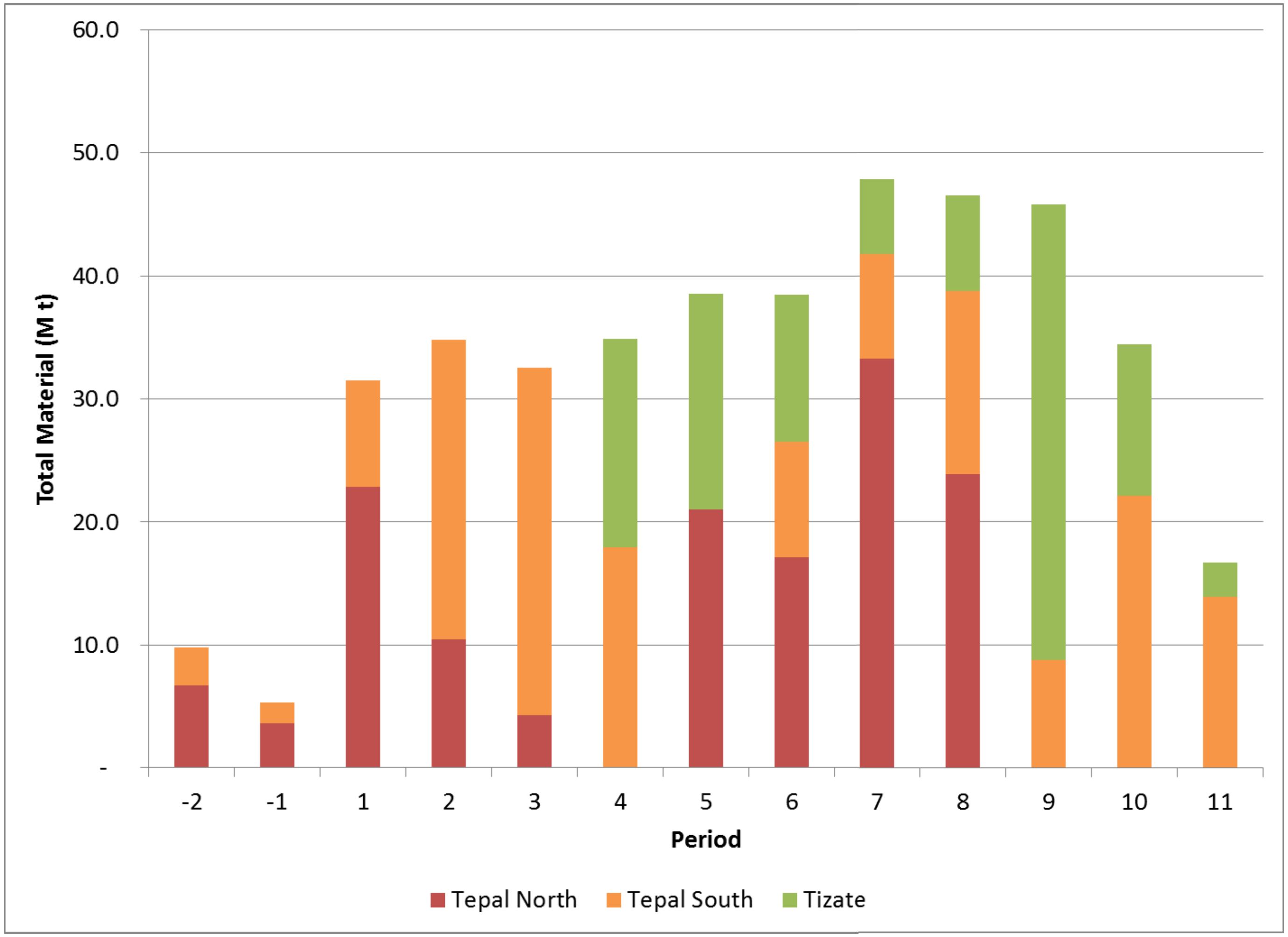


Table 16-6: Annual Mine Production Schedule

Year	Units	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Oxide Ore	Mt	11.8	2.1	1.5	3.8	1.2	-	2.0	0.8	0.5	-	-	-	-	-
Sulphide Ore	Mt	137.8	0.1	0.4	13.1	12.7	11.6	11.5	12.3	12.7	13.9	14.5	12.9	13.6	8.5
NAG Waste	Mt	97.7	6.5	2.4	10.7	10.6	8.0	11.4	10.1	10.6	7.4	10.2	4.2	5.4	0.2
PAG Waste	Mt	169.9	1.1	1.0	4.0	10.4	13.0	10.0	15.4	14.7	26.6	21.9	28.7	15.3	7.9
Total Mined	Mt	417.2	9.8	5.3	31.5	34.8	32.5	34.9	38.5	38.5	47.8	46.6	45.8	34.4	16.7
Strip Ratio	w:o	1.8	3.5	1.8	0.9	1.5	1.8	1.6	1.9	1.9	2.4	2.2	2.5	1.5	1.0
Ox. Stockpile (In/Out)	Mt	-	2.1	(0.5)	1.6	(1.0)	(2.2)	-	-	-	-	-	-	-	-
Ox. Stockpile Inventory	Mt	-	2.1	1.6	3.2	2.2	-	-	-	-	-	-	-	-	-
Sl. Stockpile (In/Out)	Mt	-	0.1	0.4	-	-	(0.3)	-	(0.1)	(0.1)	-	-	-	-	-
Sl. Stockpile Inventory	Mt	-	0.1	0.5	0.5	0.5	0.2	0.2	0.1	-	-	-	-	-	-
Oxide Ore Milled	Mt	11.8	-	2.0	2.2	2.2	2.2	2.0	0.8	0.5	-	-	-	-	-
Sulphide Ore Milled	Mt	137.8	-	-	13.1	12.7	11.9	11.5	12.4	12.8	13.9	14.5	12.9	13.6	8.5

Note that values may not sum due to rounding.

Figure 16-16: Annual Oxide Ore Head Grade

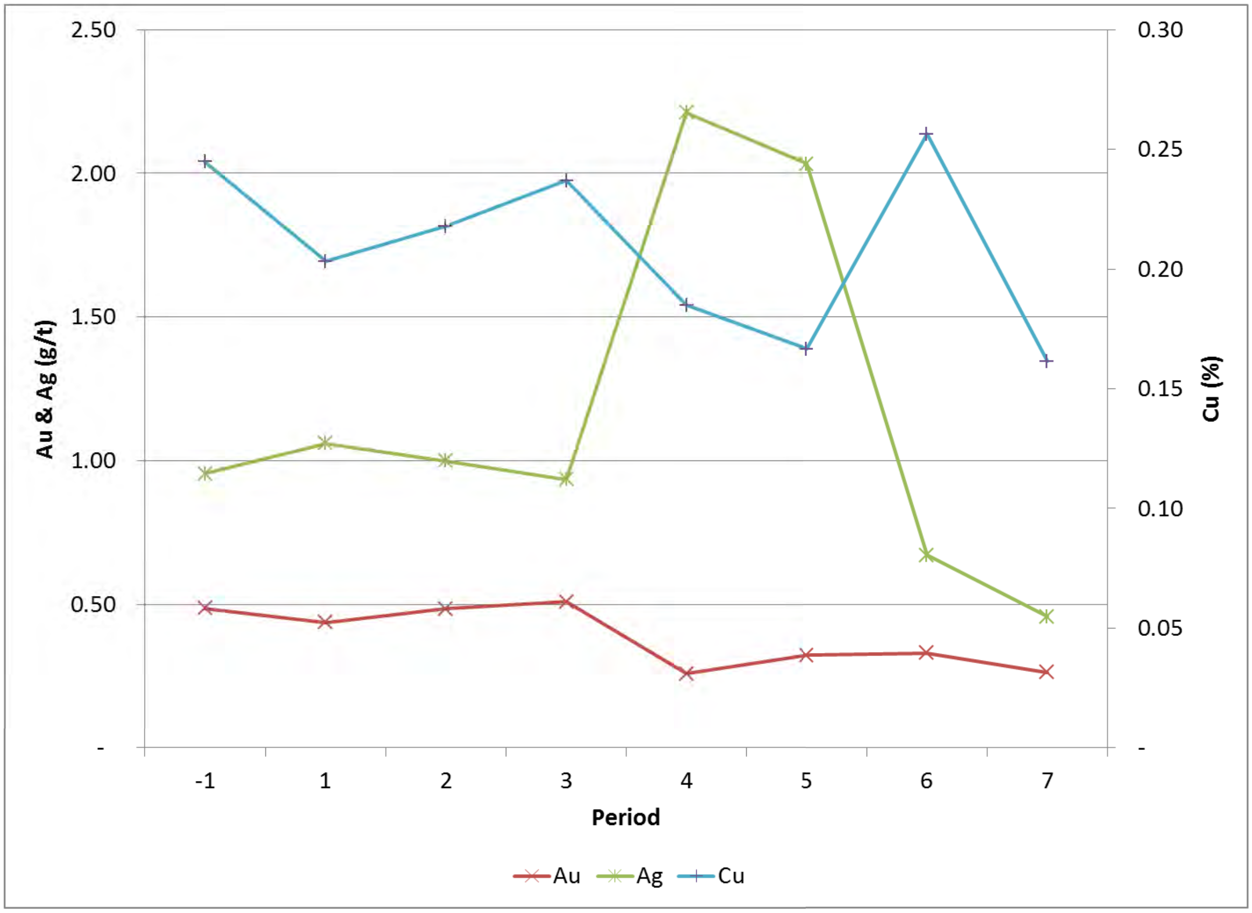


Figure 16-17: Annual Sulphide Ore Head Grade

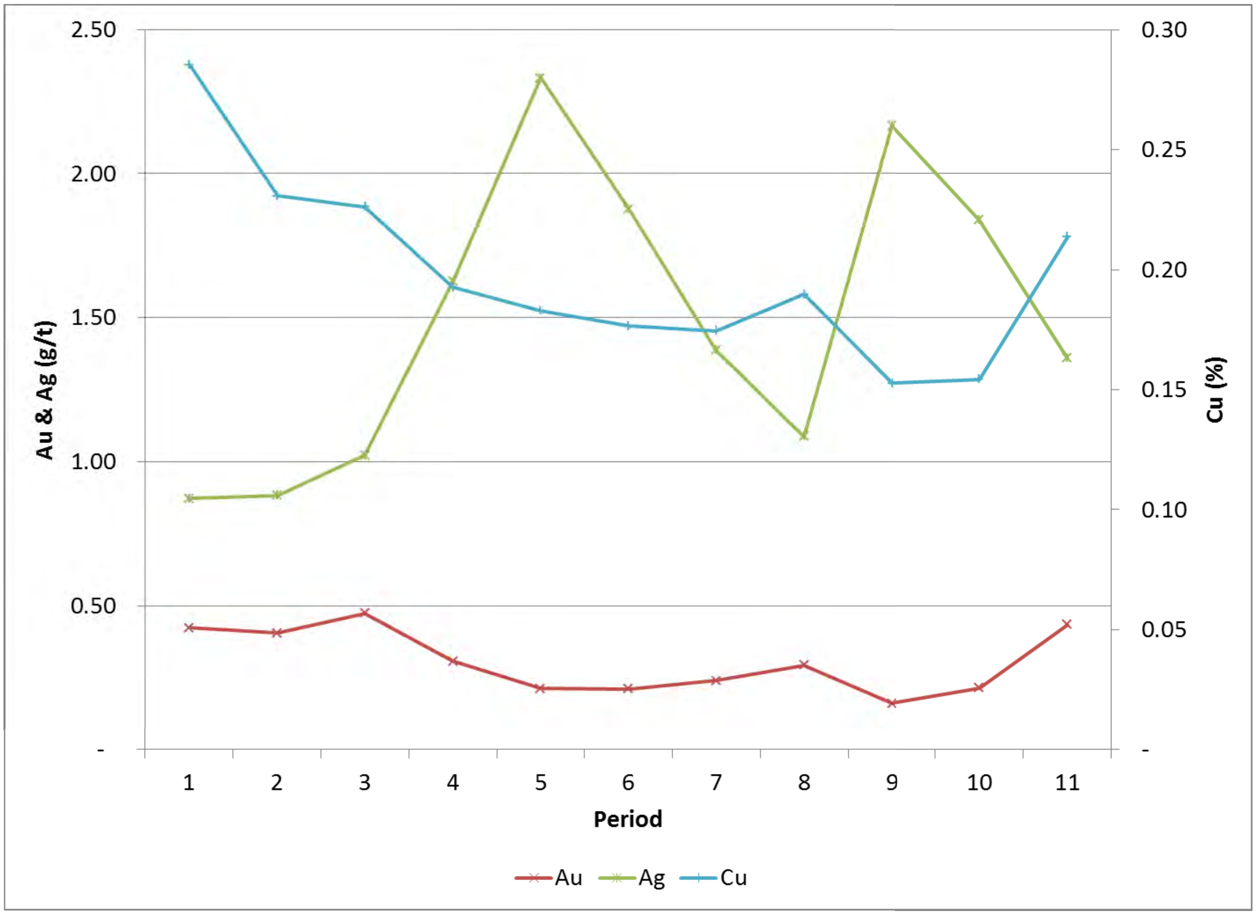
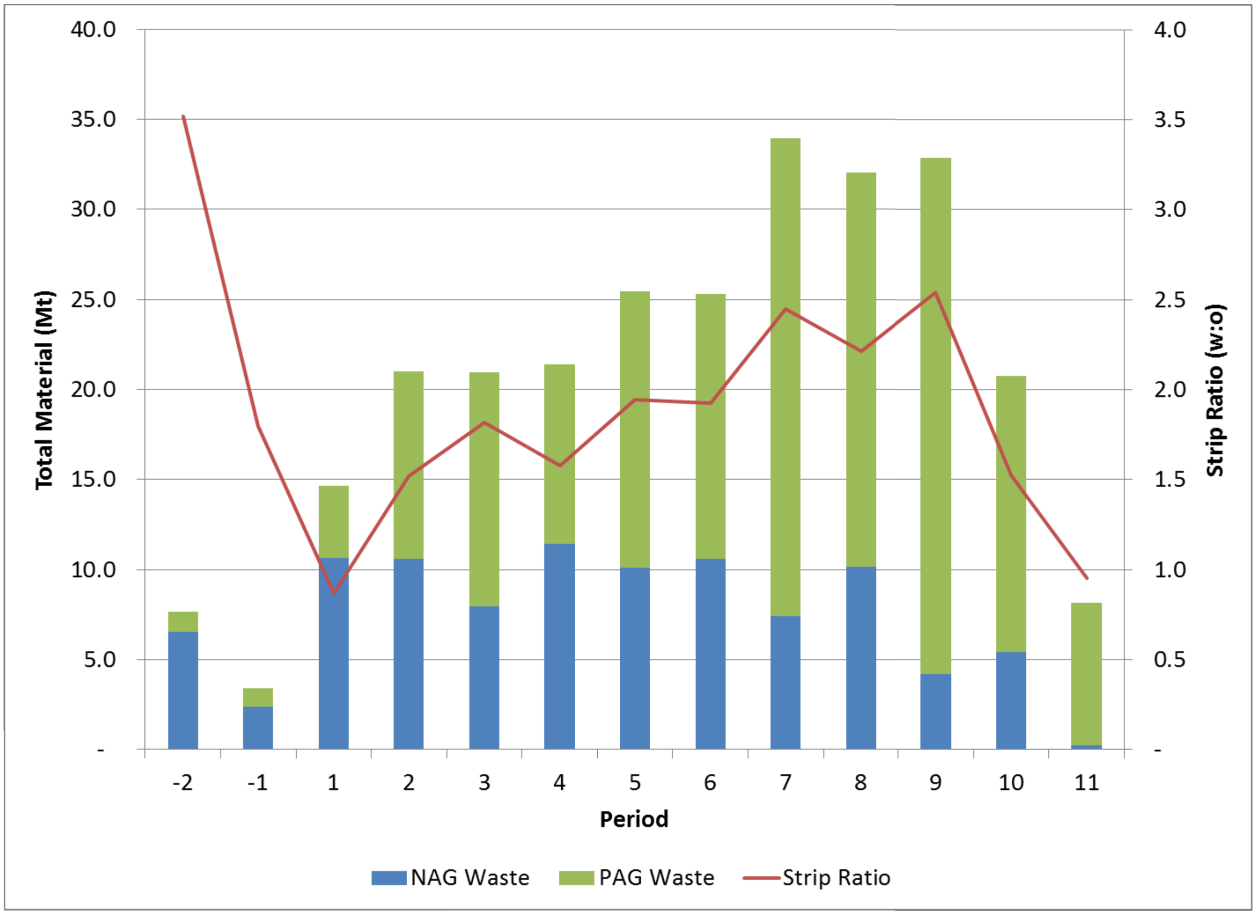


Figure 16-18: Annual Waste Mined



16.7 Mine Operations

16.7.1 Mine Equipment

16.7.1.1 Mine Equipment Parameters

The mine is planned to operate 365 days per year in 2 x 12 hour shifts per day. Equipment is expected to have long-term mechanical availability of 85%. Utilization or use of available hours has been assumed to be 95%. This would give a total of 7,074 operating hours per year.

It should be noted that 85% mechanical availability is an overall accepted standard for mining operations that perform their own maintenance. Maintenance and repair contract (MARC) structures would be considered during final project execution and would likely increase the average equipment availability.

The operations efficiency is assumed to be 85%. The net (or effective) operating hours per shift were estimated to be 10.2 hours, and accounts for breaks, travel, and other non-productive time.

Detailed equipment productivity calculations were made on an annual basis for shovels, trucks and drills. Support equipment was factored on an annual basis according to material movement and / or assumed operating requirements.

16.7.1.2 Mine Equipment Requirements

Mining equipment has been selected based on the following criteria:

- Annual ore and waste production requirement
- Pit design parameters and working bench height
- Productivity and operating costs.

A single original equipment manufacturer (OEM) was planned for drills, shovels, trucks and support equipment. A single supplier serves to reduce maintenance and supply chain direct and indirect costs. On-site OEM maintenance support personnel would be reduced to one supplier; parts procurement, shipping and storage is minimized; shop space and tooling are reduced; personnel safety and training requirements are reduced; and parts are interchangeable between units.

Annual major mining equipment required for the operation is recorded in Table 16-7. Annual mine support equipment required is shown in Table 16-8.

A waste stripping contractor would be used to supplement the owner's fleet in production Years 6 to 10. The contractor's equipment would be scaled to work in tandem with the owner's fleet.

Table 16-7: Owner's Production Equipment Fleet

YEAR	-2	-1	1	2	3	4	5	6	7	8	9	10	11
CAT 6050 Hydraulic Shovels	1	1	2	2	2	2	2	2	2	2	2	2	2
CAT 994H Wheel Loader	-	-	1	1	1	1	1	1	1	1	1	1	1
CAT 992K	-	1	1	1	1	1	1	1	1	1	1	1	1
CAT 789D Haul Trucks	3	3	8	10	10	10	10	10	10	10	10	10	10
CAT MD6540	1	1	2	3	3	3	3	3	3	3	3	3	3
CAT D10T Track Dozer	3	3	4	4	4	4	4	4	4	4	4	4	4
CAT 844H Rubber Tire Dozer	-	1	1	1	1	1	1	1	1	1	1	1	1
CAT 16M Motor Grader	1	1	2	2	2	2	2	2	2	2	2	2	2
CAT 777 Water Truck	1	1	1	1	1	1	1	1	1	1	1	1	1

Table 16-8: Owner's Support Equipment Fleet

YEAR	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Excavator	1	1	1	1	1	1	1	1	1	1	1	1	1
Airtrack Drill (Secondary Drill & Blast)	1	1	1	1	1	1	1	1	1	1	1	1	1
Tool Carriers / Small Loaders	1	1	1	1	1	1	1	1	1	1	1	1	1
Skid Steer Loader	1	2	2	2	2	2	2	2	2	2	2	2	2
Lube / Service Truck	1	1	1	1	1	1	1	1	1	1	1	1	1
Welding Service Truck	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel Truck	1	1	1	1	1	1	1	1	1	1	1	1	1
Dump Truck	1	1	1	1	1	1	1	1	1	1	1	1	1
60 t Crane	1	1	1	1	1	1	1	1	1	1	1	1	1
Fork Lift	1	1	1	1	1	1	1	1	1	1	1	1	1
Flat Deck	1	2	2	2	2	2	2	2	2	2	2	2	2
Light Plants	3	6	6	6	6	6	6	6	6	6	6	6	6
Pickups	4	8	12	12	12	12	12	12	12	12	12	12	12
Blaster’s Pickup	1	1	1	1	1	1	1	1	1	1	1	1	1
40 Passenger Bus	1	2	2	2	2	2	2	2	2	2	2	2	2
Ambulance	1	1	1	1	1	1	1	1	1	1	1	1	1

16.7.1.3 Loading Equipment

The primary loading fleet would consist of two CAT 6050 diesel hydraulic front shovels. The shovels are planned to load ore and waste as necessary. A CAT 994H wheel loader would support the shovel fleet. The loader would supply ore to the mill from the pit and stockpile, cleanup up low productivity faces and provide backup during shovel maintenance events.

The bucket on the shovels has been sized to four-pass load CAT 789D haul trucks in a cycle time of approximately three minutes.

The shovels and wheel loader would be fueled and serviced in the pit, and moved away from the face during breaks.

Over the life-of-mine, each shovel would operate approximately 75,000 hours. Major maintenance over-hauls are planned every 20,000 hours. The CAT 994H is expected to operate 30,000 hrs. Annual loading fleet gross operating hours are shown in Figure 16-19.

Contract loading equipment required in production Year 7 through 9 would include CAT 994H, or equivalent capable of loading CAT 789D haul trucks.

16.7.1.4 Haul Trucks

CAT 789D diesel haul trucks would be used to haul ore and waste to destinations around the pit.

Haul profiles for each pit, bench and dump destination were developed using Vulcan. Truck cycle time was estimated for each haul profile. Based on this information, the truck requirement for each year of the mine life was established. Annual truck hours and the required number of units are shown in Figure 16-20.

The truck fleet is planned to operate an average of 80,000 hours per unit. The oldest trucks in the fleet would operate 90,000 hours. Each unit would see major component replacement at the 15,000 hour point.

Contract trucks required in production Year 6 through 10 are proposed to be CAT 789D or equivalent.

16.7.1.5 Drilling Equipment

For the drilling fleet it is planned to use three CAT MD6540 diesel rotary units capable of drilling 250mm diameter ($9^{7/8}$ inch), 15m deep holes in a single pass. 15m deep holes would accommodate the proposed 10m working bench height, 1.5m subgrade and up to 3.5m of irregular bench floor. Annual gross operating hours and number of units are shown in Figure 16-21.

Contract drilling equipment required in production Year 7 and 8 is proposed to be CAT MD6540, or equivalent.

Figure 16-19: Annual Loading Fleet Hours

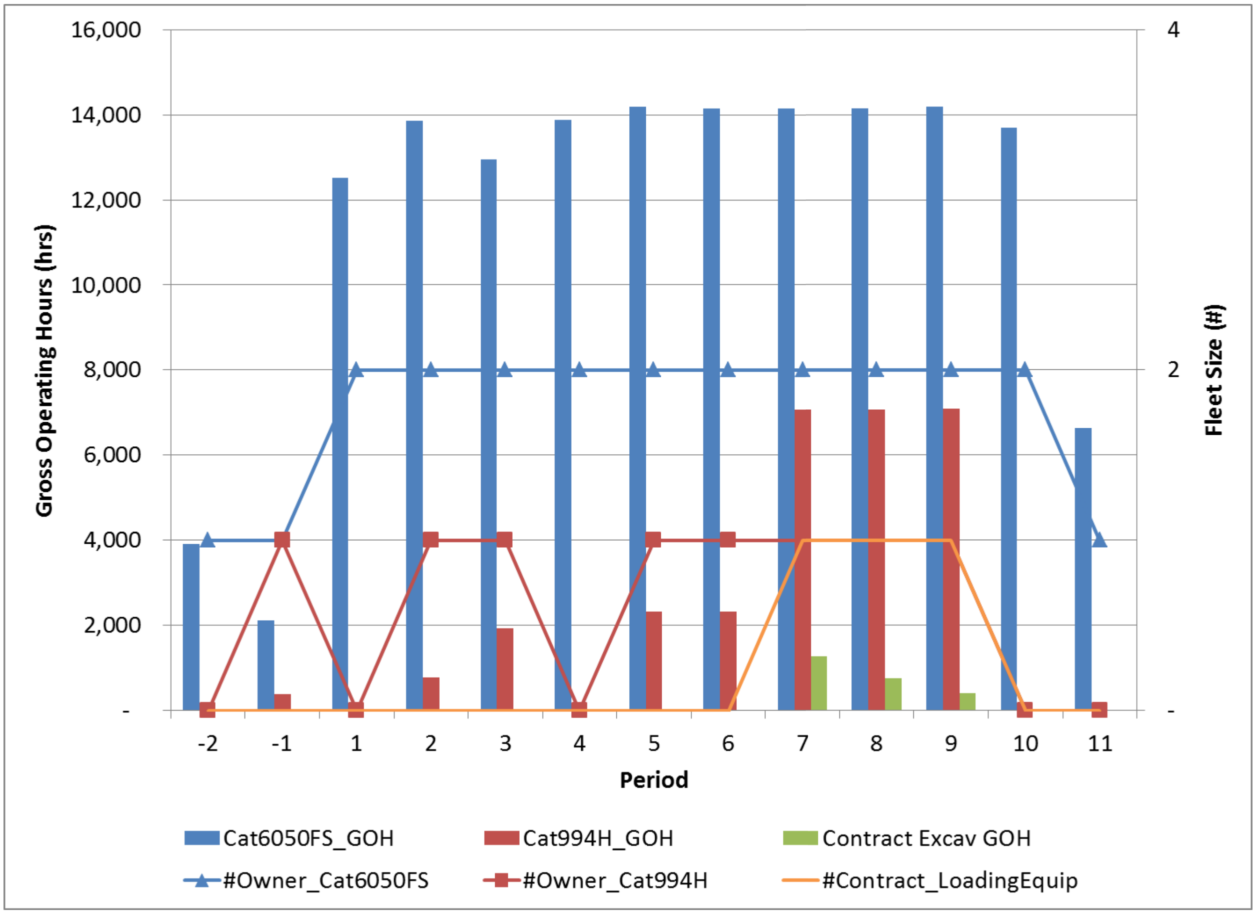


Figure 16-20: Annual Truck Fleet Gross Operating Hours and Number of Units

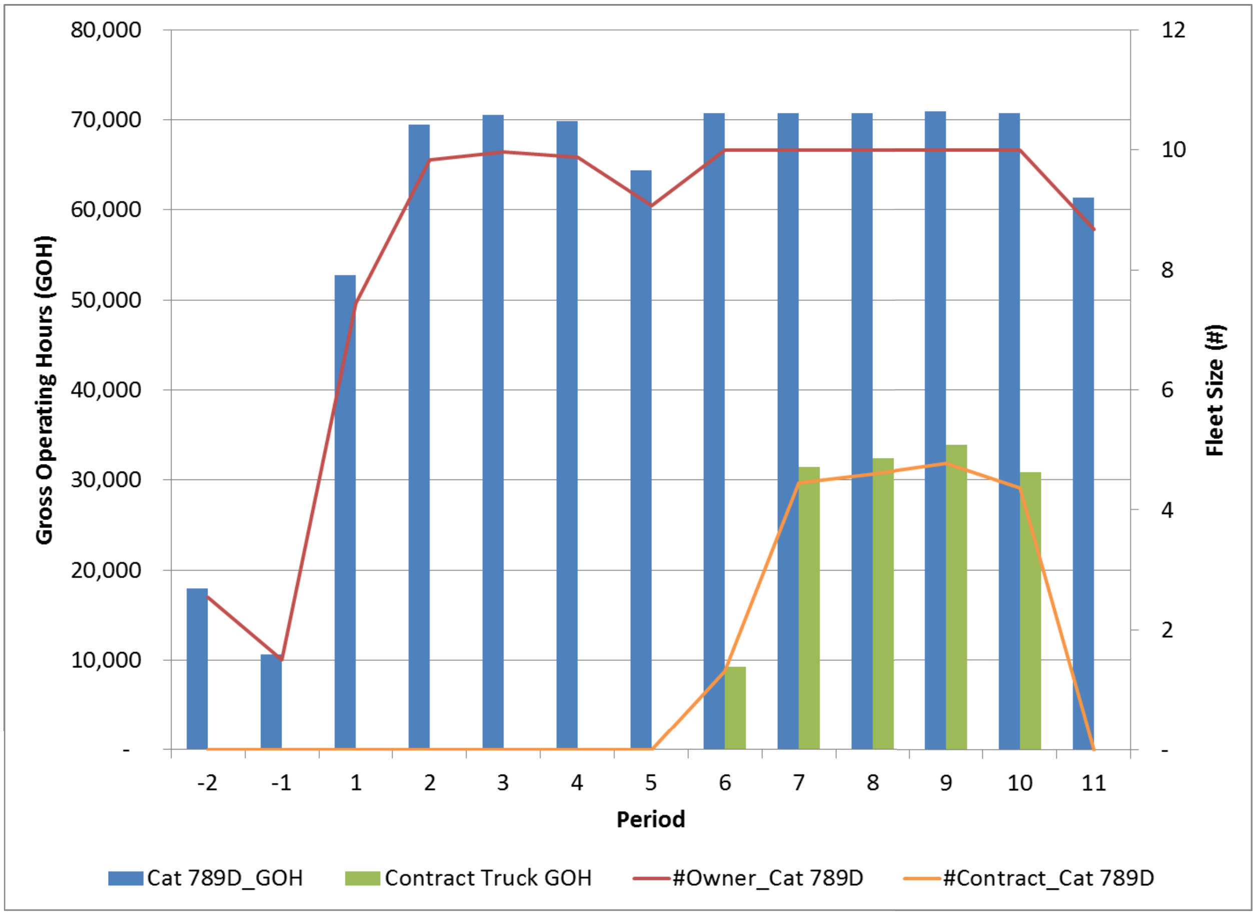
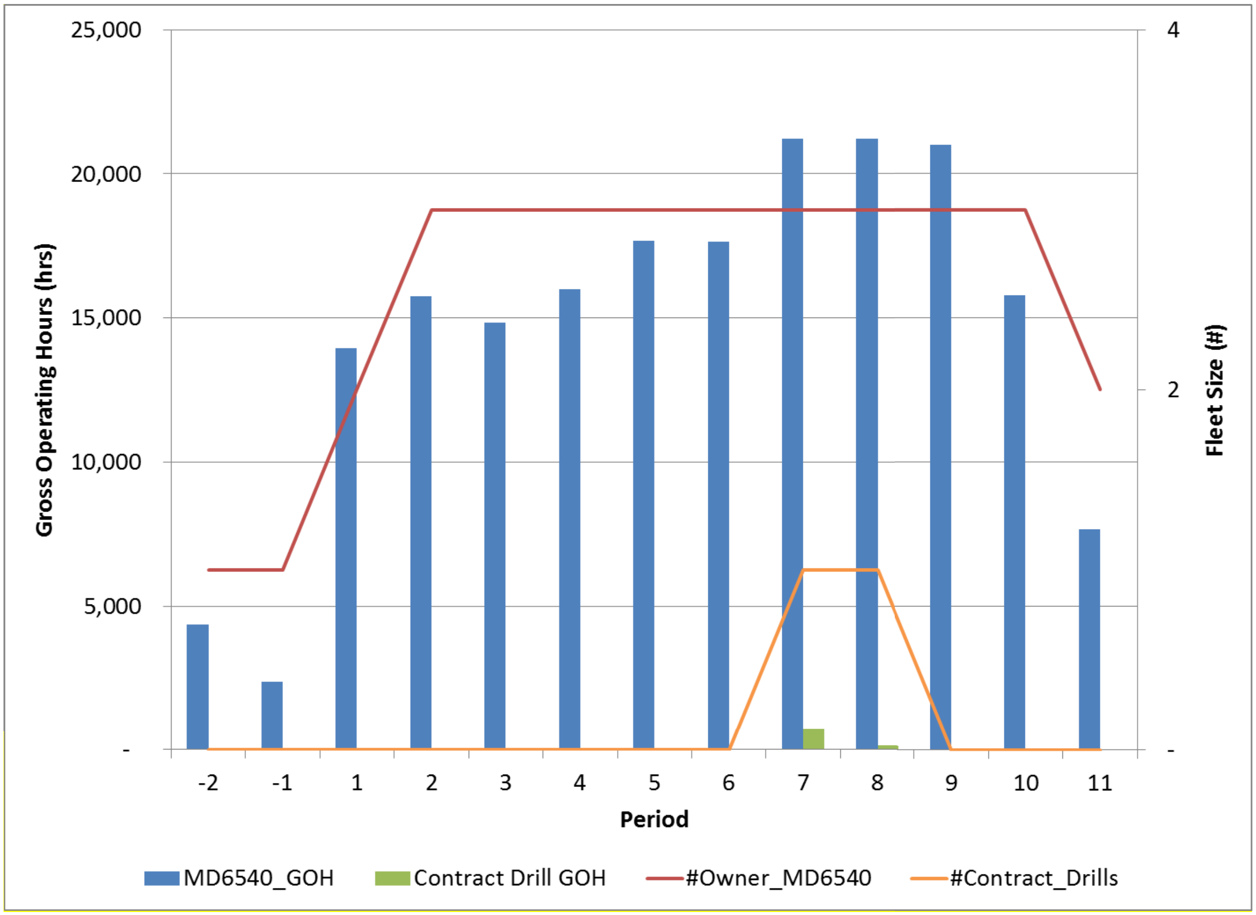


Figure 16-21: Annual Drill Fleet Gross Operating Hours and Number of Units



Secondary drilling and blasting (hard toes, boulders, construction work, etc.) would be completed by one diesel powered, hydraulic top hammer airtrack drill capable of drilling 100mm holes over 20m deep. The machine is capable of drilling angled holes and would be employed for inclined drain holes and anchor placement holes as required to support the final wall slope.

16.7.1.6 Mine Support Equipment

Major mine support equipment would consist of track dozers, rubber tire dozers, motor graders and water trucks.

Track Dozers – Primary production dozing requirements would include waste dump construction, bench cleaning and road construction. A total of four CAT D10T track dozers would be required for full production through the mine life.

Rubber Tire Dozers – A CAT 844H rubber tire dozer would handle in-pit spill rock removal, pre- and post-blast bench preparation and site-wide light duty dozing

Motor Graders – The importance of haul road maintenance to increase production and reduce tire costs would be critical. To achieve this, steady haul road grading efforts would be necessary to remove spill debris, place surface material, and repair roads after inclement weather. The planned haul roads would require two CAT 16M graders to maintain the road running surface.

Water Trucks – Water trucks would operate during warm, dry months to keep dust levels to a minimum in order to improve safety and productivity (through improved visibility and reduced dust exposure) and reduce environmental impact. The water truck would also serve as an auxiliary fire truck. One CAT 777 water truck would be required through the mine life.

16.7.1.7 Equipment Productivity

Average major mine equipment productivity per gross operating hour (GOH) is recorded in Table 16-9.

Table 16-9: Major Mining Equipment Productivity

MODEL	UNITS	VALUE
CAT MD6540 Drill	m/GOH	23.0
CAT 6050 Hydraulic Shovel	t/GOH	2,500
CAT 994H Wheel Loader	t/GOH	1,300
CAT 789D Haul Truck	t/GOH	470

16.7.2 Mine Equipment Maintenance

The focus of the equipment selection was on minimizing product variability, service, support technicians, on-site maintenance, and warehouse space, while maximizing parts commonality and overall performance and reliability.

The selected OEM would have proven their integrated equipment and maintenance service capability to existing major Mexican open pit mining operations. The maintenance philosophy consists of procuring the equipment fleet described previously, with a comprehensive planning, supply chain, warehouse and maintenance support package direct to the mine site.

The major equipment truck shop facility would be constructed and maintained by the owner. This facility would house the production, light vehicle, welding, tire shop and electrical instrumentation bays and warehouse. Complete with overhead cranes, office, compressors, HVAC and major tooling, this building would support the integrated maintenance and supply chain team of owner and OEM supplier personnel. Separate fuel and lube and wash bays would be constructed.

Other OEM suppliers would be required in the following areas:

- Production and civil construction fleet
- Tires
- Light vehicles.

Personnel from selected suppliers would be scheduled on a regular rotation to provide the core of the technical equipment maintenance services. These suppliers, in coordination with the owner's maintenance planning, production and administration personnel, would operate out of the shop facilities and ensure that all preventative, scheduled, and re-build maintenance is conducted efficiently.

The shop warehouse would serve as an extension of the supplier's warehouse and internal inventory control to ensure necessary parts are adequately stocked, controlled and revised as necessary to achieve the planned reliability rates. The integrated owner and multiple suppliers maintenance team would be structured as an alliance, whereby redundant personnel are eliminated and common services are provided by the best suited and shared personnel amongst the on-site group.

Ongoing routine services, wash bay, and maintenance labour would remain the responsibility of the owner; however, they would be coordinated to meet the overall equipment supplier's maintenance needs. Planning and supply chain staff would be integrated both on site and off site to the equipment supplier network, thus eliminating the need for separate maintenance planning and tracking software (other than what is currently used by the supplier systems).

16.7.3 Drill & Blast

16.7.3.1 Blast Pattern Design

Blast pattern design parameters are shown in Table 16-10. Powder factors of approximately 0.26kg/t are consistent through both ore and waste in all rock types for production blasting. A technique of buffer blasting would be employed for pit wall definition. The final two rows of holes are planned to have reduced sub-drill, and would be loaded at reduced powder factor.

Table 16-10: Blast Pattern Design Parameters

Pattern	Burden	Spacing	Stemming	Sub-drill	Explosives	Powder Factor
Units	**m**	**m**	**m**	**m**	**kg/hole**	**kg/t**
Ore 250mm	6.5	6.5	4.5	1.5	292	0.26
Waste 250mm	6.5	6.5	4.5	1.5	292	0.26
Buffer (row 1) 250mm	6.5	6.5	5.0	0.0	209	0.19
Buffer (row 2) 250mm	6.5	6.5	4.5	0.8	261	0.23

16.7.3.1 Explosives

Explosives would be supplied by a single service provider. Explosives consumption is based on production requirements and powder factors described in Section 16.8.1. Explosives are planned to consist of ammonium nitrate fuel oil (ANFO) and emulsion mixtures. All ammonium nitrate required for the year would be transported via highway to the site where it would be stored in two 50t silos. Blasting requirements suggest an ammonium nitrate consumption rate of approximately 24tpd. Given this consumption rate, it is expected the site would have a four day supply of ammonium nitrate. Mixing and delivering explosives to the hole would be the responsibility of the selected supplier. Mine operations personnel would be responsible for the blasting pattern design and for tie-ins.

16.7.4 Mine Personnel

This section describes the methods used to estimate mine operations, maintenance and technical services personnel requirements. Excluded are personnel required to operate the processing plant, warehouse and site general administration.

16.7.4.1 Organization Structure

For costing purposes, personnel are subdivided into three main categories as summarized in the Table 16-11.

Table 16-11: Mine Operations Organizational Summary

Area	Total Personnel	Per Shift
Units	#	#
Mine Operations	87	24
Mine Maintenance	46	12
Technical Services	13	N/A
Total Mine Personnel	**146**	

16.7.4.2 Mine Operations

Mine Operations is proposed to be the largest work force at the mine site, and consists of four areas:

Supervision – Would beresponsible for the direction of the mine equipment, drilling and blasting operations and safety and welfare of the equipment operators and blast loading personnel.

Load and Haul – The Load and Haul area would include equipment operators skilled in running shovels, loaders, excavators, trucks, tracked dozers and graders.

Drill and Blast – The Drill and Blast area would include skilled drill operators and blast loading personnel.

Support Equipment and Mine Services – Support and mine services personnel would be required to support mining operations, as well as provide site-wide services such as freight handling, crane operation, aggregate crushing and general site maintenance and electrical services. They would include supervisors, labourers and equipment operators.

Mine operations personnel are summarized in Table 16-12.

Table 16-12: Mine Operations Personnel

POSITION	QUANTITY	SCHEDULE
Supervision		
Manager	1	10x4
Superintendent	1	10x4
Supervisors	3	2x1
Load & Haul		
Shovel Operators	9	2x1
Truck Drivers	30	2x1
Support Equipment Operators	24	2x1
Mine Services	4	1x1
Drill & Blast		
Drillers	9	2x1
Blasters	2	1x1
Blaster's Helpers	4	1x1
Total	87	
Operating Crew(1)	24	

Notes:1) Operating Crew is 2 x 1 positions excluding Supervisors, Mine Services and Blast personnel.

16.7.4.1 Mine Maintenance

The Mine Maintenance area would consist of supervisors who would monitor the skilled owner maintenance personnel responsible for maintaining, repairing, fueling and lubricating the mobile mine equipment. The owner maintenance team would be supplemented with contract OEM maintenance specialists. Mine Maintenance personnel are summarized in Table 16-13.

Table 16-13: Mine Maintenance Personnel

POSITION	QUANTITY	SCHEDULE
Superintendent	1	10x4
Supervisors	3	2x1
Light Vehicle Mechanics	4	1x1
Mechanics	12	2x1
Welders	6	2x1
Electricians	3	2x1
Apprentices	6	2x1
Service, Fuel & Lube	3	2x1
Shop Labourers	6	2x1
Maintenance Planner	2	10x4
Total	46	
Maintenance Crew(1)	12	

Note: 1) Maintenance Crew is 2 x 1 positions excluding supervisors.

16.7.4.1 Technical Services

Technical services personnel would be responsible for mine engineering, geology, surveying and IT / communication services. The number of personnel required is recorded in Table 16-14.

Table 16-14: Technical Services Personnel

POSITION	QUANTITY	SCHEDULE
Superintendent	1	10x4
Planning Engineers	2	10x4
Project Engineers	2	10x4
Geology Chief	1	10x4
Geologist	1	10x4
Surveyors	2	1x1
Ore Control Technicians	2	1x1
Helpers	2	1x1
Total	13	

17 Recovery Methods

17.1 Introduction

Allnorth Consultants Ltd. (Allnorth) was engaged by the Owner to carry out the mineral process design for the prefeasibility study on the Tepal project. The results of the metallurgical testing established that the Tepal project resource contains gold, copper, and silver. It has two components, a surface oxide layer and a deeper sulphide deposit. The oxide layer is planned to be processed using a conventional gyratory crusher, SAG & ball mill grinding circuit at 56,000tpd followed by settled storage in a pond. A dredge would recover this material at 6850tpd and pump it to a CIL circuit.

The sulphide material is planned to be processed through the same grinding circuit at a rate of 40,00tpd for Tepal North zone (NZ) and 35,000tpd for Tepal South zone (SZ) and Tizate. Milled material would be fed to a conventional copper flotation circuit. The copper concentrate is planned to be thickened and then dewatered using filter presses for shipping to smelters for final processing. A pyrite flotation concentrate made from copper rougher flotation tailings, combined with the first copper cleaner tailings, would constitute the feed to a second dedicated CIL circuit. Gold and silver from both CIL circuits would be extracted from activated carbon and poured to make doré bars.

17.2 Oxide Plant Design

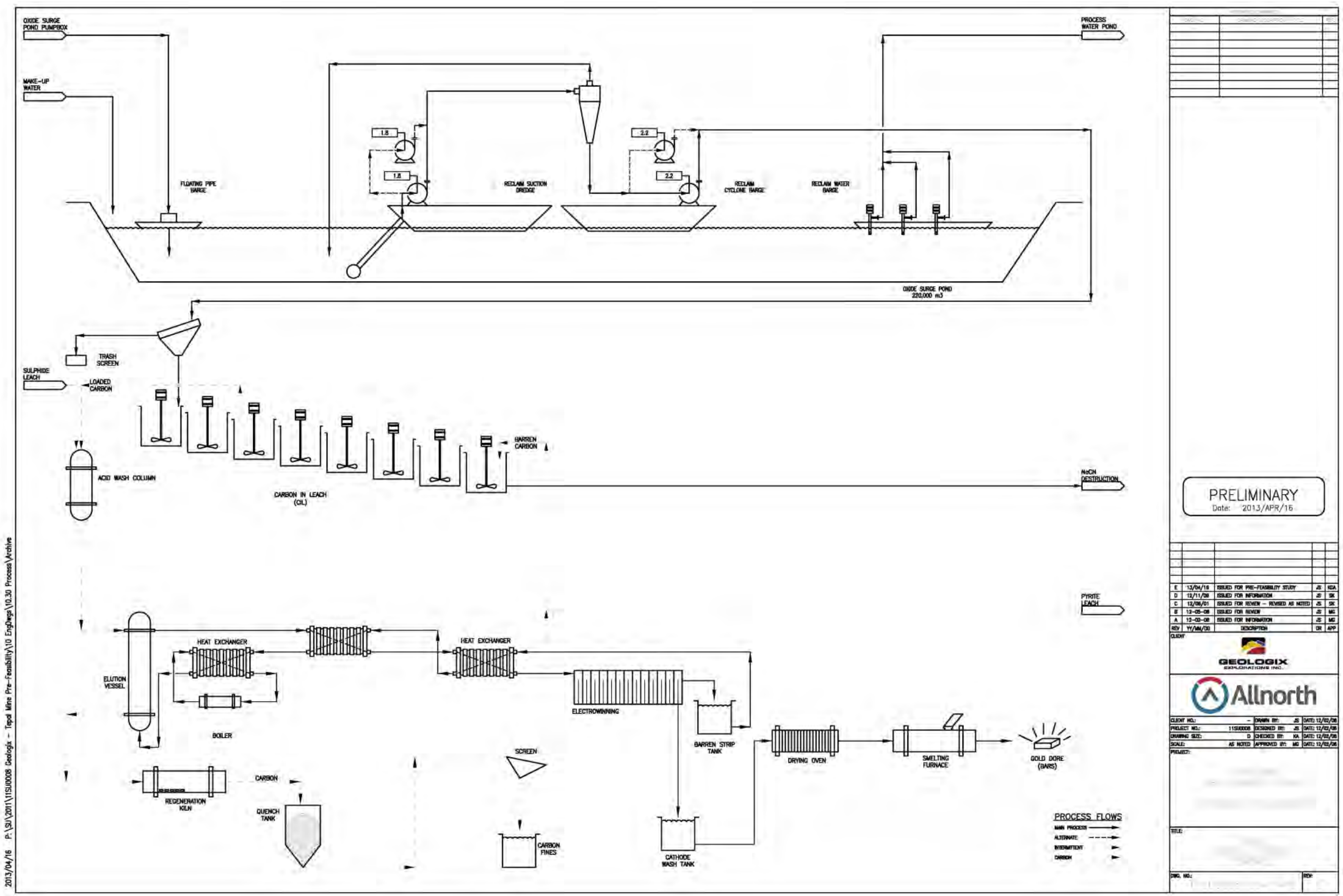
Mined ore would be transported by haul trucks and offloaded in a section of the ore yard at the rate of 56,000tpd for 4 days out of a 32 day cycle. Run-Of-Mine (ROM) ore would be put through the gyratory crusher, the SAG mill and ball mill circuit at a rate of 56,000tpd . The milled product would be sent to a storage pond where it would settle to the bottom. Surplus decanted water would be recycled.

During the four days of oxide grinding and for the following 28 days, the settled solids would be recovered by a commercial dredge, thickened, and sent to an eight tank CIL circuit where it would be leached by a weak cyanide solution. Gold and silver would be recovered from the loaded carbon using a 5-tonne carbon plant followed by electrowinning and smelting in a refinery to produce doré bars.

A Heap Leach option was investigated, priced and designed by Knight Piesold and Allnorth. This option was determined to be uneconomic relative to the CIL option. The CIL option had the benefit of higher recovery, lower capital cost and increased mining efficiency.

Figure 17-1 below shows the Oxide Overall Process Flowsheet.

Figure 17-1: Oxide Overall Process Flowsheet



17.2.1 Major Design Criteria

The proposed grinding circuit would process oxide ore at a nominal rate of 56,000tpd for four days of a 32-day cycle. The reclaim rate from the oxide surge pond would be 6,850tpd. The major criteria used in the design is summarized in Table 17-1.

Table 17-1: Major Design Criteria for Oxide Ore

Criteria	Units	Value
Operating Days	d	4 of 32
Operating Hours	h/d	24
Daily Process Rate	t/d	56,000
Crushing Availability	%	70
Primary Crushing Rate	t/h	3,333
Grinding Availability	%	92
Grinding Process Rate	t/h	2,536
SAG Mill Feed Size, 80% passing	mm	150
Ball Mill Product Size, 80% passing	µm	150
Ore Specific Gravity	N/A	2.45
Drop Weight Index, 80% hardest	kWh/m^3	4.1
Bond Ball Mill Index, 80% hardest	kWh/t	12.4

17.2.2 Oxide Surge Pond

A 220,000m^3 lined storage pond is planned to be built by excavating a 10m deep rectangular containment pond with 2.5:1 slopes on all four side surfaces. Spoil from the excavation would be used to construct a 6.2m high berm around the excavation with the inside surfaces continuous with the excavation. Access for servicing would be via a ramp up the side of the berm then descending down to the bottom. A perimeter road is also planned on the top of the berm. If required, flocculant would be added to the pond feed to help settle the solids.

17.2.3 Oxide Storage Pond Flows

Freshly ground solids from the mill would be pumped to a corner of the pond. After passing under the berm road, the pipeline would be made of flexible pipe sitting on floats located every 2-3m. The discharge end of the pipe would consist of multiple spigots over a 50m length. At the beginning of pond filling, this pipe would descend to the bottom and cross to the far end while floating on three meters of water. During filling, the end of the pipe would be towed back and forth to distribute the solids. At the completion of filling it would be floating along the opposite side of the pond at the water line, and during the next 28 days it would slowly descend to the starting position as the water level drops. Filling would stop with one metre of water above the settled solids. An additional metre of freeboard would be included, enough to prevent overtopping in a 200km/h wind on the few days in the cycle when the pond is at a maximum level.

Feed to the pond would be about 28% solids by weight. While the pond fills, it is expected that the solids would settle to 50% solids or greater by volume, or about 71% solids by weight. Surplus water would be removed by barge mounted pumps feeding a flexible pipe on floats. The location of this pump barge would stay much the same horizontally throughout the cycle but it would have 11-13m of vertical travel. A walkway would be built on the floats supporting the pipe. This would allow the crew to reach the barge where small boats would be available for travel on the water as needed.

Solids reclaim would be achieved by using a cutter head dredge with a 6m cutting depth. This dredge is of proven design for tailings pond reclaim. The dredge pump would feed a cyclone pack mounted on a separate barge that would be towed by the dredge. Cyclone overflow would be collected and directed by pipe to the vicinity of the dredge suction inlet. With a net flow of water towards the suction inlet, the solids in the cyclone overflow would rise until equilibrium is reached. Cyclone underflow would enter a pump and be sent to oxide leaching via a flexible pipe on floats. A density gauge on the pump discharge would be used to choose how many cyclones are open on the cyclone pack and controlling the outlet density at 50% solids by weight. To avoid tangling with the feed pipe, the barge would be connected to the mid-line of the long side of the pond and have enough flexible pipe to reach the far corners.

17.2.4 Pond Water Balance

Grinding would generate enough heat to raise the temperature of the water through the mills by 6-10 degrees. As planned, the pond would contain 5 hours of available water at all times. Starting from an ambient temperature of 38ºC in the summer months re-circulated water would soon exceed the recommended 70ºC limit for HDPE pipe. Hot solution would also be detrimental to cyanide leaching. In practice 1,500-2,500m^3/h of oxide storage pond reclaim water would be recirculated to grinding via the cyclone feed pump box and the balance would be pumped to the tailings pump box for delivery to the TSF to cool off. The balance of the grinding needs would come from the process water pond. For 28 days out of the 32-day cycle the pond would require a net inflow of water to repulp the solids to the desired 50% solids, giving a pulp density in which carbon has neutral buoyancy during leaching.

17.2.5 Oxide Leaching

Laboratory work showed that most or all of the soluble gold was liberated at six hours. A Carbon-In-Leach (CIL) tank size was chosen with one hour of residence time per carbon stage and six stages, giving a 500m^3 tank. Two leach tanks were added to the oxide circuit with the option to put carbon in the first two tanks if desired, for a net residence time of 8 hours. The tanks would be arranged in a hexagonal pattern to minimize the footprint. The available site has a natural slope and the tanks would sit on a series of descending steps. Any one tank could be taken off line for maintenance. It is proposed to use liquid lime to maintain a ph of 11. At the end of the CIL circuit the pulp would enter a two tank cyanide destruct module using hydrogen peroxide as the reagent. Carbon from the CIL circuit would be sent to the Carbon Plant and Refinery. Refer to Section 17.4.

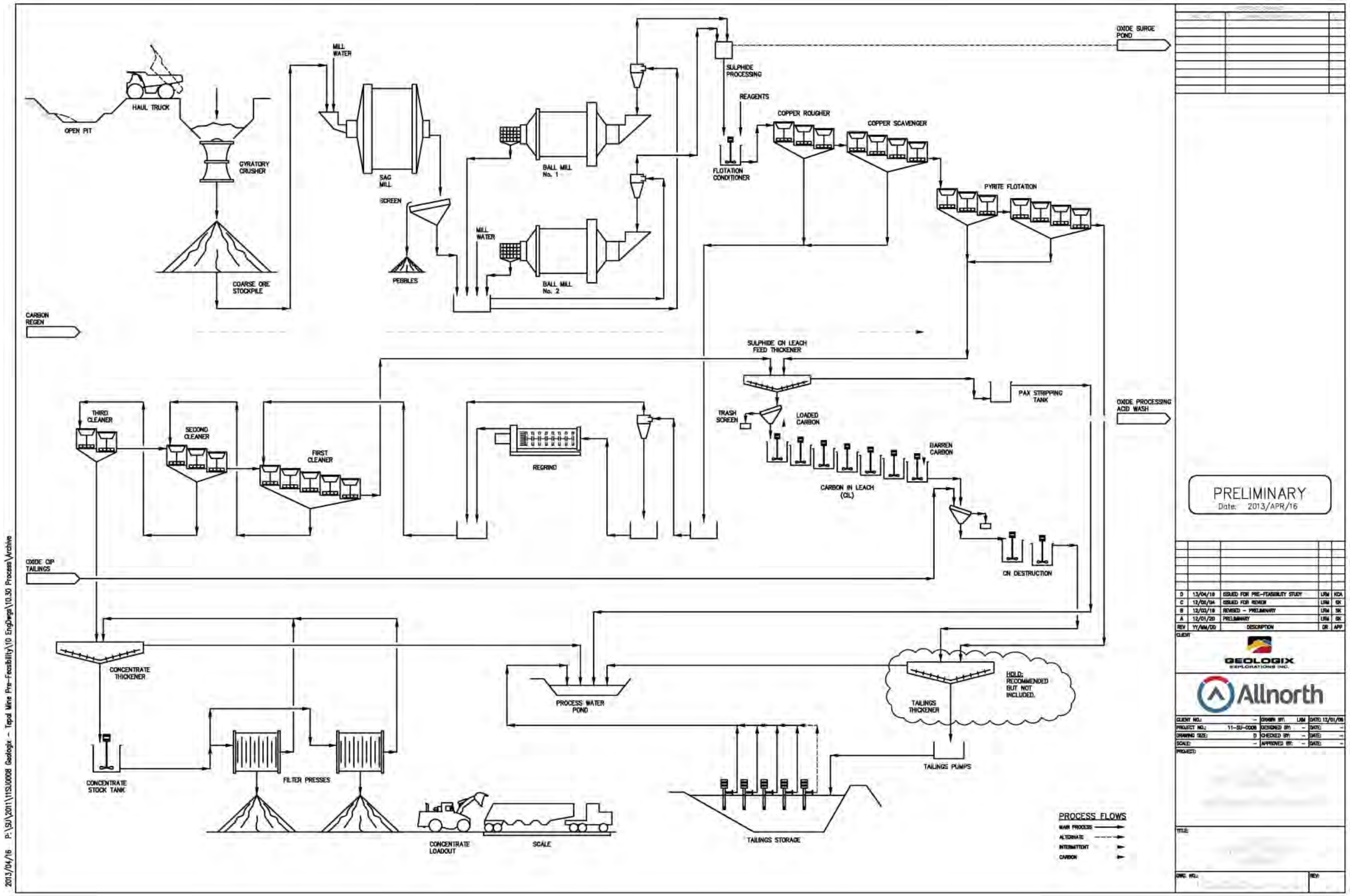
17.3 Sulphide Plant Design

The sulphide concentrator was designed to process 40,000tpd of NZ and 35,000tpd of SZ and Tizate ore in order to maximize recovery while maintaining simplicity of operation and a conventional processing layout. The ROM would be reduced through three stages of comminution then the copper minerals along with some gold and silver would be recovered by flotation. The copper rougher/scavenger concentrates would be reground and cleaned to a final commercial concentrate grade and then dewatered. The produced copper-gold concentrate would be trucked off site to a copper smelter.

Rougher/scavenger tailings would be sent to pyrite flotation. Pyrite concentrate and the first copper cleaner tails would be combined and thickened for feed to a sulphide CIL circuit. Loaded carbon is planned to be sent to the combined oxide and sulphide carbon plant and refinery where doré bars would be produced. The flotation tailings would be pumped to the TSF. A reclaim barge would recover water from the TSF for re-use.

Figure 17-2 below shows the Sulphide Overall Process Flowsheet.

Figure 17-2: Sulphide Overall Process Flowsheet



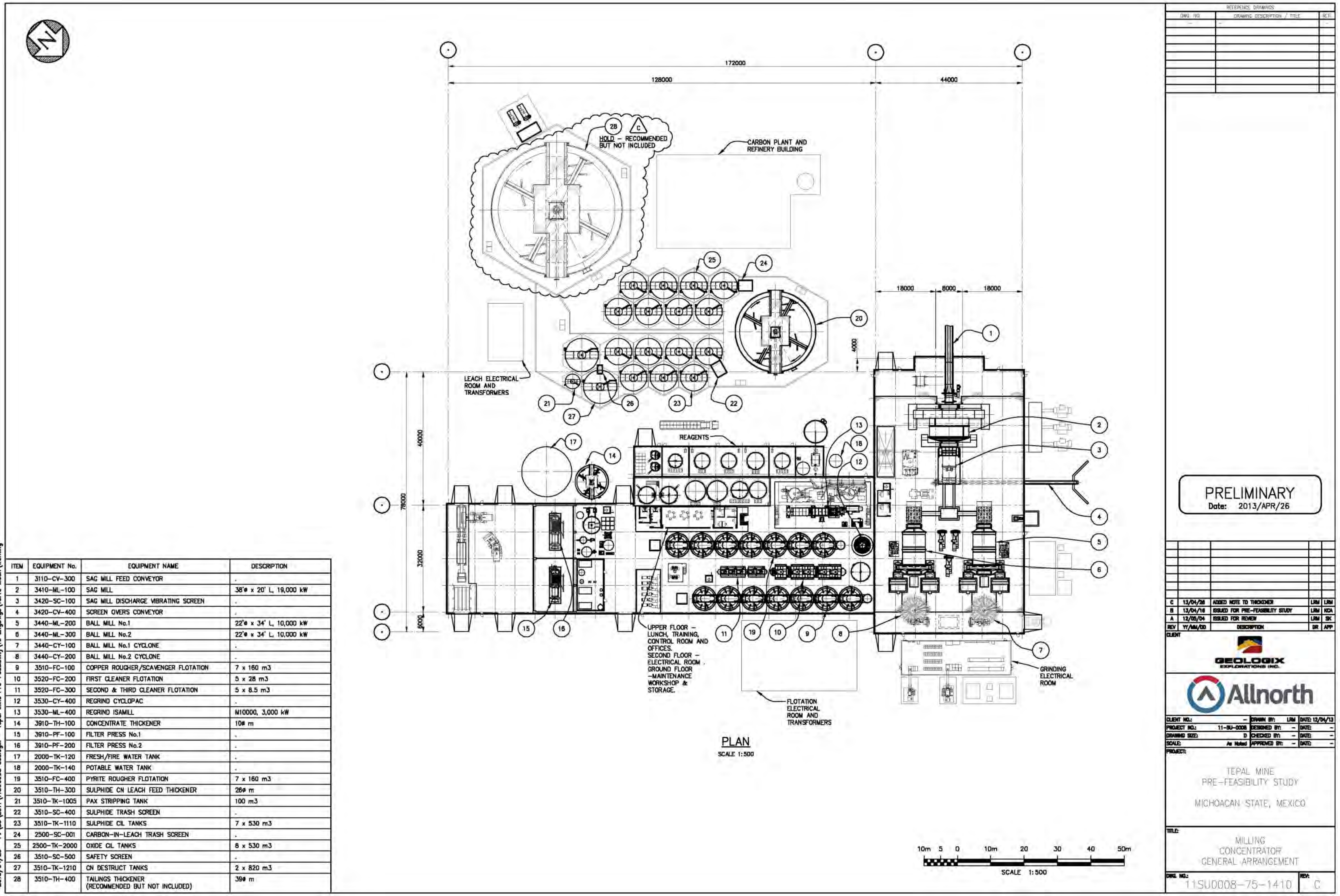
The following unit operations and facilities are proposed for the sulphide process plant:

- Primary crushing
- Conveying to coarse ore stockpile
- Coarse ore reclaim
- Primary grinding circuit
- Copper rougher/scavenger flotation
- Rougher/scavenger concentrate regrinding
- Three stage copper cleaner flotation
- Copper concentrate thickening, pressure filtration and stockpiling
- Pyrite flotation, one stage only
- Pyrite concentrate and first copper cleaner tails leaching
- Tailings disposal to the tailings storage facility.

Figure 17-3 below shows the Process Plant Layout.

It should be noted that the tailings thickener in the drawing is recommended for water balance efficiency but has not been capitalized. Furthermore, the current water balance is not adjusted for the tailings thickner, which would be a positive benefit to tailings pumping capital costs and net overall water balance. This will be further evaluated during the next stage of project development.

Figure 17-3: Process Plant Layout



ITEM	EQUIPMENT No.	EQUIPMENT NAME	DESCRIPTION
1	3110-CV-300	SAG MILL FEED CONVEYOR	.
2	3410-ML-100	SAG MILL	38'ø x 20' L, 19,000 kW
3	3420-SC-100	SAG MILL DISCHARGE VIBRATING SCREEN	.
4	3420-CV-400	SCREEN OVERS CONVEYOR	.
5	3440-ML-200	BALL MILL No.1	22'ø x 34' L, 10,000 kW
6	3440-ML-300	BALL MILL No.2	22'ø x 34' L, 10,000 kW
7	3440-CY-100	BALL MILL No.1 CYCLONE	.
8	3440-CY-200	BALL MILL No.2 CYCLONE	.
9	3510-FC-100	COPPER ROUGHER/SCAVENGER FLOTATION	7 x 160 m3
10	3520-FC-200	FIRST CLEANER FLOTATION	5 x 28 m3
11	3520-FC-300	SECOND & THIRD CLEANER FLOTATION	5 x 8.5 m3
12	3530-CY-400	REGRIND CYCLOPAC	.
13	3530-ML-400	REGRIND ISAMILL	M10000, 3,000 kW
14	3910-TH-100	CONCENTRATE THICKENER	10ø m
15	3910-PF-100	FILTER PRESS No.1	.
16	3910-PF-200	FILTER PRESS No.2	.
17	2000-TK-120	FRESH/FIRE WATER TANK	.
18	2000-TK-140	POTABLE WATER TANK	.
19	3510-FC-400	PYRITE ROUGHER FLOTATION	7 x 160 m3
20	3510-TH-300	SULPHIDE CN LEACH FEED THICKENER	26ø m
21	3510-TK-1005	PAX STRIPPING TANK	100 m3
22	3510-SC-400	SULPHIDE TRASH SCREEN	.
23	3510-TK-1110	SULPHIDE CIL TANKS	7 x 530 m3
24	2500-SC-001	CARBON-IN-LEACH TRASH SCREEN	.
25	2500-TK-2000	OXIDE CIL TANKS	8 x 530 m3
26	3510-SC-500	SAFETY SCREEN	.
27	3510-TK-1210	CN DESTRUCT TANKS	2 x 820 m3
28	3510-TH-400	TAILINGS THICKENER (RECOMMENDED BUT NOT INCLUDED)	39ø m

17.3.1 Major Design Criteria

The concentrator is planned to process sulphide ore at a nominal rate of 40,000tpd for NZ and 35,000tpd for SZ and Tizate. The major criteria used in the design are summarized in Table 17-2.

Table 17-2: Major Design Criteria for Sulphide Ore

Criteria	Units	Value
Operating Days	d	365
Operating Hours	h/d	24
Daily Process Rate	t/d	35,000 – 40,000
Crushing Availability	%	70
Primary Crushing Rate	t/h	2,083
Grinding & Flotation Availability	%	92
Grinding & Flotation Process Rate	t/h	1585
SAG Mill Feed Size, 80% passing	mm	150
Ball Mill Product Size, 80% passing	µm	150
Concentrate Regrind Size, 80% passing	µm	25
Ore Specific Gravity	N/A	2.72
Drop Weight Index, 80% hardest, Tepal N & S	kWh/m^3	8.3
Bond Ball Mill Work Index, 80% hardest, Tepal N & S	kWh/t	17.5
Drop Weight Index, 80% hardest, Tizate	kWh/m^3	10.3
Bond Ball Mill Work Index, 80% hardest, Tizate	kWh/t	20.0

The SAG mill and Ball mills were sized by a comminution specialist based on the drop weight index and the Bond ball mill work index for the Tepal North and South ore deposits and Tizate dposit. Hardness variability work indexes for the three deposits can be found in Tables 13-5, 13-6 and 13-7.

The flotation cells were sized and selected based on estimated slurry flow rates and retention times as determined from laboratory tests. Typical scales up factors were applied to the laboratory determined retention times.

17.3.2 Primary Crushing

The gyratory crusher is proposed as a permanent installation that would take ROM ore and produce a product of 80% passing 150mm. Haul trucks are planned to supply ROM material to the primary crusher dump pocket, where they would unload from one of two dump aprons. The dump pocket would have a hydraulic rock breaker to reduce any oversize rocks that may clog the crusher feed. The gyratory crusher would process the sulphide ROM ore at a rate of 2,083tph. The crushed material would discharge from the underside of the crusher into a hopper from which an apron feeder would meter the flow onto the sacrificial primary crusher discharge belt conveyor. The material would then feed into the coarse ore stockpile belt conveyor which would elevate the material to deposit onto the coarse ore stockpile.

A dust collection and suppression system would be installed to control fugitive dust generated at the crusher, material transfer points and other operations.

The primary crushing installation is proposed to include the following key equipment:

- One gyratory crusher - 1,370 x 1,900mm (54 x 75")
- One apron feeder
- One hydraulic rock breaker
- One crusher outfeed conveyor
- One stockpile feed conveyor
- One dust collection/suppression system.

17.3.3 Stockpile and Reclaim

The coarse ore stockpile would hold one day of live storage of the crushed material, or 35,000 tonnes. Three apron feeders would reclaim the material with two operating, and one on standby, during normal operation. The apron feeders would meter the flow onto the SAG mill feed conveyor at a controlled rate. The SAG feed conveyor would be equipped with a belt scale.

A dust collection and suppression system would be installed to control fugitive dust generated in the reclaim tunnel and the material transfer points.

17.3.4 Primary Grinding and Classification

The primary grinding circuit is proposed to incorporate a SAG mill and two ball mills. The process rate would be 1,585tph (35,000tpd) of solids for Tepal South and Tizate ore. For Tepal North ore, the process rate is planned to be 1,812tph (40,000tpd) due to slightly softer ore. The mine plan has been adjusted to feed Tepal North phase 1 ore in the first two years. This has the benefit of higher throughput and higher head grades.

The SAG mill would be fed at a controlled rate by the reclaim apron feeders under the coarse ore stockpile. Lime would be added to the SAG mill feed belt conveyor to raise the pH of the slurry to 10.5, which would aid copper flotation. A SAG mill ball bin and feeder would feed fresh grinding media onto the SAG mill feed belt conveyor to maintain the grinding charge.

The SAG mill discharge containing 70% solids by weight would pass over a screen to remove over-size pebbles. The pebbles would be conveyed outside the building to a discharge pile for manual re-entry into the process, storage for future processing, or disposal, depending on the ore characteristics.

The SAG mill screen underflow would combine with both ball mill discharges into one common pump box. The two ball mills would be in closed circuit with two cyclone clusters and two slurry underflow streams, one to each ball mill. The combined overflow slurry streams would feed the copper rougher/scavenger flotation circuit. The cyclone overflow particle size is proposed to be 80% passing 150µm and contain approximately 28% solids by weight. Cyclone underflow to the ball mills would be approximately 72% solids by weight, and the circulating load would be approximately 300% of new mill feed. Ball charge systems would add grinding media as required for maintaining grinding charge.

For four days out of every thirty two days, the circuit would be fed with oxide ore and the cyclone overflow would be sent to the oxide ssurge pond.

The grinding circuit would include the following key equipment:

- One SAG Mill – 11.6Ø x 6.1m (38Ø x 20ft.), 19MW
- Two vibrating screens (one operating, one standby)
- Two ball mills – 6.7Ø x 10.4m (22Ø x 34ft.), 10MW
- Three cyclone feed slurry pumps (two operating, one standby)
- Two cyclone clusters.

17.3.5 Copper Rougher/Scavenger Flotation

The overflow slurry from the cyclone clusters would gravity flow to the flotation conditioning tank. Reagents would be added to the conditioning tank to prepare the slurry as feed to the copper rougher/scavenger flotation cells. The concentrate would flow to the regrind cyclone - feed pump box. The rougher/scavenger tailings would feed the pyrite flotation circuit.

Flotation reagents would be dry lime added directly to the SAG feed and also in the conditioning and cleaner banks as a slaked lime. Reagent 3418 would be used as the collector, and methyl isobutyl carbinol (MIBC) as the frother.

The copper rougher/scavenger flotation cells would include one bank of seven 160m^3 mechanical flotation tank cells, four cells as the rougher and three as the scavenger.

17.3.6 Regrind Circuit

The concentrate from the copper rougher/scavenger would be the regrind cyclone feed slurry. The regrind cyclone feed would be pumped to the regrind cyclone cluster. The cyclone overflow would bypass the regrind circuit and feed to the regrind product pump box. The cyclone underflow containing approximately 70% of the feed to the cyclone, would be approximately 55% solids by weight and would report to the regrind IsaMill™ feed pump box.

The regrind circuit would include the following equipment:

- One regrind IsaMill™ and associated equipment

- One cyclone cluster
- Two cyclone feed pumps (one operating, one standby).

17.3.7 Cleaner Flotation

Regrind product is planned to be pumped to the first cleaner flotation cells. Concentrate from the first cleaner cells would be pumped to the second cleaner flotation cells while second cleaner concentrate would be pumped to the third cleaner flotation cells. Concentrate from the third cleaner flotation cells would be the final copper concentrate. Tailings from stage three would flow by gravity into the feed of stage two, and those of stage two would flow into the feed of stage one. Tailings from the first cleaner would flow by gravity into the sulphide leach thickener feed pump box.

The cleaner flotation circuit would include the following equipment:

- One bank of five (5) $30m^3$ first cleaner mechanical flotation cells
- One bank of five (5) $8.5m^3$ second and third cleaner mechanical flotation cells
- Two first cleaner concentrate pumps (one operating, one standby)
- Two second cleaner concentrate pumps (one operating, one standby)
- Two third cleaner concentrate pumps (one operating, one standby).

17.3.8 Concentrate Dewatering and Handling

The concentrate from the third cleaner is planned to be processed by a high rate thickener for preliminary dewatering and then sent to a filter press for final drying. The filter press concentrate cake would be stockpiled for shipment to the smelter.

Flocculant would be added to the thickener feed to accelerate the settling process. Thickener overflow would be sent to the process water pond. The 55% solids thickener underflow would be pumped to the concentrate stock tank for storage prior to being fed to the filter press. The filter press would produce cake of 8% moisture for on-site storage before shipment off-site for smelting.

The concentrate dewatering facility would include the following key equipment:

- One 10m diameter high rate thickener
- One concentrate stock tank
- Two filter presses
- Slurry pumps, including high pressure pumps for the filter press.

17.3.9 Pyrite Flotation

Laboratory testing has shown that some gold reports with pyrite if the latter is concentrated. Tailings from the copper rougher/scavenger would feed a pump box. Reagents would be added to prepare the slurry for feed to the pyrite rougher/scavenger flotation cells. Pyrite concentrate would be combined with the first copper cleaner tails in the sulphide leach thickener feed pump box. Special gold promoters could be added to recover harder to float gold. Pyrite rougher/scavenger tailings would be the final tailings and would be pumped to the tailings pond or to a tailings thickener if one is installed.

Potassium Amyl Xanthate (PAX) would be used as the collector, and MIBC as the frother. If one is proven successful, a gold-silver promoter like Aero208 would be used. A suitable flocculant would be added to the pyrite leach thickener.

The pyrite rougher/scavenger flotation cells would consist of one bank of seven 160m^3 mechanical flotation tank cells, four cells as the rougher and three as the scavenger, identical to the copper rougher/scavenger bank.

17.3.10 Sulphide (Pyrite) Leaching

Pyrite concentrate combined with the first copper cleaner tailings would be pumped to a dedicated thickener and thickened to about 45% solids, giving a density at which the carbon should have neutral buoyancy. Approximately 55% of the feed gold and silver would be in the sulphide leach thickener underflow. Sulphide leaching would take place in one leach tank followed by six carbon-in-leach (CIL) tanks with the provision to put carbon in the first tank. All tanks would be identical to those used in the oxide leach, arranged in the same hexagonal pattern to minimize the footprint, and would sit on a series of descending steps. Any one tank could be taken off line for maintenance. Average residence time would be 12 hours due to lower mass than for the oxide leach.

At the end of the CIL circuit, the pulp would enter a two tank cyanide destruct module using hydrogen peroxide as the reagent. This facility would simultaneously treat the combined tailings from the oxide and sulphide leach circuits.

Loaded carbon from the sulphide leach circuit would be sent to the same carbon plant provided for the Oxide Leach where gold and silver would be recovered. Water overflowing the leach feed thickener would be sent through a column of activated carbon to recover any excess reagent from the pyrite flotation stage. This would prevent activation of pyrite in the copper circuit when the water is recycled. This carbon would be cleaned periodically by sending it to the regeneration kiln.

17.3.11 Tailings

Combined oxide and sulphide leach tailings would enter a cyanide destruct module as previously noted. Cyanide free tailings would then be combined with the pyrite rougher tails and pumped to the tailings pond. During the four days of oxide grinding the tailings stream would only contain oxide leach tailings solids and water decanted from the oxide surge pond. The reclaim water pumps would be housed on a reclaim barge at the TSF. The pumps would send reclaim water back to the process water pond.

17.4 Carbon Plant and Refinery

Loaded carbon from both sulphide and oxide leaching would be sent to an acid wash vessel and treated by circulating approximately 3% hydrochloric acid solution to remove scale and other impurities. After neutralization the carbon would be pumped to a Zadra Strip vessel. Gold and silver would be stripped from the carbon by circulating a hot caustic solution through the vessel at about

135°C and a pressure of 345-480kPa. The strip solution would be heated using a combination of plate and frame heat exchangers and an electric hot water heater. After reaching stripping temperature, the solution would flow upward through the strip vessel. The gold-laden solution would exit the top of the strip vessel, flow through the cool down heat exchanger, and flow by gravity to 1.35m^3 electrowinning cells. Here, gold and silver would plate onto stainless steel cathodes or fall to the tank bottom as a fine sludge. Strip solution from the electrowinning cells would gravity flow to a barren solution tank. Gold and silver laden stainless steel cathodes would be taken to a cathode wash tank then cleaned by a high pressure spray. The resulting gold sludge would be separated from the wash solution by a plate and frame filter press. Sludge would be collected, mixed with fluxes, and then melted in an electric induction furnace to produce a doré bar suitable for shipping to a refinery.

After every second strip, carbon from the strip vessel would be transferred to a 125kg/h rotary regeneration kiln at approximately 700°C. It would then be quenched in water, screened to remove fines, and stored in a carbon storage tank to be re-used in the leach circuits. New carbon would be periodically added to the circuit to make up for the fines taken out of the circuit by a sizing screen. The fines would be dried and stored in sacks or barrels for off-site treatment or sale.

Table 17-3: Major Design Criteria for Carbon Plant and Refinery

Item	Units	Value
Doré Production per year, max	oz	324,000
Au solution grade, max	g/t Au	20
Ag solution grade, max	g/t Ag	120
Cu head grade	%Cu	0.2
Metal Recovery from Solution	%	98
Carbon Loading (Target)	g/t	6,000
Carbon Desorption Method	Pressure Zadra	
Precious Metal Recovery Method	Electrowinning	
Electrowinning Location	Refinery	
Electrowinning Type	SS Anodes	
Smelting Furnace Location	Indoor	
Smelting Furnace Type	Tilting Induction	

17.5 Reagents Handling and Storage

To ensure workplace safety, environmental integrity, and to optimize recovery, various reagents would be added to the process where required.

Reagents used in the process would include:

- Solid Lime
- Lime Slurry

- Aero 3418
- Fuel oil
- Frother (MIBC)
- Potassium Amyl Xanthate
- Gold Promoter (A208, to be verified)
- Flocculant(s)
- Caustic Soda
- Sodium Cyanide
- Hydrogen Peroxide
- Hydrochloric Acid.

Each reagent is proposed to have its own preparation system which includes a bulk handling system, a mixing tank if required, and a storage tank. Fresh water would be used for reagent preparation. The mixing and holding tanks would be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. The reagent preparation areas would be equipped with appropriate ventilation, eye-wash stations, safety showers, fire and safety protection, and Material Safety Data Sheets.

Dry lime is planned to be added to the SAG mill feed belt. Lime would be delivered in bulk and pneumatically unloaded into a silo. The lime silo would have seven days of storage. Some quick lime would be slaked on site, and the milk of lime would be pumped to the points of addition using a closed loop system. Lime content would be 15% by weight.

The collector 3418A would be added to the conditioning tank and flotation circuits to modify the mineral particle surfaces and enhance the floatability of the copper mineral particles into the froth concentrate. The strength of the solution would be 20% by weight. PAX would be used as the sulphide collector in the pyrite circuit.

The pre-mixed bulk liquid reagents, including MIBC, A208 (if required), would not be diluted. Metering pumps would be connected to the bulk containers, and pump directly to the points of addition.

Flocculant would be prepared in the standard manner as a dilute solution of less than 0.5% solution strength for conditioning and further diluted prior to use.

17.6 Energy, Water and Process Materials

17.6.1 Energy Load

The electrical power requirement at the incoming feeder of the main substation was estimated to be as follows:

- Maximum demand 68,700kW

- Average demand 62,700kW
- Connected load 95,900kW
- Annual consumption 492,305,200kW-h/year.

17.6.2 Instrumentation and Control System

The process control system would be a PLC-based control system, as shown in Figure 17-4 and Figure 17-5.

17.6.3 Primary Crushing Area

A graphical operator workstation (GOW) in the operator control booth at the primary crusher would allow operator control of the primary crushing process.

17.6.4 Sulphide Concentrator

Four operator control booths are planned in the sulphide concentrator. A GOW in the gyratory crusher room would be small, and would also be used for operator training. A GOW in the grinding operator control booth would allow control of the grinding process; including the coarse ore reclaim system. A GOW in the flotation operator control booth would allow control of the flotation operation, including regrinding and process water pumping. A GOW in the dewatering operator control booth would allow control of the dewatering process.

17.6.5 Pit Dewatering

The pit dewatering pumping operations would be controlled from panel-mounted graphical control panels located in the pump control stations at the rims of the North Pit, South Pit and Tizate Pit.

17.6.6 Tailings Reclaim Water Pump Station and Seepage Collection Pond Pumps

A panel-mounted graphical control panel at the tailings reclaim water pump station would allow control of the tailings barge pumps, and the seepage collection pond pumps. It would also be possible to monitor remotely.

17.6.7 Oxide Plant

A Graphical Operator Workstation in the operator control booth in the oxide plant would allow operator control of oxide processing, cyanide destruction, oxide tanks, sulphide tanks, tailings pumps, and oxide surge pond dredge and reclaim pumps.

17.6.8 Carbon Plant and Refinery

Process control for the Carbon Plant and Refinery would be provided by the plant vendor. Outputs from the plant would be sent to the overall supervisory system.

Figure 17-4: Process Control System Diagram (sheet 1)

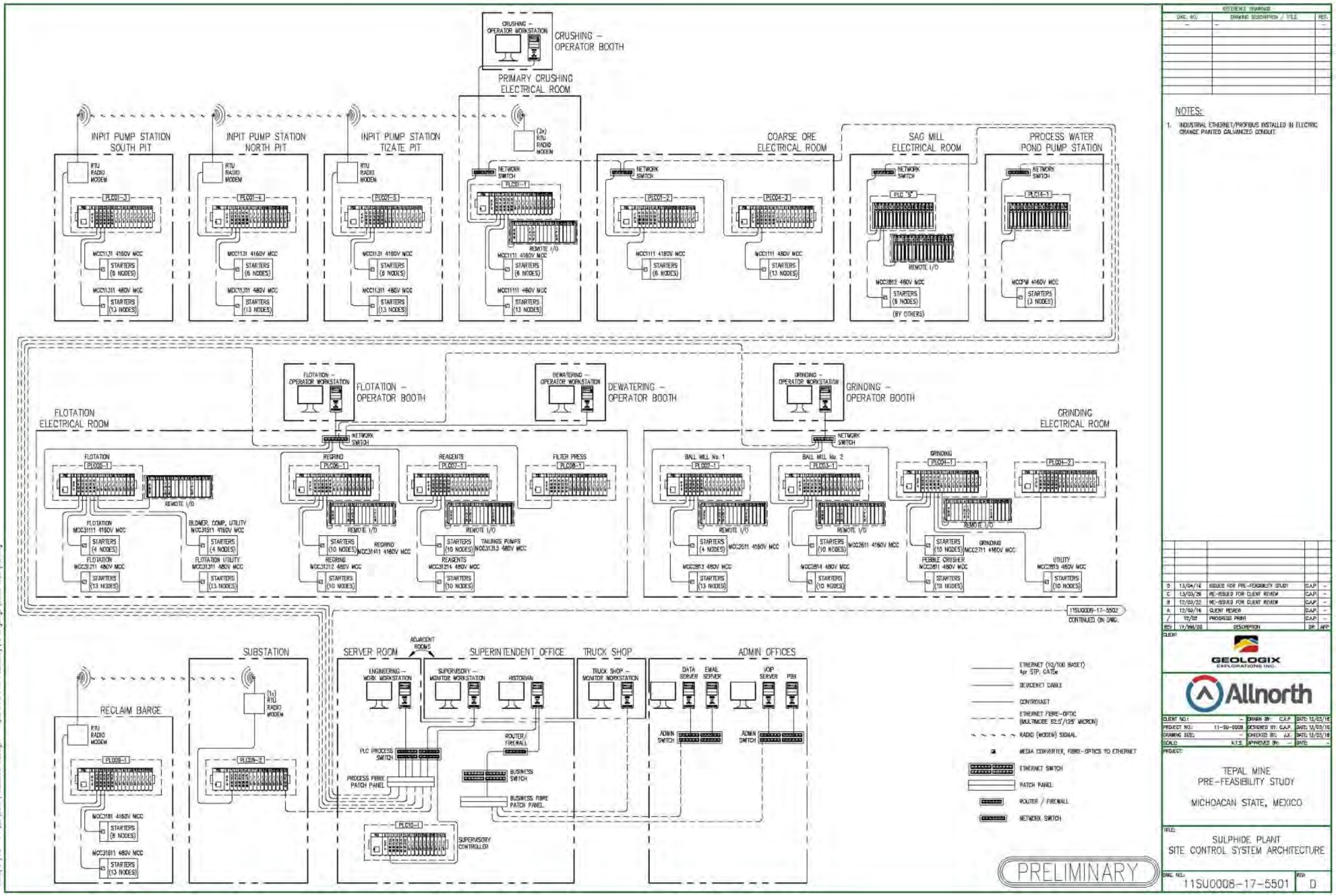
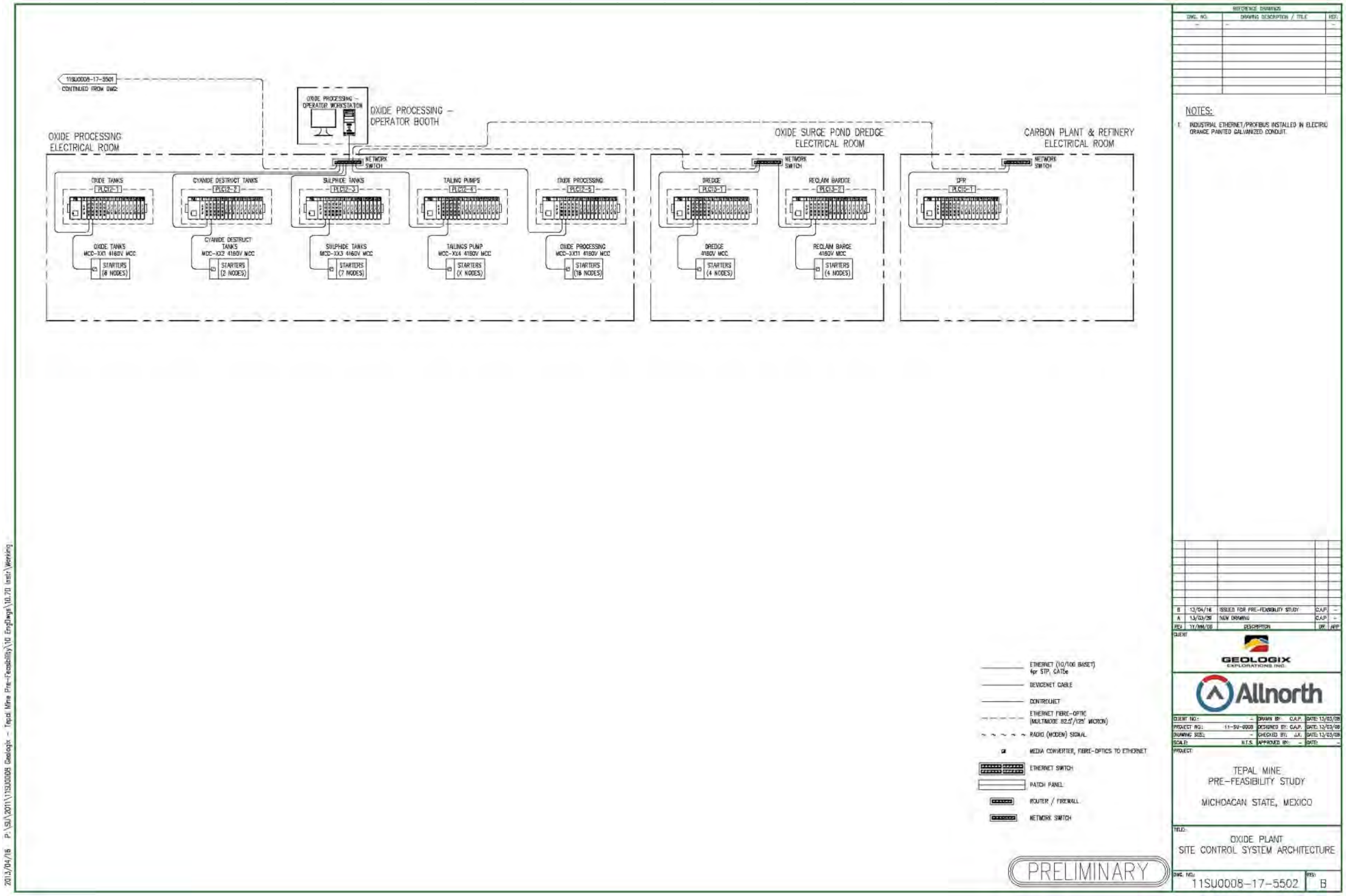


Figure 17-5: Process Control System Diagram (sheet 2)



17.6.9 Process Communication System

The process communication system would be based on an Ethernet fibre-optic network. (Refer to Figure 17-4 and Figure 17-5).This network would provide communication between the process controllers in the electrical rooms at primary crushing, coarse ore, SAG mill, grinding, flotation, oxide processing, process water pond, main substation and server room.

Radio-based (MODEM) process control communication would be provided between the controller at the primary crushing electrical room and the controllers at the in-pit pump station controller South Pit, in-pit pump station controller North Pit, and in-pit pump station controller Tizate Pit. Radio-based process control communication would also be provided between the controller at the main substation and the controller at the reclaim barge.

Cable-based Ethernet links would provide process control communication between the controllers in the electrical rooms and the primary crushing, grinding, flotation, dewatering and oxide processing operator control booths. Likewise cable-based Ethernet links would provide process control communication between the controllers in the Server room and the engineering workstation, supervisory monitor workstation, and the historian workstation.

A firewall router would connect the process communication system with the business communication system.

17.6.10 Business Communication System

The business communication system would be based on an Ethernet cable network. This network would provide communication with the truck shop monitor workstation, the data server, e-mail server, VIOP server, PBX and other business computers on the site.

17.6.11 Water Supply

The mill operation would be supplied with two separate water supply systems. The process water pond would feed all process requirements of the mill. Reclaimed water from the tailings pond would comprise most of the process water with the balance supplied as fresh water from the fresh water tank. The thickener overflow water would report to the process water pond after clarifying. The water supply is covered in detail in Section 18.3.

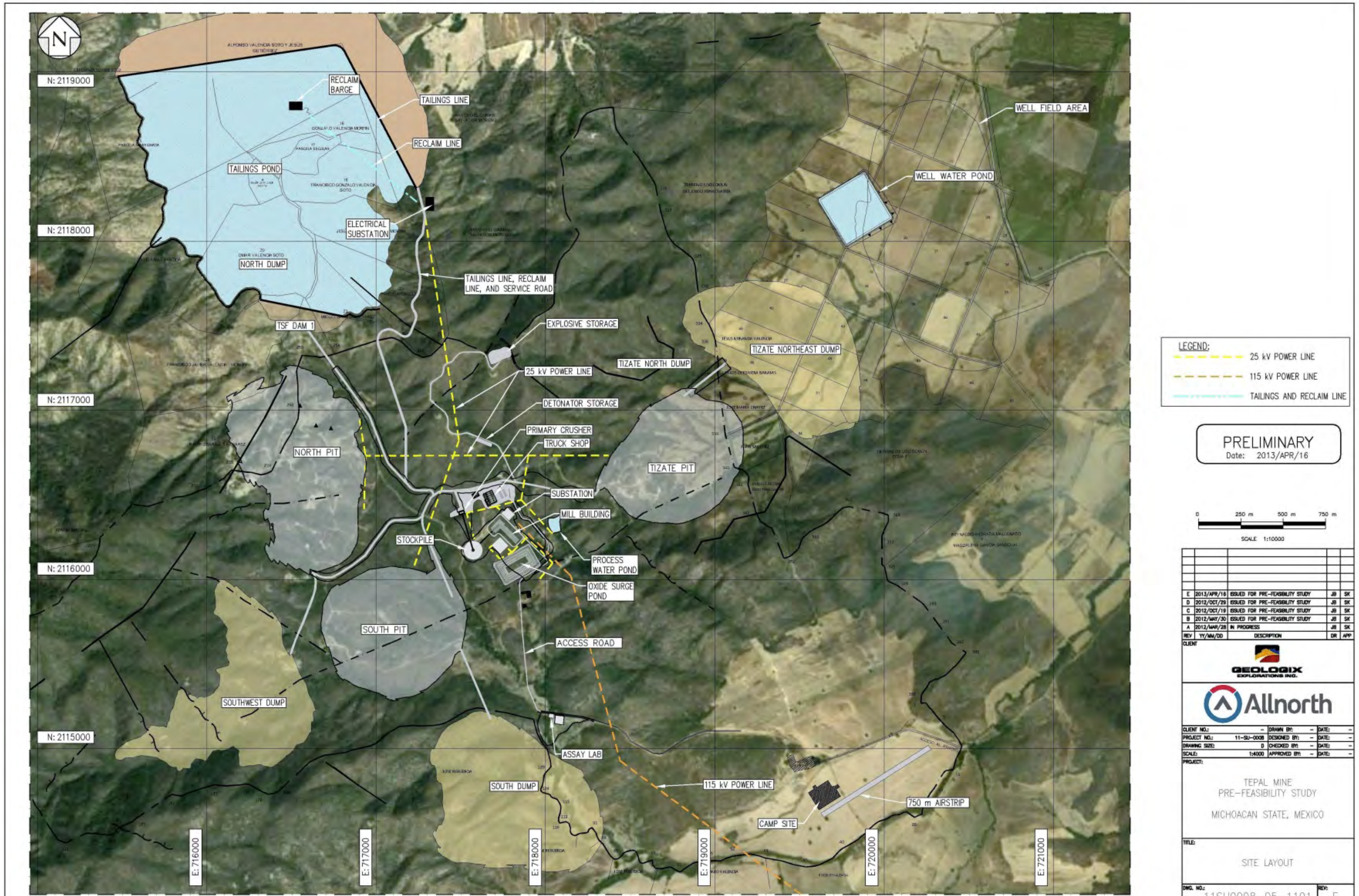
18 Project Infrastructure

The proposed services and ancillary facilities required for the project include the following:

- Plant site access road
- Haul roads
- Waste rock dumps
- TSF
- Truck shop
- Service roads
- Power supply from the Comisión Federal de Elecricado grid, transmission to site, and project site distribution
- Oxide surge pond
- Process plant
- Assay laboratory
- CIL, carbon plant and refinery facilities
- Fuel storage and dispensing
- Security, scale house, administration and first aid facilities
- Fresh water supply, fire/fresh water storage and distribution, sewage collection and treatment, drainage and runoff settling ponds, and process water pond
- Temporary housing facilities for construction personnel
- Permanent accommodation complex
- Laydown areas and parking
- 750m long airstrip.

These are shown in Figure 18-1 below.

Figure 18-1: Site Layout



18.1 Access Roads

18.1.1 Site Access Roads

The proposed site access road would be an 8km basic one-lane “farm-road” with two stream crossings and no major elevation changes. The site access road would be upgraded prior to construction of the mine to facilitate travel between the main road and the Tepal site. There would be a small crossing over the irrigation channel at the T-junction of the main road and the site access road that would have to be temporarily expanded for 40’ long trucks to cross. The site access road would be a type D road as per to Mexican standards, with a width of 10m, shoulders, fore-slopes, and storm water runoff ditches on both sides of the road. The existing road would be widened, raised with a sub-base, and resurfaced with gravel. Two “arizona type” stream crossings would be required. These crossings would be constructed of reinforced concrete within the stream channel.

18.2 Power Supply

CFE, the local Mexican power utility company, would supply electrical power to the Project. The main supply cost was origiginally estimated by CFE at 85MW but the current peak demand was estimated at 68 MW, therefore the power supply cost is being conservative.

18.2.1 Main Substation

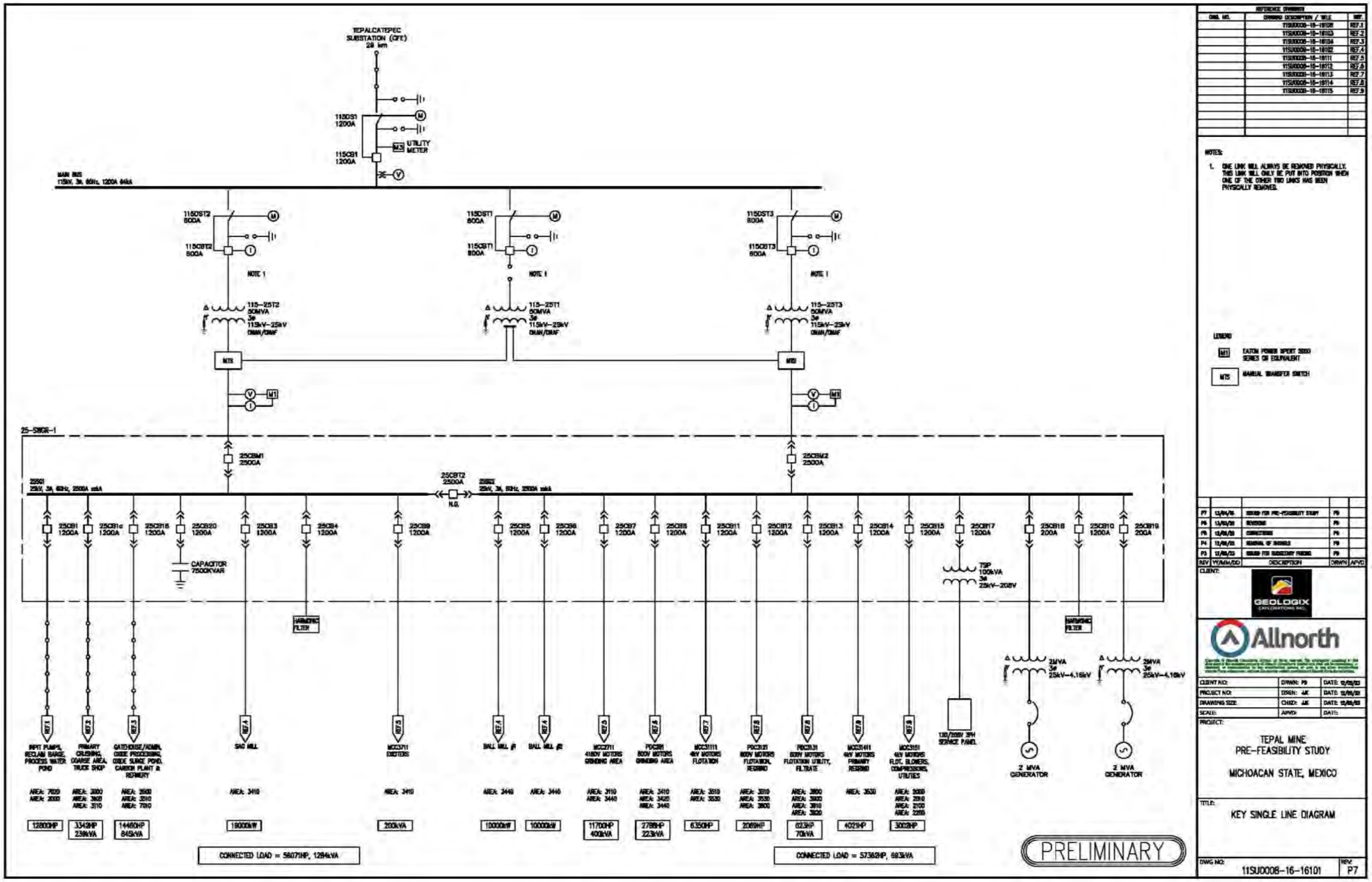
The incoming power line is planned to be a new 115kV overhead line running from the Tepalcatepec substation to the project main substation (Refer to Figure 18-1). The 115kV power transmission system feeding the Tepalcatepec substation would be upgraded.

Two main transformers are proposed to provide the 25kV power for the entire plant. A third main transformer would be mounted in place as a non-connected spare main transformer. This spare main transformer could be brought into commission to replace one duty main transformer by the removal and installation of 115kV links. This would be done to ensure minimum plant downtime during a possible outage of one of the main transformers.

18.2.2 Site Power Distribution

The site power distribution at 25kV would be as shown in the key single line diagram of Figure 18-2. Two 25 kV buses would be connected to the two main transformers. These buses would, in turn, be feeding two sections of the total plant.

Figure 18-2: Key Single Line Diagram



18.2.3 Emergency Power Distribution

The permanent emergency power distribution system is planned from the emergency power generators at the main substation, through the main 25kV buses (bus tie link closed, main transformer circuit breakers opened) to those loads which require emergency power (Refer to Figure 18-2).

The emergency power generation would be activated within 30 seconds of losing the main power supply.

All loads which require emergency power would be fed from the emergency generators through their normal feeding paths. Provision would be made for supplying a total emergency demand load of 1,600kW.

Uninterrupted UPS power to instrumentation and control systems would be supplied by distributed UPS units throughout the plant. These UPS units as critical loads would receive 60Hz power from the emergency power generators. Emergency fire pumps would be powered by local diesel engine based units.

18.2.4 On Site Power Lines

The site overhead line power distribution would be made by 25kV overhead lines running from the main substation to the truck shop, primary crusher and coarse ore stockpile; from the main substation to the North Pit, South Pit, Tizate Pit and process water pond; and from the main substation to the gatehouse and administration building, oxide plant, oxide surge pond and carbon plant and refinery (Refer to Figure 18-2).

18.3 Water Management

18.3.1 Site Water Balance

A monthly mine site water balance has been developed for each distinct phase of the mine life. The modelling was based on the estimated mean monthly hydrometeorological conditions and the input parameters summarized in Table 18-1. A description of the project phases that were modelled is presented in Table 18-2.

Table 18-1: Water Balance Input Parameters

General design Criteria	
Project Location	Michoacán State, Mexico, approximately 70km west of Apatzigan and 170km south of Guadalajara City
Site Elevation	Approximately 420 to 600masl
Mine Production	
Total Ore Milled	150Mt (11.8Mt Oxide, 137.8Mt Sulphide)
Avg LOM Throughput	38,700tpd
Life of Mine	Approximately 11.5 years
Water in Ore	8% of ore (Sulphide to mill), by weight (Micon)
Sulphide Tailings Characteristics	
Slurry Composition	30% solids by weight
Tailings Solids Specific Gravity	2.84 (Micon)
Initial Tailings Density	1.2t/m^3 (assumed)
Final Settled Density	1.4t/m^3 (achieved after 3 years, assumed)
Hydrologic Parameters	
Mean Annual Precipitation	794mm
Mean Annual Evaporation	2,551mm (estimated lake evaporation)
Beach Rewetting Loss Ratio	1.0497 x 10^{-3} l/s/tpd (Wels & Robertson, 2003)
Catchment Areas	
TSF1	2.8Mm^2
North Pit - TSF2	2.2Mm^2
South Pit	4.8Mm^2
Plant Site	1.6Mm^2
Tizate Pit	3.2Mm^2
Runoff Coefficients	
Undisturbed Catchments	0.1
Dry Beach & Open Pits	0.7
Concrete Surfaces and Ponds	1.0

Table 18-2: Mine Phase Description

Phase	Description
1 (Years -2 to -1)	• Water accumulation in Tailings Storage Facility 1 (TSF1) begins • Mill starts processing oxide ore in the latter part of Year -1
2 (Years 1 to 3)	• Mill is operating, tailings deposition to TSF1 begins • Initial tailings dry density = 1.2t/m^3 • Runoff from all catchments on site reports to TSF1 • South Pit groundwater inflows are pumped to TSF1
3 (Years 4 to 8)	• Tailings dry density increases to 1.4t/m3 • Mining in the Tizate Pit commences in Year 4 • Groundwater inflows from the North Pit, South Pit, and Tizate Pit are pumped to TSF1
4 (Years 9 to 11)	• Mining ceases in the North Pit; the pit then becomes Tailings Storage Facility 2 (TSF2) • Tailings deposition to TSF1 ceases, and tailings are deposited in TSF2 • Runoff from the decommissioned TSF1 is pumped to TSF2 • Groundwater inflow from the South Pit and Tizate Pit is pumped to TSF2

The water balance model developed for the project tracked inflows and outflows on a monthly basis from the following facilities:

- Tailings storage facilities (TSF1 and TSF2)
- Process plant (mill)
- All three open pits
- Disturbed and undisturbed catchments within the project site.

The model was used to estimate the net change to water stored on site in each month using the following primary sources of water at the site:

- Precipitation on the mine facilities and their catchment areas
- Fresh water make-up, which was assumed to be obtained from groundwater.

The major losses of water included evaporation from ponds and wetted surfaces, and water lost in the tailings voids.

A representative year was modelled for each project phase, meaning that multiple years within a phase were not linked together. This simplification to the study was warranted because the site water storage returns to zero during the dry season within each average precipitation year.

The site is predicted to be in a water deficit condition under average precipitation conditions, with the deficit increasing during dry years; therefore, make-up water would likely be required during all phases of the mine life. A summary of the results for all the four project phases under average and 1-in-20 dry conditions is shown in Table 18-3.

Table 18-3: Summary of Results

Phase	Initial Pond Volume (Mm^3)	Final Pond Volume (Mm^3)	TSF Make-up Water Required (Mm^3/yr)	Total Annual Water Deficit (Mm^3/yr)	Maximum Water Make-up Rate (l/s)
Average Year Precipitation					
1	0	1	n/a	n/a	n/a
2	0	n/a	2.7	3.6	209
3	0.5	n/a	1.0	1.9	147
4	1.7	n/a	0	1.0	34
1-in-20 Year Dry Scenario					
1	0	0.5	n/a	n/a	n/a
2	0	n/a	3.7	4.6	210
3	0	n/a	1.9	2.8	148
4	0	n/a	1.0	2.0	130

Notes:

1)The “Total annual water deficit” and “Maximum water make-up rate” values include fresh water requirements to the mill (approximately 100 m^3/h)

2)The “Maximum water make-up rate” is the maximum monthly TSF make-up water required plus the mill fresh water requirement for the given month.

The water balance indicates that the tailings storage facility would operate in a water deficit on an average annual basis. As expected, the deficit would increase under dry conditions. The average annual make-up water requirements range from 3.6Mm^3 in Phase 2 to 1Mm^3 in Phase 4. The corresponding annual make-up water requirements under 1-in-20 year dry conditions range from 4.6Mm^3 in Phase 2 to 2Mm^3 in Phase 4.

18.3.2 Seepage Collection Pond Design

Six seepage collection and water management ponds are proposed at the project site:

- Seepage collection pond 1
- Seepage collection pond 2
- Seepage collection pond 3
- Seepage collection pond 4
- Seepage collection pond 5
- Water management pond 1.

The ponds are shown on Figure 18-3.

Seepage collection ponds 1, 2 and 3 would collect surface runoff from the downstream embankment face and seepage from the embankment drains. Water collected in the seepage collection ponds would be recycled to the TSF.

Seepage collection ponds 4 & 5 downstream of the waste rock dumps would provide collection points for surface runoff and seepage from the waste rock dumps. Water from seepage pond 5 would be pumped to the well field storage pond and water from seepage pond 4 would flow by gravity to the site-wide stormwater collection pond.

Runoff flowing towards the Tizate Pit would be collected in water management pond 1. The water would be pumped directly to a channel flowing to the site-wide stormwater collection pond.

18.3.3 Site-Wide Stormwater Ponds

A site-wide stormwater pond located at the southeast of the site would collect water during the rainy season. The pond has been sized for a 1-in-10 year 24-hour storm event over the following catchment areas:

- North pit indirect catchment area
- South pit indirect catchment area
- Plant indirect catchment area
- Plant direct catchment area
- Site-wide stormwater pond direct catchment area.

The water in the pond would be pumped to the process water tank continuously during the wet season to maintain the storm storage capacity. The total storage capacity of the site-wide stormwater pond would be approximately 250,000m^3.

18.3.4 Water Well Fields and Storage pond

A well field would be needed to provide make-up water for the project. The average annual make-up water requirement, based on the site water balance model for mean climatic conditions, would range from 1Mm^3 in Phase 4 to 3.6Mm^3 in Phase 2. The well field would likely be located east of the Northeast Dump and may require approximately 10-15 wells. This estimate is based on a desktop review of the regional groundwater resource (Geologix Exploration, Environmental Base Line, Phase I & II, Tepal Project, Tepalcatepec, Michoacán). A 500,000m^3 water storage pond would be included in the design to store water during the wet season and to help buffer the demand on the well field during the dry season. The well field and well storage pond locations are shown on Figure 18-3.

The maximum well field pumping rate has been calculated based on the Phase 2 water requirement, which would be approximately 3.6Mm^3 per year. This represents the largest annual water shortfall during the project life. Pumping from the well field would likely be on a 24 hours per day, 7 days per week basis during the dry season and for 12 hours per day, 7 days per week during the wet season.

Less water would be needed for make-up during the wet season; therefore, excess water withdrawal beyond the process requirements would be stored in the water storage pond.

Additional hydrogeological studies would be required to determine the actual location of the well field(s) and the approximate number of wells and depths of completion. A suitable aquifer would need to be identified and field tested.

18.4 Waste Management

18.4.1 Waste Rock Storage

Three waste rock dump areas have been proposed at the site: the Southwest, South and Northeast Dumps. General dimensions are presented in Table 18-4.

Table 18-4: Waste Rock Storage Dimensions

Dump	Height (m)	Elevation (m)
Southwest Dump	180	700
South Dump	80	580
Northeast Dump	100	520

Overburden would consist primarily of weathered oxide material. Waste rock would generally consist of hard oxide and sulphate rock types such as tonalite, altered volcanics, and volcanics.

The overall final slope of the waste storage sites would be established at 2H:1V to facilitate reclamation.

The layout of the waste rock dumps is shown on Figure 18-3.

18.4.2 Tailings Storage

The Tepal mill is planned to operate at a nominal throughput of approximately 38,700tpd over the 11.5 year mine life generating a total of approximately 150Mt of tailings that would be stored in two separate TSFs.

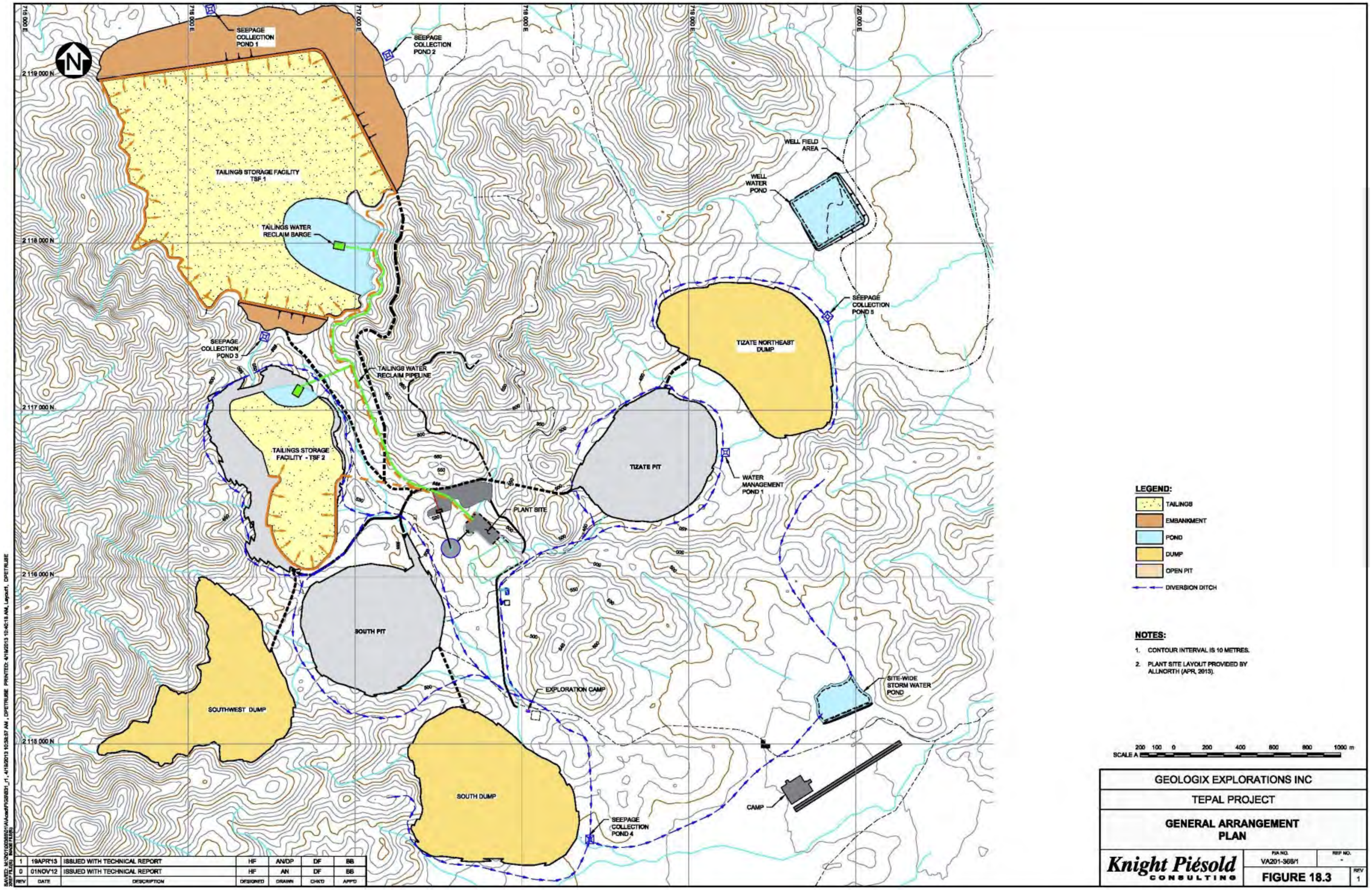
The proposed tailings storage facility 1 (TSF1) is located approximately 2km northwest of the plant site, and was designed to store a total of approximately 120Mt of tailings, process water, surface runoff, and incident precipitation. The remaining 30Mt of tailings would be deposited into the North Pit, which would become tailings storage facility 2 (TSF2) once tailings deposition begins in the pit.

18.4.2.1 Tailings Storage Facility 1 Design

The location of TSF1 was selected based on an alternatives assessment that considered economic, environmental, and operational factors.

The TSF would comprise the two embankments shown on Figure 18-4: the main embankment to the north of the impoundment and the saddle embankment at the south.

Figure 18-3: General Arrangement Layout



The starter embankment would be constructed with 3H:1V upstream and downstream slopes as shown on Figure 18-4. The embankment would be underlain with a two meter thick filter blanket over the entire downstream foundation to manage seepage through the embankment and limit pore pressure build-up in the downstream shell zone. The starter embankment would serve as a water retaining dam prior to deposition of the first tailings in Year -1.

The embankments would be developed in stages throughout the life of the project using the downstream construction method for Stage 2 and the centreline construction method for Stages 3 and 4. The initial embankment would be constructed as a water retaining structure with a vertical filter and transition zone running longitudinally along the length of the low permeability dam core. Shell zones would be constructed at 2.5H:1V upstream and 2.5H:1V downstream slopes using oxide waste rock from the open pits.

Seepage from the TSF would be intercepted by the high permeability vertical chimney drain within the embankment. Seepage would flow through the continuous filter and transition drain into a series of foundation drains at select low points in the embankment footprint. The foundation drains would generally be aligned perpendicular to the embankment centreline.

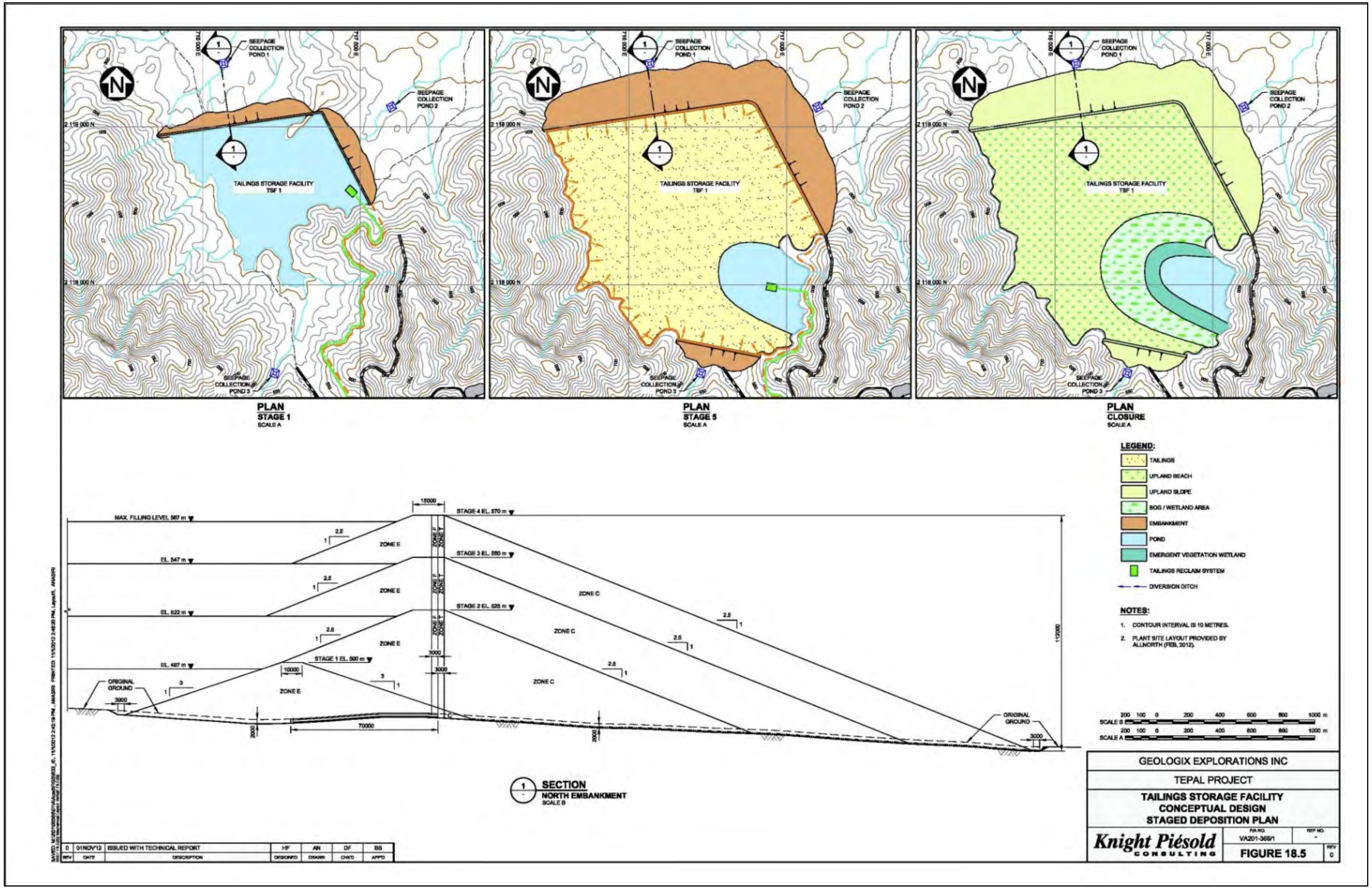
The foundation drains and outlets would be placed in excavated trenches within the embankment foundation. The drain would comprise Zone D material (clean gravel) surrounded by Zone F (filter sand). The foundation drain outlets would daylight into the seepage collection ponds constructed at topographic low points downstream of the embankment.

Water in the seepage collection ponds would be monitored and recycled to the TSF by a system of pumps and pipes.

The final TSF embankment is proposed to be approximately 112m high and 2250m long, with a total fill volume of approximately 33.5Mm^3. Oxide waste rock from the North and South Pits would be used for the construction of the TSF embankment shell zones (32.3m^3) and processed material would be used for the construction of the drainage zones (1.2Mm^3).

Construction is planned to be staged to minimize capital expenditure and defer costs where possible. The starter facility would provide adequate capacity for start-up water collection. Four additional stages (Stage 2 through 5) of construction would occur at 2-5 year intervals over the approximately 11 year mine life. Closure of the TSF would include capping of the facility with oxide waste rock and topsoil. Details of the TSF layout from construction to final closure are shown on Figure 18-4.

Figure 18-4: Tailings Storage Facility Design Layout



18.4.2.2 Tailings Discharge and Reclaim

The tailings slurry would be pumped through a 32” pipeline at approximately 30% solids (by weight) and discharged around the perimeter of the TSF. The total pumping power capacity would be 3.3MW.

The reclaim water pipeline would consist of a 22” HDPE pipeline. The reclaim system is designed to deliver the process water requirements for a nominal LOM throughput of 38,700tpd. The water would be pumped from the TSF supernatant pond to a process water tank at the mill for reuse in the process. The reclaim pumps would be mounted on a floating barge and a booster pump station would be located between the barge and the mill head tank. Each pump station would deliver approximately half of the total dynamic head. The average elevation of the tailings supernatant pond would increase steadily over the life of the project resulting in lower pumping head requirements over the life of the project.

18.5 Plant Site Facilities

The location of the mill building, truck shop and ancillary support buildings areal planned centrally on the mine site, outside the 150m blast zone radius established between each open pit. It has been determined that rock is close to the existing ground surface at this location to adequately support equipment foundations. The facilities contained on the plant site would consist of a crusher, stock pile and underground reclaim system, mill process building, truck shop and truck wash, re-fueling station, mine dry, security and scale house building, administration offices, assay lab and carbon plant & refinery facilities as shown in Figure 18-5.

18.5.1 Mill Building and Concentrate Load-out

The grinding and concentrate processing equipment is planned to be housed in a pre-engineered metal building as shown in Figure 18-6. The mill building would be covered with a metal roof to protect against rain and sun. However the perimeter of the building would remain mostly open with limited wall cladding installed around the uppermost portion of the building to act as rain skirting.

The grinding area would contain the SAG and ball mill equipment, including the associated equipment for the grinding circuits. The proposed design of the griding area is approximately 78m by 44m. It would have several platforms at various levels to support equipment, and allow access where necessary. The majority of the grinding area would be serviceable by a 30t heavy-duty overhead crane.

The flotation and concentrate area is planned to be 128m by 32m, and would be located adjacent to the grinding area. This area would be accessible by 3 overhead cranes. The flotation and regrind areas would each be serviced by 20t overhead cranes. The filter presses would have their own 10t crane. Adjacent to this part of the process building would be the concentrate thickener and the fresh water tank. It is from the far end of the concentrate area that the final concentrate would be shipped by truck to a smelter for further processing. The flotation area would also contain storage, electrical rooms, offices, and space for training purposes.

Figure 18-5: Plant Site Layout

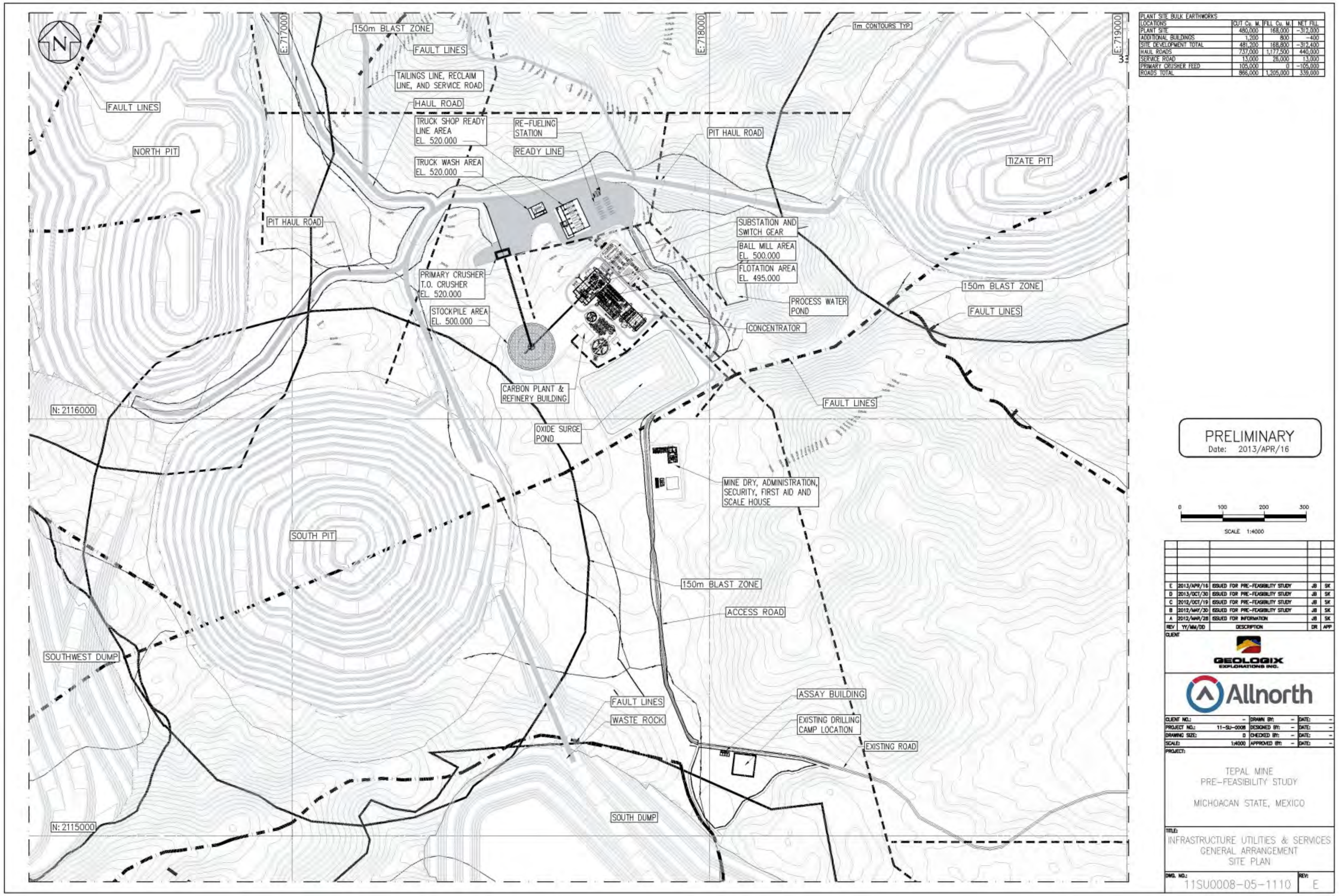
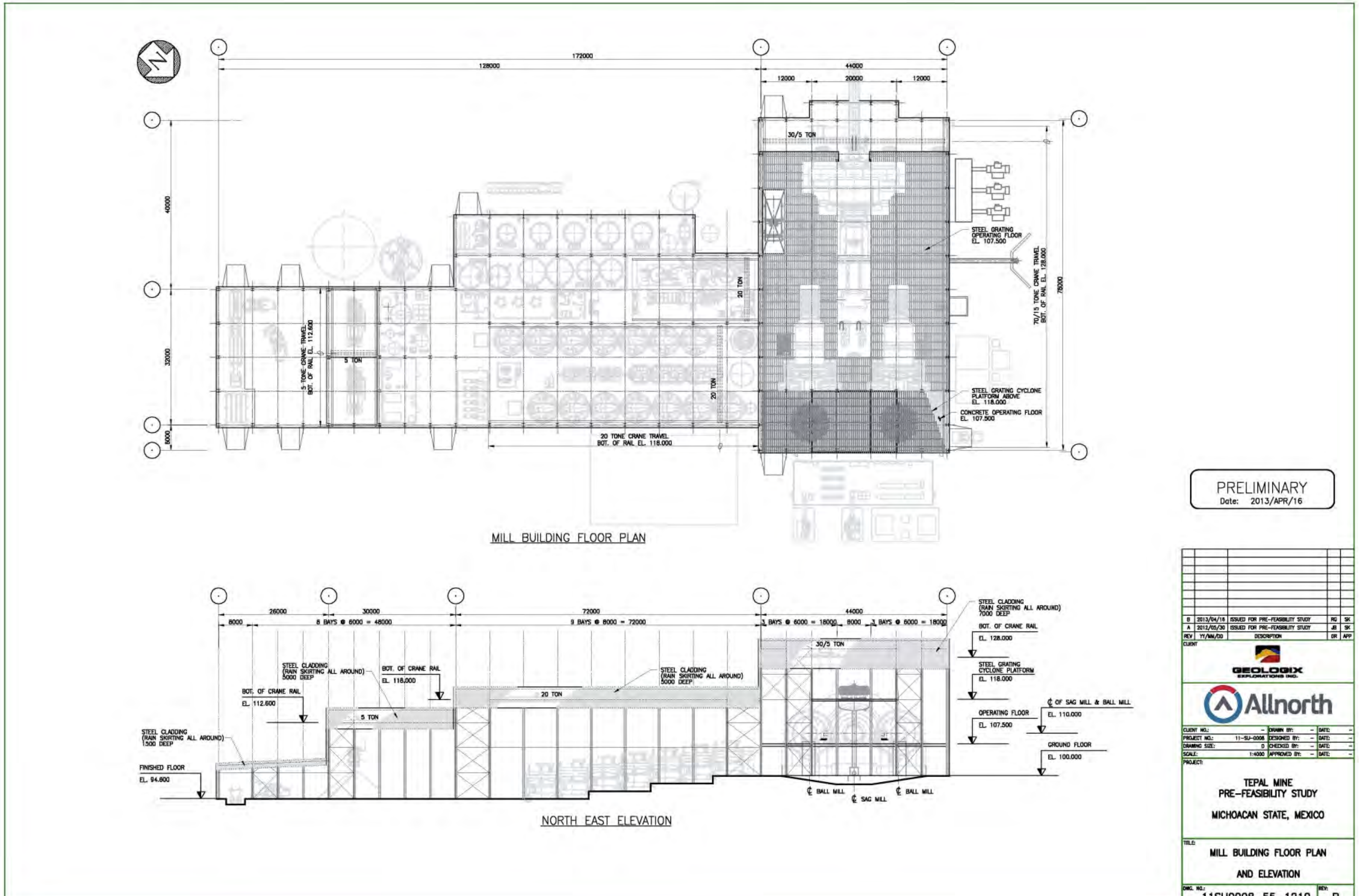


Figure 18-6: Mill Building Layout



The south side of the mill building would house the reagent mixing and storage area. This area, and all related equipment and supplies would be located under a lower roof adjacent to the main flotation area of the mill building. These two areas would have fire separation as required.

18.5.2 Leaching Area

The carbon-in-leach tanks for the gold recovery process would be located to the South of the mill building. The sulphide leach feed thickener, oxide CIL tanks, cyanide destruct tanks, PAX strip tank, and sulphide CIL tanks would all be located in a tank-farm type arrangement. The natural grade in this area is sloping down west to east. The grading and location of these tanks would use this natural slope in order to minimize structural work required to support various tank elevations.

18.5.3 Carbon Plant and Refinery Area

The carbon plant and refinery area is proposed to have a self-contained building located just south of the mill building, adjacent to the leaching area. This building, inclusive of steel structure and foundations would all be provided by the equipment vendor.

18.5.4 Oxide Surge Pond

Ground oxide is to be stored in an oxide surge pond that is planned to be located south of the mill building and the leaching areas. An access road is planned to enter from the north as would two pipe trestles.

18.5.5 Primary Crusher

The location of the crusher is planned to the north-west of the plant site. 30m wide haul roads, constructed using earth/rock fill would provide access to the crusher pad from the various pits. A mechanically stabilized earth (MSE) wall would be required at the crusher pad. See Figure 18-7 for a cross section through the crusher. The main structure of the crusher area is planned to be made up of concrete, while all access and maintenance platforms would be constructed from steel.

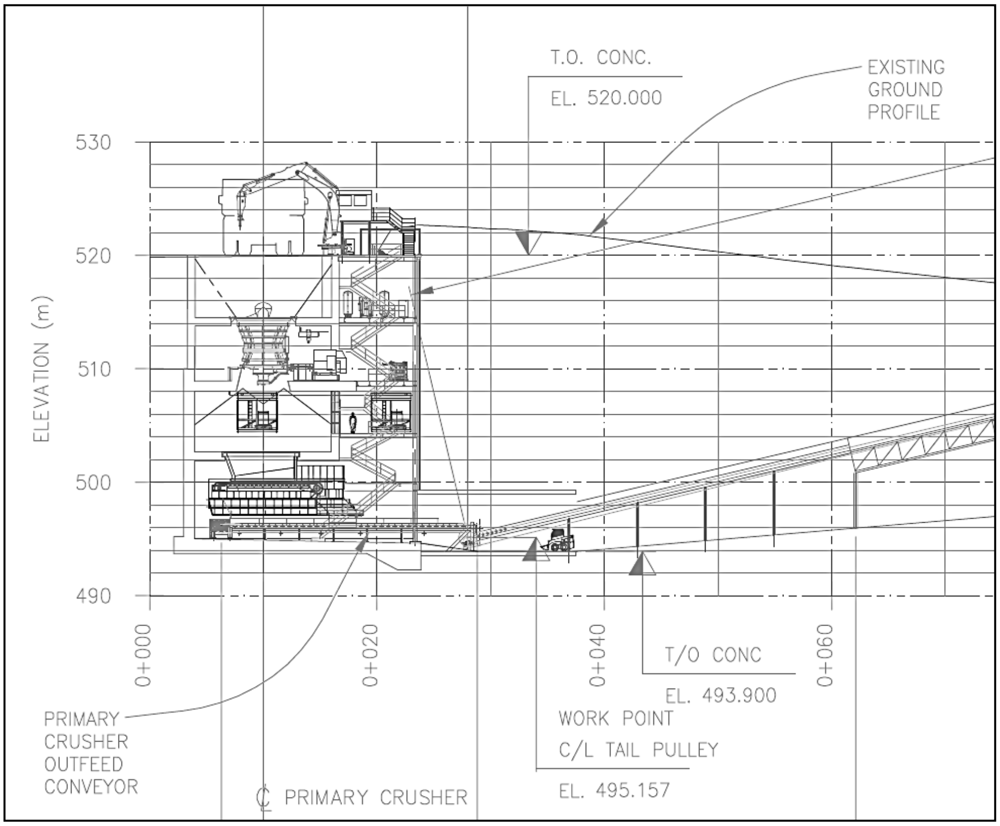
18.5.6 Assay Lab

The assay lab would be located south of the plant site. It would be in an enclosed building constructed near the existing camp facilities. This building would be equipped with the necessary analytical instruments to provide all routine assays for the geology, mining, processing, and environmental departments. The major pieces of equipment would include:

- Wet chemical lab ware fume hoods and bench surfaces
- Atomic absorption spectrophotometer (AAS)
- Fire assay equipment
- Sulphur and carbon determination furnace (Leco).

It is planned that the assay lab would undertake all necessary test work to monitor metallurgical performance and, more importantly, to improve process flow sheet unit operations and efficiencies. The laboratory would be equipped with laboratory crushers, ball and stirred mills, test sieves and shakers, flotation cell filtering and settling equipment, balances, pH and oxidation reduction potential meters, etc.

Figure 18-7: Primary Crusher



18.6 Ancillary Facilities

18.6.1 Truck Shop and Truck Wash Facility

The truck shop is proposed to be housed in a pre-engineered metal building. It would be approximately 60m by 42m, and would also contain electrical, machine and weld shops, small vehicle repair shop and space for parts storage. Unlike the mill building, the truck shop would be mostly enclosed to protect against contaminates such as dust. Mine trucks and other vehicles would be serviced using a 50t heavy duty overhead bridge crane. The welding bay would be open to the outdoors in order to minimize HVAC requirements.

Facilities for washing the trucks would be located near the truck shop. The truck wash would consist of a concrete slab and sump, with all washing equipment installed therein. No superstructure would be required.

18.6.2 Security and Scale House

The security and scale house is planned to be located near the south west corner of the mill site. This would be the location from which all persons and vehicles entering and leaving site would be monitored and controlled. The building would include an ambulance and first aid room, workplace monitoring office and security/scale house facilities. The building would be constructed with masonry walls on top of a concrete slab on grade.

Closed circuit cameras would provide feeds to a screen monitor located inside the building. The structure would be equipped with a telephone system facilitating communications both on and off site (for emergency purposes). As well, security personnel would be equipped with base station radios. The offices would be also equipped with desk top computers linked to the site computer network.

18.6.3 Detonator and Explosive Facilities

Separate detonator and explosive facilities would be located on a remote service road north of the plant site. Both facilities would be located outside of the 150m blast radius of each pit. One would house explosives inside a small building structure, with ammonium nitrate storage placed outside. The other would house detonators inside a small brick building. The structures for both of these facilities would be built on top of a concrete slab on grade with masonry walls and metal roof construction.

A chain link fence would be installed around both facilities for security purposes.

18.6.4 Fuel Storage

Diesel fuel for the mining fleet is planned to be stored and dispensed at the refueling station located east of the truck shop. The on-site facility plans to house 150,000 litres or roughly 3 days' supply of diesel fuel consumption. Two double walled 75,000 litre tanks would be installed initially changing to four when demands reach 250,000 litres per day.

A positive draining concrete apron measuring 15m x 12m would be installed at the re-fueling station and connected to a sump and oil water separator for containment of any leaked fuel.

B train tanker trucks would park at the opposite end to the storage tanks for replenishing purposes. This area would also require a sump and connected oil water separator.

Two separate 30,000 litre double walled tanks would also be required for storage and re-fuelling of highway diesel and gasoline vehicles.

18.6.5 Temporary Construction Camp

The location of the temporary camp is planned at the south west corner of the plant site near the security building. The camp would be constructed using single story pre-fabricated modular trailer units. Each unit would be joined together and supported on concrete cinder blocking and enclosed with plywood skirting to finished grade. The construction camp would be built in stages in order to accommodate the build-up of personnel from the early onset of construction activity to the estimated peak of approximately 350 workers in the camp. The balance of the construction period workforce, which will peak at around 500 workers, will be recruited from the local communities.

18.6.6 Permanent Accommodation Complex

Upon the completion of the construction period the temporary construction camp will be reduced to accommodate approximately 120 permanent workers.

18.6.7 Administration Building

A single story administration building is proposed close to the security and scale house, near the south west corner of the plant site. The building would contain offices for up to 23 management and support staff employees. Washrooms, meeting and lunchrooms would also be included. The building would be constructed using local masonry building methods over a concrete slab on grade.

18.6.8 Mine Dry

The single story mine dry would be located between the security and administration buildings. It would provide lockers and showers for 350 workers at the beginning and end of shifts. In addition, it would include separate change rooms and washrooms for 40 staff members and 30 women employees.

The building would be built using masonry wall construction placed on top of a concrete slab on grade. The roof and walls would be adequately vented to reduce condensation and additional windows would be placed around the uppermost portion of the exterior walls for indirect lighting and ventilations purposes.

18.6.9 Sewage Collection and Treatment

A sewage collection and treatment system would be located besides the administration building. It would consist of a bio disk treatment facility to filter out solid waste and a below ground weeping bed to treat waste water. Raw sewage and grey water would be first routed to the bio disk facility where solids would be filtered out and later pumped and removed from the site. The remaining filtered sewage water would then be distributed to the filter bed system for infiltration into the surrounding soil.

19 Market Studies and Contracts

19.1 Market Studies

A preliminary market study on the potential concentrate sales from the Tepal project were completed by Exen Consulting Services, an independent industry participant, who provided indicative terms and an analysis of the market conditions with respect to the copper concentrate and doré to be produced at the Tepal mine. These terms are considered to be in line with the current market conditions and have been considered in the economic analysis of this report. The indicative terms were reviewed and found to be acceptable by Matt Bender, QP.

The study revealed that none of the impurities commonly found in copper concentrates are at levels which should be of any concern to copper smelters for Tepal's copper concentrate. The projected levels of the deleterious elements are well below typical penalty threshold which means no penalties would be incurred and this would make the concentrates particularly attractive to some buyers as they can be used to offset impurities in more complex feeds. Although Tepal's copper concentrate is not considered "high grade" in the marketplace, its low level of deleterious elements would be very attractive to most smelters and, as such, would be readily saleable at terms in line with standard industry payables and benchmarks.

An increase in the volume of complex qualities in Mexico has resulted in a shortage of clean, blendable material. In certain instances, traders have had to import clean concentrates to blend down impurities to meet smelter limits – this is an expensive practice which has resulted in high domestic demand for clean material, regardless of copper grade, for existing blend operations as well as for those looking for a base clean feed.

Concentrate transportation would be conducted using trucks from the mine site to Lazaro Cardenas. Shipment and port handling costs were estimated based on Exen's recent work with other clients. The study recommends that as the project advances towards development, a more detailed marketing report and logistics study is undertaken to ensure the accuracy of the terms.

Table 19-1 outlines the smelter terms and concentrate transportation costs used in the economic analysis.

Table 19-1: Treatment, Shipping & Refining Cost Parameters used in Economic Analysis

Category	Unit	Value
Copper Concentrate – Sulphide Flotation		
Moisture Content	%	8%
Smelter Payables		
Cu Payable	%	96.5%
Min. Cu deduction	% Cu/tonne	1%
Au Payable	%	97%
Min. Au deduction	g/t concentrate	0.0
Ag Payable	%	90%

Category	Unit	Value
Min. Ag deduction	g/t concentrate	30.0
Treatment & Refining Costs		
Cu TC	$/dmt concentrate	50.00
Cu RC	$/payable lb	0.05
Au RC	$/payable oz	5.00
Ag RC	$/payable oz	0.50
Transport Costs		
Ocean freight to Japan	$/wmt	60.00
Truck freight to Port	$/wmt	36.73
Representation at Port	$/wmt	1.00
Port charges	$/wmt	10.50
Insurance	$/wmt	1.93
Losses	$/wmt	7.50
Subtotal	**$/wmt**	**117.66**
	$/dmt	**108.25**
Doré Production – Sulphide Cyanidation		
Smelter Payables		
Au Payable	%	99.9%
Min. Au deduction	g/t	0.0
Ag Payable	%	97.0%
Min. Ag deduction	g/t	0.0
Treatment & Refining Costs		
Au RC	$/payable oz	7.50
Ag RC	$/payable oz	1.40
Doré Production – Oxide Cyanidation		
Smelter Payables		
Au Payable	%	99.9%
Min. Au deduction	g/t	0.0
Ag Payable	%	97.0%
Min. Ag deduction	g/t	0.0
Treatment & Refining Costs		
Au RC	$/payable oz	7.50
Ag RC	$/payable oz	1.40

19.2 Contracts

No contractual arrangements for concentrate trucking, port usage, shipping, smelting or refining exists at this time. Furthermore, no contractual arrangements have been made for the copper concentrate or the precious metal doré at this time.

19.3 Metal Prices

The base and precious metal markets benefit from terminal markets around the world (London, New York, Tokyo and Hong Kong) and fluctuate on an almost continuous basis. Historical metal prices are shown in Figures 19-1 to 19-3 and demonstrate the change in metal prices from 1998 through February 2013.

Figure 19-1: Copper Price (USD $/lb)

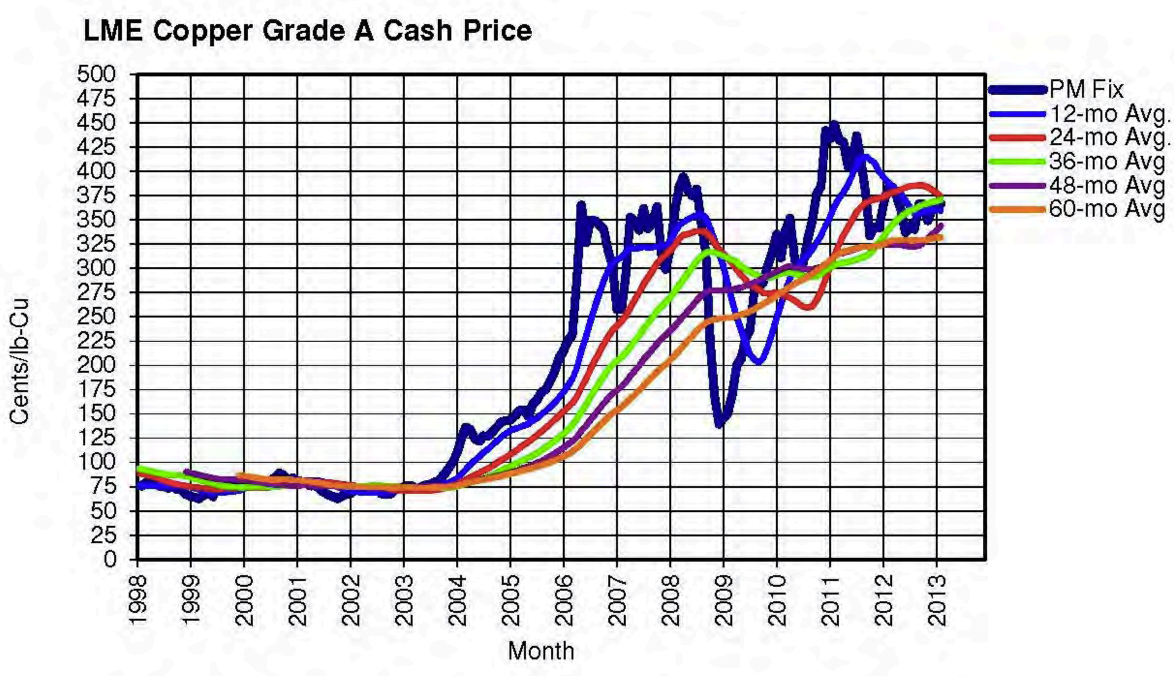
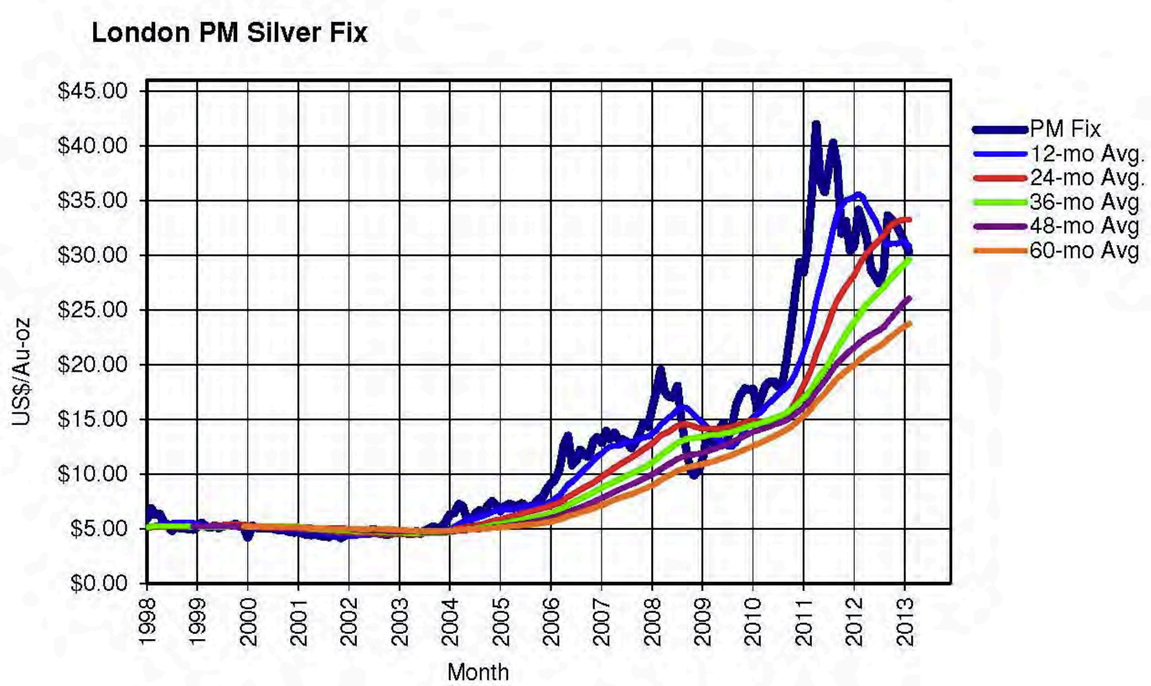


Figure 19-2: Gold Price (USD $/oz)

Figure 19-3: Silver Price (USD $/oz)



The metal prices used in the Base Case economic analysis are the four-year trailing average through February 2013. Three additional cases were calculated, using the three-year and five-year trailing average through February 2013 and the metal prices that were used for Whittle, which were 10% below spot prices as at January 31, 2013.

Table 19-2 summarizes the metal prices and exchange rates used to run various scenarios in the economic analysis.

Table 19-2: Metal Prices for Scenarios used in the Economic Analysis

Parameter	Units	Three-Year Trailing Average	PFS Base Case Four-Year Trailing Average	Five-Year Trailing Average	Whittle Parameter Pricing
Copper Price	USD $/lb	3.71	3.44	3.32	3.15
Gold Price	USD $/oz	1,518	1,390	1,286	1,400
Silver Price	USD $/oz	29.58	26.03	23.68	26.00
Exchange Rate	MEX:USD	13:1	13:1	13:1	13:1
Exchange Rate	CDN:USD	1.00	1.00	1.00	1.00

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Studies Introduction

Environmental baseline studies have been carried out for Geologix by Clifton Associates Ltd. out of Guadalajara, Jalisco, México. Baseline studies were completed in 2010 in both the rainy season and the dry season and further studies in 2011. Results from the baseline studies by Clifton Associates are summarized below.

The Tepal Project is located within the warm, sub-humid climatic zone. Annual average temperature is 22°C with annual variations ranging from 21 to 36°C from May to October and from 15 to 33°C from November to April. There are 60 to 89 days of rain from May to October during which 700 to 800mm of rain falls with the majority occurring in August and September. The dry season occurs from November to April when there is 0 to 25mm of precipitation and only 1 to 29 days of rain. Annual evaporation ranges from 600 to 700mm. Winds at the project site are predominantly from the northeast.

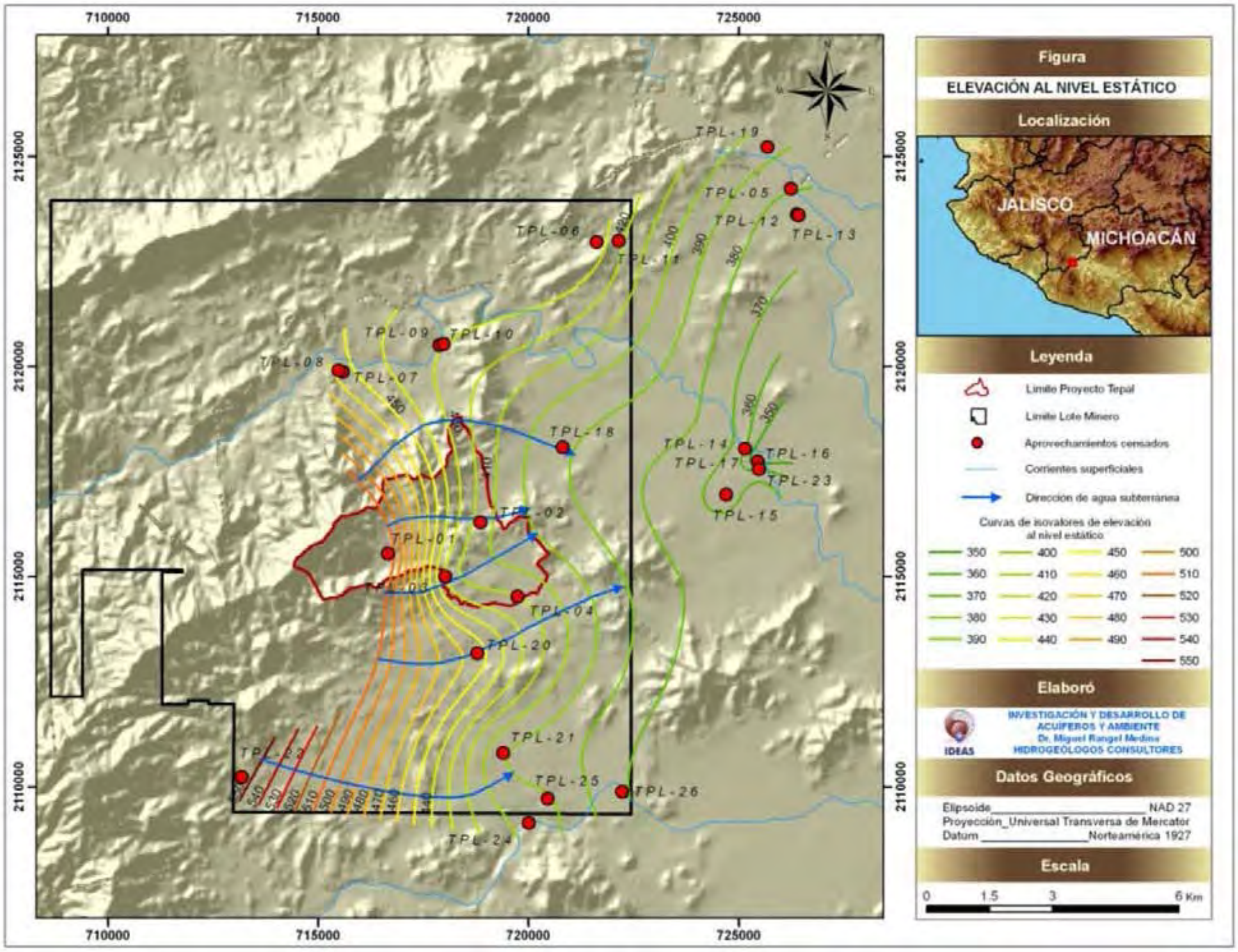
20.1.1 Aquatic Resources

The Tepal Project is located in the headwaters of the Tepalcatepec River. El Cascalote, La Laja, Los Lobos are the main ephemeral creeks from the property that lead to the main Tequiluca Creek which is a tributary to the Tepalcatepec River basin, which is 11,860km^2 and designated as Hydrological Region 18 by the National Water Commission. The Tepalcatepec River joins with other large drainages that are part of the overall Balsas River drainage of 35,046km^2 that reaches tidewater on the Pacific coast at the border of the states of Guerrero and Michoacán. The Balsas River drainage is influenced by agriculture, industry, cities, and the Infiernillo reservoir and hydroelectric dam.

Surficial water quality around the project is influenced by the mineralized rocks and by the agricultural activities in the area. Water samples were collected in May and November 2010; February, July and October 2011, and April 2012. Water quality is high in aluminum and iron, typical of weathered soils in tropical climates and has elevated levels of copper due to the local mineralization. Dissolved solids are high and there are high levels of nitrogen, phosphorous and coliforms related to the local agriculture (Clifton Associates Ltd., 2011).

Ground water is estimated to be 40-60m deep on high grounds and 6-15m deep in lower areas of unconsolidated materials. According the National Water Commission, the project lies within the Apatzingán aquifer No. 1620, and groundwater generally flows east from the project site (Figure 20-1).

Figure 20-1: Ground Water Static Levels and Flow Direction in the Tepal Project Area (IDEAS, 2011)



20.1.2 Terrestrial Resources

The project is in the tropical sub-deciduous, deciduous forest zone and consists of forest, agricultural and ranch lands. Within the forested zones, trees are generally not spiny and range from 4 to 10m in height with densities of 2104 to 3308 individuals per hectare. The shrub layer ranges in height from 3 to 6m and is dense in areas where there are fewer trees. Drier areas have some columnar and candelabra-form cacti. The most common species in this zone include *Bursera ariensis, B. diversifolia, B. hintonii, Ceiba aesculifolia, Conzattia multiflora, Ficus cotinifolia, F. goldmanii, F. kellermanni, F. petiolaris, Heliocarpus reticulatus* and *Agave pedunculifera*. There are two threatened plant species in the area under NOM-059-SEMARNAT-2010, *Cephalocereus senilis* (local name El Viejito) and *Tabebuia chrysantha* (local name Amapa).

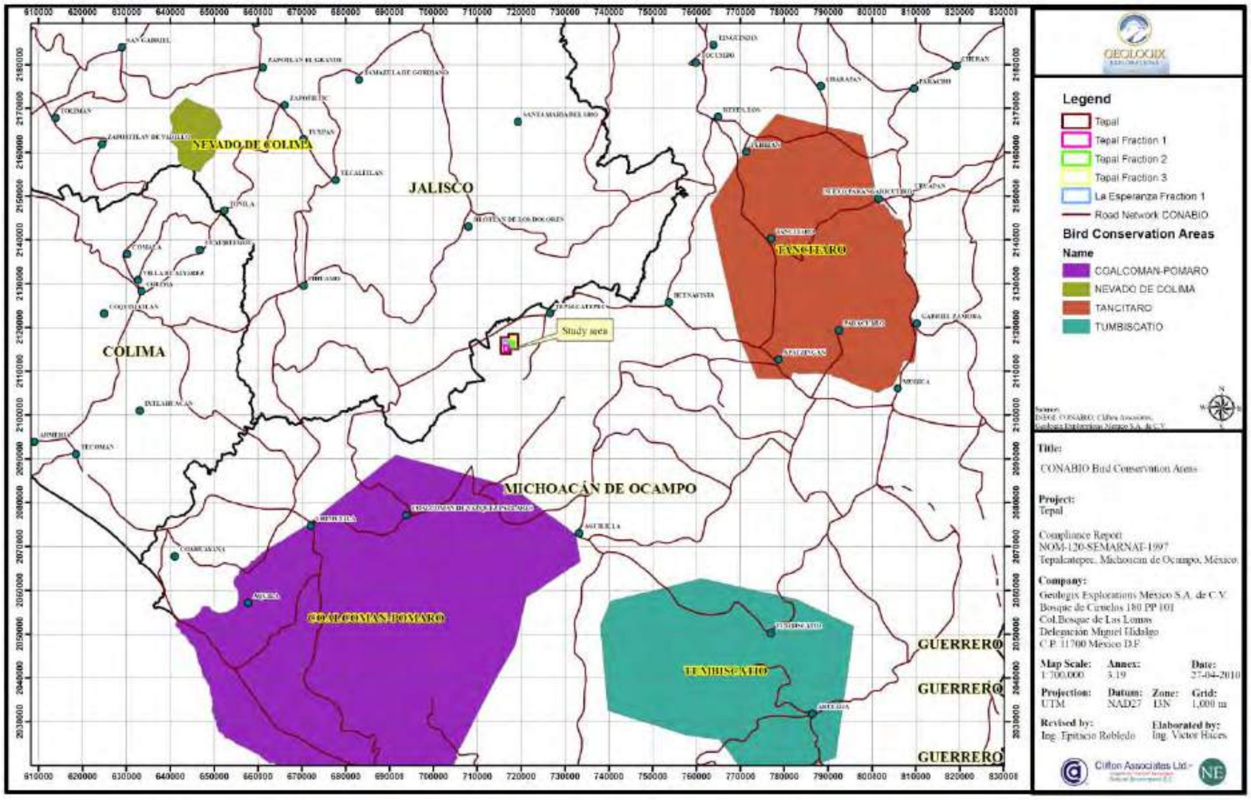
In the ranch lands to the northeast and southeast of the concessions, vegetation is dominated by spiny and xerophilic woody forest species.

Animals include various amphibians, reptiles, mammals and birds. Mammals in the area include skunk (*Mephistis macroura* and *Conepatus mesoleucus*), racoon (*Procyon lotor*), ringtail (*Bassariscus astutus*), rabbit, armadillo (*Dasypus novemcinctus*), bobcat (*Lynx rufus*), grey fox (*Urocyon cinereoargenteus*), opossum (*Didelphis virginiana* and *Tlacuatzin canescens*), squirrel (*Spermophilus annulatus*), mouse (*Peromyscus melanosis, Peromyscus levipes* and *Liomys pictus*), jaguarundi (*Herpailurus yagouaroundi* and *Felis yagouaroundi tolteca*), coyote (*Canis latrans*), weasel (*Mustela frenata*), coati (*Nasua narica*), bats (*Micronycteris, Choeronycteris mexicana, Glossophaga leachii, Glossophaga morenoi, Glossophaga soricina, Leptonycteris curasoae, Artibeus jamaicensis, Desmodus rotundus, Nyctinomops macrotis*), collared peccary (*Tayassu tajacus*), white-tailed deer (*Odocoileus virginianus*).

Under NOM-059-SEMARNAT-2010, the jaguarondi (*Herpailurus yagouaroundi*) and two species of bat (*Choeronycteris mexicana* and *Leotoycteris curasoae*) are threatened. There are five protected reptiles in the project area under NOM-059-SEMARNAT-2010 including Mexican spiny-tailed iguana (*Ctenosaura pectinata*), chameleon (*Phrynosoma asio*), Mexican pine snake (*Pituophis deppei*), rattlesnake (*Crotalus durissus*), and river turtle (*Kinosternon hirtipes*).

Birds in the area include red headed duck, ring-necked duck, blue-winged teal, loud pheasant, the mourning dove and pigeon, chicken, and American widgeon. Other bird species in the region include *Zenaida asiática, Zenaida macroura, Columbina passerina, Leptotila verreauxi, Aratinga canicularis, Bolborhynchus lineola, Calocitta Formosa, Aphelocoma coeruslescens, Corvus corax, Myadestes obscurus, Mimus polyglottos, Toxostoma curvirostre, Setophaga rutinilla, Cyanerpes cyaneus, Piranga rubra, Cardinalis cardinales, Pheucticus melanocephalus, Guiraca caerulea, Passerina amoena, Passerina cyanea, Passerina versicolor, Passerina ciris, Spiza americana, Sporophila torqueola, Chondestes grammacus, Tiaris olivácea, Amphispiza bilineata, Quiscalus mexicanus, Icterus parisorum, Casicus melanicterus*, and *Carpodacus mexicanus*. There is one threatened bird species (Barred Parakeet, *Bolborhynchus lineola*) and two specially protected bird species (Orange-fronted Conure, *Aratinga canicularis* and Red-tailed Hawk, *Buteo jamaicansis*) in the area listed under NOM-059-SEMARNAT-2010.

Figure 20-2: Important Bird Areas near the Project



The project is not in a protected area. The closest conservation areas are Important Bird Areas (IBAs), the Coalcomán-Pómaro (MX025) is located approximately 32km south, southwest of the project from the highlands and canyons to the coast; and Tancítaro (MX005) located approximately 50 km northeast of the project and is also part of a Flora and Fauna Protected Area established in 2009 (Figure 20.2; Vidal et al, 2009; www.conanp.gob.mx).

20.2 Waste and Water Management

20.2.1 Waste Characterization

Waste Characterization studies were carried out by pHase Geochemistry Inc., Vancouver, British Columbia. The results of their work are summarized below.

A geochemical characterization has been carried out to access the potential for acid rock drainage and metal leaching (ARD/ML) from waste rock and tailings associated with the Tepal Project. The program consisted of characterization of drill core representing in-pit waste as well as tailings products from metallurgical testing. Standard static test methods were used.

Mineralization on the property is characteristic of a porphyry copper-gold deposit, consisting of structurally controlled zones of stockwork and disseminated copper sulphides with elevated gold values. Almost all mineralization is hosted within three small tonalite intrusives surrounded by volcanics. Primary sulphide mineralization consists of chalcopyrite and pyrite with minor pyrrhotite, bornite, sphalerite, molybdenite and galena. Minerals associated with the overlying oxide zone include malachite and chalcocite with minor azurite, tenorite and chrysocolla.

20.2.2 Waste Rock Static Testwork

The waste rock static test program on drill core was represented by 300 samples with 100 samples collected from each of the three deposits. Sample selection considered the various rock types intersected in the core as well as an appropriate range of sulphur and copper contents, alterations and mineral zones. Acid-base accounting (ABA) and ICP-metals testing were completed.

Results indicated 67% of samples tested from Tepal North were classified as potentially acid generating (PAG) compared to roughly 40% of samples from each of the Tepal South and Tizate sample sets. This does not infer the same proportion of total waste would be classified as PAG. A relatively small proportion of samples representing each of the three deposits classified as uncertain (<15%) with the remaining samples (25% at Tepal North to ~45-50% at Tepal South and Tizate) classified as non potentially acid generating (NAG).

With respect to rock type, a large proportion of tonalite (73% of samples tested) at Tepal North classified as PAG compared to Tepal South (58% of samples) and Tizate (48% of samples). For all three deposits, >75% of late dyke and overburden samples typically classify as NAG. The altered volcanic samples at Tepal North consistently classified as PAG, whereas the unaltered volcanics at Tepal South predominantly classified as NAG.

In relation to the in-situ oxidation state, the majority (>50%) of oxide samples at Tepal South and Tizate classified as NAG; whereas the majority of oxide samples at Tepal North classified as PAG. Low neutralization potential to acid potential ratios (NP/AP) on which classifications are based may be somewhat conservative for the oxide samples as values for both sulphide (and resulting AP) and NP are low, as is typical in highly weathered material. However, weakly acidic pH values for a number of these samples in Tepal North and Tizate support their potential to generate acid.

A preliminary evaluation of sulphur cut-offs for classification of PAG from NAG rock for each main rock type was completed in an effort to assess the volumetrics of PAG versus NAG rock at Tepal. Evaluation of laboratory test work and spatial analysis of sample locations indicated that geological rock types and elemental analysis could be used to estimate PAG/NAG volumes and locations.

To classify PAG and NAG rock for materials handling during mining, a separate folder was created in the SURPAC resource block model. Geological limits were respected for oxide/sulphide boundaries and non-mineralized volcanics defined by fault contacts. Statistical analysis of test work indicated that a range of sulphur contents (from 0.25% to 1%) were suitable to define NAG material depending on the rock type and oxidation. These sets of criteria were incorporated into the block model to create preliminary spatial volume estimation for mine planning and materials movement. Mine scheduling utilized to strategically place waste rock in locations which would facilitate a closure plan. Cut-offs used would require verification with on-going testwork, but provide a preliminary basis for this assessment.

The testwork to assess the metal leaching potential is currently underway with preliminary indications that there may be some metals of potential concern. The potential for these metals to become mobilized and leach would be further examined in the on-going test program via leach extraction tests and planned kinetic testwork to follow. It is expected that greater metal leaching potential would likely exist in rocks from the hypogene or sulphide zone of the deposits rather than the already leached oxide zones, as well as from the narrow transition between these zones.

20.2.3 Tailings Testwork

The static testing completed to date on metallurgical tailings has been conducted on 10 samples of bulk rougher tailings produced from variability testing completed by G&T Metallurgical. The tailings are representative of the Tepal North (3 samples), Tepal South (3 samples) and Tizate (4 samples) deposits. Testwork completed to date includes quantitative X-Ray Diffraction analyses (QXRD), ABA, ICP-metals and net acid generation tests with metals analysis of leachates.

The mineralogical composition of the tailings included quartz, plagioclase and muscovite/illite with accessory chlinochlore, calcite (1-10%) and pyrite (1-5%), +/- K-feldspar, dolomite, ankerite, siderite and gypsum. Substantial variability in both sulphur (acid potential) and neutralization potential resulted in a range of classifications. Based on ABA results, four of the ten samples classified as PAG, another four classified as uncertain and two classified as NAG. Those from the Tizate deposit mainly classified as uncertain, and those from the Tepal North and Tepal South deposits were predominantly classified as PAG. Net acid generation tests, which add a strong oxidant to the sample in the form of hydrogen peroxide and measure the response, corroborate the ABA results for all but two samples. In these two, the test would suggest NAG behaviour while the ABA test provided classifications of uncertain and PAG. Preliminary results also indicate that there may be some potential metal leaching. As a result, it is recommended that tailings impoundment design and management should assume that the some of the tailings would have potential for acid generation and metal leaching.

Additional testwork is recommended to help further define the potential for acid generation and metal leaching from waste and tailings and refine segregation and mining sequencing strategies. Waste rock testwork should include synthetic precipitation leaching, meteoric water mobility leaching, and humidity cell tests with samples chosen based on current results. Tailings testwork should include leaching tests and humidity cell tests on samples from future metallurgical testing (pHase Geochemistry Inc., 2012).

20.2.4 Waste Management

PAG waste rock would be segregated and strategically disposed of in waste rock dumps. PAG waste rock dumps would be designed so that drainage from the dumps with higher potential to carry contaminants flows towards the pits and infiltration of water through the dumps is minimized with engineered caps during ongoing reclamation and after closure. Seepage or runoffs from the dumps would need to be monitored during operations, closure and post-closure and managed and mitigated as required.

Tailings disposal should be scheduled so that material with lower acid generation and metal leaching potential is placed adjacent to the dam and is used to cap the tailings where possible.

20.2.5 Water Management

Clean waterwould be kept separate from water that comes in contact with tailings, pit walls, waste rock and/or ore in order to minimize the amount of water that needs to be managed.

During construction, diversions, check dams, silt fences and hay bales are recommended to be used to minimize erosion and suspended solids in water leaving the site.

During operations, surface and seepage water from the pit, waste rock dumps and tailings impoundment would be collected and used in the process plant. Additional make-up water may be needed and would be obtained from groundwater wells. Any surplus water would be stored in the tailings impoundment for use in the process during the dry season. If necessary, evaporation may be enhanced with sprayers within the impoundment to prevent the need for a discharge.

At final closure, any PAG waste rock dumps and the tailings impoundment would be capped and revegetated to minimize infiltration and prevent acid generation and leaching over the long-term. Seepage would be collected, analysed and recycled back or treated if necessary until seepage water quality meets standards for direct release.

20.3 Social and Environmental Management

A number of documents have been completed that provide background for a management system and plans including the environmental baseline and internal stakeholder maps and consultation plans. The Environmental Impact Assessment which in this case includes the Risk Assessment and Change of Land Use studies would also be part of a management system.

There are a number of management plans that are specifically important for this project. The waste management plan is important due to the potentially acid generating potential of some of the waste rock; the water management plan is important due to the proximity of the project to the surrounding communities and agricultural areas; dust suppression would be important given the silty soils and dry conditions at site; the public consultation and disclosure plan and security plans are important due to the proximity of the project to communities and the potentially volatile nature of illicit activities and perception of mining by active NGOs in Mexico; and, the hazardous materials management plan is important due to the proposed cyanide gold leach processing.

It is recommended that the project consider adopting the International Voluntary Principles of Security and Human Rights and the International Cyanide Management Code. In addition, it is recommended that a Security Risk Assessment be completed during the project feasibility stage so that appropriate costs can be included in the financial analysis, security plans can be developed, and so that future financiers' requirements would be satisfied.

20.4 Permitting Requirements

The main environmental legislation in México is the General Law of Ecological Equilibrium and Environmental Protection (LGEEPA) that governs environmental impact assessments, environmental management, protection of natural resources (air, water, flora and fauna), and enforcement thereof. Other applicable environmental legislation includes the General Law for Sustainable Forestry Development, the General Law for the Prevention and Integral Management of Waste (LGPGIR), and the National Water Law. In addition, there are a Mexican Official Standards set by SEMARNAT that would apply to the project during construction and operation with respect to air emissions, discharges, biodiversity, noise, mine wastes, tailings, hazardous wastes, soils, health and safety, etc.

The project exploration activities at the Tepal project site are regulated by a standardized set of environmental protection measures specified under NOM-120-SEMARNAT-2011 for exploration projects in agricultural zones, livestock, or uncultivated lands and in zones with dry and temperate climates in which grow vegetation of arid tropical scrub and tropical deciduous forest, forests of conifers or oaks. These environmental protection measures have been implemented and are reported to government annually.

In order to go forward with project exploitation, the project requires a local area Environmental Impact Assessment (MIA-P) to be completed. A Change of Land Use authorization is also needed before the project can be constructed for which the application is submitted at the same time as the MIA-P.

Once the Environmental Impact Assessment is submitted for review, the government publishes an announcement to allow for public review of the proposed project. If the government receives requests, they will conduct formal public hearings. The government also requests that the company publish announcements in the local papers to provide an opportunity for public comment.

Once the project design is complete, preparation of the Environmental Impact Assessment and permit applications including the Change of land Use application are estimated to take approximately three months. Government review, comment and approval are estimated to be completed in the following three to six months; however, it should be noted that permitting can sometimes be delayed with requests for information or for political reasons.

Following the main project approval and receipt of the Change of Land Use authorization, there are a number of permits that need to be acquired from various ministries for various activities on site. Key permits include approval from the National Water Commission for construction of the tailings dam in creek basins that are considered federal zones, approval from the National Water Commission for water discharges (if any), and approval from the Secretary of National Defense for explosives storage and use.

These permits all assume that Geologix has acquired the necessary surface titles, rights and agreements for the project lands.

20.5 Social and Community Aspects

There are five communities located near the project including La Estanzuela (population ~30), La Ciénega (population ~50), Nuevo Corongoros, Colomotitán, and the larger community of Tepalcatepec (population ~22,152). The Tepalcatepec area, which includes the communities mentioned above, has two preschools, seven primary schools, three secondary schools, and one preparatory school. In the past, a technical institute was being considered to help with technical training for mines in the area. In the past, Geologix has had difficulty in finding skilled workers locally for exploration. It is recommended that the company support initiatives to set up a technical institute locally to help build capacity of the local workforce.

Health facilities in Tepalcatepec include a family medical unit, Instituto Mexicano del Seguro Social (MSS) and a medical centre, Instituto de Seguridad Social al Servicio de Trabajadores del Estado (ISSSTE). The medical facilities in Tepalcatepec are limited for the expected number of construction workers and may not be able to treat expatriate workers. The project would need to include an on-site medical clinic, paramedics, doctors, ambulance and medical emergency evacuation plan.

Labour collective agreements would be developed and agreed following Federal Labour Laws. It is recommended that a strategy and plan be developed in conjunction with labour relations experts and legal counsel prior to construction for engaging workers, contractors and unions for conformance with Federal Labour Laws and international standards if financing is sought.

Cultural and heritage resource studies were completed by the technical specialist of INAH, the National Institute for Anthropology and History, in November, 2011. No pre-hispanic artifacts were found; however, one area of significant interest was identified as “La Hacienda Vieja,” near the old house located near the proposed South Pit. Geologix received a clearance letter that allows for project activities without further authorization with the exception of these two areas. INAH has catalogued and archived these two sites and given clearance for development in these areas. The ninth term in the INAH authorization is that if an archaeological artifact is found by workers, work must be suspended and INAH must be contacted immediately to determine the required actions.

20.6 Mine Closure Requirements

It is recommended that local communities be consulted prior to implementing closure and reclamation plans.

Mine closure and reclamation would include removal of the process plants, powerline and ancillary facilities. Pits would be closed out by constructing a perimeter berm and installing cautionary signs next to steep pit walls. The waste rock and tailings areas would be capped where necessary to minimize water infiltration on PAG material and to prepare the site for revegetation. Disturbed areas would be revegetated with native species. Site roads that would not be required by the surrounding communities would be barred to prevent access, scarified, graded where needed, and revegetated.

Although a payment is made to government to compensate for land disturbance, the payment is not returned to the proponent for reclamation purposes. For this PFS, it is assumed that reclamation costs would be borne partially during operations with concurrent reclamation of the dumps with the remainder at the end of the mine life. If the project decides to seek international debt financing, the majority of reclamation costs would be required to be set aside in the form of a security during the construction phase to meet international financing requirements. It is assumed that waste management plan would be designed to avoid water treatment after closure to the extent possible.

21 Capital & Operating Costs

21.1 Capital Cost Estimates

21.1.1 Capital Costs Summary

The capital cost estimate was prepared using first principles, applying project experience and avoiding the use of general industry factors. The estimate is derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in the study. Given that assumptions have been made due to a lack of available engineering information, the accuracy of the estimate and/or ultimate construction costs arising from the engineering work cannot be guaranteed. The target accuracy of the estimate is ±25%.

Costs are expressed in US dollars with no escalation unless stated otherwise. Foreign exchange rates of CDN$1.00:US$1.00 and MX13.00:US$1.00 are used where applicable.

The estimate is based on the assumption that contractors would mobilize only once to carry out their work and are not already mobilized on site performing other work.

Total life of mine capital costs are estimated to be $397M. Pre-production capital costs amount to $354M. Capital costs during production years total $44M.These costs are summarized in Table 1-7. The capital costs do not include mining fleet as it is accounted for in operating costs through leasing. Contingency for the project totals $39M. Individual contingency rates were applied to each of the capital cost categories, with most rates being 15-20%. Some of the capital costs did not have any contingency applied as direct quotes were obtained from suppliers. This resulted in a blended contingency rate of 8.7%. A listing of the supplier quotes received for the project can be found on Table 21-8.

Table 21-1: Capital Cost Summary ($M)

Category	Pre-Production	Production	Total Capital Costs	% of Total
Capitalized Pre-Stripping	21.5	0.0	21.5	5.4
Support Equipment	3.3	1.1	4.4	1.1
Tailings	34.7	42.0	76.7	19.3
Process Plant	229.6	0.0	229.6	57.8
Indirects	20.1	0.0	20.1	5.0
Owner's Costs	13.4	0.0	13.4	3.4
Salvage Value	0.0	-34.4	-34.4	-8.6
Closure	0.0	27.3	27.3	6.9
Contingency (8.7%)	31.3	7.6	38.9	9.8
Total Capital Costs	**353.8**	**43.6**	**397.4**	**100.0**

21.1.2 Costs not included

The following costs have not been in included in the capital cost estimate:

- Taxes – detailed in Section 22
- Working Capital – detailed in Section 22.

21.1.3 Direct Costs

21.1.3.1 Construction Labour

Labour costs include offloading, handling, installation, testing, and commissioning of equipment and materials, carried out on the basis of a scheduled "5 x 10 hour" work week.

The labour costs are based on carrying out the construction work on a number of firm priced contracts.

The local Mexican labour rate of MXP200 per hour is based on 2010 and 2011 actual Mexican contractor rates for skilled workers working 5 days x10 hours per day work-week plus allowance for premium time.

21.1.3.2 Contractor Purchases, Materials

Material costs are Delivery Duty Paid Mill-site unless noted otherwise.

21.1.3.3 Civil/Structural

Structural costs are allowances based on Allnorth's engineering experience and in-house cost data applied to general building sizes. Civil costs are allowances based on volumes established with minimal engineering.

21.1.3.4 Equipment

Mechanical equipment costs are based on major equipment shown on flow diagrams and any other lists, notes, etc. Supplier cost information used in the estimate is identified with the equipment description. Quotes were received from reputable vendors for all major equipment.

21.1.3.5 Piping

Piping costs are based on flow sketches and marked-up layouts, from which long pipe runs were estimated and applied to unit pricing. Other piping is based on Allnorth's in-house cost data.

Pump costs are estimated from information gathered in the flow sheets and equipment list for sizing and power requirements.

21.1.3.6 Electrical

Electrical costs are based on an estimated number of motors and total connected horsepower.

Non-process items, such as lighting, communications, etc. are allowances.

21.1.3.7 Building Mechanical Services

Building mechanical costs are allowances based on historical data.

21.1.3.8 Taxes and Duties

Duties and customs/brokerage are included in equipment cost, where applicable. Wherever not quoted as part of the delivery costs of US and/or Canadian equipment, duties and customs/brokerage are included in the Indirect Costs which Allnorth is not calculating as part of this study.

Value added taxes such as HST and GST are excluded. Mexican IVA taxes are excluded.

21.1.3.9 Premium Time Allowance

An allowance for premium costs for unscheduled overtime is included in the composite labour rate. Premium time rates in Mexico are paid for time worked beyond 50 hrs/wk @ time-and-one-half and for time worked on statutory holidays at double-time.

21.1.3.10 Suppliers Supervision and Commissioning Assistance

Suppliers' services, wherever included in their quotations, include supervision of equipment installation by the contractor, training services and manuals, as well as support for commissioning of the equipment and systems.

21.1.4 Mining Capital Costs

The mining capital estimate includes mobile production and support equipment, non-mobile equipment and capitalized stripping. Quotes for production equipment were received from several manufacturers and include estimates for shipping, assembly, commissioning, fire suppression, tires, first-fills, etc. Non-equipment includes cap magazine, pumps, engineering office equipment (Global positioning system (GPS), computers, etc.), voice-radios, etc. All costs incurred during mining sulphide and oxide ore in Years -2 and -1, prior to mill operation are capitalized. Mining capital is summarized in Table 21-2. The production equipment listed in this table is assumed to be leased and is considered in the operating costs of the project for the purpose of the economic analysis.

Table 21-2: Mining Capital Cost Breakdown

Category	Total Cost ($M)	% of Total
Production Equipment (Leased)	88.3	77.3
Capitalized Pre-Stripping	21.5	18.9
Support Equipment (incl. Non-Equipment)	4.4	3.8
Total Mining Capital Costs	**114.2**	**100.0**

21.1.5 Tailings Storage Facility and Water Management

A cost estimate was developed for the Tailings Storage Facility (TSF) and site wide water management systems that includes the capital expenditures (initial and sustaining) and closure costs over the life of the project.

The major cost items in the estimate include:

- Construction of temporary and permanent access roads
- Construction of surface runoff diversion channels
- Construction of seepage collection and sediment control ponds
- Construction of water management ponds
- Fresh water supply
- Construction of the TSF
- Installation of pump and pipeline systems for TSF reclaim water, tailings distribution, seepage pump back and open pit dewatering.

Delivery of the following materials from the mine to the TSF is included in the mining cost estimate, and subsequently excluded from the waste and water management estimate:

- Zone E (overburden and highly weathered bedrock)
- Zone C (rockfill)
- Zones F and T (Filter and Transition – to be processed on-site by a contractor).

The estimate is separated into two phases:

- Initial capital costs (Pre-Production, Years -2 and -1)
- Sustaining capital costs (Production, Years 1-11).

The initial capital cost includes construction activities prior to the start of mill operation, while sustaining capital includes costs incurred during production.

The estimate was developed by identifying a scope of work and the activities required to achieve that scope. Estimate quantities were derived from the waste and water management PFS design figures and drawings where sufficient detail existed.

Quotes from the Mexican earthworks contractor Ingeniera de Cuidades South America (ICSA) were used where available for the major earthworks activities associated with the TSF, water management systems, and roads. Unit rates were developed from first principles by estimating the size and production rate of an appropriate equipment fleet for items where quotes were not available. Assumptions about the location of the various construction material sources such as borrow pits were incorporated into the earthworks estimates. Quantities reported in the estimate are based on neat-line material take-offs developed from the design drawings with allowances for material lost due to overbuilding. Losses associated with materials processing have also been included where applicable.

Annual maintenance and replacement costs for pumps and pipework were estimated as a percentage of the total capital cost for major components.

A summary of the capital cost estimate is presented in Table 21-3.

Table 21-3: Tailings Capital Cost Breakdown

Costs	Years	$M
Initial Capital	Year -2 &-1	34.2
Capitalized Operating Costs during Pre-Production	Year -2 & -1	0.5
Sustaining Capital	Years 1 through 11	42.0
Total Cost		76.7

21.1.6 Processing & Infrastructure

The total capital cost estimate for the process plant is $230M. These costs occur in the pre-production period Years -3 to -1. A summary of these costs is presented in Table 21-4.

Table 21-4: Processing & Infrastructure Capital Cost Breakdown

Category	Total Cost ($M)	% of Total
Site Development, Runway, Roads	5.0	2.2
Common Services	3.4	1.5
Oxide Leach	24.5	10.7
Crushing	17.5	7.6
Grinding & Classification	79.3	34.6
Flotation	39.2	17.1
Mill Building & Common Services	8.5	3.7
Concentrate	4.0	1.7
Buildings	4.2	1.8
Power Line, Electrical, Instrumentation	43.8	19.1
Total Process Plant Capital Costs	**229.6**	**100.0**

21.1.7 Indirect Costs

Indirect capital costs total $20M and occur during the pre-production period. Table 21-5 provides a breakdown of the indirect costs included in the total capital costs.

Table 21-5: Indirect Capital Cost Breakdown

Category	Total Cost ($M)	% of Total
Project Engineering/Design	9.5	47.4
Freight	4.0	20.0
Commodity/Minor Spares	3.0	15.0
Suppliers' Installation Services	1.3	6.4
Suppliers' Training/Commissioning Services	1.3	6.4
Permits	1.0	5.0
Total Indirect Capital Costs	**20.1**	**100.0**

21.1.8 Owner's Costs

Owner's costs for the life of the project total $13M. Table 21-6 provides a breakdown of the owner's costs.

Table 21-6: Owner's Costs Breakdown

Category	Total Cost ($M)	% of Total
Camp	3.5	26.2
Operating Spares	1.8	13.8
First Fills & Commissioning Spares	1.2	9.2
Capitalized G&A Labor & Support (Pre-Production)	6.8	50.8
Total Owner's Costs	**13.4**	**100.0**

21.1.9 Contingency

Contingency for the project total $39M. A blended contingency was applied to estimate the total contingency cost. The contingencies were calculated individually by the parties estimating each capital cost category. The breakdown of the contingency calculation by category of capital cost is demonstrated in Table 21-7.

Table 21-7: Breakdown of contingency

Category	Contingency Value ($M)	Contingency %
Mine Equipment	0.0	0.0
Tailings	15.3	20.0
Process Plant	16.9	7.4
Indirects	3.4	17.0
Owner's Costs	1.1	8.1
Closure	2.2	8.1
Total Contingency	**38.9**	**8.7**

21.1.10 Closure Cost

Closure cost for the project is estimated to be $27M. Of this cost, $25M accounted for the closure and reclamation of the TSF. An additional $2M was allocated for the closure and demo of mill facility foundations. Closure costs are set to occur in Year 12, one year after the end of production. Salvage value is accounted for in 2027 amounting to $34M. This amounts to 10% of the mine equipment and process plant capital costs.

21.1.11 Vendor and Supplier Quotations

All major mining and milling equipment have a fixed price quotation. Contractor quotations derived from design drawings have been received for the majority of construction costs and tailings facility construction. A listing of the supplier quotes received for the project can be found on Table 21-8.

Table 21-8: Vendor and Supplier Quotations

Type of Equipment	Qoutation	Status
Mill Processing Equipment	**Manufacturer Quotation**	
Primary Crusher Gyratory	Metso	F.O B. Vendor Location*
Primary Crusher O/F Conveyor	Transcontinental Engineered Products	F.O B. Vendor Location*
Coarse Ore Stockpile Conveyor	Transcontinental Engineered Products	F.O B. Vendor Location*
SAG Mill Feed Conveyor	Transcontinental Engineered Products	F.O B. Vendor Location*
SAG Mill	FLSmidth	F.O B. Vendor Location*
SAG Mill Motor	FLSmidth	F.O B. Vendor Location*
Ball Mills c/w motors	Metso	F.O B. Vendor Location*
Ball Mill Cyclone Cluster	FLSmidth	F.O B. Vendor Location*
Copper Rougher & Scavenger Flotation Cells Bank	FLSmidth	F.O B. Vendor Location*
First Cleaner Flotation Bank	FLSmidth	F.O B. Vendor Location*
Second & Third Cleaner Flotation Bank	FLSmidth	F.O B. Vendor Location*
Regrind Mill	Xstrata Technologies	F.O B. Vendor Location*
Regrind Mill Accessories	Xstrata Technologies	F.O B. Vendor Location*
Filter Press	Pure World Diemme	F.O B. Vendor Location*
Concentrate Thickener	FLSmidth	F.O B. Vendor Location*
ADR Plant	FLSmidth	F.O B. Vendor Location*
ADR Plant Building	FLSmidth	F.O B. Vendor Location*
Overflow Clarifier Thickener	FLSmidth	F.O B. Vendor Location*
Lime Hydration & Feed System	Industrial Kiln & Dryer Group	F.O B. Vendor Location*

** Delivery of equipment is included in PFS economics at 3-5% of capital cost*

Type of Equipment	Qoutation	Status
Mining Equipment	**Manufacturer Quotation**	
Mining Fleet Complete	Caterpillar (Tracsa) Mexico	Delivered on Site, Assembled
Mining Fleet Lease Rates	Caterpillar (Tracsa) Mexico	Delivered on Site, Assembled
Tires	Kal Tire Grimaldi (Mexico)	Delivered on Site
Ancillary Equipment	JDS	FOB Various Locations
Infrastructure (constructed or delivered on site)	**Supplier/Contractor Quotation**	
Steel Buildings	Corey (Mexico)	Constructed
Airstrip	CYAM	Constructed
Concrete	Codessa	Delivered on Site
Rebar	Acceros Murrilo	Delivered on Site
Earthworks	ICSA	Constructed
HDPE Liners	Technoplasticos	Constructed
HDPE Pipe	Wolsely	Constructed
Roads & Bridges	ICSA	Constructed
Construction Camp	CYAM	Constructed
Pumps, Fittings & Pipe	Xylem - Delivered	Delivered on Site
Permanent Camp	CYAM	Constructed
Powerline	CFE & DPA	Constructed
Powerline Right of Way	DPA	Constructed
Property Power Distribution	DPA	Constructed
Supply Water Wells	Affesa	Constructed
Consumables	**Supplier/Contractor Quotation**	
Mill Balls	Molycop	Delivered on Site
Flotation Reagents	Cytec de México,Grupo Celanese, Disosa	Delivered on Site
Cyanide	El Sauzal, Timmins, Argonaut	Delivered on Site
Lime	Grupo Calhira	Delivered on Site
Fuel	Pemex	Delivered on Site
Caustic Soda	Dupont	Delivered on Site
Explosives	Dyno, Orica	Delivered on Site
Detonators	Dyno, Orica	Delivered on Site
Lubricants	Mobil (Mexico)	Delivered on Site

21.2 Operating Cost Estimates

21.2.1 Operating Cost Summary

The operating cost estimate was prepared using first principles, applying project experience and avoiding the use of general industry factors. Inputs are derived from engineers, contractors and suppliers who have provided similar services to other projects. In addition, input was provided by Geologix personnel, based on their valuable experience working in Mexico.

Operating costs in this section of the report include mining, processing, tailings, and administration up to the production of concentrate from the site. Mine operating costs incurred during the construction phase (pre-production Years -2 and -1) are capitalized and form part of the capital cost estimate. Concentrate transportation, treatment and refining charges, and royalties are discussed in Section 22 .

Operating costs are presented in 2013 US dollars on a calendar year basis. No escalation or inflation is included. Average annual operating costs over the life of mine are $163M and are summarized in Table 21-9. Figures 21-1 and 21-2 show the breakdown and distribution of the life of mine operating costs by category.

Table 21-9: Average Annual Operating Costs

Category	$M	%
Mining	54.9	33.6
Processing - Sulphide Flotation	76.3	46.8
Processing - Sulphide Cyanidation	10.8	6.6
Milling & Processing - Oxide Cyanidation	6.0	3.7
G&A	7.3	4.5
Tailings	0.5	0.3
Mine Equipment Leasing	7.4	4.5
Total Average Annual Operating Costs	**163.2**	**100**

Figure 21-1: Breakdown of Operating Costs by Year

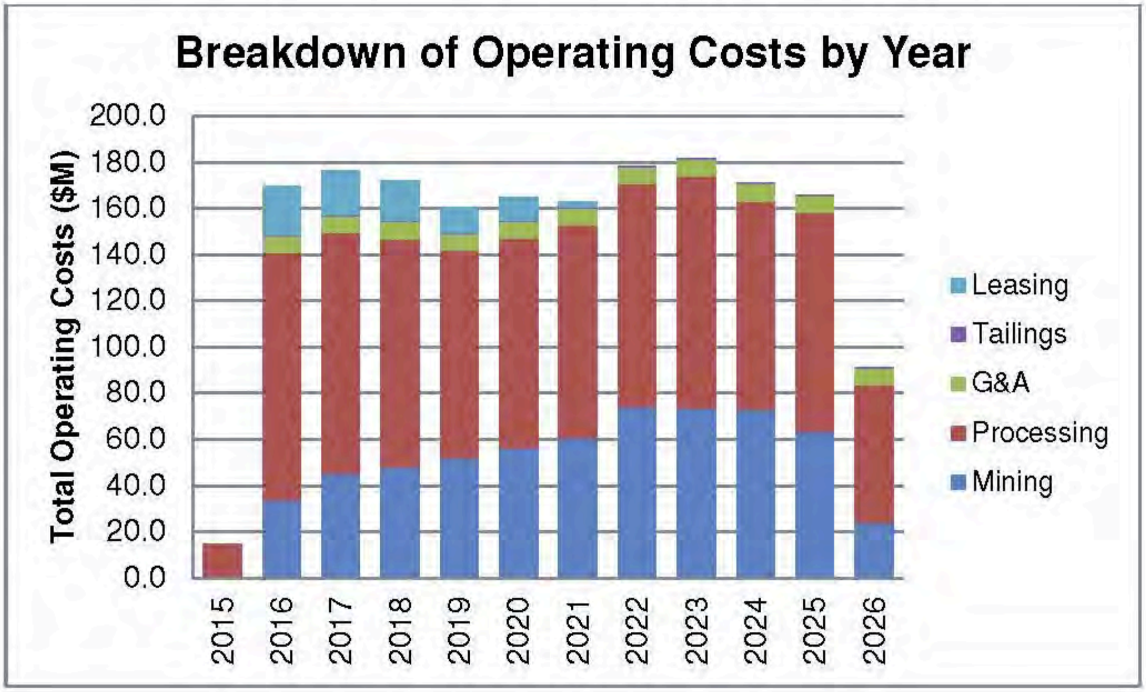
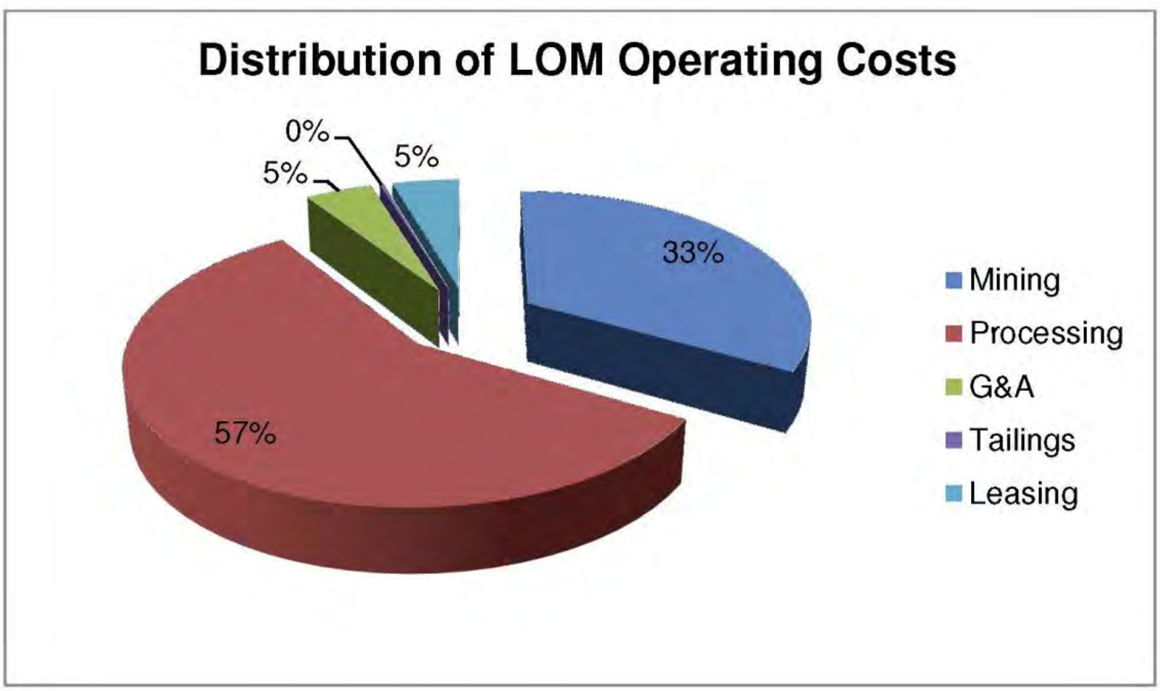


Figure 21-2: Distribution of LOM Operating Costs



21.2.2 Labour

Labour is a significant portion of annual operating cost. Labour rates include base wage and allowances for overtime, insurance, tax, benefits, and bonuses.

Labour costs assume that operating personnel would work 12 hour shifts on a two week on, one week off schedule. Supervisory, technical and administration personnel are assumed to work on a ten day on, four day off schedule.

Site labour is a fixed cost and independent of mining rates; therefore, the mining workforce levels are sized to meet peak year requirements and are not decreased during years of lower waste stripping.

Employee organization, number of personnel and total expenditure are recorded in Table 21-10.

Table 21-10: Planned Workforce

Department	Average number of personnel during production	Average Annual Cost during Production ($M)	%
Mining	145	2.9	42.5
Processing	110	1.8	26.5
G&A	42	2.1	31.0
Total	**297**	**6.7**	**100.0**

21.2.3 Mining Costs

Mining cost totaled an average of $1.50/tonne mined. Total LOM mining operating costs total $604M. Pre-stripping is not included in this cost and is capitalized (see Section 21.1.3). Figures 21-3 through 21-5 show the breakdown and distribution of the mining operating cost by category.

Figure 21-3: Breakdown of Mining Operating Cost by Activity

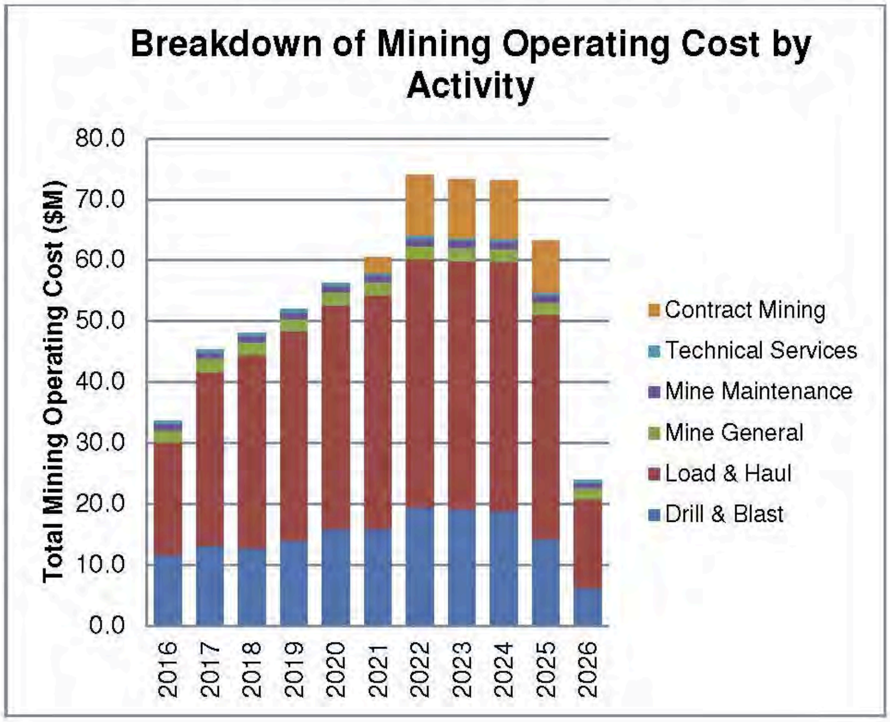


Figure 21-4: Distribution of Mining Operating Costs by Activity

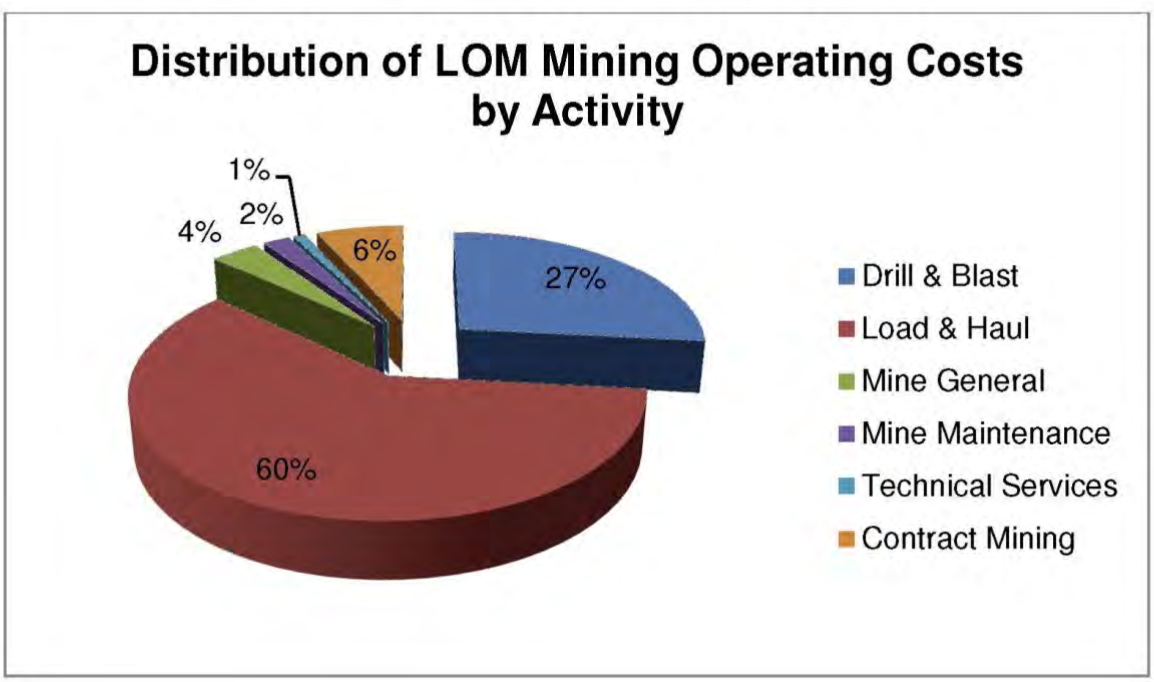
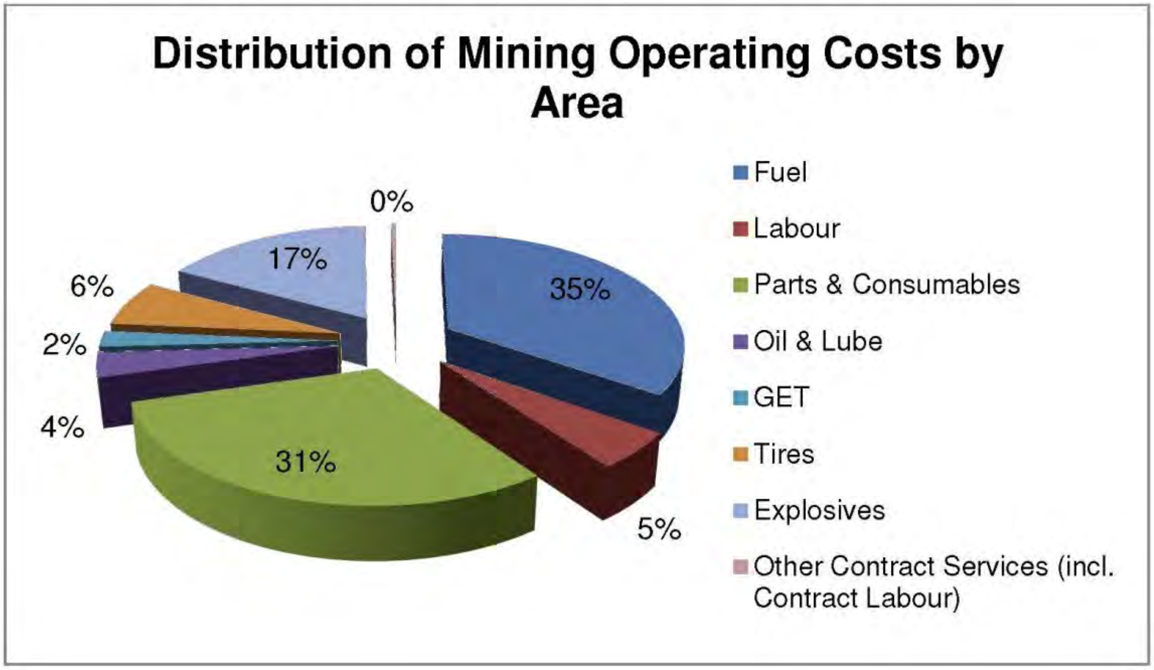


Figure 21-5: Distribution of Mining Operating Costs by Area



21.2.4 Processing

The processing operating costs are broken down into two categories:

- Processing costs for both sulphide and oxide material (Years 1-6)
- Processing costs for sulphide material only (Years 7-11).

Table 21-11 breaks out the operating costs that were used in the economic analysis. Figures 21-6 and 21-7 show the distribution of average annual operating costs for the two categories listed above.

Table 21-11: Processing Operating Costs

Category	$/tonne	Average Annual Cost ($M)	
		Years 1-6	Years 7-11
Power	3.1	45.7	39.5
Major Consumables	1.77	26.2	22.7
Reagents	1.78	26.3	21
Labour	0.13	1.8	1.8
Maintenance & Miscellaneous	0.26	3.8	3.3

Figure 21-6: Average Annual Processing Costs for Years 1-6

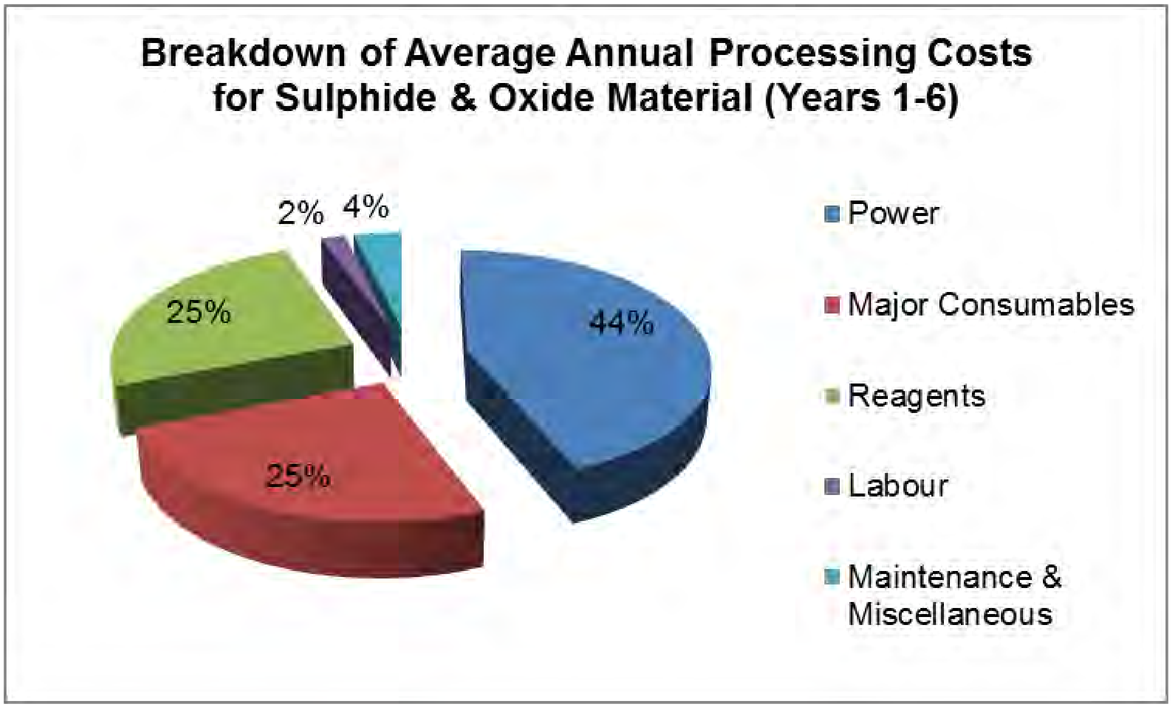
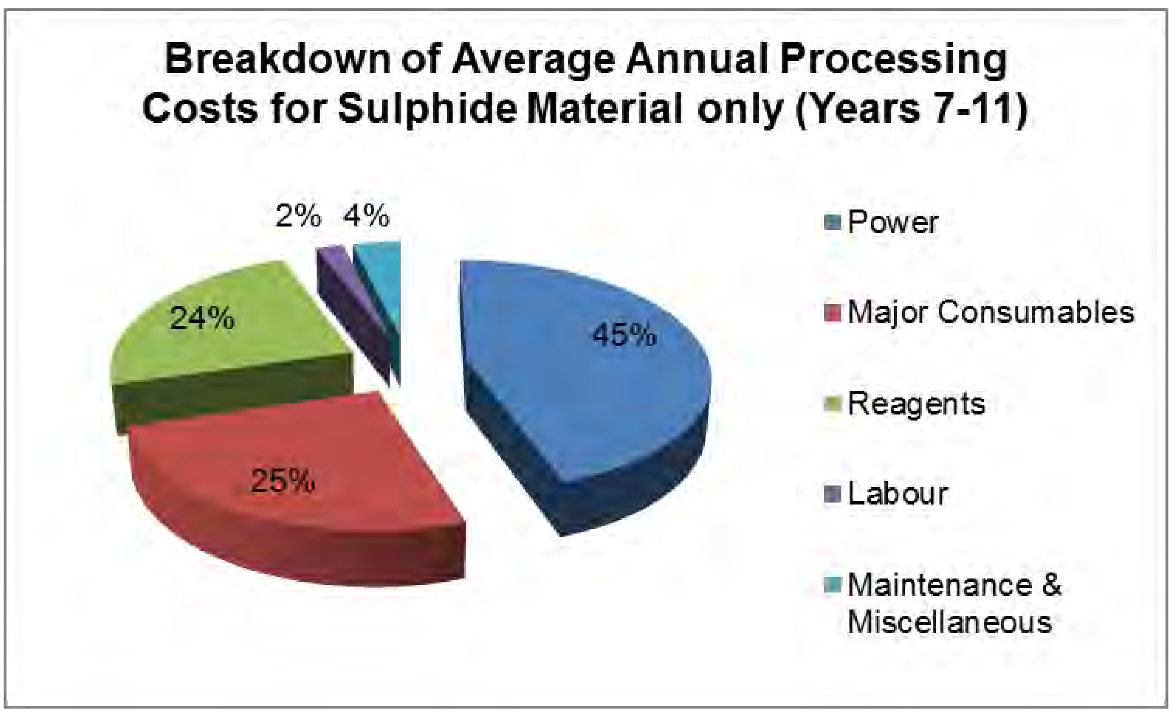


Figure 21-7: Average Annual Processing Costs for Years 7-11



Power in all cases was based on the power use estimate completed by Allnorth. Average annual power usage is 478MkW with a cost of $0.089/kWhr.

Table 21-12 outlines the manpower cost for the mill.

Table 21-12: Manpower Costs for Mill

Process	Count	Annual $/person	Total Annual Cost
Manager	1	$100,828	$100,828
Superintendents	2	$73,463	$146,927
Supervisors	10	$47,841	$478,408
Laboratory Chief	1	$60,338	$60,338
Maintenance Planner	1	$63,788	$63,788
Electrical Chief	1	$60,338	$60,338
Electrical Engineer	1	$40,543	$40,543
Metallurgy Technicians	2	$26,328	$52,657
Electrical Instrumentalists	6	$29,619	$177,716
Electromechanical Technicians	6	$26,328	$157,970
Sample Preparers	2	$22,012	$44,024
Welders	3	$10,448	$31,344
Mechanic	3	$10,740	$32,220
Mechanic Lubricator	3	$7,956	$23,868
Crusher Operators	3	$9,857	$29,570
Grinding Operators	5	$9,857	$49,283
Sulphide/Float Operators	3	$9,857	$29,570
Leach Operators	2	$9,857	$19,713
Tailings and water Operators	3	$9,857	$29,570
Dewater/Load-out/Reagents Ops	3	$9,857	$29,570
Elution/Refiner Operators	2	$9,857	$19,713
Mill Helpers	3	$7,919	$23,757
Reagents Preparers	3	$7,919	$23,757
Crusher Helpers	5	$7,919	$39,595
General Helpers	6	$7,919	$47,515

21.2.5 Tailings & Site Water Management

The total estimated LOM operating cost for the tailings storage facility and site water management is $5M These costs include maintenance, replacement, and fuel costs associated with the mechanical (pump and pipework) systems for the tailings storage facility, water management ponds, and open pit dewatering system during mine operation.

Fuel requirements were estimated based on the pumping power requirements throughout the life of the project and a unit rate for fuel of $0.82/L.

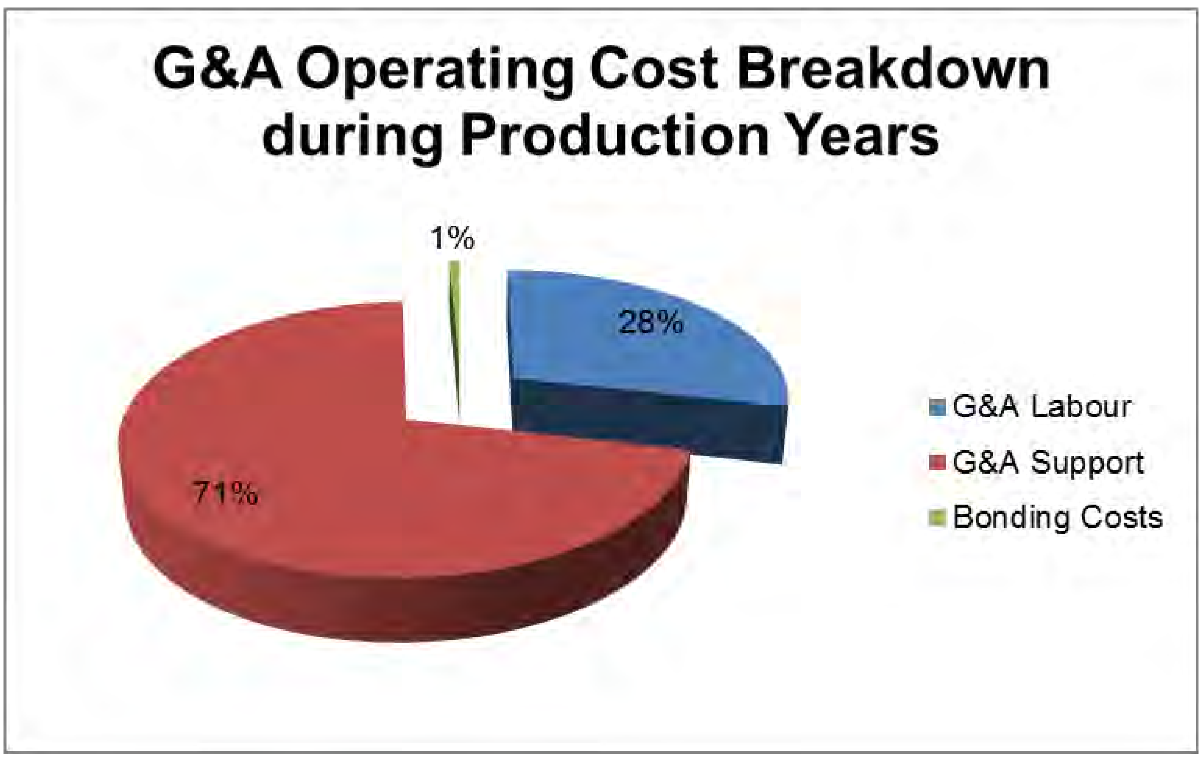
21.2.6 General and Administration

Average annual G&A costs during production total $7M. This includes labour and supplies for site administration, human resources, materials management, finance, and security. The G&A costs incurred during pre-production are capitalized and are included in the Owner's Costs. This amount amounts to $7M. Table 21-13 shows a breakdown of the average total G&A costs broken down by labour and support materials and services. Figure 21-8 demonstrates the breakdown of the LOM G&A costs during production by category.

Table 21-13: Average G&A cost per Production Year

Category	Average annual cost during production ($M)	% of Total
G&A Labour	2.1	29.2
G&A Support	5.1	70.8
Total G&A Cost	**7.2**	**100**

Figure 21-8: Breakdown of G&A Costs during Production Years



22 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variations in metal prices, ore production, grades, operating costs, capital costs and discount rates to determine their relative importance as project value drivers.

This technical report contains forward-looking information regarding projected mine production rates, construction schedule and forecast of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Section 21 of this report and are presented in 2013 dollars. The economic analysis has been run with no inflation (constant dollar basis).

22.1 Assumptions

Four metal price scenarios were evaluated to estimate the economic value potential of each and use the results as a comparative tool to better understand the value drivers in each scenario. In addition, the economic model was tested using leased mining fleet and a no-lease, up-front capital outflow during pre-production and Year 1 of production for the mining fleet. All costs, metal prices and economic results are reported in US dollars (USD). All three cases and the Whittle shell identical LOM plan tonnage and grade estimates (Table 22-1). On-site and off-site costs and production parameters were also held constant for each scenario evaluated.

Table 22-1: LOM Plan Summary for all Cases

Category	Units	Value
Sulphide Ore	M tonnes	137.8
Oxide Ore	M tonnes	11.8
Total Ore	**M tonnes**	**149.6**
Waste	M tonnes	267.6
Total Mined	**M tonnes**	**417.2**
Strip Ratio	w:o	1.79
Sulphide Ore Head Grade		
Cu	%	0.20
Au	g/t	0.30
Ag	g/t	1.50
Oxide Ore Head Grade		
Cu	%	0.22
Au	g/t	0.42
Ag	g/t	1.25
LOM Payable Metals		
Cu	M lbs	503.1
Au	k oz	1,164
Ag	k oz	2,952

Other economic factors common to all three cases include the following:

- Discount Rate of 7% (sensitivities using other discount rates have been calculated for each scenario)
- Closure cost of $27.3M
- Salvage Value of $34.4M
- Nominal 2013 dollars
- No inflation
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment
- Working capital calculated as three months of sulphide ore operating costs (including processing, G&A, and tailings operating costs)
- Results are presented on 100% ownership and do not include management fees or financing costs
- Exclusion of all pre-development and sunk costs (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.).

Table 22-2 outlines the metal prices, MEX:USD and CDN:USD exchange rates assumptions used in the various scenarios of the economic analysis. Trailing averages were through February 2013. Prices used for Whittle were approximately 10% below spot price at January 31, 2013.

Table 22-2: Metal Prices and Exchange Rates by Scenario

Parameter	Units	Three-Year Trailing Average	PFS Base Case Four-Year Trailing Average	Five-Year Trailing Average	Whittle Parameter Pricing
Copper Price	USD $/lb	3.71	3.44	3.32	3.15
Gold Price	USD $/oz	1,517.50	1,389.95	1,286.13	1,400.00
Silver Price	USD $/oz	29.58	26.03	23.68	26.00
Exchange Rate	MEX:USD	13:1	13:1	13:1	13:1
Exchange Rate	CDN:USD	1.00	1.00	1.00	1.00

22.2 Mine Production Statistics

Mine production is reported as the ore and waste material resulting from the mining operation. Annual production figures were obtained from the mine plan developed for this study. The LOM ore and waste quantities and head grades are presented in Table 22-3.

22.3 Revenues & NSR Parameters

Mine revenue is derived from the sale of concentrate and doré into the international marketplace. No contractual arrangements for concentrate smelting or refining exist at this time. Details regarding the smelter terms used for the economic analysis can be found in the Market Studies Section 19 of this report. Revenues from doré production are set to begin in 2015 and revenues from concentrate sales are set to begin in 2016. Concentrate and doré sales are set to end in 2026, 11 years after the start of sulphide ore processing (12 years after oxide ore processing). Tables 22-4 and 22-5 indicate the NSR parameters that were used in the economic analysis. Figure 22-1 shows a breakdown of the amount of concentrate produced during the mine life – a total of 908k dmt of copper concentrate is produced from 2016 to 2026.

Table 22-6 shows the amount of payable metal for the life of the project. Figure 22-2 demonstrates the breakdown of LOM Net Smelter Return by metal. Figures 22-3 to 22-7 show the breakdown of payable copper, gold and silver by year and processing method. Total Net Smelter Return (net of royalty payments) for the base case amounted to $3,150M.

Table 22-3: LOM Production & Head Grades

Category	Units	Values
TEPAL NORTH PIT		
Sulphide Ore	M tonnes	50.2
Cu	%	0.23
Au	g/t	0.35
Ag	g/t	0.96
Oxide Ore	M tonnes	7.0
Cu	%	0.23
Au	g/t	0.47
Ag	g/t	0.96
TEPAL SOUTH PIT		
Sulphide Ore	M tonnes	31.0
Cu	%	0.20
Au	g/t	0.43
Ag	g/t	1.12
Oxide Ore	M tonnes	2.3
Cu	%	0.20
Au	g/t	0.48
Ag	g/t	1.10
TIZATE PIT		
Sulphide Ore	M tonnes	56.7
Cu	%	0.17
Au	g/t	0.19
Ag	g/t	2.17
Oxide Ore	M tonnes	2.5
Cu	%	0.18
Au	g/t	0.26
Ag	g/t	2.20

Table 22-4: NSR Parameters used in Economic Analysis – Copper Concentrate

Category	Unit	Value
TEPAL - Recovery to Cu Concentrate		
Cu Recovery	%	88.2
Au Recovery	%	62.4
Ag Recovery	%	27.4
TIZATE - Recovery to Cu Concentrate		
Cu Recovery	%	85.9
Au Recovery	%	58.0
Ag Recovery	%	59.6
Concentrate Grade		
Cu - Tepal	%	25.7
Cu - Tizate	%	26.9
Au	g/t	Variable
Ag	g/t	Variable
Moisture Content	%	8
Smelter Payables		
Cu Payable	%	96.5
Min. Cu deduction	% Cu/tonne	1
Au Payable	%	97
Min. Au deduction	g/t in concentrate	0.0
Ag Payable	%	90
Min. Ag deduction	g/t in concentrate	30.0
Treatment & Refining Costs		
Cu TC	$/dmt concentrate	50.00
Cu RC	$/payable lb	0.05
Au RC	$/payable oz	5.00
Ag RC	$/payable oz	0.50
Transport Costs		
Ocean freight to Japan	$/wmt	60.00
Truck freight to Port	$/wmt	36.73
Representation at Port	$/wmt	1.00
Port charges	$/wmt	10.50
Insurance	$/wmt	1.93
Losses	$/wmt	7.50
Subtotal	**$/wmt**	**117.66**
	$/dmt	**108.25**
Royalties - Cu	%	2.5
Royalties - Au	%	2.5
Royalties - Ag	%	2.5

Figure 22-1: LOM Concentrate Production

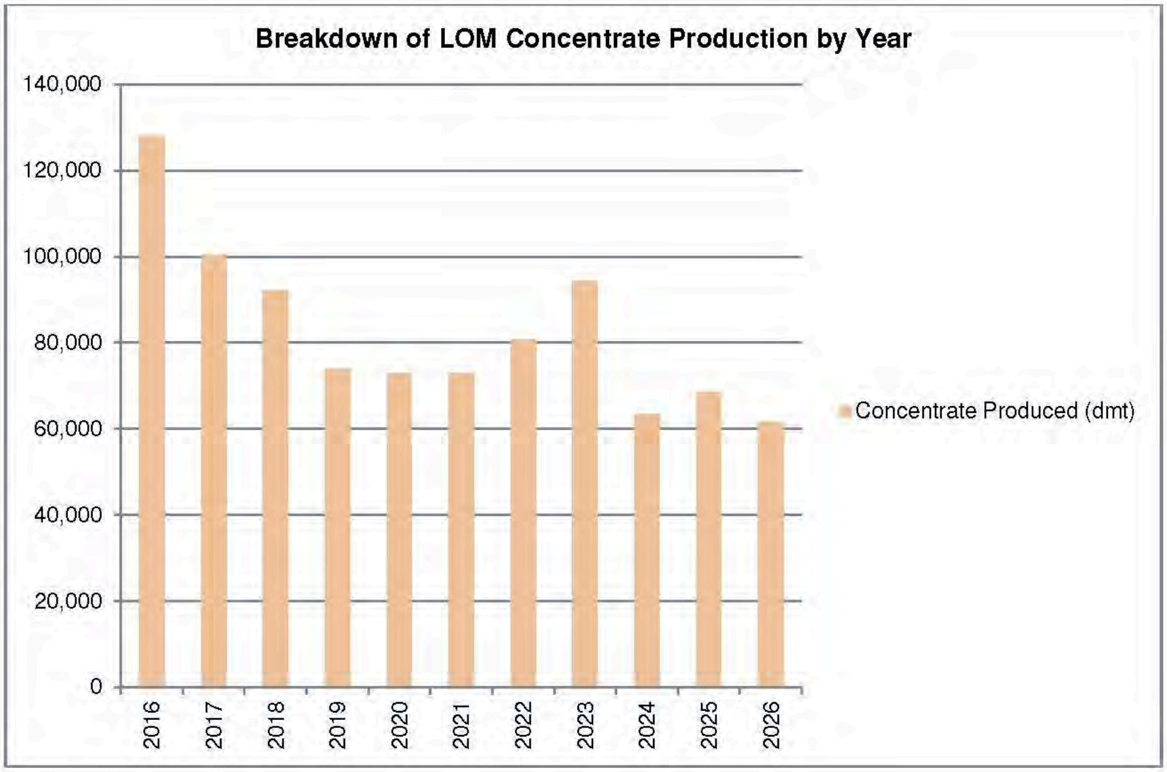


Table 22-5: NSR Parameters used in Economic Analysis – Gold and Silver Doré

Category	Unit	Value
SULPHIDE CYANIDATION		
TEPAL - Recovery		
Au Recovery	%	17.3
Ag Recovery	%	13.6
Losses to Solution	%	0.0
TIZATE - Recovery		
Au Recovery	%	20.5
Ag Recovery	%	12.2
Losses to Solution	%	0.0
Smelter Payables		
Au Payable	%	99.9
Min. Au deduction	g/t	0.0
Ag Payable	%	97.0
Min. Ag deduction	g/t	0.0
Treatment & Refining Costs		
Au RC	$/payable oz	7.50
Ag RC	$/payable oz	1.40
Operating Costs		
Processing Cost - Tepal	$/tonne milled	0.87
Processing Cost - Tizate	$/tonne milled	0.87
Sulphide Cyanidation feed	%	12.0
OXIDE CYANIDATION		
TEPAL - Recovery		
Au Recovery	%	83.2
Ag Recovery	%	63.3
Losses to Solution	%	0.0
TIZATE - Recovery		
Au Recovery	%	75.2
Ag Recovery	%	55.9
Losses to Solution	%	0.0
Smelter Payables		
Au Payable	%	99.9
Min. Au deduction	g/t	0.0
Ag Payable	%	97.0
Min. Ag deduction	g/t	0.0

Category	Unit	Value
Treatment & Refining Costs		
Au RC	$/payable oz	7.50
Ag RC	$/payable oz	1.40

Table 22-6: LOM Payable Metal

Category	Unit	Value
Payable Cu	LOM M lbs	503
Payable Au	LOM k oz	1,164
Payable Ag	LOM k oz	2,952

Figure 22-2: NSR Breakdown by Payable Metal

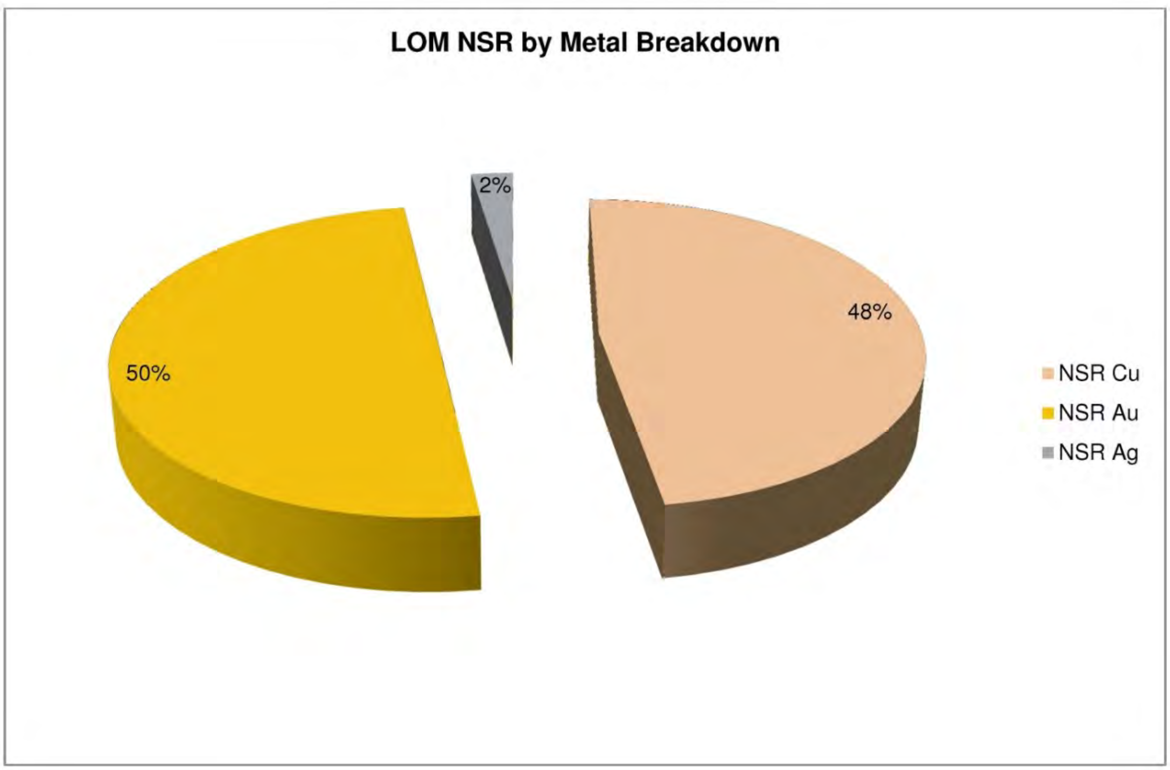


Figure 22-3: LOM Payable Copper by Year

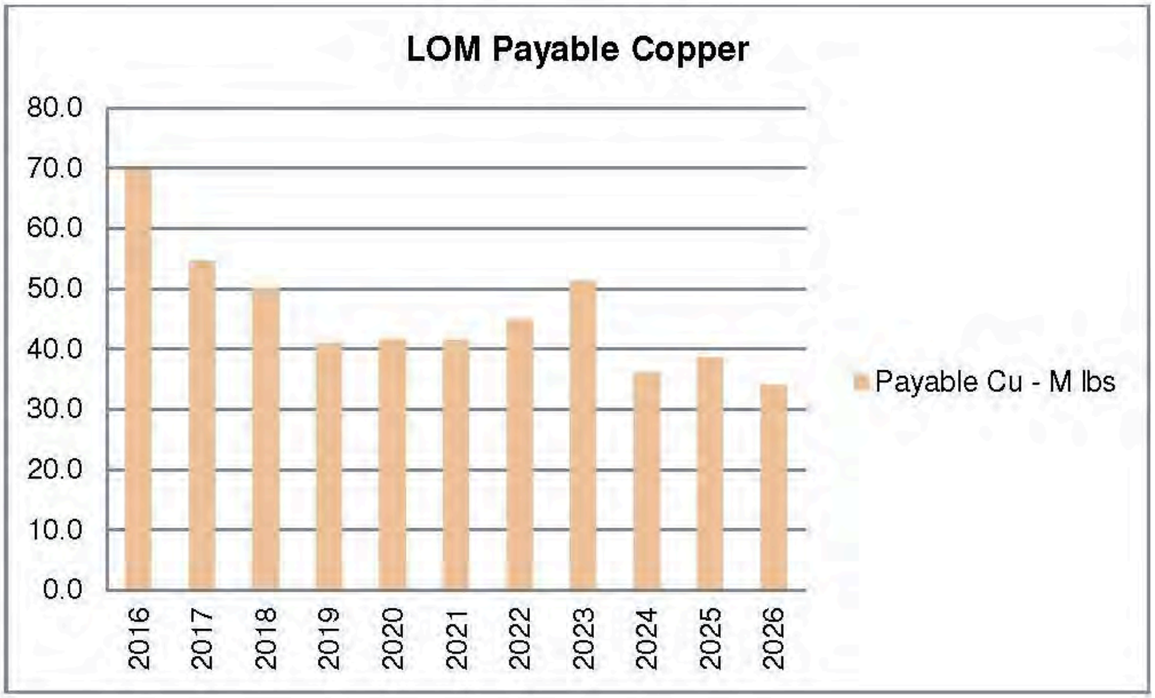


Figure 22-4: LOM Payable Gold by Year

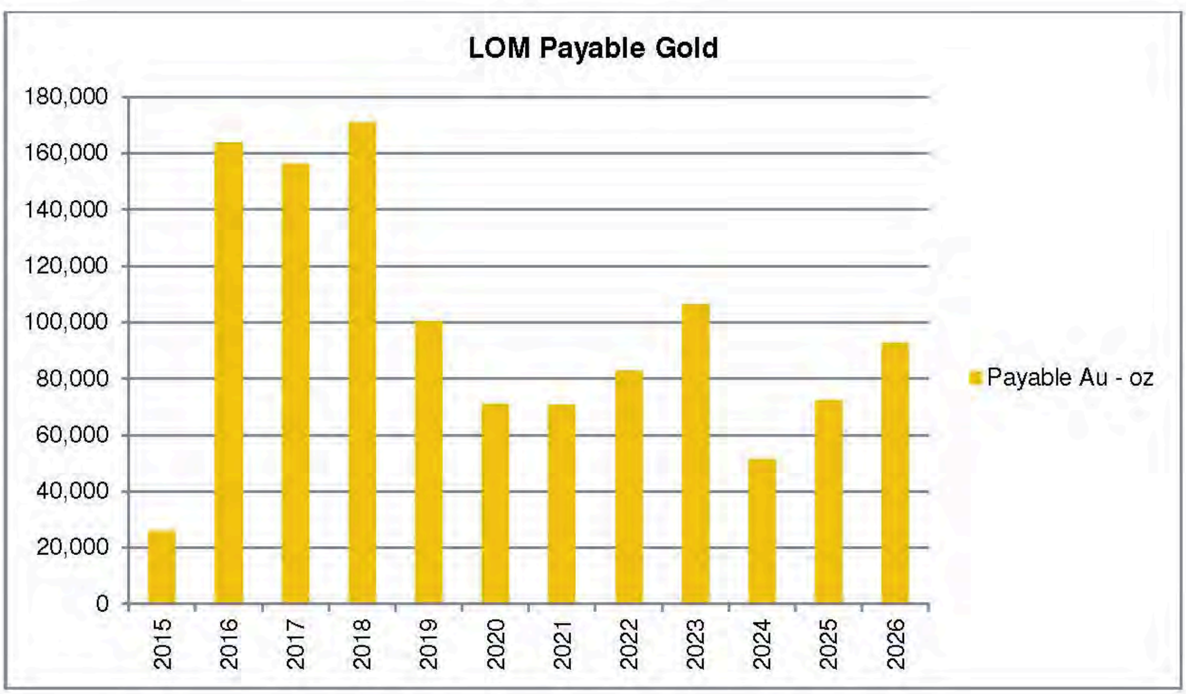


Figure 22-5: Breakdown of Payable Gold by Processing Method

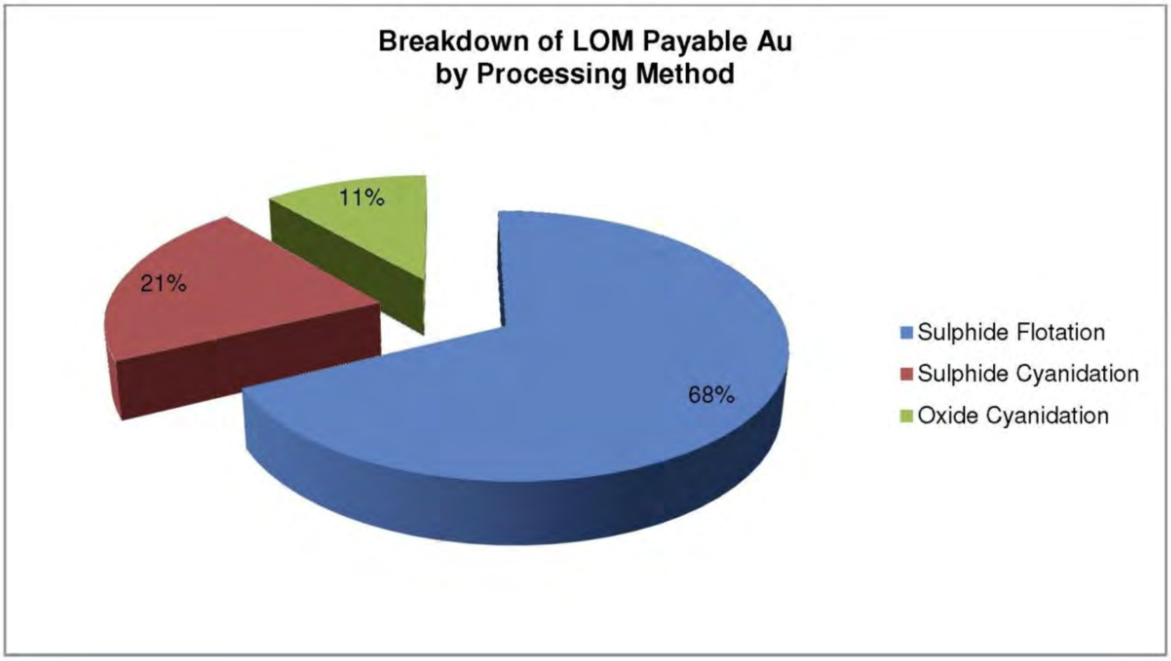


Figure 22-6: LOM Payable Silver by Year

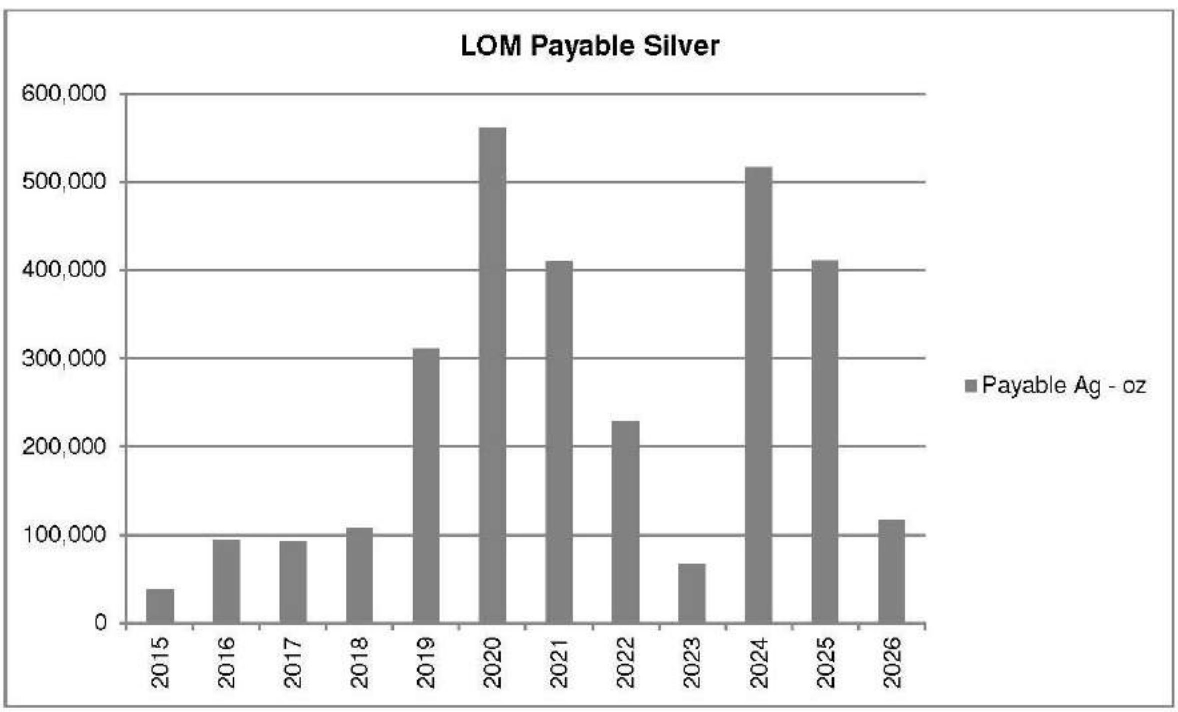
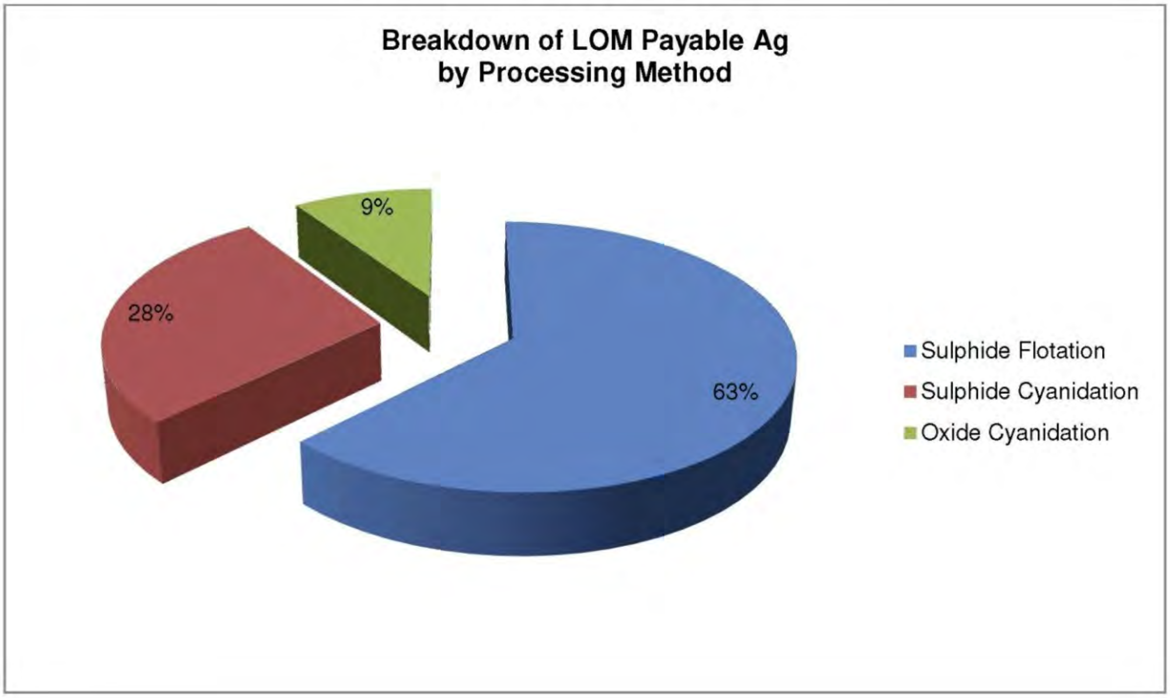


Figure 22-7: Breakdown of Payable Silver by Processing Method



22.4 Summary of Capital Cost Estimates

From 2013 to 2015, the pre-production capital costs amount to $354M. This includes costs for site development, processing plant, on-site infrastructure, camp construction, pre-production operating costs, etc. This also assumes leasing of the mining fleet totaling $88M. Leasing of mine equipment fleet was assumed to determine the project value for all scenarios. A blended 9% contingency is included in this total pre-production capital cost.

Sustaining capital costs amount to $44M and occur from 2016 to 2027 with most of the costs occurring in 2016. The majority of these costs account for tailings earthworks. A blended 9% contingency is also included in this sustaining capital costs. Sustaining capital costs without a leased mining fleet amounts to $95M. The leasing costs incurred to calculate the project value are accounted for in the mine's operating costs.

Closure costs amount to $27M and occur in 2027. This includes $25M for tailings, waste dumps and site closure and $2.0M for the closure of mill facility foundations.

Salvage value is accounted for in 2027 amounting to $34M. This amounts to 10% of the mine equipment and process plant capital costs.

Table 22-7 shows a breakdown of all of the capital costs considered in the economic analysis of the project. The table reflects the capital costs using a leased mine equipment fleet.

Detailed information on capital costs are found in Section 21 of this report. Figure 22-8 and 22-9 show breakdowns of the pre-production and production capital costs assuming leasing of mine equipment fleet.

Table 22-7: Capital Cost Summary

Category	Pre-Production	Production	Total Capital Costs	% of Total
Capitalized Pre-Stripping	21.5	0.0	21.5	5.4
Support Equipment	3.3	1.1	4.4	1.1
Tailings	34.7	42.0	76.7	19.3
Process Plant	229.6	0.0	229.6	57.8
Indirects	20.1	0.0	20.1	5.0
Owner's Costs	13.4	0.0	13.4	3.4
Salvage Value	0.0	-34.4	-34.4	-8.6
Closure	0.0	27.3	27.3	6.9
Contingency (8.7%)	31.3	7.6	38.9	9.8
Total Capital Costs	**353.8**	**43.6**	**397.4**	**100.0**

Figure 22-8: Breakdown of Pre-Production Capital Costs

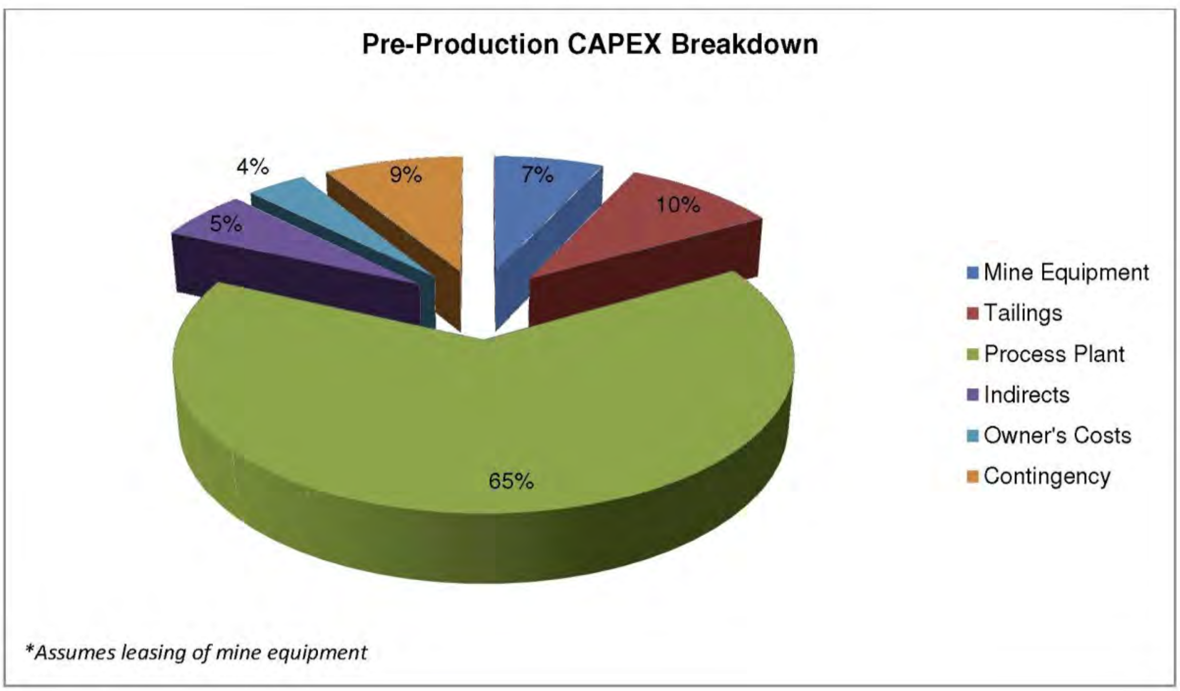
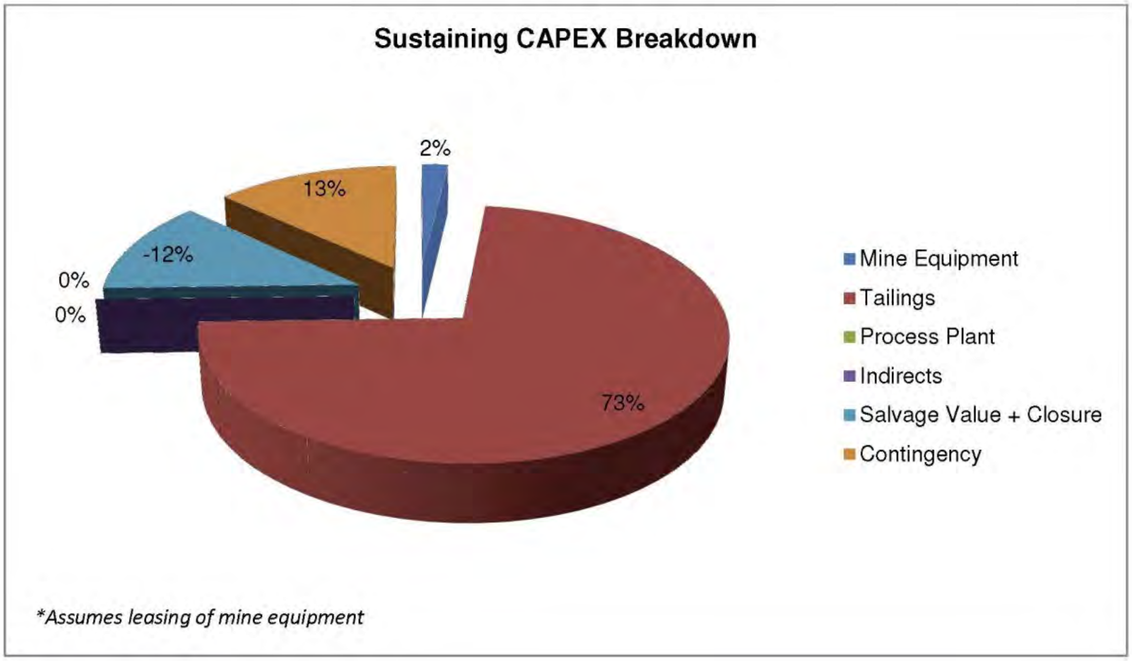


Figure 22-9: Breakdown of Sustaining and Closure Costs & Salvage Value



22.5 Summary of Operating Cost Estimates

Total operating costs amount to $1,809M (including leasing of mine equipment fleet). Without the leased equipment, total operating cost amount to $1,728M for the life of the project.

This translates to an average cost of $12.27/tonne of ore mined assuming leasing of mining equipment, and $11.60/tonne of ore mined assuming no leasing of mining fleet. The breakdown of these costs is shown in Tables 22-8 and 22-9.

Table 22-8: Summary of Annual Operating Costs (including pre-production ore & associated costs) – with Leased Mine Equipment Fleet

Category	$M	%
Mining	54.9	33.6
Processing - Sulphide Flotation	76.3	46.8
Processing - Sulphide Cyanidation	10.8	6.6
Milling & Processing - Oxide Cyanidation	6.0	3.7
G&A	7.3	4.5
Tailings	0.5	0.3
Mine Equipment Leasing	7.4	4.5
Total Average Annual Operating Costs	**163.2**	**100**

Table 22-9: Summary of Annual Operating Costs (including pre-production ore & associated costs) – No Lease of Mining Fleet

Category	$M	%
Mining	54.9	35.2
Processing - Sulphide Floatation	76.3	49.0
Processing - Sulphide Cyanidation	10.8	7.0
Processing & Milling - Oxide Cyanidation	6.0	3.8
G&A	7.3	4.7
Tailings	0.5	0.3
Mine Equipment Leasing	0.0	0.0
Total Average Annual Operating Costs	**155.8**	**100**

22.6 Leasing

The economic analysis assumes that all mine equipment fleet would be leased with Caterpillar. The total value of the mine equipment to be leased is $88M. The terms used to calculate the lease payments were taken from a quote dated December 2012 and include the following:

- 5% interest rate
- 5 year term
- 15% up-front payment of equipment leased.

Lease payments (including up-front payments) for the life of mine total $100M. The economic model was tested without a leasing option – the results of this using the four-year trailing metal price Base Case scenario were an after-tax $NPV_{7\%}$ of $346M, 26.3% IRR, and 3.3 year payback period.

22.7 Taxes

The project has been evaluated on an after-tax basis in order to reflect a more indicative value of the project. Geologix commissioned PwC in Vancouver, BC to prepare a tax model for the post tax economic evaluation of the project with the inclusion of applicable Mexican income taxes. These tax calculations have been used in the economic analysis presented in this report. The tax calculation uses an inflation factor of 3.5% per year, a 5% employee profit sharing, and a 28% Mexican corporate tax rate. Total taxes for the life of the project amount to $234M.

22.8 Economic Results

The project is economically viable with an after-tax internal rate of return (IRR) of 27.7% and a net present value at 7% ($NPV_{7\%}$) of $345M for the Base Case which was calculated based on four-year trailing average metal prices as of February 28, 2013 and a CDN:USD exchange rate at par. In addition, three additional scenarios were measured based on three-year, five-year trailing average metal prices as of February 28, 2013 and metal prices used for Whittle.

The scenario using three-year trailing average prices resulted in the highest performance and project value due to the highest metal prices of all scenarios evaluated in this study. The calculated Base Case resulted in the second highest project value of all three cases primarily due to higher metal prices than the scenario evaluated using five-year trailing prices and metal prices used to run Whittle. The metal prices used in the five year trailing scenario yielded the lowest economic value of the project, however, still showed positive results.

Figure 22-10 shows the projected cash flows for the project used in the economic analysis. Tables 22-10 to 22-13 show the economic results of each of the three cases calculated. In addition, Tables 22-14 to 22-17 show the sensitivity of pre-tax and after-tax NPV at various discount rates using the Base Case metal pricing.

Figure 22-10: Annual and Cumulative Cash Flows for the Project Life

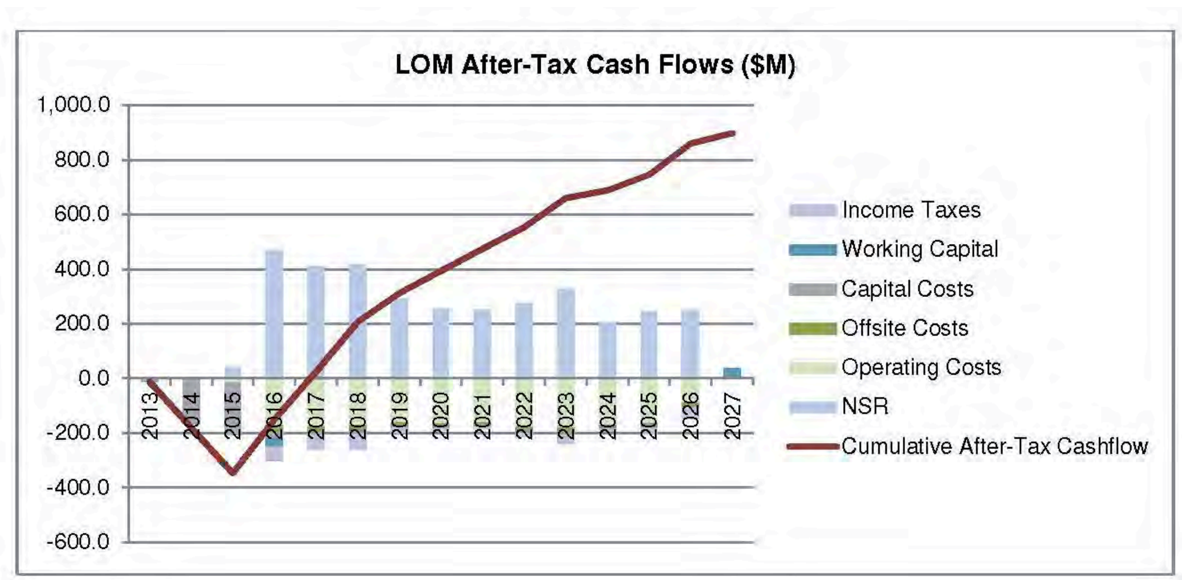


Table 22-10: Summary of Base Case Economic Results (Four-Year Trailing Average Metal Prices)

Category	Unit	Value
Mine Life	Years	11.5
Average Plant Throughput	M tpa	13.0
Payable Cu LOM	LOM M lbs	503.1
Average Payable Cu (Year 1-7)	M lbs/yr	49.0
Payable Au LOM	LOM k oz	1,164
Average Payable Au (Year 1-7)	k oz/yr	116.6
Payable Ag LOM	LOM k oz	2,952
Average Payable Ag (Year 1-7)	k oz/yr	257.8
Cash Cost (Net of By-Product Credits)	$/Payable Cu lb	0.62
	$/Payable Au oz	170
Cash Cost (Net of By-Product Credits incl. of Sustaining Capital)	$/Payable Cu lb	0.81
	$/Payable Au oz	251
Cash Cost (Net of By-Product Credits incl. Total Capital)	$/Payable Cu lb	1.58
	$/Payable Au oz	587
Unit OPEX (Offsite Costs + Operating Costs)	$/tonne ore	13.40
Avg Annual Cashflow during production	$ M	86.8
Pre-Production Capital with Leased Equipment	$ M	353.8
Sustaining & Closure Capital	$ M	43.6
Total Capital + Contingency	$ M	397.4
Pre-Tax $NPV_{7\%}$	$ M	495.1
Pre-Tax IRR	%	35.9
Pre-Tax Payback Period	Years	2.7
After-Tax $NPV_{7\%}$	$ M	344.8
After-Tax IRR	%	27.7
After-Tax Payback Period	Years	3.2

Table 22-11: Summary of Results using Three-Year Trailing Average Metal Prices

Category	Unit	Value
Mine Life	Years	11.5
Average Plant Throughput	M tpa	13.0
Payable Cu LOM	LOM M lbs	503.1
Average Payable Cu (Year 1-7)	M lbs/yr	49.0
Payable Au LOM	LOM k oz	1,164
Average Payable Au (Year 1-7)	k oz/yr	116.6
Payable Ag LOM	LOM k oz	2,952
Average Payable Ag (Year 1-7)	k oz/yr	257.8
Cash Cost (Net of By-Product Credits)	$/Payable Cu lb	0.31
	$/Payable Au oz	50
Cash Cost (Net of By-Product Credits incl. of Sustaining Capital)	$/Payable Cu lb	0.50
	$/Payable Au oz	132
Cash Cost (Net of By-Product Credits incl. Total Capital)	$/Payable Cu lb	1.28
	$/Payable Au oz	468
Unit OPEX (Offsite Costs + Operating Costs)	$/tonne ore	13.45
Avg Annual Cashflow during production	$ M	103.8
Pre-Production Capital with Leased Equipment	$ M	353.8
Sustaining & Closure Capital	$ M	43.6
Total Capital + Contingency	$ M	397.4
Pre-Tax $NPV_{7\%}$	$ M	675.2
Pre-Tax IRR	%	44.2
Pre-Tax Payback Period	Years	2.4
After-Tax $NPV_{7\%}$	$ M	474.5
After-Tax IRR	%	34.1
After-Tax Payback Period	Years	2.9

Table 22-12: Summary of Results using Five-Year Trailing Average Metal Prices

Category	Unit	Value
Mine Life	Years	11.5
Average Plant Throughput	M tpa	13.0
Payable Cu LOM	LOM M lbs	503.1
Average Payable Cu (Year 1-7)	M lbs/yr	49.0
Payable Au LOM	LOM k oz	1,164
Average Payable Au (Year 1-7)	k oz/yr	116.6
Payable Ag LOM	LOM k oz	2,952
Average Payable Ag (Year 1-7)	k oz/yr	257.8
Cash Cost (Net of By-Product Credits)	$/Payable Cu lb	0.86
	$/Payable Au oz	224
Cash Cost (Net of By-Product Credits incl. of Sustaining Capital)	$/Payable Cu lb	1.05
	$/Payable Au oz	305
Cash Cost (Net of By-Product Credits incl. Total Capital)	$/Payable Cu lb	1.83
	$/Payable Au oz	641
Unit OPEX (Offsite Costs + Operating Costs)	$/tonne ore	13.37
Avg Annual Cashflow during production	$ M	76.1
Pre-Production Capital with Leased Equipment	$ M	353.8
Sustaining & Closure Capital	$ M	43.6
Total Capital + Contingency	$ M	397.4
Pre-Tax $NPV_{7\%}$	$ M	379.7
Pre-Tax IRR	%	30.1
Pre-Tax Payback Period	Years	3.0
After-Tax $NPV_{7\%}$	$ M	261.5
After-Tax IRR	%	23.2
After-Tax Payback Period	Years	3.5

Table 22-13: Summary of Results using Whittle Parameter Pricing

Category	Unit	Value
Mine Life	Years	11.5
Average Plant Throughput	M tpa	13.0
Payable Cu LOM	LOM M lbs	503.1
Average Payable Cu (Year 1-7)	M lbs/yr	49.0
Payable Au LOM	LOM k oz	1,164
Average Payable Au (Year 1-7)	k oz/yr	116.6
Payable Ag LOM	LOM k oz	2,952
Average Payable Ag (Year 1-7)	k oz/yr	257.8
Cash Cost (Net of By-Product Credits)	$/Payable Cu lb	0.59
	$/Payable Au oz	292
Cash Cost (Net of By-Product Credits incl. of Sustaining Capital)	$/Payable Cu lb	0.78
	$/Payable Au oz	374
Cash Cost (Net of By-Product Credits incl. Total Capital)	$/Payable Cu lb	1.55
	$/Payable Au oz	709
Unit OPEX (Offsite Costs + Operating Costs)	$/tonne ore	13.38
Avg Annual Cashflow during production	$ M	79.0
Pre-Production Capital with Leased Equipment	$ M	353.8
Sustaining & Closure Capital	$ M	43.6
Total Capital + Contingency	$ M	397.4
Pre-Tax $NPV_{7\%}$	$ M	414.6
Pre-Tax IRR	%	32.2
Pre-Tax Payback Period	Years	2.9
After-Tax $NPV_{7\%}$	$ M	286.7
After-Tax IRR	%	24.8
After-Tax Payback Period	Years	3.4

Table 22-14: Base Case NPV for Various Discount Rates (Four-Year Trailing Average Metal Prices)

Discount Rate Sensitivity	Pre-Tax $NPV_{x\%}$ ($M)	After-Tax $NPV_{x\%}$ ($M)
0%	924.6	690.1
5%	590.3	421.2
7%	**495.1**	**344.8**
8%	453.6	311.6
10%	380.6	253.3

Table 22-15: NPV for Various Discount Rates using Three-Year Trailing Average Metal Prices

Discount Rate Sensitivity	Pre-Tax $NPV_{x\%}$ ($M)	After-Tax $NPV_{x\%}$ ($M)
0%	1,212.0	897.0
5%	794.5	568.2
7%	675.2	474.5
8%	623.0	433.6
10%	531.3	361.8

Table 22-16: NPV for Various Discount Rates using Five-Year Trailing Average Metal Prices

Discount Rate Sensitivity	Pre-Tax $NPV_{x\%}$ ($M)	After-Tax $NPV_{x\%}$ ($M)
0%	741.1	558.4
5%	459.6	326.9
7%	379.7	261.5
8%	344.9	233.1
10%	283.8	183.3

Table 22-17: NPV for Various Discount Rates using Whittle Parameter Metal Prices

Discount Rate Sensitivity	Pre-Tax $NPV_{x\%}$ ($M)	After-Tax $NPV_{x\%}$ ($M)
0%	793.6	596.1
5%	498.5	355.0
7%	414.6	286.7
8%	378.0	257.0
10%	313.8	204.9

22.9 Sensitivity

The sensitivity charts Tables 22-18 to 22-25 and Figures 22-11 to 22-18, below, show IRR and NPV variations from the Base Case with respect to changes in metal prices, ore production, head grades, operating costs and capital costs, holding all other inputs constant. The results below show that the project is most sensitive to metal price and head grade and least sensitive to changes in capital costs in all four scenarios.

Table 22-18: Base Case Pre-Tax $NPV_{7\%}$ Sensitivity Results (Four-Year Trailing Average Metal Prices)

Pre-Tax $NPV_{7\%}$ Sensitivity Table ($M)			
Variable	**-15%**	**100%**	**+15%**
Metal Price	181.4	495.1	808.9
Ore Production	292.1	495.1	698.2
Head Grade	199.1	495.1	791.2
Operating Costs	659.4	495.1	330.9
Capital Costs	550.8	495.1	439.5

Table 22-19: Baser Case After-Tax $NPV_{7\%}$ Sensitivity Results (Four-Year Trailing Average Metal Prices)

After-Tax $NPV_{7\%}$ Sensitivity Table ($M)			
Variable	**-15%**	**100%**	**+15%**
Metal Price	117.1	344.8	570.8
Ore Production	198.2	344.8	491.1
Head Grade	130.1	344.8	558.1
Operating Costs	463.1	344.8	226.0
Capital Costs	400.5	344.8	289.2

Table 22-20: Pre-Tax $NPV_{7\%}$ Sensitivity Results using Three-Year Trailing Average Metal Prices

Pre-Tax $NPV_{7\%}$ Sensitivity Table ($M)			
Variable	**-15%**	**100%**	**+15%**
Metal Price	334.4	675.2	1,015.9
Ore Production	445.2	675.2	905.2
Head Grade	352.1	675.2	998.3
Operating Costs	839.4	675.2	510.9
Capital Costs	730.9	675.2	619.5

Table 22-21: After-Tax NPV7% Sensitivity Results using Three-Year Trailing Average Metal Prices

After-Tax $NPV_{7\%}$ Sensitivity Table ($M)			
Variable	**-15%**	**100%**	**+15%**
Metal Price	228.7	474.5	719.9
Ore Production	308.8	474.5	640.2
Head Grade	241.5	474.5	707.2
Operating Costs	592.8	474.5	356.2
Capital Costs	530.2	474.5	418.8

Table 22-22: Pre-Tax $NPV_{7\%}$ Sensitivity Results using Five-Year Trailing Average Metal Prices

Pre-Tax $NPV_{7\%}$ Sensitivity Table ($M)			
Variable	**-15%**	**100%**	**+15%**
Metal Price	83.3	379.7	676.1
Ore Production	194.0	379.7	565.4
Head Grade	100.9	379.7	658.5
Operating Costs	543.9	379.7	215.5
Capital Costs	435.4	379.7	324.0

Table 22-23: After-Tax $NPV_{7\%}$ Sensitivity Results using Five-Year Trailing Average Metal Prices

After-Tax $NPV_{7\%}$ Sensitivity Table ($M)			
Variable	**-15%**	**100%**	**+15%**
Metal Price	38.8	261.5	475.2
Ore Production	126.7	261.5	395.4
Head Grade	54.3	261.5	462.5
Operating Costs	380.0	261.5	141.6
Capital Costs	317.2	261.5	205.8

Table 22-24: Pre-Tax $NPV_{7\%}$ Sensitivity Results using Whittle Parameter Pricing

Pre-Tax $NPV_{7\%}$ Sensitivity Table ($M)			
Variable	**-15%**	**100%**	**+15%**
Metal Price	113.0	414.6	716.3
Ore Production	223.7	414.6	605.6
Head Grade	130.6	414.6	698.6
Operating Costs	578.9	414.6	250.4
Capital Costs	470.3	414.6	358.9

Table 22-25: After-Tax NPV$_{7\%}$ Sensitivity Results using Whittle Parameter Pricing

After-Tax NPV$_{7\%}$ Sensitivity Table ($M)			
Variable	**-15%**	**100%**	**+15%**
Metal Price	63.2	286.7	504.1
Ore Production	148.3	286.7	424.4
Head Grade	78.5	286.7	491.4
Operating Costs	405.1	286.7	167.0
Capital Costs	342.4	286.7	231.1

Figure 22-11: Base Case Pre-Tax NPV$_{7\%}$ Sensitivity Results (Four-Year Trailing Average Metal Prices)

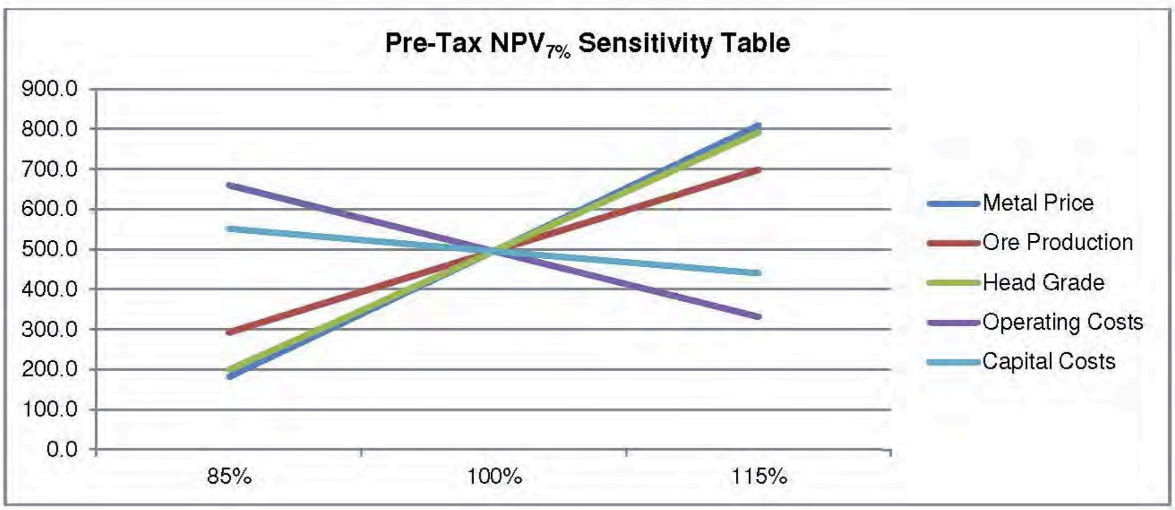


Figure 22-12: Base Case After-Tax NPV$_{7\%}$ Sensitivity Results (Four-Year Trailing Average Metal Prices)

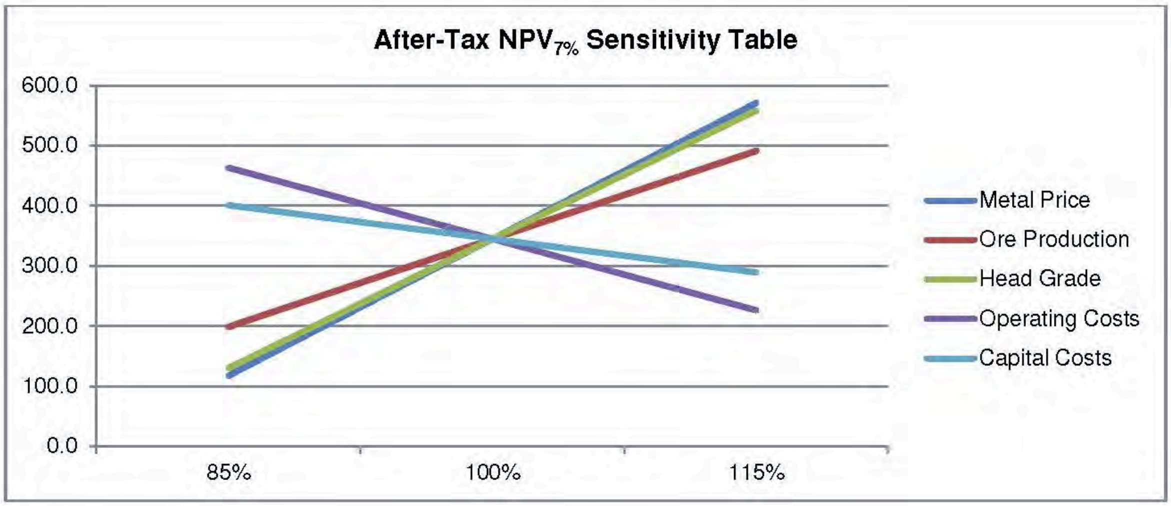


Figure 22-13: Pre-Tax NPV$_{7\%}$ Sensitivity Results using Three-Year Trailing Average Metal Prices

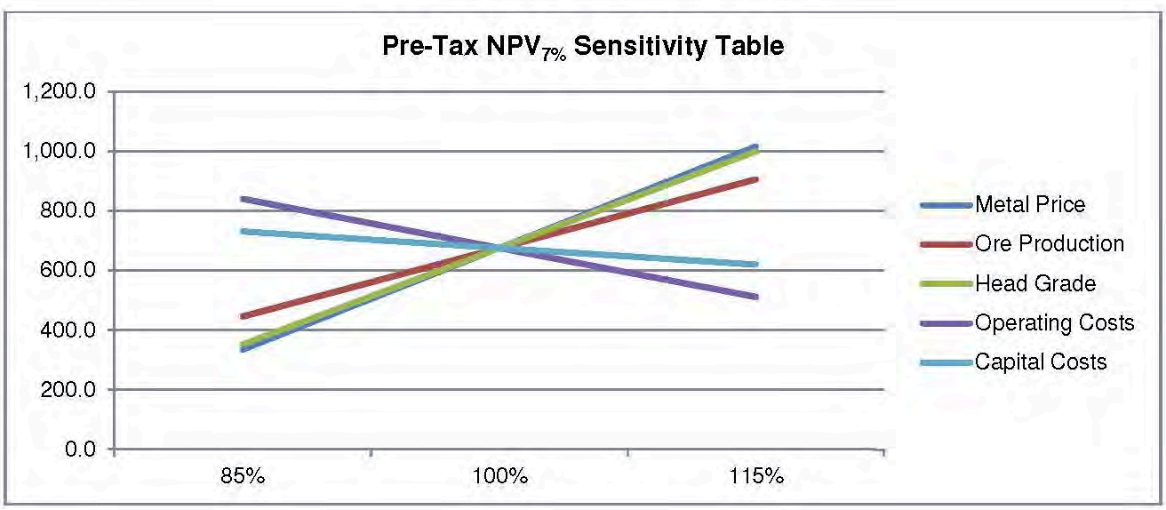


Figure 22-14: After-Tax NPV$_{7\%}$ Sensitivity Results using Three-Year Trailing Average Metal Prices

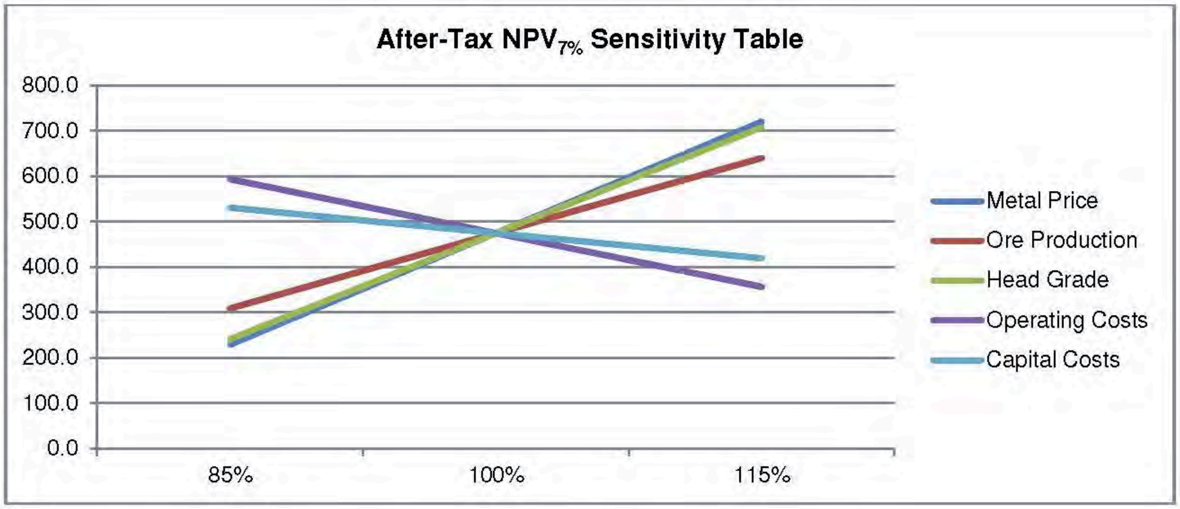


Figure 22-15: Pre-Tax NPV$_{7\%}$ Sensitivity Results using Five-Year Trailing Average Metal Prices

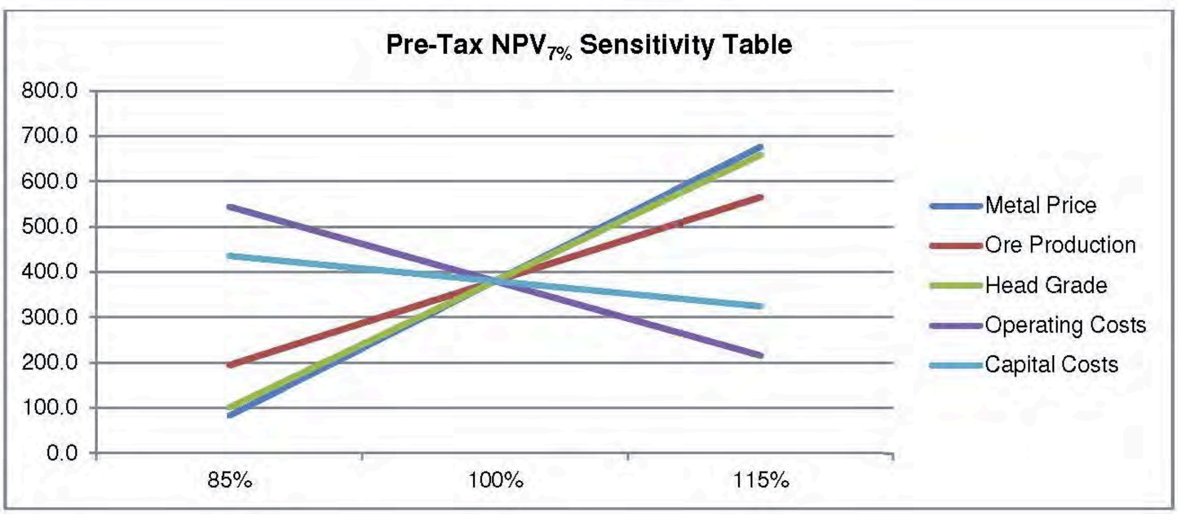


Figure 22-16: After-Tax NPV$_{7\%}$ Sensitivity Results using Five-Year Trailing Average Metal Prices

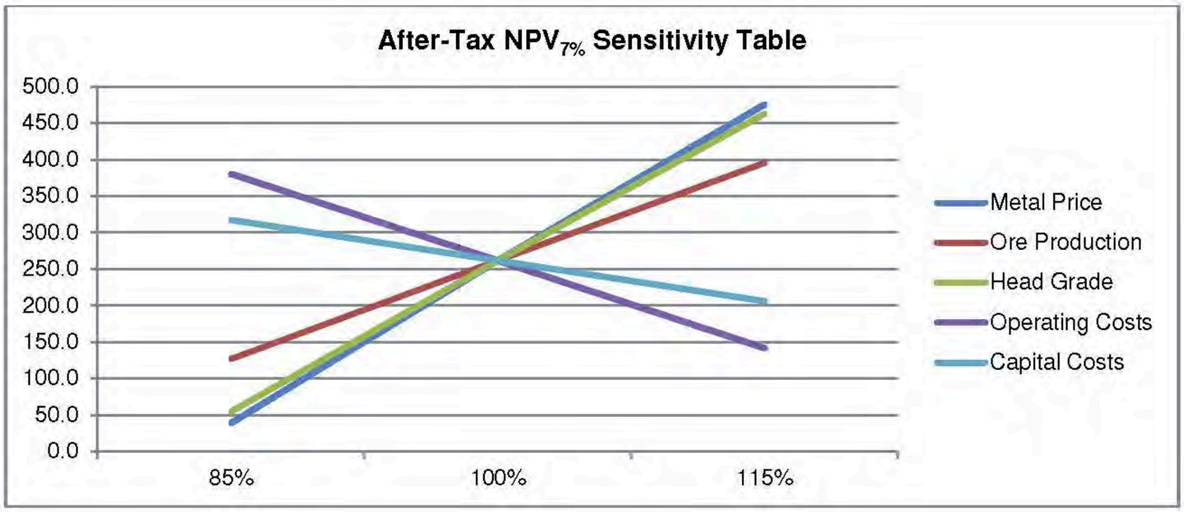


Figure 22-17: Pre-Tax NPV$_{7\%}$ Sensitivity Results using Whittle Parameter Pricing

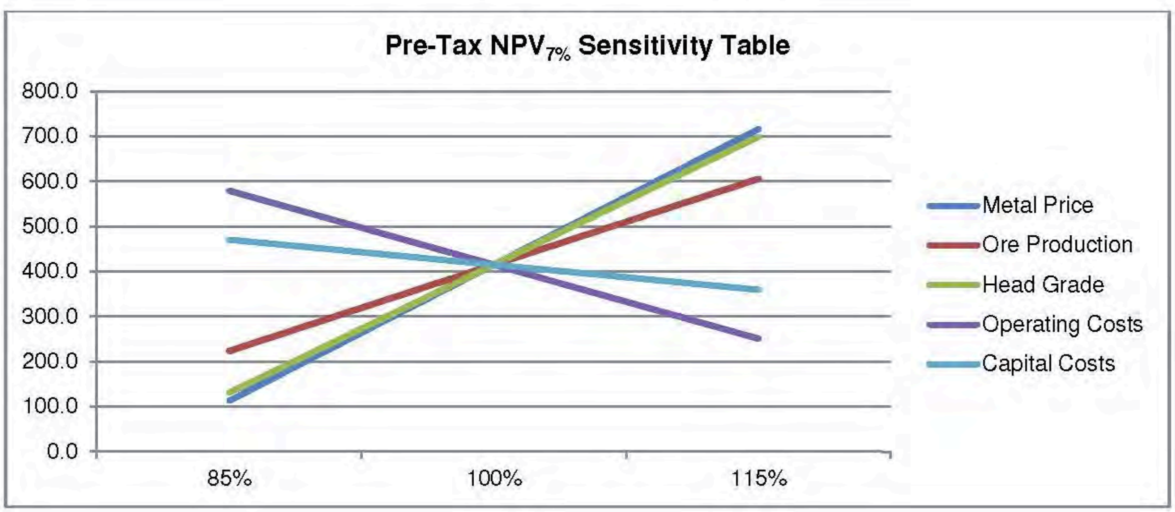
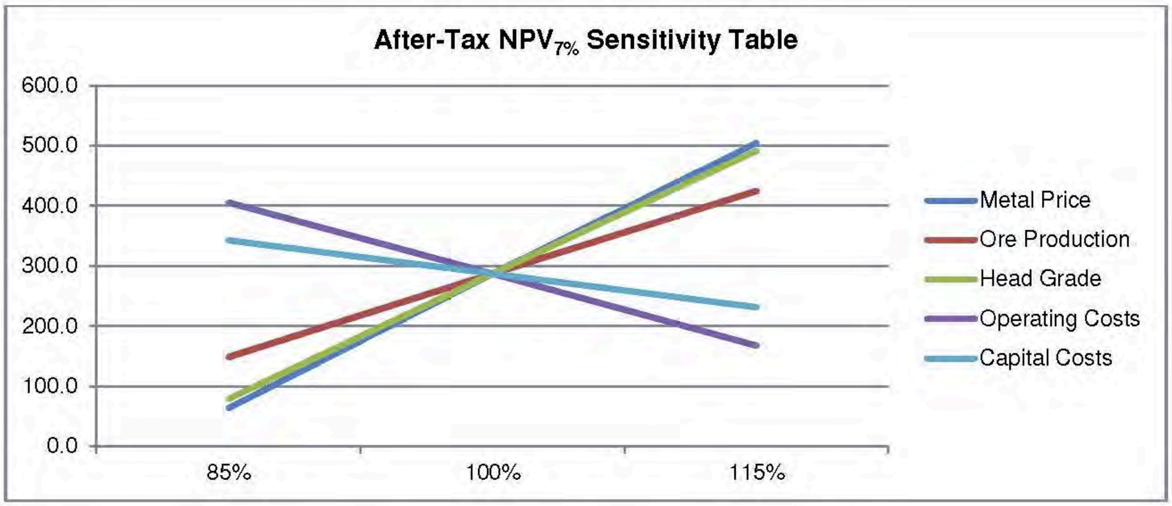


Figure 22-18: After-Tax $NPV_{7\%}$ Sensitivity Results using Whittle Parameter Pricing



22.10 Metal Price Sensitivity Analysis

A sensitivity analysis was performed to test the volatility of the project based on the changes of a specific commodity price in the Base Case calculation.

The prices of copper, gold and silver were each tested to show the changes in NPV and IRR. Tables 22-26 to 22-28 show the results of these sensitivity tests.

Table 22-29 shows the details of the economic model for the Base Case scenario.

Table 22-26: Copper Price Sensitivity Analysis (holding gold and silver prices constant)

Cu Price $/lb Sensitivity	Pre-Tax $NPV_{7\%}$ ($M)	After-Tax $NPV_{7\%}$ ($M)	Pre-Tax IRR	After-Tax IRR
$2.25	$135.0	$78.6	16.8%	12.5%
$2.50	$210.7	$138.2	21.4%	16.3%
$2.75	$286.3	$193.7	25.7%	19.6%
$3.00	$362.0	$248.6	29.6%	22.7%
$3.15	$407.4	$281.5	31.8%	24.5%
$3.50	$513.3	$357.9	36.8%	28.3%
$3.70	$573.8	$401.5	39.5%	30.4%

Table 22-27: Gold Price Sensitivity Analysis (holding copper and silver prices constant)

Au Price $/oz Sensitivity	Pre-Tax $NPV_{7\%}$ ($M)	After-Tax $NPV_{7\%}$ ($M)	Pre-Tax IRR	After-Tax IRR
$750.00	$31.1	$0.7	9.1%	7.1%
$900.00	$139.9	$87.4	16.2%	12.6%
$1,000.00	$212.4	$140.4	20.5%	15.9%
$1,200.00	$357.4	$245.5	28.7%	22.2%
$1,300.00	$429.9	$297.8	32.6%	25.1%
$1,400.00	$502.4	$350.1	36.3%	28.0%
$1,500.00	$574.9	$402.3	39.9%	30.8%

Table 22-28: Silver Price Sensitivity Analysis (holding copper and gold prices constant)

Ag Price $/oz Sensitivity	Pre-Tax $NPV_{7\%}$ ($M)	After-Tax $NPV_{7\%}$ ($M)	Pre-Tax IRR	After-Tax IRR
$12.00	$472.1	$328.2	35.2%	27.0%
$15.00	$477.0	$331.8	35.3%	27.2%
$18.00	$481.9	$335.3	35.5%	27.3%
$20.00	$485.2	$337.7	35.6%	27.4%
$25.00	$493.5	$343.6	35.9%	27.6%
$26.00	$495.1	$344.8	35.9%	27.7%
$29.00	$500.0	$348.4	36.1%	27.8%

Table 22-29: Tepal Cash Flow Summary, Part 1

Geologix Explorations Inc. - Tepal

Cash Flow Summary

	Unit	LOM	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
			-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Metal Price																		
Cu	$/lb	**3.44**	3.44	3.44	3.44	3.44	3.44	3.44	3.44	3.44	3.44	3.44	3.44	3.44	3.44	3.44	3.44	3.44
Au	$/oz	**1,390**	1,390	1,390	1,390	1,390	1,390	1,390	1,390	1,390	1,390	1,390	1,390	1,390	1,390	1,390	1,390	1,390
Ag	$/oz	**26**	26	26	26	26	26	26	26	26	26	26	26	26	26	26	26	26
Production																		
Sulphide Ore	M tonnes	**137.8**	0.0	0.1	0.4	13.1	12.7	11.6	11.5	12.3	12.7	13.9	14.5	12.9	13.6	8.5	0.0	0.0
Oxide Ore	M tonnes	**11.8**	0.0	2.1	1.5	3.8	1.2	0.0	2.0	0.8	0.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Ore	M tonnes	**149.6**	0.0	2.2	1.9	16.9	13.8	11.6	13.5	13.1	13.2	13.9	14.5	12.9	13.6	8.5	0.0	0.0
Total Waste	M tonnes	**267.6**	0.0	7.7	3.4	14.6	21.0	21.0	21.4	25.4	25.3	34.0	32.1	32.9	20.8	8.1	0.0	0.0
Total Mined	M tonnes	**417.2**	0.0	9.8	5.3	31.5	34.8	32.5	34.9	38.5	38.5	47.8	46.6	45.8	34.4	16.7	0.0	0.0
Strip Ratio	w:o	**1.8**	0.0	3.5	1.8	0.9	1.5	1.8	1.6	1.9	1.9	2.4	2.2	2.5	1.5	1.0	0.0	0.0
Daily Mining Rate	tpd	**87,864**	0	26,924	14,579	86,017	95,388	89,097	95,611	105,318	105,439	131,086	127,544	125,100	94,254	45,725	0	0
Head Grade																		
Milled Sulphide Ore (Tepal + Tizate)	**M tonnes**	**137.8**	0.0	0.0	0.0	13.1	12.7	11.9	11.5	12.4	12.8	13.9	14.5	12.9	13.6	8.5	0.0	0.0
Cu	%	**0.20**	0.00	0.00	0.00	0.29	0.23	0.23	0.19	0.18	0.18	0.17	0.19	0.15	0.15	0.21	0.00	0.00
Au	g/t	**0.30**	0.00	0.00	0.00	0.42	0.41	0.47	0.31	0.21	0.21	0.24	0.29	0.16	0.21	0.43	0.00	0.00
Ag	g/t	**1.49**	0.00	0.00	0.00	0.87	0.88	1.02	1.62	2.33	1.88	1.39	1.09	2.17	1.84	1.36	0.00	0.00
Milled Oxide Ore (Tepal + Tizate)	**M tonnes**	**11.78**	0.0	0.0	2.0	2.2	2.2	2.2	2.0	0.8	0.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Cu	%	**0.22**	0.00	0.00	0.24	0.20	0.22	0.24	0.18	0.17	0.26	0.16	0.00	0.00	0.00	0.00	0.00	0.00
Au	g/t	**0.42**	0.00	0.00	0.49	0.44	0.48	0.51	0.26	0.32	0.33	0.26	0.00	0.00	0.00	0.00	0.00	0.00
Ag	g/t	**1.25**	0.00	0.00	0.95	1.06	1.00	0.93	2.21	2.03	0.67	0.46	0.00	0.00	0.00	0.00	0.00	0.00
Sulphide Processing & Recovery																		
Sulphide Flotation																		
Combined Recovery (Tepal + Tizate)																		
Cu Recovery	%	**87%**	0%	88%	88%	88%	88%	88%	87%	86%	86%	87%	88%	86%	86%	88%	0%	0%
Au Recovery	%	**61%**	0%	62%	62%	62%	62%	62%	61%	58%	59%	61%	62%	58%	59%	61%	0%	0%
Ag Recovery	%	**41%**	0%	27%	27%	27%	27%	27%	41%	58%	54%	40%	28%	57%	51%	36%	0%	0%
Contained Metal																		
Cu in Concentrate	M lbs	**523.1**	0.0	0.0	0.0	72.5	56.9	52.2	42.7	43.1	42.9	46.6	53.4	37.4	40.1	35.2	0.0	0.0
Au in Concentrate	oz	**813,527**	0	0	0	110,846	103,037	112,951	68,810	49,255	51,049	64,709	85,255	39,012	55,584	73,019	0	0
Ag in Concentrate	oz	**2,954,940**	0	0	0	100,346	98,555	106,890	245,896	539,399	415,787	244,898	139,490	516,122	412,624	134,935	0	0
Combined Cu Concentrate Grade (Tepal + Tizate)	%	**26.2%**	0.0%	25.7%	25.7%	25.7%	25.7%	25.7%	26.2%	26.8%	26.7%	26.2%	25.7%	26.8%	26.6%	26.0%	0.0%	0.0%
Cu Concentrate Production	dmt	**908,444**	0	0	0	127,966	100,364	92,149	73,901	72,903	72,953	80,740	94,249	63,336	68,449	61,435	0	0
	wmt	**981,120**	0	0	0	138,203	108,393	99,521	79,814	78,735	78,789	87,199	101,789	68,403	73,925	66,349	0	0
Moisture Content	%	**8%**	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%
Au in Cu Concentrate	g/dmt	**27.2**	0.0	0.0	0.0	26.9	31.9	38.1	29.0	21.0	21.8	24.9	28.1	19.2	25.3	37.0	0.0	0.0
Ag in Cu Concentrate	g/dmt	**110.1**	0.0	0.0	0.0	24.4	30.5	36.1	103.5	230.1	177.3	94.3	0.0	253.5	187.5	68.3	0.0	0.0
Payable Cu in Cu Concentrate	M lbs	**503**	0	0	0	70	55	50	41	42	41	45	51	36	39	34	0	0
	tonnes	**228,189**	0	0	0	31,608	24,790	22,761	18,624	18,834	18,740	20,311	23,284	16,349	17,514	15,374	0	0
Payable Au in Cu Concentrate	oz	**789,121**	0	0	0	107,521	99,946	109,562	66,746	47,777	49,517	62,768	82,698	37,841	53,917	70,828	0	0
Payable Ag in Cu Concentrate	oz	**1,847,900**	0	0	0	0	1,577	16,210	157,154	422,175	310,880	150,320	0	409,530	311,943	68,112	0	0
Total Smelter Payable from Cu Concentrate	**$ M**	**2,875.5**	**0.0**	**0.0**	**0.0**	**389.2**	**327.0**	**325.3**	**238.1**	**220.2**	**219.0**	**245.2**	**291.5**	**187.2**	**215.9**	**216.8**	**0.0**	**0.0**
Sulphide Cyanidation																		
Combined Recovery (Tepal + Tizate)																		
Au Recovery	%	**19%**	0%	17%	17%	17%	17%	17%	19%	20%	20%	19%	17%	20%	20%	18%	0%	0%
Ag Recovery	%	**13%**	0%	14%	14%	14%	14%	14%	13%	12%	12%	13%	14%	12%	13%	13%	0%	0%
Metal Production																		
Au Dore	oz	**243,397**	0	0	0	30,731	28,566	31,315	21,174	17,191	17,314	19,728	23,662	13,560	18,482	21,673	0	0
Ag Dore	oz	**851,653**	0	0	0	49,807	48,918	53,055	78,359	114,460	95,990	80,764	68,857	110,830	101,231	49,382	0	0
Payable Au from Sulphide Cyanidation	oz	**243,153**	0	0	0	30,701	28,538	31,284	21,153	17,173	17,297	19,709	23,639	13,546	18,464	21,651	0	0
Ag Payable from Sulphide Cyanidation	oz	**826,103**	0	0	0	48,312	47,450	51,463	76,008	111,026	93,110	78,341	66,792	107,505	98,194	47,900	0	0
Total Payable from Sulphide Cyanidation	**$ M**	**359.5**	**0.0**	**0.0**	**0.0**	**43.9**	**40.9**	**44.8**	**31.4**	**26.8**	**26.5**	**29.4**	**34.6**	**21.6**	**28.2**	**31.3**	**0.0**	**0.0**
Oxide Processing & Recovery																		
Combined Recovery (Tepal + Tizate)																		
Au Recovery	%	**82%**	0%	0%	83%	83%	83%	83%	75%	78%	83%	83%	0%	0%	0%	0%	0%	0%
Ag Recovery	%	**62%**	0%	0%	63%	63%	63%	63%	56%	58%	63%	63%	0%	0%	0%	0%	0%	0%
Metal Production																		
Au Dore	oz	**132,170**	0	0	25,851	25,645	27,981	29,824	12,530	6,174	3,975	189	0	0	0	0	0	0
Ag Dore	oz	**286,724**	0	0	38,617	47,359	43,857	41,623	79,738	29,132	6,146	250	0	0	0	0	0	0
Payable Au from Oxide Cyanidation	oz	**132,038**	0	0	25,826	25,620	27,953	29,794	12,517	6,168	3,971	189	0	0	0	0	0	0
Payable Ag from Oxide Cyanidation	oz	**278,122**	0	0	37,459	45,938	42,542	40,375	77,346	28,258	5,962	243	0	0	0	0	0	0
Total Payable from Oxide Cyanidation	$ M	**190.8**	0.0	0.0	36.9	36.8	40.0	42.5	19.4	9.3	5.7	0.3	0.0	0.0	0.0	0.0	0.0	0.0

Table 22-30: Tepal Cash Flow Summary, Part 2

Geologix Explorations Inc. - Tepal

Cash Flow Summary

			2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
	Unit	LOM	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Total Revenues (Sulphide + Oxide Processing)	$ M	3,425.7	0.0	0.0	36.9	469.9	407.8	412.6	288.9	256.3	251.2	274.9	326.1	208.9	244.1	248.2	0.0	0.0
Offsite Costs																		
Total TC/RC - Cu Concentrate	$ M	75.4	0.0	0.0	0.0	10.4	8.3	7.7	6.2	6.2	6.1	6.7	7.7	5.4	5.8	5.2	0.0	0.0
Total RC - Sulphide Cyanidation	$ M	3.0	0.0	0.0	0.0	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.2	0.0	0.0
Total RC - Oxide Cyanidation	$ M	1.4	0.0	0.0	0.2	0.3	0.3	0.3	0.2	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Treatment & Refining Charges	$ M	79.8	0.0	0.0	0.2	11.0	8.8	8.3	6.6	6.5	6.4	6.9	8.0	5.6	6.1	5.4	0.0	0.0
Net Invoice Value (NIV)	$ M	3,345.9	0.0	0.0	36.6	458.9	399.0	404.3	282.3	249.8	244.8	268.0	318.2	203.3	238.1	242.8	0.0	0.0
Total Transportation Costs	$ M	115.4	0.0	0.0	0.0	16.3	12.8	11.7	9.4	9.3	9.3	10.3	12.0	8.0	8.7	7.8	0.0	0.0
Total Royalties	$ M	80.8	0.0	0.0	0.9	11.1	9.7	9.8	6.8	6.0	5.9	6.4	7.7	4.9	5.7	5.9	0.0	0.0
Total Offsite Costs	$ M	196.2	0.0	0.0	0.9	27.3	22.4	21.5	16.2	15.3	15.2	16.7	19.6	12.9	14.4	13.7	0.0	0.0
Net Smelter Return (Including Royalties)	$ M	3,149.7	0.0	0.0	35.7	431.6	376.6	382.8	266.1	234.5	229.6	251.3	298.5	190.3	223.6	229.1	0.0	0.0
Operating Costs																		
Mining	$ M	603.6	0.0	0.0	0.0	33.7	45.4	48.1	52.0	56.3	60.5	74.0	73.3	73.2	63.2	24.0	0.0	0.0
Processing Cost - Sulphide Flotation	$ M	839.3	0.0	0.0	0.0	79.6	77.2	72.3	70.2	75.7	77.8	84.4	88.2	78.8	83.0	52.0	0.0	0.0
Combined Processing Cost - Sulphide Cyanidation (Tepal + Tizate)	$ M	119.2	0.0	0.0	0.0	11.3	11.0	10.3	10.0	10.8	11.1	12.0	12.5	11.2	11.8	7.4	0.0	0.0
Combined Oxide Milling & Processing Costs (Tepal + Tizate)	$ M	80.4	0.0	0.0	14.7	16.2	15.9	16.1	9.7	4.3	3.3	0.2	0.0	0.0	0.0	0.0	0.0	0.0
Tailings	$ M	5.1	0.0	0.0	0.0	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.0	0.0
G&A	$ M	80.3	0.0	0.0	0.0	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	0.0	0.0
Leasing Cost	$ M	80.9	0.0	0.0	0.0	21.1	19.1	17.3	11.1	10.1	2.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Operating Cost	$ M	1,808.9	0.0	0.0	14.7	169.6	176.2	171.9	160.8	165.0	162.8	178.4	181.8	170.9	165.8	91.1	0.0	0.0
Net Operating Income	$ M	1,340.9	0.0	0.0	21.1	262.0	200.4	210.9	105.3	69.5	66.8	72.9	116.7	19.4	57.8	138.0	0.0	0.0
Capital Costs																		
Mine Equipment	$ M	25.9	0.0	14.9	9.9	0.1	0.0	0.0	0.0	0.0	0.0	1.0	0.0	0.0	0.0	0.0	0.0	0.0
Tailings & Water Management	$ M	76.7	0.0	23.3	11.4	16.0	4.8	7.9	2.5	1.9	0.2	4.2	3.8	0.2	0.2	0.2	0.0	0.0
Site Development, Runway, Roads	$ M	5.0	0.0	4.6	0.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Common Services	$ M	3.4	0.0	1.1	2.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Oxide Leach	$ M	24.5	0.0	11.3	13.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Crushing	$ M	17.5	0.0	9.5	8.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Grinding & Classification	$ M	79.3	0.0	37.4	42.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Flotation	$ M	39.2	0.0	2.4	36.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Mill Buildings & Common Services	$ M	8.5	0.0	4.2	4.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Concentrate	$ M	4.0	0.0	0.1	3.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Buildings	$ M	4.2	0.0	1.6	2.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Powerline, Electrical & Instrumentation	$ M	43.8	7.9	22.2	13.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Salvage Value	$ M	-34.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-34.4	0.0
Indirects	$ M	20.1	0.7	8.2	11.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Closure	$ M	27.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.0	27.2	0.0
Pre-Contingency Capital	$M	358.5	12.1	142.5	167.8	16.1	4.8	7.9	2.5	1.9	0.3	5.2	3.8	0.2	0.2	0.2	-7.2	0.0
Contingency	$ M	38.9	1.1	15.2	15.1	4.9	1.4	0.7	0.2	0.2	0.0	0.5	0.3	0.0	0.0	0.0	-0.6	0.0
Total Capital	$ M	397.4	13.1	157.7	182.9	21.0	6.2	8.6	2.7	2.1	0.3	5.7	4.2	0.3	0.2	0.2	-7.8	0.0
Lease Payments during Pre-Production	$ M	18.9	0.0	10.9	8.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Working Capital	$ M	0.0	0.0	0.0	0.0	29.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-29.4	0.0
Cash Flow																		
Net Cash Flow	$ M	924.6	-13.1	-168.6	-169.9	211.6	194.2	202.4	102.6	67.4	66.5	67.2	112.5	19.2	57.6	137.8	37.2	0.0
Cumulative Net Cash Flow	$ M	0.0	-13.1	-181.8	-351.6	-140.1	54.1	256.5	359.1	426.5	493.0	560.2	672.7	691.9	749.5	887.3	924.6	924.6
Pre-Tax NPV7%	$ M	495.1																
Pre-Tax IRR	%	35.9%																
Pre-Tax Payback Period	Years	2.7																
Corporate Income Tax	$ M	234.4	0.0	0.0	0.0	41.0	43.7	45.8	14.0	2.6	0.0	4.9	25.3	3.3	15.5	38.3	0.0	0.0
Net After-Tax Cash Flow	$ M	690.1	-13.1	-168.6	-169.9	170.6	150.4	156.5	88.6	64.8	66.5	62.4	87.2	15.8	42.2	99.5	37.2	0.0
Cumulative After-Tax Cash Flow	$ M	0.0	-13.1	-181.8	-351.6	-181.0	-30.6	125.9	214.5	279.4	345.8	408.2	495.4	511.3	553.4	652.9	690.1	690.1
After-Tax NPV7%	$ M	344.8																
After-Tax IRR	%	27.7%																
After-Tax Payback Period	Years	3.2																

23 Adjacent Properties

The following section is taken from Micon Technical report, March 29, 2012

Micon is unaware of any mineral exploration or mining in adjacent properties.

The closest active exploration property is La Verde. This porphyry copper deposit is owned by Catalyst Copper Corporation and is approximately 95 km due east of the Tepal property. There are two deposits on the property (West and East Hill). It has a Measured and Indicated Mineral Resource of 354Mt grading 0.41% Cu and 0.043g/t Au and 2.3g/t Ag at a cut-off of 0.2% Cu. There is an additional Inferred Mineral Resource of 168Mt grading 0.41% Cu, 0.058g/t Au and 2.3g/t Ag at a cut-off of 0.2% Cu. This is a global in-situ mineral resource not constrained to an economic pit (Catalyst Copper News Release, January 20, 2012).

The Cerro Pelon deposit on the San Isidro porphyry copper property is 115km southeast of the Tepal property. The property was owned by Aquiline Resources Inc. in the 1990s. The property has been drilled and there are coincidental geophysical and geochemical anomalies that have defined the Cerro Pelon deposit. The latest data indicates that the deposit as exposed on surface is 500 by 200m and extends to at least 300m depth.

ASARCO (now Grupo Mexico or GMEXICO) mined several breccia bodies at Inguaran from 1971-1982 and extracted some 7Mt of ore grading 1.2% Cu (Osoria et al., 1991). Gold, silver and tungsten were bi-products in the concentrates. The property is presently owned by Rome Resources Ltd. of Surrey, British Columbia. The Inguaran Copper Mine is 140km southeast of the Tepal property.

24 Other Relevant Data & Information

24.1 Project Execution

24.1.1 Introduction & Philosophy

The project execution plan for the Geologix Tepal project is based on principles tested and proven in the development projects in Mexico. These principles include:

- Safety in design, construction and operations is paramount to success
- Simple, passive environmental solutions; minimizing disturbance footprint
- Fit-for-purpose design, construction, and operation
- Due to the high cost of transportation, consolidate construction and operational needs to the extent practical
- Common equipment fleet purchased by Owner at onset and used for construction needs to the extent practical
- Efficient operations; minimize site labor requirements
- Negotiated contracts with suppliers, contractors, and engineers with proven track records in Mexican mine developments
- No nonsense project management; decisive decision making
- Early completion of project components turned over to operations
- Elimination of superfluous management organizations.

24.1.2 Project Management Team

Project management would be provided by an integrated team comprised of the Owners project management personnel and the project management consultant (PM consultant). The project management team (PM team) would oversee the detailed engineering, procurement, and construction management activities for the project. The PM team would also coordinate the work of the engineering subcontractor and other specialized consultants as required.

The PM team would be responsible for all project activities from detailed design through to commissioning and turnover to operations. The PM team would be available to backstop the operations teams with key supervision and management assistance when the operations personnel assume control of project components as they are completed.

24.1.3 Project Procedures

The PM team would prepare and publish a project procedures manual (PPM) early in the development of the project. This manual would describe standard project templates, procedures, and forms for use in the engineering, procurement, construction, and project disciplines. Some of the major procedures are listed below for reference:

- Engineering
  - Engineering section would be supplemented by procedures utilized by the selected engineering contractor(s).
- Procurement
  - Designation of authority guideline
  - Purchase order & contract execution procedure
  - Purchase order & contract change procedure
  - Invoice approval and payment procedures.
- Logistics
  - Procedures as required to support the freight and logistics plan.
- Construction
  - Quality assurance procedures
  - Health and Safety procedures
  - Environmental procedures.
- Project Controls
  - Project change procedure
  - Project cost procedures
  - Project schedule procedures
  - Project risk procedures.

24.1.4 Project Controls Systems

In keeping with the “fit for purpose” execution philosophy, a suitable owner approved cost and budget control system with minimum complexity would be utilized. As the Owner is embedded into the PM team, it is envisioned that project reporting would be concise and contain pertinent project progress information only. Project reporting would track budget, committed, actual and forecasted quantities and costs. Earned value would be implemented as required for specific critical sub-projects only (i.e. concrete installation or building erection).

The project management team would utilize Primavera as the primary scheduling software. All scheduling would be performed utilizing the critical path method (CPM).

24.1.5 Procurement Strategy

In general, the PM consultant would oversee the selection and tendering of all tagged equipment and bulk materials and commodities as a function of managing the engineering sub-contractor. Tagged equipment is defined as uniquely designed and engineered equipment and assemblies required for the project as documented in the project equipment lists. Bulk materials are not generally specifically engineered items and are not identified on the project equipment list. All bulk materials for the project would be purchased, tracked and referenced to applicable specifications and standards.

Construction bulks to support the initial site activities would need to be factored based on estimated quantities and procured by the Owner to meet the mobilization timeline of Jan 1 2014, given that detailed engineering may not be sufficiently advanced to generate MTO’s.

Process equipment considered to be “long delivery,” would have to be selected and conditionally committed to earlier than required by site delivery schedules, in order to receive the vendor’s certified drawings and allow detailed design of the civil and structural components of the project to be completed in a timely manner to receive the equipment. Some long-lead delivery equipment and materials are targeted for delivery at the project site in early & mid-2014; these items would be identified during detailed engineering in order that commitments could be made sufficiently in advance.

24.1.6 Freight & Logistics

A detailed Freight and Logistics Plan would be developed for the project. The plan would address the requirements for ocean freight, air freight and truck freight, as well as personnel transport to support the project schedule.

24.1.7 Contracting Strategy

The contracting strategy would be established by the PM team at the onset of the project, which would address each contract battery limit, detailed scope of work and the cost structure of each. Contract work packages would be divided into manageable scopes, and awarded to contractors “best fit” for the work. Contractors would be pre-qualified by the PM team based on their ability to execute the work in a safe and efficient manner, as demonstrated by past performance. Opportunities for qualified local contractors would be given consideration when determining the work packages, providing that they can meet bid requirements and are available to provide value to the project through competitive pricing.

Contracts that extend into operations, such as camp catering, would be structured in conjunction with the Owner’s operations personnel to ensure that operational needs are properly addressed.

24.1.8 Development Schedule

Metallurgical testing would continue immediately, as well as field geotechnical and hydrogeological programs to support feasibility study. It is anticipated a feasibility study would be completed by Q4 2013/Q1 2014. The length of time required for detailed engineering has some flexibility and can be staged following the test-work or can run parallel to the feasibility.

The construction schedule would be premised on the Owner procuring and engaging their operations equipment fleet upon commencement of construction, supplemented by contractor’s equipment as required to execute the schedule. The anticipated construction schedule should take 24 months or less as shown in Figure 24-1.

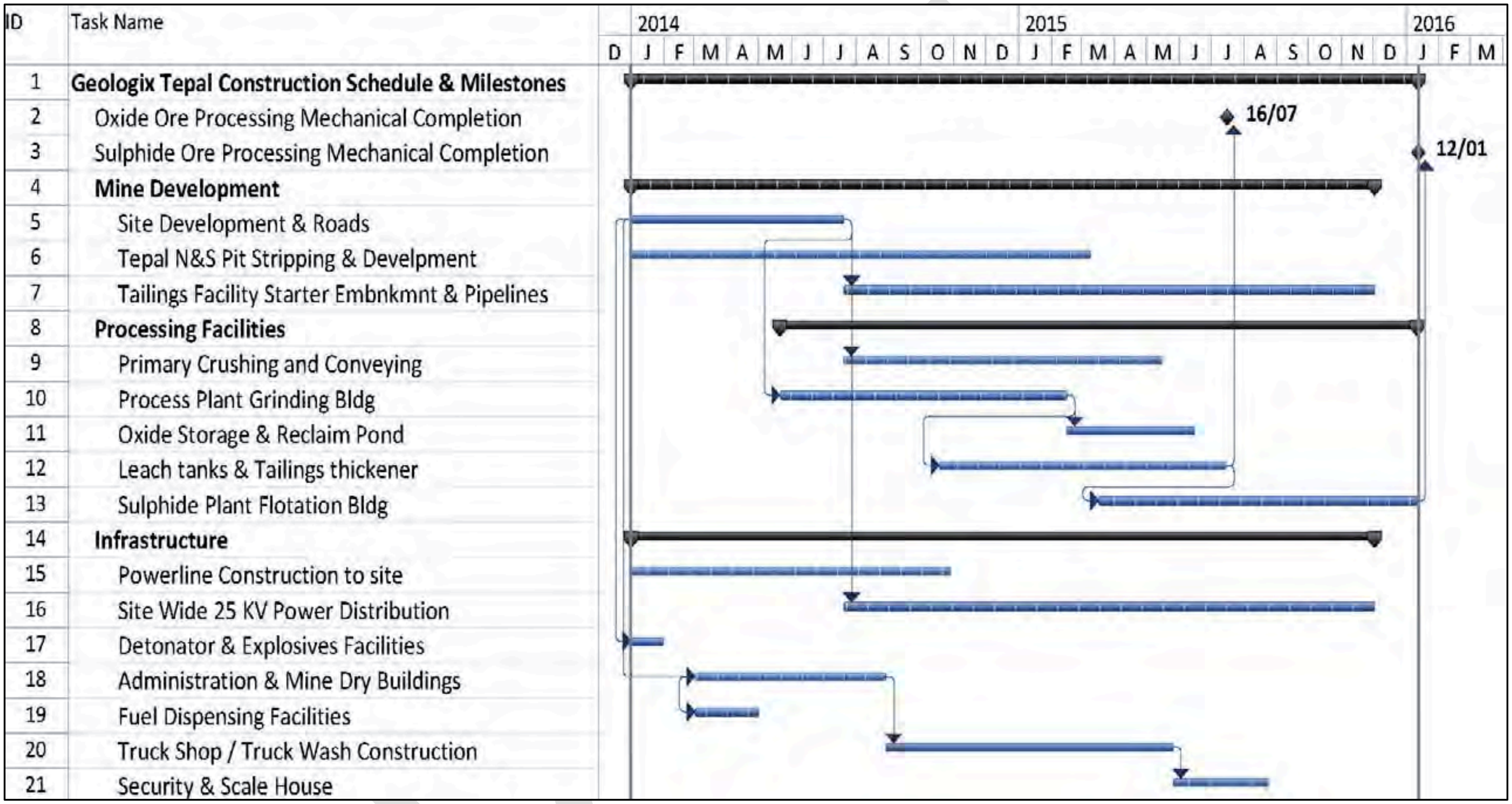
The general site earthworks, mine pit overburden stripping and oxide ore stockpiling would be executed simultaneously during the first 6 months of construction, and staged to be completed as required to support the schedule milestones. The first phase of TSF would commence as soon as practical in the construction phase to ensure its readiness to receive waste in the first quarter of 2016. The milestones that would need to be achieved are oxide ore processing in Q3 of 2015, followed by sulphide ore processing by Q1 2016.

The mine infrastructure construction activities would commence in parallel with the site development activities: the explosives storage facility, administration and mine dry facilities should commence as soon as possible to support the construction activities. The power line to site and site power distribution systems would also need to commence as soon as practical after project financing approval to ensure the success of the schedule.

The construction of the process facilities would be staged in three phases: the grinding circuit, followed by the leaching and tailings circuit to support the oxide processing by Q3 2015, with the flotation circuit to process sulphide ore by Q1 2016.

Due to the site's seasonal rainfall and dry climate, the water supply system construction timing would have to be adjusted to ensure that water would be available for start-up. This construction can also be staged with early construction of the fresh water storage pond to catch enough water for a partial year's production.

Figure 24-1: Summary Project Execution Schedule



25 Interpretations & Conclusions

25.1 Mineral Resources

Industry standard exploration practices have been used to evaluate the Tepal Project. There is adequate geological and other pertinent data available to have generated this new mineral resource estimate.

The current mineral resource estimate has expanded the tonnage of the deposits. It has also upgraded the interpolated mineralization into higher confidence mineral resource categories in the Tepal North and South deposits. This is due primarily by the definition and delineation drilling completed in 2011.

There is a future opportunity to expand the Geologix mineral inventory below the South Zone pit. Higher-grade mineralization has been discovered by deep drilling, below the soft pit. This Inferred mineral resource may be economically extracted by utilizing underground mining methods, if future economic studies and programs, warrant it.

The preliminary feasibility study has shown the project has positive economics. Micon feels that the extra cost of in-fill drilling of the Tizate zone to upgrade the mineral resource categories will only add a minor increase in the NPV and therefore is not warranted.

Mining of the three deposits will most likely add some peripheral inferred mineral resource blocks to the production and increase the economics of the project.

25.2 Mineral Reserves

Mineral Reserves have been modified from Mineral Resources by including geotechnical, mining, processing and economic factors. The Mineral Reserves are a subset of the Mineral Resources and are summarized in Table 15-1 of Section 15. The stated Reserves conform to CIM guidelines.

This study did not identify any mining, metallurgical, infrastructure or other relevant factors that may materially affect the estimate of Mineral Reserves.

25.3 Mining Methods

Tepal is proposed to be a conventional open pit mining operation.

Pit designs for Tepal North, South and Tizate were developed based on the results of Whittle open pit optimization and incorporate geotechnical recommendations provided by Knight Piésold Ltd. The pits would be mined in sequence targeting the highest value ore first to facilitate early capital payback and improve project economics.

The mine plan is based on a fleet of diesel equipment supplied by Caterpillar (Tracsa), Mexico. The fleet would include: 6050 hydraulic shovels, 994H and 992K wheel loaders, 789D trucks, MD6540 rotary drills and D10T track dozers. A waste stripping contractor would supplement the mine fleet in Years 6 to 10.

Two years were allocated for construction of both the mill and site infrastructure. Mining at this time would be focused on supplying non-acid generating waste to construct the starter tailings dam and preparing the pit for full-scale operation. Oxide milling wouldl commence during the latter half of the second construction year. Commissioning of the sulphide circuit at design capacity would be completed at the end of the second construction year. Production would begin immediately afterwards and continue for a total of 11 years.

A total of 11.8Mt of oxide and 137.8Mt of sulphide ore and 267.6Mt of waste are estimated to be mined at an average daily rate of 88,000tpd. The life of mine stripping ratio is expected to be 1.8:1 waste to ore. The mine would have two waste types: non-acid generating (NAG) and potentially acid generating (PAG). 68.7Mt of NAG would be delivered to and placed as part of the construction of the tailings storage facility. The remaining NAG and PAG waste would be stored in engineered dumps adjacent to the open pits.

Tepal project would employ industry standard mining methods and equipment.

25.4 Metallurgy

There are three sources of gross metal value (GMV) from the Tepal resources. They are chalcopyrite (copper sulphide with interstitial gold and silver) in a quartz matrix, an iron pyrite (iron sulphide with interstitial gold and silver) encased in a secondary quartz/gangue matrix, and a surface oxide layer containing copper minerals (in decreasing amounts; tenorite, malachite, azurite and covellite) which also contain gold and silver values.

Currently three pits are planned to be mined that include the North Zone (NZ), South Zone (SZ) and Tizate Zone. Metallurgically, the NZ and SZ can be considered similar (grade and hardness) while Tizate should be considered slightly different (grade and hardness).

The metallurgical testwork performed for this prefeasibility study has followed industry standard practices. Geologix commissioned grinding, flotation, leaching and heap leach leach tests to support this study. G&T Metallurgical Services in Kamloops, Canada conducted the sulphide flotation and leaching recovery tests while McClelland Laboratories Inc. in Sparks, USA performed the oxide cyanide leach and column tests.

The concentrates had minor element assays performed to determine if any deleterious elements would diminish the value when calculating a Net Smelter Return (NSR) for this resource. No fatal flaws or deleterious elements were identified.

The copper concentrate is unusually clean owing to the quartz matrix containing the chalcopyrite. There is good separation between chalcopyrite and pyrite due to the faster chalcopyrite flotation kinetics. Fortunately there is little contamination of pyrite in the copper concentrate, which should make the concentrate easy to market.

25.5 Processing

Based on the metallurgical results, a standard copper concentrator equipped with SAG, ball mills, copper rougher flotation, copper cleaning regrind and flotation, pyrite flotation, pyrite CIL, carbon plant and refinery is proposed. There is also an oxide circuit to recover gold and silver ina CIL circuit, carbon plant and refinery.

The saleable products are a copper concentrate with gold and silver values from copper flotation and a gold/silver doré bar from the ADR plant.

The proposed process plant employs industry standard design and equipment.

25.6 Tailings

The Tepal project would generate a total of approximately 150Mt of tailings that would be stored in two separate tailings storage facilities (TSF). Tailings Storage Facility 1 (TSF1) would be located approximately 2km northwest of the plant site and is designed to store a total of approximately 120Mt of tailings, process water, surface runoff, and incident precipitation. The remaining 30Mt of tailings would be deposited into the inactive North Pit, which would become Tailings Storage Facility 2 (TSF2) once tailings deposition begins in the pit.

The final TSF embankment will be approximately 112m high and 2250m long, with a total fill volume of approximately 33.5Mm^3. Oxide waste rock from the North and South Pits would be used for the construction of the TSF embankment shell zones (32.3m^3) and processed material will be used for the construction of the drainage zones (1.2Mm^3).

Construction would be staged to minimize capital expenditure and defer costs where possible. The starter facility will provide adequate capacity for start-up water collection. Four additional stages (Stage 2 through 5) of construction would occur at 2-5 year intervals over the mine life. Closure of the TSF will include progressive capping of the facility with oxide waste rock and topsoil.

The proposed TSF employs industry standard design methods.

25.7 Environment and Social

Consultation, social and environmental studies and programs are proceeding in concert with the project stage. There are no environmental or social risks identified that would prevent project permitting. Continued waste characterization and water management would be essential components in ongoing project planning to minimize long-term liabilities for the project.

25.8 Capital & Operating Costs

The capital cost estimate was prepared using first principles, applying project experience and avoiding the use of general industry factors. The estimate is derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in the study. Given that assumptions have been made due to a lack of available engineering information, the target accuracy of the estimate is ±25%.

Costs are expressed in US dollars with no escalation unless stated otherwise. Foreign exchange rates of CDN$1.00:US$1.00 and MX13.00:US$1.00 are used where applicable.

The operating cost estimate was prepared using first principles, applying project experience and avoiding the use of general industry factors. Inputs are derived from engineers, contractors and suppliers who have provided similar services to other projects. In addition, input was provided by Geologix personnel, based on their valuable experience working in Mexico.

Operating costs in this section of the report include mining, processing, tailings and administration up to the production of concentrate from the site. Costs are expressed in US dollars with no escalation unless stated otherwise. Foreign exchange rates of CDN$1.00:US$1.00 and MX13.00:US$1.00 are used where applicable.

25.9 Financial Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. Geologix commissioned PwC in Vancouver, BC to prepare a tax model for the post tax economic evaluation of the project with the inclusion of applicable Mexican income taxes.

Four metal price scenarios were evaluated to estimate the economic value potential of each and use the results as a comparative tool to better understand the value drivers in each scenario. In addition, the economic model was tested using leased mining fleet and a no-lease, up-front capital outflow during pre-production and year one of production for the mining fleet. All costs, economic results and metal prices are reported in US dollars (USD).

The economic modeling developed for this project followed industry standard methods.

25.10 Conclusions

The financial analysis of the prefeasibility study demonstrates that the project has positive economics and warrants consideration for advancement to feasibility level engineering by Geologix.

Standard industry practices, equipment and processes were used in this study. The Qualified Persons for this report are not aware of any unusual significant risks or uncertainties that could affect the reliability or confidence in the project based on the data and information available to date.

25.11 Risks

As with most mining projects, many risks could affect the economic outcome of the project. Most of these risks are external and largely beyond the control of the project proponents. They can be difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. Table 25-1 identifies what are currently deemed the most important internal project risks, potential impacts, and possible mitigation approaches, excluding those external circumstances that are generally applicable to all mining projects (e.g., changes in metal prices, exchange rates, smelter terms, transport costs, investment capital availability, government regulations, local and regional project support, etc.).

Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

Table 25-1: Preliminary Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Water Supply	A source of make-up water supply has been identified but not fully investigated. A lack of water supply could delay project start-up or cause cost overruns.	Continued collection and analysis of data relating to ground and surface water needs to be continued and, in particular, a well drilling and testing program needs to be undertaken to ensure an adequate supply. Additional wells could be drilled or alternate sources of water found to mitigate potential shortfalls
Water Geochemistry	If ML/ARD testing indicates that the waste dumps require lining or special treatment then the CAPEX costs would increase.	Additional testing and modeling of the geochemical water balance is needed to better define water management strategies.
PAG Rock	The volume of PAG rock has been estimated based on available information. If the volume of PAG rock increases with more testwork and modelling the costs could increase.	Further definition of PAG rock.
Resource Modeling	Resource volumes that were estimated using industry standard methods, but are still subject to some variation. Variability of grade and discontinuity of orebodies can be the biggest issues of a resource model that is not representative of the orebody.	Good grade control, careful mapping and regular resource model reconciliations can significantly reduce the risk of an un-representative model.
Metallurgical Recoveries	The metallurgical recoveries in this study are based on numerous tests but results may vary when the actual orebody is mined. A drop in recoveries would have a direct impact on the project economics.	Continued testwork and optimization during the plant operations would help improve recoveries if results are below expectations.
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of project success. A significant increase in OPEX would reduce the after tax NPV and may adversely affect the project economics. If OPEX increases then the mining cut-off grade would increase and, all else being equal, the size of the reserves would decrease. A significant increase in CAPEX would reduce the after tax NPV and may adversely affect the project economics.	Well-developed and controlled construction and operating plans, along with experienced personnel will greatly mitigate potential cost overruns.
Development Schedule	The project development could be delayed for a number of reasons and a change in schedule would alter the project economics.	Well-developed and controlled construction and operating plans, along with experienced personnel would greatly mitigate potential schedule overruns. Contingency planning would be conducted for project execution to help mitigate

		variances.
Permits	The ability to secure all of the permits to build and operate the project is of paramount importance. Failure to secure the necessary permits could stop or delay the project.	The development of close relationship with the communities and government along with a thorough Environmental and Social Impact Assessment and a project design that gives appropriate consideration to the environment and local people is required.
Geotechnical	The geotechnical nature of the open pit wall rock, including the nature of faults and secondary geological structures could impact pit slopes. Pit slopes could be increased or decreased and thus alter the pit designs, mineable tonnes, and strip ratio.	Further field investigations may be advisable for the next level of study.
Ability to Attract Experienced Professionals	The ability of Geologix to attract and retain competent, experienced professionals is a key success factor for the project. High turnover or the lack of appropriate technical and management staff at the project could result in difficulties meeting project goals.	The early search for professionals as well as the potential to provide living arrangements other than in a camp may help identify and attract critical people. A well-planned, comprehensive training program for local people would help increase the local content and likely improve employee retention.

25.12 Opportunities

Significant opportunities exist that could improve the economics, timing, and/or permitting potential of the project. Most of these opportunities are also potential risks, as explained in the previous section. For example, metallurgical recoveries present both a risk and opportunity.

Opportunities not previously mentioned are presented in Table 25-2, excluding those that are typical to all mining projects, such as increases in metal prices. Further information and evaluation is required before these opportunities can be included in the project economics.

Table 25-2: Project Opportunities

Opportunity	Explanation	Potential Benefit
High Potential Benefit		
Exploration Potential	The expansion of known mineral resources and the addition of new deposits may be possible with further resource drilling and could potentially extend mine life. Based on preliminary geophysical results, the Tepal area has several exploration targets that justify drilling.	The expansion of the deposit resources could potentially lead to a longer project life and/or greater operating flexibility and potentially higher throughput justification. This becomes particularly important if higher grade mineral resources are defined that defer lower grade mineral resources currently utilized in the economic analysis.
Project Strategy and Optimization	Typically, feasibility study mine planning and scheduling can be improved upon with detailed engineering. In addition, leasing financing, streaming and other financial factors can be improved with further investigation.	Detailed optimization of the mine plan could result in improved economics.
NAG Rock	The volume of NAG rock has been estimated based on available information. If the volume of NAG rock increases with more testwork and modelling CAPEX and OPEX costs would decrease.	More NAG rock available to be placed in the tailing facility dam and in waste rock piles around the site would likely reduce CAPEX and OPEX.
Tailings Dam Construction using PAG rock	The entire tailings containment dam construction is done using NAG rock and, as such, a higher level of pre-stripping is required to provide sufficient NAG material to the dam.	If PAG rock could be used to some degree in dam construction, some pre-stripping could be delayed, thus delaying some CAPEX.

26 Recommendations

The financial analysis of the prefeasibility study demonstrates that the project has positive economics and warrants consideration for advancement to feasibility level engineering by Geologix.

26.1 Mining Methods

26.1.1 Geotechnical and Hydrogeology

- Additional hydrogeological geotechnical site investigations should be carried out at the open pits and tailings storage facility area to support the feasibility level design.
- A hydrogeological site characterization should be carried out to better constrain the following aspects of the design:
    - Open pit groundwater inflows during operations and post-closure.
    - Groundwater supply well field location and design.

26.1.2 Geotechnical

A component of mining Tepal North Phase 1 pit is to construct a temporary haul road on the west highwall that is used to access the second phase of mining in that pit. The haul road is mined out as part of Tepal North Phase 2. An additional geotechnical investigation is recommended to ensure that the highwall is stable given the temporay haul road.

26.1.3 Blast Pattern Design

A uniform blast pattern design is used for all pits and both oxide and sulphide ore and waste. There may be an opportunity to reduce the overall powder requirement by tailoring the blast design to specific rock characteristics and steepen the final slope angle given good highwall control blasting by developing specific blast pattern designs and explosives loading requirements for the different pits and rock types and highwall control..

26.1.4 Tailings Dam Design

The current tailings storage facility design is based on the use of only NAG material in the construction. The opportunity to use some PAG material in the dam construction would defer stripping until later in the mine life and improve overall project economics.

26.2 Processing and Metallurgical Testing

Additional process studies are being considered during the next phase of project development. Further sulphide and oxide process tests are required to obtain data in order to complete the design of unit operations.

The mine schedule in this PFS assumes a combined ore from the North, South and Tizate zones, but uses separate processing testwork data. This would have to be updated once the NZ/SZ: Tizate feed ratio is calculated by the mine schedule and a representative composite can be made.

The current PFS has determined that there are potential opportunities to optimize the economics of the resource. This includes flotation of the sulphide mineralization's chalcopyrite mineral then selling the chalcopyrite concentrate (with copper, gold and silver credits), a flotation of the sulphide mineralization's pyrite mineral followed by CIL to recover gold and silver and an oxide cyanide leach to recover gold and silver from the oxide. The loaded carbon from both CIL circuits would feed a common carbon plant and refinery to produce doré.

Silver recovery from the pregnant solution requires further investigation. If sufficient silver is recovered to solution, a Merrill Crowe circuit would be required to replace the carbon circuit. Since this is typically more expensive, a rough cost analysis should be performed before testing for an ADR or Merrill Crowe unit.

26.2.1 Sulphide Process

- Operating costs for ball and mill/crusher liner consumptions require abrasion tests on nominal and worst case (Tizate) mineralization. Gyratory Crusher work indexes are required since the mineralization is usually hard which requires authentication of crusher rates.
- Tailing characterization, including pumping characteristics, settling rates (if a thickener is to be used as per the water balance recommendations), final compaction for water release, etc.
- Dewatering the concentrates that requires thickener settling tests combined with filter tests to meet the design criteria of less than 8% water. Currently there are no data on dewatering.
- Auto ignition tests are recommended if a high sulphur concentrate is to be produced.
- Water chemistry on recirculating, steady state water streams to determine if there are any problem substances such as carbonate, pH, oxidation reduction potential, etc. The water balance may be an issue since there are wet and dry seasons as outlined in KP's water report, with the majority of makeup water coming from wells.

26.2.2 Pilot Plant

Along with re-testing chalcopyrite recovery, it is recommended that large scale bulk tests or a pilot plant be run to check the recovery of molybdenum from the chalcopyrite concentrate. A significant amount of concentrate would be produced from the pilot plant that would enable testing the further separation of molybdenum minerals from the chalcopyrite concentrate. Previous bench tests did not produce enough product to test this separation.

A pilot scale test would also produce enough pyrite concentrate and copper cleaner tails (both combined makes the pyrite leach feed) to test the leach, adsorption, and refining of this part of the gold recovery.

26.2.3 Oxide Mineralization

Further metallurgical tests are needed in order to make the decision on processes to recover gold from the oxide mineralization. It is unclear at this point whether a pilot run is required since the oxide mineralization is variable. Therefore it should be determined whether a large representative composite is practical.

Comminution tests to determine potential bottlenecks such as crusher packing and overgrinding to slimes in the SAG and ball mills. If excessive slims are generated, the effects on dewatering and pond containment will require further study.

Agitated tank leach data is required. At present only bottle roll tests have been completed on the oxide ore (along with column leach tests not being considered in this PFS). Recovery kinetics are required to determine tank sizes, etc. A closed circuit tank pilot should be considered to determine the pregnant solution metals assay.

The ADR plant recovery of metal in solution to a doré bar requires further testing to design solution flows, carbon loading, fouling by copper or other deleterious compounds, etc. Note that the oxide leach pregnant solution would be combined with the pyrite leach solution. The ratio of each would need to be determined. It is recommended a vendor be contacted for input when developing the tests.

26.2.4 Miscellaneous

- Optimize cyanidation leaching kinetics for sulphide and oxide mineralization; residence time, CN concentration, percent solids, etc.
- Investigate and test cyanide destruction.
- Test the SART system for the removal of copper from cyanide solution.

26.3 Environment and Social

- Additional testwork is recommended to help further define the potential for acid generation and metal leaching from waste and tailings and refine segregation and mining sequencing strategies. Waste rock testwork should include synthetic precipitation leaching, meteoric water mobility leaching, and humidity cell tests with samples chosen based on current results. Tailings testwork should include leaching tests and humidity cell tests on samples from future metallurgical testing.
- It is recommended that a Security Risk Assessment be completed during the project feasibility stage so that appropriate costs can be included in the financial analysis, that security plans can be developed, and that future financiers' requirements would be satisfied.
- It is recommended that Geologix continue to move forward with land acquisitions, consultation programs and environmental studies for permitting and potential international financing when the project moves to the feasibility stage. The estimated budget for these programs shown in Table 26-1 does not include land purchase or lease costs.

26.4 Feasibility Study

Conduct a feasibility–level study on the Tepal Project, based on the current prefeasibility design, suitable to support both financing and detailed engineering.

26.5 Estimated Cost of Recommended Work Programs

The estimated cost of the next stage of work is presented in Table 26-1.

Table 26-1: Summary of Estimated Costs of Recommended Work Programs

Item	Cost in US$
Mining Methods	**515,000**
Geotechnical and Hydrogeology Study	300,000
Geotechnical Evaluation for High wall Stability	100,000
Blast Pattern Design	15,000
Tailings Dam Design	100,000
Processing and Metallurgy	**480,000**
Testing for ADR and/or Merrill Crowe	5,000
Sulphide Process Testing	100,000
Pilot Plant	250,000
Oxide Mineralization Testing	50,000
Miscellaneous	75,000
Environment and Social	**2,125,000**
Additional Testwork	100,000
Security Risk Assessment	25,000
Environmental Studies, consultation, land acquisition	2,000,000
Feasibility Study	**1,000,000**
TOTAL	**4,120,000**

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Appendix A – Qualified Person Certificates

CERTIFICATE OF AUTHOR

I, Matt R. Bender do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report on the Prefeasibility Study of the Tepal Project, Michoacán, Mexico", with an effective date of March 19, 2013 prepared for Geologix Explorations Inc.;

2. I am currently employed as Director of Metallurgy with JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver BC V6C2T6;

3. I am a graduate of Colorado School of Mines, Golden, CO USA with a BSc. in Metallurgical Engineering, 1987. I have practiced my profession continuously since 1987;

4. I am a Registered Professional Metallurgical Engineer in Nevada and Colorado, USA;

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 independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

7. I have not visited the Tepal Project site;

8. I am responsible for and/or shared responsibility for Sections 1, 2, 3, 4, 13.4, 13.5, 19, 21, 22, 24, 25, 26, 27;

9. I have had no prior involvement with the property that is the subject of this Technical Report;

10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading; and

11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: March 19, 2013
Signing Date: April 30, 2013

"Signed and Sealed"

Matt R. Bender

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CERTIFICATE OF AUTHOR

I, Michael E. Makarenko do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report on the Prefeasibility Study of the Tepal Project, Michoacán, Mexico", with an effective date of March 19, 2013 prepared for Geologix Explorations Inc.;
2. I am currently employed as Project Manager, Mining with JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver BC V6C2T6;
3. I am a graduate of University of Alberta with a BSc. in Mining Engineering, 1988. I have practiced my profession continuously since 1988;
4. I am a Registered Professional Mining Engineer in Alberta and the Northwest Territories;
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 independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;
7. I have visited the Tepal Project site in September 4-6, 2012;
8. I am responsible for and/or shared responsibility for Sections 15 and 16, excluding Sections 16.2 and 16.3;
9. I have had no prior involvement with the property that is the subject of this Technical Report;
10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading; and
11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: March 19, 2013
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CERTIFICATE OF AUTHOR

I, Mark W. Dobbs do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report on the Prefeasibility Study of the Tepal Project, Michoacán, Mexico", with an effective date of March 19, 2013 prepared for Geologix Explorations Inc.;
2. I am currently employed as the Tucson Division Manager of Allnorth Americas LLC, with a business address at 3556 E Lizard Rock Place, Tucson, Arizona, 85718;
3. I am a graduate of the University of British Columbia in Mechanical Engineering (1993). I have practiced my profession continuously since 1993 and have worked in Mining and Mine Plant Design for the last seven (7) years.
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #22906) and Professional Engineers Ontario (License #100115013)
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 independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;
7. I have visited the Tepal Project site on January 9 to 13, 2012;
8. I am responsible for and/or shared responsibility for Sections 17 and 18, excluding Sections 18.3 and 18.4;
9. I have had no prior involvement with the property that is the subject of this Technical Report;
10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading; and
11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: March 19, 2013
Signing Date: April 30, 2013

"Signed and Sealed"

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micon INTERNATIONAL LIMITED | mineral industry consultants

CERTIFICATE OF AUTHOR

I, David Kent Makepeace do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report on the Prefeasibility Study of the Tepal Project, Michoacán, Mexico", with an effective date of March 19, 2013 prepared for Geologix Explorations Inc.;

2. I am currently employed as a Senior Geologist with Micon International Limited, 205 – 700 West Pender Street, Vancouver, British Columbia, V6C 1G8;

3. I am a graduate of Queen's University with a B.A.Sc. in Geological Engineering in 1976 and a graduate of the University of Alberta with an M.Eng. in Environmental Engineering in 1994. I have practiced my profession continuously since 1976.;

4. I am a Registered Professional Engineer in Alberta (29367) and British Columbia (14912);

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 independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

7. I have visited the Tepal Project site in January 8 to 12, 2012;

8. I am responsible for Section 5, 6, 7, 8, 9, 10, 11, 12, 14, 20, 23, 25.1, 25.7 and 26.3 and share in the responsibility of Section 25.11 and 25.12.

9. I have had no prior involvement with the property that is the subject of this Technical Report;

10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading; and

11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: March 19, 2013
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CERTIFICATE OF AUTHOR

I, Michael Godard do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report on the Prefeasibility Study of the Tepal Project, Michoacán, Mexico", with an effective date of March 19, 2013 prepared for Geologix Explorations Inc.
2. I am currently employed as President, Godard Mineral Processing Ltd, 5399 Patrick Street, Bunaby, BC, V5J 3B2, and Associate to Micon International Ltd, Suite 205, 700 West Pender Street, Vancouver, BC, V6C 1G8.
3. I am a graduate of Unversity of British Columbia with a BaSc in Metallurgicl Engineering, (1985). I have practiced my profession continuously since then.
4. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia.
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 independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101.
7. I have visited the Tepal Project site in January 8 to 13, 2012.
8. I am responsible for and/or shared responsibility for Section 13 (exclusive of 13.4 and 13.5).
9. I have had no prior involvement with the property that is the subject of this Technical Report.
10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: March 19, 2013
Signing Date: April 30, 2013

"Signed and Sealed"

Michael Godard

Michael Godard, P. Eng.

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CERTIFICATE OF AUTHOR

ISO 9001, ISO 14001
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I, Bruno Borntraeger, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Technical Report on the Prefeasibility Study of the Tepal Project, Michoacán, Mexico", with an effective date of March 19, 2013 prepared for Geologix Explorations Inc.;
2. I am currently employed as Specialist Engineer/Project Manager with Knight Piésold Ltd. Inc. with an office at Suite 1400 – 750 West Pender Street, Vancouver BC V6C 2T8;
3. I am a graduate of University of British Columbia with a Bachelor of Applied Science in Geological Engineering, May 1990. I have practiced my profession continuously for over 22 years;
4. I am a Registered Professional Engineer (20926) with the Association of Professional Engineers and Geoscientists of British Columbia;
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 independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;
7. I visited the Tepal Project site from September 18 to 22, 2011;
8. I am responsible for and/or shared responsibility for Sections 16.2, 16.3, 18.3 and 18.4;
9. I have had no prior involvement with the property that is the subject of this Technical Report;
10. As of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading; and
11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: March 19, 2013
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