NI 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT ON THE TEPAL PROJECT, MICHOACÁN, MEXICO

Prepared for:

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NOTICE

JDS Energy & Mining, Inc. prepared this 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.

Geologix Explorations Inc. filed this Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

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1 Executive Summary

1.1 Introduction

Geologix Explorations Inc. (Geologix) commissioned JDS Energy & Mining Inc. (JDS) to complete a preliminary economic assessment (PEA) of the Tepal Project (Tepal or Project) located in Michoacán State, Mexico.

The purpose of this study is to revise and update the previous pre-feasibility study (PFS) filed in 2013 titled "Technical Report on the Pre-feasibility Study of the Tepal Project Michoacán, Mexico" with an effective date of March 19, 2013 (JDS, 2013).

This study documents the following changes from the 2013 PFS including:

- A revised process flow sheet consisting primarily of:
    - o Reduction of the sulphide flotation throughput from an average 37,000 t/d to 22,000 t/d;
    - o Change from batch grinding oxide material in the SAG and ball mills to an independent oxide crushing and grinding circuit; and
    - o Increase of oxide carbon-in-leach (CIL) retention time from 8 hours to 24 hours.
- Mining operating costs based on contractor mining rates;
- Revised Whittle pit optimization at lower metal prices;
- Revised mining schedule based on changes to the process plant;
- Updated capital and operating cost estimates (CAPEX and OPEX) based on revised designs and more recent equipment budgetary pricing; and
- Updated economic base case metal prices to:
    - o US$1,250/oz gold (Au);
    - o US$2.50/lb copper (Cu); and
    - o US$18.00/oz silver (Ag).

1.2 Project Description and Location

The Tepal Property is located in the municipality of Tepalcatepec, Michoacán State in south-western Mexico as shown in Figure 1.1. The Property is centred 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 550 m. The climate in the area is hot and relatively dry.

Figure 1.1: Location Map of the Tepal Property

Source: Micon (2012)

The Property has been explored by several exploration companies over the past 30 years. Geologix currently owns 100% of the concessions, which are subject to a 2.5% net smelter return (NSR).

1.3 Access and Ownership

1.3.1 Access

The Property can be accessed year-round by paved highway Carretera Federal 120 which traverses the southeastern portion of the Property. The last 8 km to the centre of the Property is on dirt roads.

A series of all-weathered roads and the Morelia-Lazaro Cárdenas Autopista (tollway) can be used to reach both the capital of Michoacán State, Morelia or Mexico's main west coast port of Lazaro Cárdenas. Lazaro Cárdenas is approximately three and a half hours driving time by vehicle.

Two international airports service the area. The General Francisco J. Mujica International Airport (Morelia) is approximately four and a half hours' drive northeast of the Property, while the Ixtapa Zihuatanejo International Airport is approximately five hours south of the Property. The closest domestic airport to the Property is the Pablo L. Sidar Airport in Apatzingán which is approximately one hour drive southeast of the Property.

1.3.2 Ownership

The Tepal Property consists of five contiguous concessions totalling 1,406 ha. 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).

Table 1.1 lists the current five concessions of the Property.

Table 1.1: Tepal Property Concessions

Lot	Concession number	Issuance date	Years in force	Hectares
Tepal	219924	6/5/2003	14 years	986
Tepal Fracción 1	216874	4/6/2002	17 years	140
	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(according to former title)	
Tepal Fracción 2	216875	4/6/2002	17 years	70
	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(according to former title)	
Tepal Fracción. 3	216876	4/6/2002	17 years	90
	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(according to former title)	
La Esperanza Fracción 1	216873	4/6/2002	23 years	120
	(title substituted 199423 La Esperanza 19/04/94)	(title substituted 199423)	(according to former title)	
		La Esperanza 19/04/94)		

Source: Geologix (2017)

Arian Silver de Mexico S.A. de C.V. (Arian) originally optioned the 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$5M to gain 100% interest in the Property, subject to a 2.5% NSR.

As of April 4, 2011, Geologix completed the purchase of the concessions from Arian, and completed Arian’s obligations to Minera Tepal. The concessions are subject to a Minera Tepal 2.5% NSR. There is a first-right-of-refusal on the Minera Tepal NSR royalty should Minera Tepal elect to sell the royalty. Other than the NSR, Geologix owns a 100% interest in the concessions.

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 parcelized land owners. Geologix has negotiated an access agreement for an extended period of time with the main private owner.

1.4 History, Exploration and Drilling

Geologix completed over 40,000 m of infill diamond drilling in 2011 after the 2010 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 2013 Preliminary Feasibility Study.

1.5 Geology & Mineralization

The Property is located within the Coastal Range of southwestern 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.

1.6 Metallurgical Testing and Mineral Processing

There are three sources of gross metal value (GMV) from the Tepal resource. 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.

There has been no additional metallurgical test work since the Tepal 2013 PFS was completed. The flow sheet for the sulphide circuit is the same, utilizing the identical product grind size, retention times and reagents. The PFS design was for 35,000 t/d and this PEA has reduced the throughput to 22,000 t/d with modifications to equipment size only.

The oxide circuit has two modifications from the 2013 PFS. Firstly, the PFS oxide grind circuit was batch processed through the sulphide SAG and ball mill and stored in a pond for reclamation and processing through the carbon-in-leach/adsorption desorption regeneration (CIL/ADR) circuit. This circuit has been removed and replaced with a stand-alone dedicated oxide crushing plant and ball mill. Secondly, the oxide CIL retention time has been increased from eight to 24 hours.

Sulphide feed hardness is variable in the three pits, with the Tepal North Zone (NZ) and Tepal South Zone (SZ) being moderately hard and Tizate being hard. Bond Work index (BWi) were completed on over 42 variability samples with hardness ranging from a low of 11.0 kWh/t to a high of 20.0 kWh/t (SRK, 2012, Grinding & Crushing Circuit Equipment Sizing for Tepal Prefeasibility Project, Memorandum). Due to this variation, the milling circuit is designed to process 22,000 t/d of NZ, SZ, and Tizate mineralized material.

The Tizate mineralized material, due to its increased hardness, will require a pebble crusher in Year 6, when milling of the Tizate mineralized material is planned to commence. The oxide feed is soft from all three areas resulting in a design capacity of 5,500 t/d through the PEA milling and CIL circuit.

The saleable products for this PEA are: 1) a sulphide copper concentrate with gold and silver values obtained from a sulphide flotation; and 2) gold and silver doré produced on-site from cyanide leaching a pyrite concentrate and the first copper cleaner tailings. The oxide circuit will produce gold and silver doré from a common refinery.

Molybdenum will be contained in the concentrate but is not considered payable in this study. 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 in Section 27.

Table 1.1 is a summary of sulphide copper concentrate recovery predictions used in the design criteria for this 2017 PEA.

Table 1.2: 2017 PEA Copper Concentrate Design Criteria Summary

Product Resource Grade	Unit	Flotation
Tepal Feed Grade		
Copper	%	0.24
Gold	g/t	0.42
Silver	g/t	0.93
Tizate Feed Grade		
Copper	%	0.17
Gold	g/t	0.20
Silver	g/t	2.17
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

Source: JDS (2017)

Table 1.3 summarizes the leach extraction of gold and silver in the pyrite flotation concentrate and copper first cleaner tailings.

Table 1.3: Pyrite Concentrate and Copper First Cleaner Tailings Leach Extraction

Product	Unit	Extraction
Pyrite Conc. and First Cleaner Tailings		
Tepal		
Gold	%	16.5
Silver	%	15.5
Tizate		
Gold	%	16.0
Silver	%	18.5

Source: G&T KM3577-25/26CN

Table 1.4 summarizes the extraction of gold and silver from the oxide leach.

Table 1.4: Oxide Leach Recovery

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

Source: G&T KM3568-03/04CN

1.7 Mineral Resource Estimate

A Mineral Resource estimate was presented on March 29, 2012, using the Ordinary Kriging method. The three deposits were defined by mineralogical models which were 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 a US$5/t equivalent value net smelter return (NSR) cut-off.

Table 1.5: Measured and Indicated Mineral Resources at US$5/t Equivalent Value Cut-Off – March 29, 2012

Deposit	Resource Category	Tonnage (000)	In-Situ Average Grade				Contained Metal	
			Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Au (koz)	Cu (Mlb)
Tepal North	Measured	14,000	0.50	0.29	0.78	0.002	228	89
	Indicated	55,000	0.30	0.21	1.01	0.002	533	252
	M + I	69, 000	0.34	0.22	0.96	0.002	761	341
Tepal South	Measured	20, 000	0.47	0.22	1.07	0.002	300	96
	Indicated	21, 000	0.45	0.2	1.17	0.002	305	91
	M + I	41, 000	0.46	0.21	1.12	0.002	605	187
Tizate	Measured	-	-	-	-	-	-	-
	Indicated	77, 000	0.18	0.17	2.29	0.006	438	285
	M + I	77, 000	0.18	0.17	2.29	0.006	438	285
Total	**Measured**	**34, 000**	**0.48**	**0.25**	**0.95**	**0.002**	**528**	**185**
	Indicated	**153, 000**	**0.26**	**0.19**	**1.67**	**0.004**	**1,276**	**628**
	M + I	**187, 000**	**0.30**	**0.20**	**1.54**	**0.004**	**1,804**	**813**

*Assumptions used to calculate soft pit constraint: Au Price US$ 1,300/oz, Cu Price US$ 3.30/lb and:

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%

CIM definitions were followed for Mineral Resource s

***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.**

Source: Micon (2012)

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

Table 1.6: Inferred Mineral Resources at US$5/t Equivalent Value Cut-Off – March 29, 2012

Deposit	Resource Category	Tonnage (kt)	In-Situ Average Grade				Contained Metal	
			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	34,000	0.15	0.15	1.70	0.007	169.8	114.8
Total	**Inferred**	**36,000**	**0.16**	**0.15**	**1.68**	**0.006**	**181.7**	**120.4**

*Assumptions used to calculate soft pit constraint: Au Price US$ 1,300/oz, Cu Price US$ 3.30/lb and:
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%
Source: Micon (2012)

1.8 Mineral Reserve Estimate

This PEA does not state a Mineral Reserve.

1.9 Mining

It is planned to mine the three deposits at Tepal (North, South and Tizate) via conventional open pit (OP) methods. Mining of the deposits will produce a total of 12.1 Mt of oxide plant feed, 78.3 Mt of sulphide feed along with 52.5 Mt of waste (0.6:1 overall strip ratio) over a 10-year mine life (excluding pre-production period). The mine design process for the deposits commenced with the development of OP optimization design parameters. These parameters included estimates of metal price, mining dilution, process recovery, off-site costs, geotechnical constraints (slope angles), and royalties.

The sulphide block value is based on the NSR of copper concentrate produced by sulphide flotation and revenue from doré bar produced by sulphide cyanidation. The value of oxide blocks was based on revenue from doré bar produced by the oxide CIL process. Table 1.7 summarizes the NSR inputs and optimization parameters.

The current life of mine (LOM) plan focuses on achieving consistent plant feed production rates, and early mining of higher value material, as well as balancing grade and strip ratios.

Table 1.7: Mine Planning Optimization Design Parameters*

Parameter	Unit	Sulphide Flotation	Sulphide Cyanidation	Oxide CIL
Metal Prices				
Copper (Cu)	US$/lb	2.25		
Gold (Au)	US$/oz	1,250		
Silver (Ag)	US$/oz	20.00		
TEPAL - Recovery				
Cu Recovery	%	88.2	-	-
Au Recovery	%	62.4	17.2	88.5
Ag Recovery	%	27.4	13.6	73.4
TIZATE - Recovery				
Cu Recovery	%	85.9	-	-
Au Recovery	%	58.0	22.0	82.5
Ag Recovery	%	59.6	12.2	70.7
Copper Concentrate				
Cu - Tepal	%	25.7	-	-
Cu - Tizate	%	26.9	-	-
Au[1]	g/t	variable	-	-
Ag[1]	g/t	variable	-	-
Moisture Content	%	8%	-	-
Smelter Payables				
Cu Payable	%	96.5	-	-
Cu Deduction	%	1.0	-	-
Au Payable	%	97.0	99.9	
Ag Payable	%	90.0	97.0	
Ag Deduction	g/t	30.0	-	-
Treatment & Refining Costs				
Cu Conc. Transport Charge	US$/dmt	90.00	-	-
Cu Refining Charge	US$/payable lb	0.09	-	-
Au Refining Charge	US$/payable oz	5.00	7.50	
Ag Refining Charge	US$/payable oz	0.50	1.40	
Transport Costs				
Ocean Freight	US$/wmt	60.00	-	-
Truck Freight to Port	US$/wmt	36.73	-	-
Representation at Port	US$/wmt	1.00	-	-
Port Charges	US$/wmt	10.50	-	-
Insurance	US$/wmt	1.93	-	-

Parameter	Unit	Sulphide Flotation	Sulphide Cyanidation	Oxide CIL
Losses	US$/wmt	7.50	-	-
Operating Costs				
Mining Cost - waste	US$/t mined	1.80		
Mining Cost – mineralized material	US$/t mined			
Processing Cost	US$/t milled	5.79	1.06	8.70
General & Administrative (G&A)	US$/t milled	0.90	-	0.90
Tailings Cost	US$/t milled	0.05	-	0.05
Royalties - Cu	%	2.5		
Royalties - Au	%			
Royalties - Ag	%			
Other Parameters				
Processing rate	t/d milled	22,000	12% of Sulp Flot. Rate	8,000
Processing rate	t/a milled	8.0		2.92
External Mining Dilution	%	5.0		
Mining Recovery	%	100		
Discount rate	%	5		
Slope angles (overall)	Deg.	varies		

Note:
*The values in this table vary slightly from those used in the economic model as parameters were further refined in the economic model as the Project progressed. The differences are not considered material to pit shape definition.
1. Au and Ag contained within Cu concentrate vary due to the head grade and recovery of those metals in the flotation process
Wmt = wet metric tonnes/ dmt = dry metric tonnes
Source: JDS (2017)

Pit optimization software utilizing the Lerch-Grossman algorithm and the parameters listed above was used to determine the optimal mining shells with the assumed overall slope angles. Preliminary mining stages were selected and mine planning and scheduling were then conducted from the selected optimal shells. The mineable resources for the Tepal deposit are presented in Table 1.8.

Table 1.8: PEA Proposed Mine Plan Summary

Description	Unit	Value
Mine Production Life	years	10
Total Process Feed Material	Mt	90.5
Total Diluted Gold Grade	g/t	0.34
Total Diluted Copper Grade	%	0.21
Total Diluted Silver Grade	g/t	1.42
Total Contained Gold	koz	1,000
Total Contained Copper	Mlbs	366
Total Contained Silver	koz	4,125
Total Waste	Mt	52.5
Total Material	Mt	142.9
Overall Strip Ratio	w:o	0.6

Note: Mineable Resources are not Mineral Reserves and do not have demonstrated economic viability.
Source: JDS (2017)

The mining sequence was divided into a number of stages at each of Tepal and Tizate and designed to maximize grade and value, reduce pre-stripping requirements in the early years, and maintain the plant at full production capacity. The mine and mill production schedule is summarized in Table 1.9.

Table 1.9: Mine and Mill Production Schedule

Parameter	Unit	Total	Year										
			Pre-prod.	1	2	3	4	5	6	7	8	9	10
Total Feed Mining	Mt	**90.5**	1.5	12.2	10.4	8.5	8.7	9.5	9.5	8.1	8.1	8.0	6.0
Total Gold Grade	g/t	**0.34**	0.58	0.52	0.46	0.47	0.31	0.40	0.27	0.21	0.21	0.19	0.20
Total Copper Grade	%	**0.21**	-	0.35	0.23	0.22	0.21	0.24	0.18	0.17	0.17	0.19	0.16
Total Silver Grade	g/t	**1.42**	0.91	0.88	0.96	1.14	0.91	0.83	1.91	2.50	1.93	1.87	1.96
Total Waste	Mt	**52.5**	12.0	3.0	6.7	3.3	4.7	6.1	6.9	3.8	3.9	1.7	0.2
Total Material Mined	Mt	**142.9**	13.5	15.2	17.0	11.8	13.5	15.6	16.3	11.9	12.1	9.8	6.3
Strip Ratio	w:o	**0.6**	8.2	0.2	0.6	0.4	0.5	0.6	0.7	0.5	0.5	0.2	0.0
Total Feed Milling	**Mt**	**90.5**	**1.4**	**10.0**	**10.0**	**10.0**	**9.7**	**9.5**	**9.5**	**8.1**	**8.1**	**8.0**	**6.0**
Total Recovered Gold	**koz**	**812**	**23**	**139**	**121**	**124**	**84**	**98**	**65**	**44**	**44**	**39**	**32**
Total Recovered Copper	**Mlbs**	**320**	**-**	**54**	**36**	**34**	**33**	**37**	**28**	**25**	**25**	**29**	**19**
Total Recovered Silver	**koz**	**2,555**	**30**	**135**	**149**	**169**	**134**	**117**	**373**	**465**	**362**	**348**	**273**

Source: JDS (2017)

1.10 Recovery Methods

The mineral processing facility will consist of copper flotation of sulphides to produce a saleable concentrate, cyanide leach of the first cleaner tailings and pyrite flotation concentrate and cyanide leach of oxide material. Two plants have been designed; one for the sulphides and one for the oxides. The sulphide circuit will consist of crushing, grinding, conventional copper flotation with regrind, concentrate dewatering, filtering and load-out. The copper rougher tailings will feed a pyrite rougher flotation circuit to produce a concentrate that will be reground and combined with the copper first cleaner tailings to be leached in a carbon-in-leach (CIL) circuit. The oxide circuit will include crushing, grinding, CIL tanks and an ADR plant. The carbon from the oxide and sulphide CIL circuits will be processed in the ADR plant to produce doré bars. The CIL tailings will feed a cyanide destruction circuit before being recombined with the pyrite rougher tailings to be pumped to the Tailings Management Facilities (TMF).

The sulphide mill process plant is designed with a nominal capacity of 22,000 t/d and the oxide circuit at 5,500 t/d. The crushing circuit will operate 18 hours per day at a utilization of 75%. The milling and leaching circuits will operate 24 hours per day, 365 days per year at an availability of 92%.

1.10.1 Oxide Plant Design

Oxide feed will be processed through the gyratory crusher one day out of five to provide 27,500 t to the crushed mineralized material stockpile. The crushed mineralized material stockpile will feed the secondary and tertiary crushing circuits at a rate of 5,500 t/d to produce a final product 80% passing (P_{80}) of 9.5 mm. The crusher product will feed one ball mill for further reduction to P_{80} of 143 microns (µm). From the grinding circuit the cyclone overflow will feed the pre-leach thickener followed by the CIL circuit. The loaded carbon from the CIL circuit will be pumped to the ADR plant to remove the precious metals. The CIL tailings will combine with the sulphide CIL tailings in the cyanide destruction circuit. The final tailings will be pumped to the TMF.

1.10.2 Sulphide Plant Design

The sulphide concentrator was designed to process 22,000 t/d sulphide feed. The run-of-mine (ROM) feed will be reduced through three stages of comminution, then the copper minerals, along with some gold and silver, will be recovered by flotation. The copper rougher/scavenger concentrates will be reground and cleaned to a final commercial concentrate grade and then dewatered. The produced copper-gold concentrate will be trucked off-site to a copper smelter.

Rougher/scavenger tailings will be sent to pyrite rougher flotation. Pyrite rougher concentrate and the first copper cleaner tailings will be combined and thickened for feed to a sulphide CIL circuit. Loaded carbon will be sent to the common oxide/sulphide ADR plant where doré bars will be produced. The rougher flotation tailings and cyanide destruction tailings will be pumped to the TMF. A reclaim barge will recover water from the TMF for re-use in the process plant as make-up water.

1.11 Infrastructure

The Project envisions the upgrading or construction of the following key infrastructure items:

- A plant site access road of approximately 8 km from the highway;
- Power supply from the Comisión Federal de Electricidad (CFE) grid, transmission to site, and Project site distribution;
- A TMF;
- Site haul roads and service roads;
- A waste rock storage area (WRSA);
- Crushing and grinding circuits;
- An oxide plant with CIL;
- A sulphide plant with copper flotation and CIL;
- An ADR plant;
- An assay laboratory;
- Security, scale house, administration and first aid facilities;
- Fresh water supply, fire/fresh water storage and distribution, sewage collection and treatment, and drainage and run-off settling ponds; and
- A permanent accommodation complex.

1.12 Environment and Permitting

Environmental baseline studies were carried out for Geologix by Clifton Associates Ltd. out of Guadalajara, Jalisco, Mexico. 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, BC. 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, more than 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 will be required for appropriate waste rock facility design and closure to manage the PAG and metal leaching material, as well as a long-term monitoring protocol.

Water management requirements for the site are planned to include groundwater wells to augment the water supply from other sources (i.e. pit seepage, tailings pond reclaim, waste rock retention) for use in the processing plant.

There will be no discharge of process water to the environment during operations. All PAG waste dumps will be capped and revegetated at closure. Seepage during closure is planned to be collected, analyzed, recycled or treated to ensure it meets standards for release to the environment.

The development of a number of social and environmental management plans will 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 will be required for the Project under the General Law of Ecological Equilibrium and Environmental Protection. In 2013, the PFS design was submitted for an Environmental Impact Manifest (MIA-P) and was approved. Design changes in the 2017 PEA are identical with respect to the MIA excepting a smaller tailings dam and WRSA. A Change of Land Use Authorization and re-submittal of the design changes for an MIA addendum are the two key items that will be required to advance the Project. Once the government approves the revised MIA-P and Change of Land Use permits, additional detailed permits are required for construction and operations.

Although the Project is located adjacent to several small communities and the larger community of Tepalcatepec, the local skilled workforce is limited. A technical institute, sponsored by the Company, will assist with local capacity building for various positions in the mine. The majority of workers will 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 could influence construction and operations, and these will need to be considered in plans in the pre-feasibility phase of the Project.

1.13 Operating and Capital Cost Estimates

1.13.1 Operating Costs

The operating cost estimate (OPEX) is based on a combination of experience, reference project, budgetary quotes and factors as appropriate with a PFS. All costs are in US$ unless otherwise specified. No allowance for inflation has been applied.

Preparation of the OPEX is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined and proven project execution strategies.

The operating cost estimate is broken into four major sections:

- Open pit mining;
- Sulphide flotation/cyanidation processing;
- Oxide CIL processing; and
- G&A costs.

These items total the Project operating costs and are summarized in Table 1.9.

The estimate is based on contract mining. The target accuracy of the operating cost is -25/+30%.

The total operating unit cost is estimated to be $9.65/t processed. The average annual, total LOM and unit operating cost estimates are summarized in Table 1.10.

Table 1.10: Breakdown of Estimated Operating Costs

Operating Costs	Avg Annual (M$)	$/t processed**	LOM (M$)
Mining*	31	3.30	299
Processing – Sulphide Flotation/Cyanidation	44	4.75	430
Processing – Oxide CIL	8	0.85	77
G&A	7	0.75	67
Total	**90**	**9.65**	**873**

*Average LOM Mining cost amounts to $2.16/t mined at a 0.6:1 strip ratio (excluding pre-production tonnes mined).
**includes all tonnes processed (both sulphide and oxide)
Numbers may not add due to rounding
Source: JDS (2017)

The main OPEX component assumptions are outlined in Table 1.11.

Table 1.11: Main OPEX Component Assumptions

Item	Unit	Value
Electrical Power Cost	$/kWh	0.089
Power Consumption	kWh/t processed	17.2
Diesel Cost (delivered)	$/litre	0.885
LOM Average Manpower	employees	325

Source: JDS (2017)

1.13.2 Capital Costs

All capital costs are stated in US$. LOM project capital costs total $301M, consisting of the following distinct phases:

- **Pre-production Capital Costs** – includes all costs to develop the Property to a 22,500 t/d sulphide circuit and a 5,500 t/d oxide circuit. Initial capital costs total $214M (including a $22M contingency) and are expended over a 24-month pre-production construction and commissioning period;
- **Sustaining Capital Costs** – includes all costs related to TMF expansion and the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $87M (including $7M in contingency) and are expended in operating Years 1 through 10; and
- **Closure Capital Costs** - includes all costs related to the closure, reclamation, and ongoing monitoring of the mine post operations. Closure costs total $23M (net of equipment salvage values), and are primarily incurred in Year 10, with costs extending into Year 15 for ongoing monitoring activities.

The capital cost estimate (CAPEX) was compiled using a combination of quotations, database costs, and database factors.

Table 1.12 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q4 2016 dollars with no escalation. The estimate is also based on the assumption that the Project will use contractor mining.

Table 1.12: Capital Cost Summary

WBS	Area	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
1000	Mining	12.4	3.0	15.4
2000	Site Development/Earthworks	5.5	0.0	5.5
3000	Sulphide Processing Plant	77.7	5.7	83.4
4000	Oxide Processing Plant	29.9	0.0	29.9
5000	Tailings and Waste Rock Management	8.6	48.5	57.1
6000	Surface Infrastructure	25.2	0.0	25.2
7000	Project Indirects	10.5	0.0	10.5
8000	EPCM	15.3	0.0	15.3
9000	Owners Costs	6.9	0.0	6.9
C100	Closure Costs	0.0	22.9	22.9
	Subtotal Pre-Contingency	**191.9**	**80.1**	**272.0**
9900	Contingency	22.3	6.6	28.9
	Total Capital Costs	**214.2**	**86.7**	**300.9**

Source: JDS (2017)

1.14 Economic Analysis

1.14.1 Main Assumptions

A preliminary market study was completed by Cliveden in Q3 2016 on the potential sale of copper concentrate and doré from the Tepal Project. The terms were reviewed and found to be acceptable by QP Gord Doerksen, P.Eng.

No contractual arrangements for shipping, port usage, or refining exist at this time. Table 1.12 outlines the terms used in the economic analysis.

This PEA is preliminary in nature and includes the use of Inferred Mineral Resources (2.0 Mt of mill feed material) that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized.

Table 1.13: NSR Assumptions Used in the Economic Analysis

Assumptions	Unit	Value
Copper Concentrate NSR Parameters		
Payable Copper	%	96.5
Copper Minimum Deduction	%	1
Payable Gold	%	97.5
Gold Minimum Deduction	g/t	0
Payable Silver	%	90.0
Silver Minimum Deduction	g/t	0
Copper Treatment Charge	US$/dmt concentrate	97.35
Copper Refining Charge	US$/payable lb	0.097
Gold Refining Charge	US$/payable oz	5.00
Silver Refining Charge	US$/payable oz	0.50
Concentrate Transportation	US$/dmt	90.04
Doré NSR Parameters		
Payable Gold	**%**	**99.9**
Gold Refining Charge	US$/payable oz	7.50
Payable Silver	%	97.0
Silver Refining Charge	US$/payable oz	1.40

Source: JDS (2017)

Table 1.14 outlines the metal prices and exchange rate used in the economic analysis.

Table 1.14: Metal Prices used in the Economic Analysis

Assumptions	Unit	Value
Cu Price	US$/lb	2.50
Au Price	US$/oz	1,250
Ag Price	US$/oz	18.00
FX Rate	MX$:US$	18

Source: JDS (2017)

1.14.2 Results

The economic results for the Project based on the assumptions made are shown in Table 1.15.

Table 1.15: Economic Results

Summary of Results	Unit	Value
Cash Cost (Net of Byproduct)	US$/oz	313
Cash Cost (incl. Sustaining CAPEX)	US$/oz	396
Capital Costs		
Pre-Production Capital	M$	192
Pre-Production Contingency	M$	22
Total Pre-Production Capital	**M$**	**214**
Sustaining & Closure Capital	M$	80
Sustaining & Closure Contingency	M$	7
Total Sustaining & Closure Capital	M$	87
Total Capital Costs Incl. Contingency	**M$**	**301**
Working Capital	M$	23
Pre-Tax Cash Flow	LOM M$	417
	M$/a	43
Taxes	LOM M$	160
After-Tax Cash Flow	LOM M$	257
	M$/a	26
Economic Results		
Pre-Tax $NPV_{5\%}$	**M$**	**299**
Pre-Tax IRR	**%**	**36**
Pre-Tax Payback	**Years**	**1.6**
After-Tax $NPV_{5\%}$	**M$**	**169**
After-Tax IRR	**%**	**24**
After-Tax Payback	**Years**	**2.3**

Source: JDS (2017)

1.14.3 Sensitivities

A simplistic sensitivity analysis was performed to determine which factors most affect the Project economics and this is discussed in Section 23. Each variable evaluated was tested using the same sensitivity values, although some may be more likely to experience significantly more fluctuation in value over the LOM (i.e. CAPEX versus metal prices). The confidence attributed to each variable in this study does not factor into the sensitivity analysis, the inter-correlation between certain variables, and for this reason is considered a simplistic approach to determine which variable will most affect the economic results of the Project.

Sensitivity analyses were performed on metal prices, mill head grade, CAPEX, and OPEX as variables. The value of each variable was changed plus and minus 20% independently while all other variables were held constant. The results of the sensitivity analyses are shown in Table 1.16.

Table 1.16 Sensitivity Results (Pre- and After- Tax $NPV_{5\%}$)

Variable	Pre-Tax $NPV_{5\%}$ (M$)			After-Tax $NPV_{5\%}$ (M$)		
	-20% Variance	0% Variance	20% Variance	-20% Variance	0% Variance	20% Variance
Metal Prices	32	299	567	-19	169	345
Head Grade	53	299	546	-4	169	331
OPEX	433	299	166	258	169	74
CAPEX	353	299	246	223	169	116

Source: JDS (2017)

1.15 Conclusions

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the positive economic result herein contained. The PEA proposes the use of industry standard equipment and operating practices. To date, the QPs are not aware of any fatal flaws for the Project.

Using the assumptions highlighted in this report, the Tepal Project offers sufficient economic potential to be advanced to the next stage of study (Preliminary Feasibility Study).

1.15.1 Risks

The most significant potential risks associated with the Project are:

- The source of make-up water supply for processing;
- Metallurgical recoveries;
- Operating and capital cost escalation;
- Unforeseen schedule delays;
- The ability to attract and retain experienced professionals; and
- The ability to raise financing and metal prices.

These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning and pro-active management.

1.15.2 Opportunities

There are significant opportunities that could improve the economics, timing, and/or permitting potential of the Project. The major opportunities that have been identified at this time, excluding those typical to all mining projects, such as changes in metal prices, exchange rates, etc., are:

- The expansion of known Mineral Resources and the addition of new deposits may be possible with further resource drilling and could potentially extend the mine life;
- The Project strategy and optimization of mine plans and development schedule;
- Metallurgy and process optimization; and
- The potential to employ good used equipment.

1.16 Recommendations

In the opinion of JDS, financial analysis of this PEA demonstrates that the Tepal Project has positive economics and warrants consideration for advancement to the pre-feasibility level engineering by Geologix. This more advanced study will further detail:

- A revised Mineral Reserve estimate;
- Processing engineering based on the PEA flowsheets;
- Project scheduling;
- Capital and operating cost estimation; and
- Economic results.

The study will improve the confidence in the Project design and execution and will result in an improved accuracy of project economics.

It is estimated that a PFS and its supporting work programs would cost approximately $800,000. A breakdown of the key components of the next study phase is summarized in Table 1.16.

Table 1.17: Preliminary Feasibility Study Cost Estimate

Item	Cost (US$)
Hydrogeology Studies (test wells and draw down testing)	50,000
Processing and Metallurgy	250,000
Pre-Feasibility Study	500,000
TOTAL	**800,000**

Source: JDS (2017)

2 Introduction

2.1 Basis of Technical Report

Geologix Explorations Inc. (Geologix) commissioned JDS Energy & Mining Inc. (JDS) to complete a preliminary economic assessment (PEA) of the for the Tepal Project (Tepal or Project) located in Michoacán State, Mexico

The purpose of this study is to revise and update the previous pre-feasibility study (PFS) filed in 2013 titled "Technical Report on the Pre-feasibility Study of the Tepal Project Michoacán, Mexico" with an effective date of March 19, 2013 (JDS, 2013).

The structure and content of this report uses National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101) guidelines.

The purpose of this study is to document the following changes from the 2013 PFS including:

- A revised process flow sheet consisting primarily of:
    - Reduction of the sulphide flotation throughput from an average 37,000 t/d to 22,000 t/d; and
    - Change from batch grinding oxide material in the SAG and ball mills to an independent oxide crushing and grinding circuit.
- Increasing the oxide CIL retention time from eight hours to 24 hours;
- Mining operating costs based on contractor mining rates;
- Revised mining schedule based on changes to the process plant;
- Updated capital and operating costs based on revised designs and more recent equipment budgetary pricing; and
- Updated metal prices to:
    - US$1,250/oz Au;
    - US$2.50/lb Cu; and
    - US$18.00/oz Ag.

2.2 Scope of Work

This report summarizes the work carried out by several consultants. The scope of work for each company is listed below, and combined, makes up the total Project scope.

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

- Compiling the technical report including historical data and information provided by other consulting companies;
- Establishing an economic framework for potentially mineable resources;
- Open pit (OP) mine planning and scheduling;
- Selecting mining equipment;
- Developing a conceptual flowsheet, specifications and selection of process equipment;
- Designing required site infrastructure, identifying proper sites, plant facilities and other ancillary facilities;
- Estimating mining, process plant and infrastructure OPEX and CAPEX for the Project;
- Establishing gold and silver recovery values for doré production on-site;
- Preparing a financial model and conducting an economic evaluation including sensitivity and Project risk analyses; and
- Interpreting the results and making conclusions that lead to recommendations to improve Project value and reduce risks.

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

- Tailings management; and
- Water management.

Micon International Limited (Micon) scope of work included:

- Review the 2012 Mineral Resource estimate and certify that it is current so it could be included in the mining plan using Measured, Indicated and Inferred Resources.

Mike Godard, P. Eng. scope of work included:

- Establishing sulphide copper concentrate recovery values based on metallurgical testing results.

2.3 Qualifications and Responsibilities

The results of this PEA are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Geologix and the respective QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions/associations. The QPs are responsible for the specific report sections as follows in Table 2.1.

Table 2.1: QP Responsibilities

QP	Company	QP Responsibility/Role	Report Section(s)
Gord Doerksen, P.Eng.	JDS Energy & Mining Inc.	Project Management, Infrastructure, CAPEX, OPEX, Economic Analysis	1,2,3,4,5,6, 18 (except 18.6,18.8),19,20,21, 22,23,24,25,26,27,28,29
Dino Pilotto, P.Eng.	JDS Energy & Mining Inc.	Mining Methods	15,16
Kelly McLeod, P. Eng.	JDS Energy & Mining Inc.	Metallurgical Recoveries, Recovery Method	13 (excluding 13.5.1), 17
Michael Godard, P.Eng.	Independent Consultant	Metallurgical Recoveries	13.5.1
Daniel Friedman, P. Eng.	Knight Piésold Ltd.	Tailings Management	18.6, 18.8
David K. Makepeace, P.Eng	Micon International Ltd.	Mineral Resource Estimate	7,8,9,10,11,12,14

Source: JDS (2017)

2.4 Site Visit

In accordance with NI 43-101 guidelines, QP Michael Godard visited the Tepal Project from January 8 to 13, 2012 and QP Dino Pilotto visited the Project from July 9 to 10, 2010. Both were accompanied by Dunham Craig of Geologix. The other QPs relied upon the observations of other JDS personnel that visited the site during previous studies.

Table 2.2: QP and Non-QP Site Visits

Qualified Person	Company	Date	Accompanied by	Description of Inspection
Dino Pilotto, P.Eng.	JDS	July 9-10, 2010	Dunham Craig	Tepal Project Site
Michael E. Makarenko, P.Eng.	JDS	September 4-6, 2012	Dunham Craig	Tepal Project Site
Michael Godard, P.Eng.	Independent Consultant	January 8-13, 2012	Dunham Craig	Tepal Project Site
David K. Makepeace, P.Eng	Micon	January 8-13, 2012	Dunham Craig	Tepal Project Site

Source: JDS (2017)

2.5 Units, Currency and Rounding

The units of measure used in this report are as per the International System of Units (SI) or "metric" except for Imperial units that are commonly used in industry (e.g., ounces (oz.) and pounds (lb.) for the mass of precious and base metals).

All dollar figures quoted in this report refer to US dollars (US$ or $) unless otherwise noted.

Frequently used abbreviations and acronyms can be found in Section 29. This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

2.6 Sources of Information

This report is based on information provided by Geologix throughout the course of JDS's investigations. Other information was obtained from the public domain. JDS has no reason to doubt the reliability of the information provided by Geologix. This technical report is based on the following sources of information:

- Discussions with Geologix personnel;
  - Dunham L. Craig, Chairman; and
  - Kiran Patankar, President and CEO.
- Review of geological exploration data collected by Geologix; and
- Additional information from public domain sources including the Micon Technical report, March 29, 2012 (Micon, 2012).

Engineering and geological information from historical documents were used in this report after determination by JDS that the work was performed by competent persons or engineering firms. Data derived from engineering companies, consultants and authors listed in the reference section of this report. The documentation received and the sources of information are listed in Section 28.

3 Reliance on Other Experts

The QPs opinions contained herein are based on information provided by Geologix and others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

Non-QP specialists relied upon for specific advice are:

- Victoria Viveash, CPA, CMA, JDS Energy & Mining Inc. for taxation guidance; and
- Cliveden Trading, for NSR terms and shipping contract advice

The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending.

The QPs did not review any licenses, permits, work contracts, or perform an independent verification of land title and tenure. The QPs have not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s), such as royalty agreements, between third parties.

4 Property Description and Location

The information presented in this section has been adapted from the March 29, 2012 Technical Report by Micon and was updated based on information provided by Geologix.

4.1 Property Description and Location

The Tepal Property is located in the municipality of Tepalcatepec, Michoacán State in south-western Mexico. The Property is centred at 19° 7' 40" Latitude and 102° 56' 8" Longitude or 2,116,257 mN and 717,161 mE, Zone 13Q (UTM - NAD 83). The average elevation is 550 m. Figure 4.1 illustrates the location and the infrastructure surrounding the Tepal Property.

Figure 4.1: Location Map of the Tepal Property

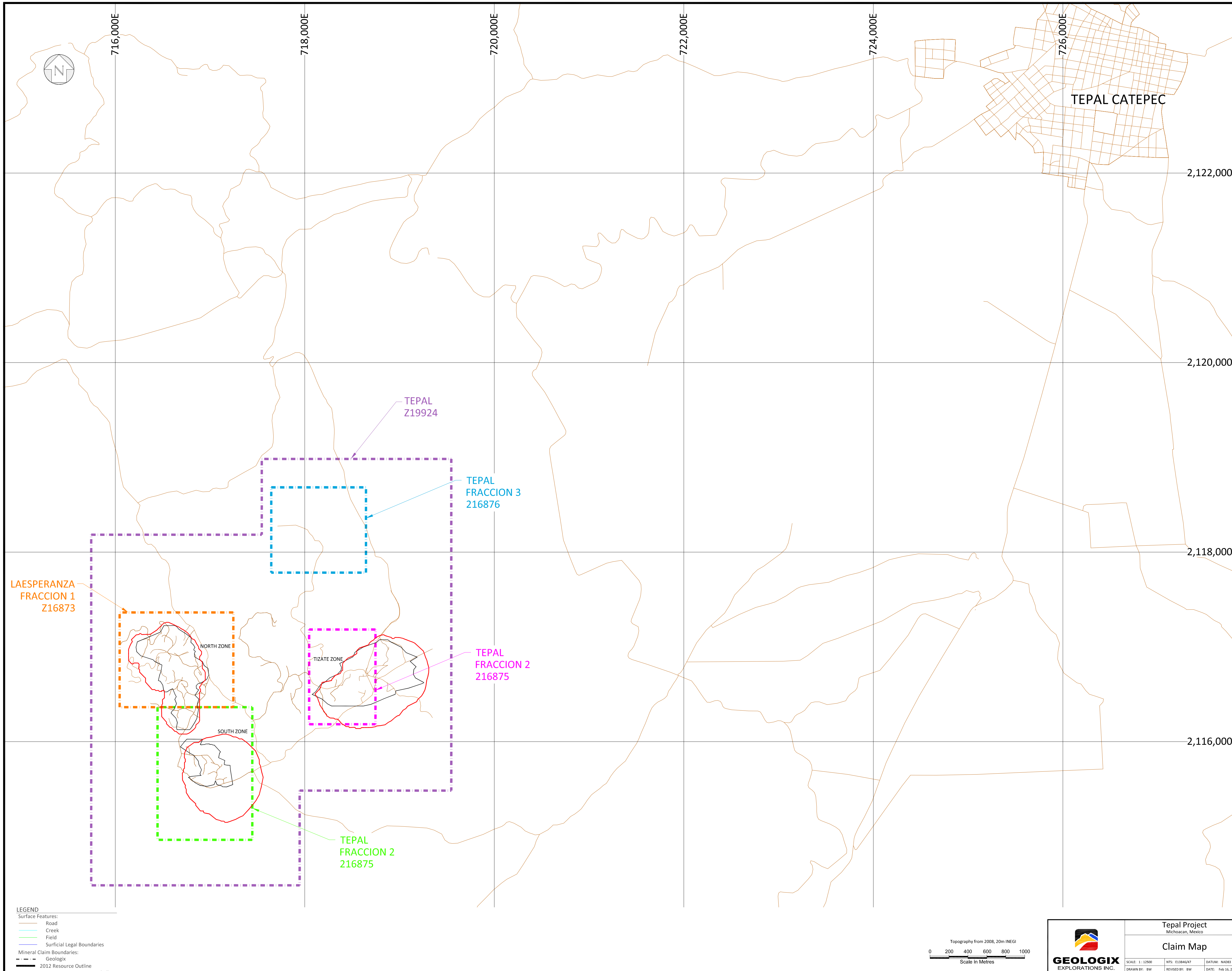
Source: Micon (2012)

The Tepal Property consists of five contiguous concessions totalling 1,406 ha. The Property has been explored by several exploration companies over the past 30 years. Geologix currently owns 100% of the concessions, and these are subject to a 2.5% NSR.

Table 4.1: Concession Titles, Tepal Project

Lot	Concession number	Issuance date	Years in force	Hectares
Tepal	219924	6/5/2003	14 years	986
Tepal Fracción 1	216874	4/6/2002	17 years	140
	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(according to former title)	
Tepal Fracción 2	216875	4/6/2002	17 years	70
	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(according to former title)	
Tepal Fracción. 3	216876	4/6/2002	17 years	90
	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(title substituted 211997 Tepalcatepec Número Uno 18/08/00)	(according to former title)	
La Esperanza Fracción 1	216873	4/6/2002	23 years	120
	(title substituted 199423 La Esperanza 19/04/94)	(title substituted 199423)	(according to former title)	
		La Esperanza 19/04/94)		

Source: Geologix (2017)


716,000E
718,000E
720,000E
722,000E
724,000E
726,000E
N
TEPAL CATEPEC
2,122,000N
2,120,000N
2,118,000N
2,116,000N
TEPAL
Z19924
TEPAL
FRACCION 3
216876
LAESPERANZA
FRACCION 1
Z16873
NORTH ZONE
TIZATE ZONE
TEPAL
FRACCION 2
216875
SOUTH ZONE
TEPAL
FRACCION 2
216875
LEGEND
Surface Features:
Road
Creek
Field
Surficial Legal Boundaries
Mineral Claim Boundaries:
Geologix
2012 Resource Outline
2012 Resource Reporting Shell
Topography from 2008, 20m INEGI
0 200 400 600 800 1000
Scale in Metres
GEOLOGIX
EXPLORATIONS INC.
Tepal Project
Michoacan, Mexico
Claim Map
SCALE: 1 : 12500
NTS: E13B46/47
DATUM: NAD83
DRAWN BY: BW
REVISED BY: BW
DATE: Feb 16, 2017
FILE NAME: TEPAL CLAIM MAP 2017-02-16
MAP No: 1

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$ 5M to gain 100% interest in the Property, subject to a 2.5% net smelter return (NSR).

As of April 4, 2011, Geologix has completed the purchase of the internal concessions 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.

NSR payments are subject to Mexican Value Added Tax (VAT) (16%) which would be paid by Geologix and applied for reimbursement. A 2.5% 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.

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 parcelized 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$20,817 (US$10,410 paid) for 2017.

Assessment work is calculated on the same basis as Property taxes. The assessment work commitment for the Property has been met for each year that Geologix has owned the concessions and sufficient assessment work credits are available to meet the requirements for 2017 with a balance in favour of approximately US$10,916,779.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

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

5.1 Accessibility

The Tepal Project can be accessed year-round by paved highway Carretera Federal 120 which traverses the southeastern portion of the Property. The last 7.5 km to the centre of the Property is on 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 three and a half hours.

Two international airports service the area. The General Francisco J. Mujica International Airport (Morelia) is approximately a four and a half hour drive northeast of the Property while the Ixtapa Zihuatanejo International Airport is approximately five hours south of the Property. The closest domestic airport to the Property is the Pablo L. Sidar Airport in Apatzingán which is approximately one 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 limit access to the Property by turning the dirt roads into mud and/or producing wash outs in places.

Average annual precipitation ranges from 500 to 700 mm (Murphy et al, 2011). The daytime temperatures range from 27 to 40°C with an average annual temperature between 28 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 500 to 700 m. The elevation immediately around the deposit ranges from 550 to 650 m. 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 proposed mill site.

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

Figure 5.1: Proposed North Zone Pit looking South

Source: Micon (2012)

Figure 5.2: Proposed Mill Area looking North to South Pit (Flat Area) and North Pit (Hill behind Pickup)

Source: Micon (2012)

5.4 Infrastructure

Tepalcatepec is the town nearest to the Property. It has a population of approximately 30,000 inhabitants. 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. There is also a secure warehouse for core and rejects sample storage near the exploration compound.

Apatzingán, located approximately 55 km southeast of Tepalcatepec, has a population of approximately 90,000. It is the closest town with scheduled domestic air service from Pablo L. Sidar Airport). Daily commuter flights are available 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 7 km east of the deposits. A major power substation is located 2 km east of the town of Tepalcatepec. La 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. 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 fed 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 3 m below the surface.

The dominant land use centred around the three deposits is non-agricultural due to the steep terrain and thick brush. However, some of the peripheral land 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 from 1972 to 1974 by means of surface geochemistry, Induced Polarization (IP) geophysics and drilling. INCO developed a historic (not in accordance with NI 43-101 guidelines) resource estimate of 27 Mt averaging 0.33% Cu and 0.65 g/t Au during this time. The methodology used to develop the estimate is unknown. This estimate was used to attract future companies to the Property. Unfortunately, INCO abandoned the Property. INCO however stressed at the time that more drilling was required to further define the width of the mineralized zones.

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 60 km of grid line, the collection of 1,268 soil samples and 50 rock chip samples from the grid, the construction of 15 km of access road and the completion of 50 reverse circulation holes (RC) totalling 8,168 m in four phases of work. Teck also undertook some metallurgical testing.

In 1994, Teck completed an historic resource estimate (not in accordance with NI 43-101 guidelines). 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 78.8 Mt grading 0.40 g/t Au and 0.25% Cu with drill indicated resources totalling 55.8 Mt grading 0.51 g/t Au and 0.26% Cu. The South Zone had a drill indicated resource of 24.3 Mt averaging 0.55 g/t Au and 0.25% Cu. The North Zone had a drill indicated resource of 31.6 Mt averaging 0.49 g/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 as per NI 43-101 guidelines 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, 2014).

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 50 m by 50 m grid, the re-analysis of 298 pulps from the Teck reverse circulation drilling program, the completion of 17

RC drill holes totalling 1,506 m 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 9.1 Mt averaging 0.90 g/t Au and containing 262 koz of gold.

The historical estimate prepared by Hecla, Teck and INCO are believed to be reliable and a good approximation of the amount and grade of mineralization found on the Property at the time they were completed. However, these historical estimates are no longer relevant as heyt precede 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 7,180 m. In April 2008, ACA Howe International Limited (ACA Howe) did a Mineral Resource estimate using an inverse weighted method to the third power (ID^3). The constrained +0.18 g/t Au mineralized zones at Tepal were interpolated to have a total Inferred Mineral Resource of 78.8 Mt grading 0.47 g/t Au and 0.24% Cu at a zero cut-off grade for approximately 1.18 M oz Au and 421.5 M lbs Cu.

In September, 2008, ACA Howe undertook a NI 43-101 Technical 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 Mt grading 0.54 g/t Au and 0.27% Cu and an Inferred Mineral Resource of 55 Mt grading 0.41 g/t Au and 0.22% Cu, constrained by a 0.18 ppm 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 10,656 m. 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 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 results up to 2010 were 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 it is generally narrow (i.e. 1 to 2 m) 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 10 x 10 x 5 m 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 off-site 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 57.8 Mt of Indicated Mineral Resources grading 0.42 g/t Au and 0.24% Cu at a cut-off grade of US$5.00 equivalent value. The deposits contained an additional 93.2 Mt grading 0.28 g/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 Priesmeyer, 2007 and 2013 with refined interpretations from Geologix's geological staff after the Micon Technical report, March 29, 2012.

7.1 Regional Geology

The Tepal Property is located within the Costal Ranges of southwestern 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 volcano-sedimentary assemblage. At least some of the intrusive rocks are coeval with the volcanic units. Neogene basalts locally overlie 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 mineralized hypabyssal intrusions at the Tepal Project are thought to be marginal phases of this batholith (Shonk, 1994).

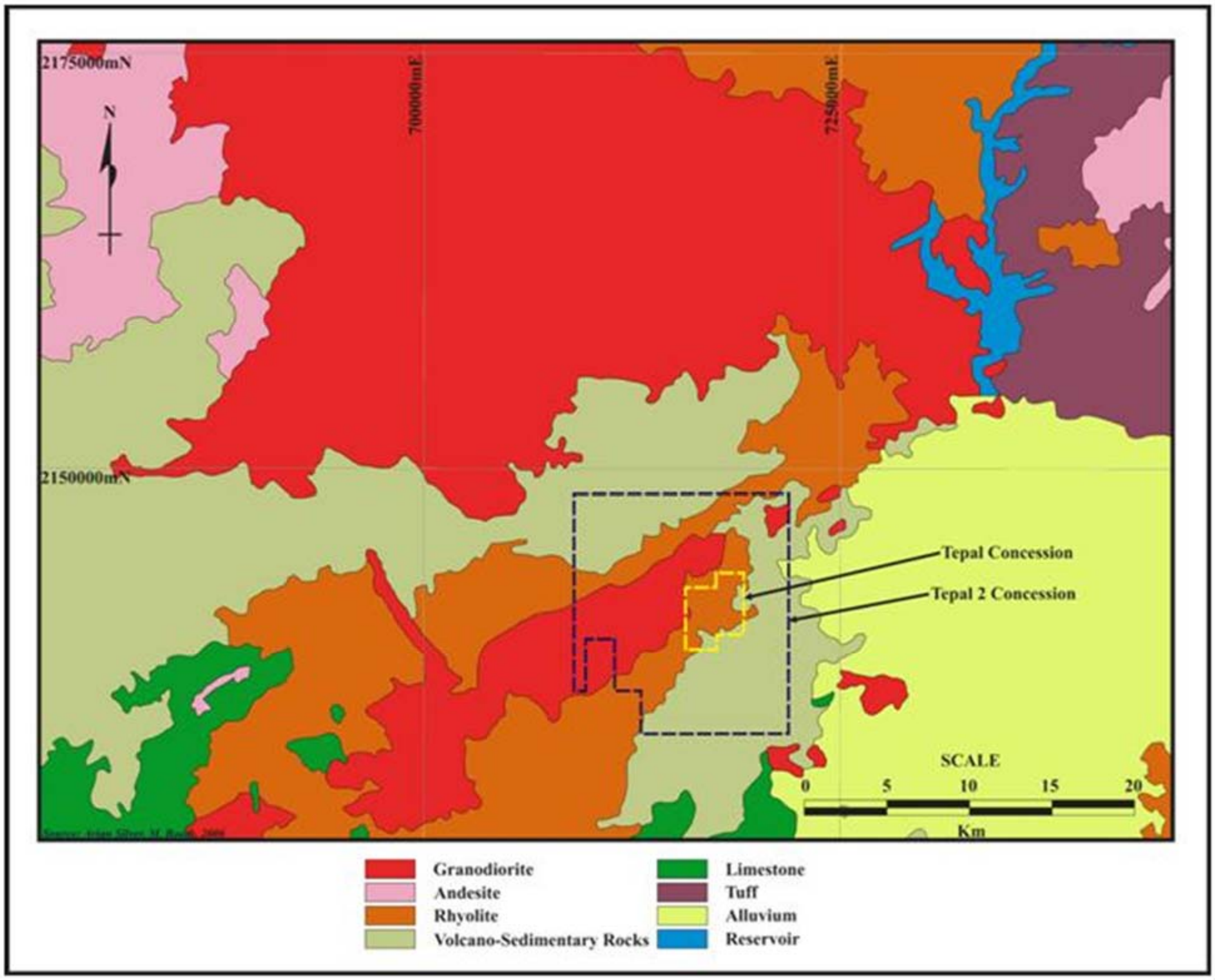
7.2 Property Geology

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 have 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



Source: Micon (2012)

7.3 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.

7.4 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 1,100 by 600 m and thicknesses of up to 200 m. The South Zone has a smaller footprint, 600 by 500 m, but a greater vertical extent of up to 400 m, 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° to 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 attitude 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 20 to 40 m on the hilltops and 0 to 20 m in the drainages. Minerals in the oxidized zone include malachite, chalcocite, minor azurite, tenorite and minor chrysocolla. Shonk (1994) indicated that a thin supergene-enriched layer exists 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 molybdenum 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 soil sampling program, which covers the core of the Property, and most Teck drill core samples were both only analyzed for Cu and Au. Anomalous levels of As, Pb and Zn have been encountered in recent drilling which have full Inductively Coupled Plasma (ICP) data. In most cases, elevated levels of these elements occur erratically in veins and mineralized structures or areas outside of the deposits.

7.5 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 to 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 and 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 2,800 km 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 10 km^2 in size. These altered zones are commonly coincident with shallow intrusives and/or dyke swarms and hydrothermal or intrusion breccias. Deposit boundaries are determined by economic factors, which outline mineralized feed 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 dyke 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 < 100 ppm Mo, but do contain elevated gold (> 0.3 g/t) and silver (>2 g/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 0.50 g/t Au (Cooke and others, 2004) to small high-grade deposits such as Ridgeway in Australia which contains 54 Mt averaging 0.77% Cu and 2.5 g/t Au (Wilson and others, 2003). The average of 112 deposits from around the world is 200 Mt averaging 0.44% Cu, 0.4 g/t Au, 0.002% Mo and 1.4 g/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 and excerpted from Priesmeyer, 2007, and modified excerpted 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 1,400 m east of the Tepal 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 copper, molybdenum, zinc and gold and anomalous copper zones were identified. In early 1973, six diamond drill holes (DDH) (57001 – 57006) were drilled in the Tepal gossan. Geologic mapping and an Induced Polarization (IP) survey were completed during the winter of 1973 to 1974. IP anomalies were found to be generally confined to geochemically anomalous copper zones. According to Shonk (1994), a summary map showing extent and strength of interpreted anomalous IP response along each line, in conjunction with molybdenum in soil anomalies, drill hole locations, and photocopies of contoured IP sections were all 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 that 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 60 km of grid line, the collection of 1,268 soil samples and 50 rock chip samples from the grid, the construction of 15 km of access road and the completion of 50 RC holes totalling 8,168 m in four phases. Total expenditures by Teck were 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 mineralized area and part of the eastern half with a line spacing of 100 m and a station spacing of 50 m over areas of known mineralization and alteration and a station spacing of 100 m outside areas of known mineralization and alteration.

In late 1993 and early 1994, Teck completed a soil sampling program. Grid lines were generally spaced 200 m apart and sample spacing was 100 m in non-anomalous areas, and over anomalous areas, line spacing was reduced to 100 m and sample spacing to 50 m. 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 coper and gold 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 copper values covers a 1.5 x 2.0 km area on the east side of the Property with three poorly defined highs. Gold 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 50 m by 50 m grid, the re-analysis of 298 pulps from the Teck reverse circulation drilling program, the completion of 17 RC drill holes totalling 1,506 m 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 some 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 are available 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 301 ppm copper in rock.

This anomaly measured approximately 1,100 m in length and 125 m 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 1,000 ppm. The largest of these strong anomalies measured approximately 300 m by 230 m 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 200 ppb or 0.2 g/t Au in rock and was open to the south and southeast. The anomaly trended 320° and measured approximately 700 m by 215 m.

Within this anomaly was a smaller, very strong anomaly in which all values exceed 910 ppb or 0.91 g/t Au. This anomaly measured approximately 230 m by 80 m 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 DDH (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 diamond drill program on Tepal Phase 1.

9.5 Geologix

During the due diligence period commencing in the fourth quarter of 2009 and continuing into the first quarter of 2010, Geologix initiated additional metallurgical test work utilizing core from historical drill programs, an IP survey over the core mineral concessions covering 1,526 ha, includinggeology, mineralization and alteration studies and preliminary economic studies as they pertain to the viability of the Tepal Project.

By the end of the first 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 10,656 m 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, Geologix initiated detailed geological mapping, prospecting, a soil geochemical grid survey, 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 150 m over the entire concession. A more detailed survey over 19 km^2 (130 line-km) was flown over the known deposit area at 75 m spacing.

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, Geologix drilled 34 RC holes totalling 4,906 m. **None of this drilling was carried out on the known mineralized zones** and was not included, for obvious reasons, in the 2012 mineral resource estimate.

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 (DDH) utilizing a Longyear 38 core rig from Boyles Brothers Drilling. Holes were collared with NX (core - 54.7 mm) and reduced to BX (42.0 mm). Sample intervals ranged from 0.2 to 3.0 m and averaged 2.0 m. 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 RC drill holes totalling 8,169 m. 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 approximately every 2 m (3 per 20-foot drill rod) for the first 24 holes and every 1.5 m (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 copper and gold at Chemex (now ALS Chemex). An additional 123 samples from selected intervals were analyzed for silver, cobalt, copper, iron, manganese, molybdenum, nickel, lead, and zinc using a multi-element Inductively Coupled Plasma (ICP) procedure.

Drilling at Tepal generally indicated that the best values were present within 150 m 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 RC drilling program totalling 1,506 m.

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 1.0 m.

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 7,180 m. Drilling was 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, Mexico.

The majority of the initial diamond drilling was carried out using HQ drill steel (core - 63.5 mm) and reduced if required to NQ (core - 47.6 mm). 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 10,656 m completed on the Tepal Property. The drill program utilized two diamond drilling rigs. 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 DDH in the totalling 41,248 m. 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 (m)
Tepal	132	23,074
Tizate	70	18,173
Total	**202**	**41,247**

Source: Geologix (2011)

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 seven in-pit geotechnical drill holes totalling 1,354 m and a total of six condemnation holes totalling 298 m.

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	64.05	64.05	0.3	0.67	0.8
TEP-11-012	South	146.5	425.9	279.45	0.26	0.54	1.3
	including	301.4	403.85	102.45	0.38	0.86	0.9
	including	303.4	370.95	67.55	0.42	1.01	1
TEP-11-015	South	0	91.1	91.1	0.25	0.67	1
TEP-11-016	South	6.2	86.1	79.9	0.26	0.88	1.4
TEP-11-018	South	0	140	140	0.27	0.59	1.4
TEP-11-020	South	0	213.4	213.4	0.21	0.39	0.5
TEP-11-026	South	309.2	498	188.8	0.4	1.04	2.7
	including	317.2	422	104.8	0.44	1.45	1.3
TEP-11-033	North	0	41.9	41.9	0.58	0.29	5.9
TEP-11-043	South	152	294.55	142.55	0.35	0.91	1.3
	including	162	274	112	0.38	1.04	1.2
TEP-11-060	North	0	96	96	0.26	0.43	2.3
TEP-11-063	North	4	67.4	63.4	0.26	0.36	1
TEP-11-064	North	0	54.5	54.5	0.29	0.43	2.1
TEP-11-065	North	0	29.95	29.95	0.39	0.41	0.5
	and	54.4	77.25	22.85	0.42	0.43	0.8
TEP-11-068	North	52.5	93.5	41	0.37	0.74	1.1
TEP-11-072	North	0	76	76	0.59	0.77	1
TEP-11-075	North	0	140.7	140.7	0.36	0.87	1.4
	and	162.75	188.9	26.15	0.23	0.53	0.8
TEP-11-084	North	0	31.5	31.5	0.3	0.14	0.7
TEP-11-089	North	0	41	41	0.78	0.45	1.8
TEP-11-093	North	0	67.95	67.95	0.64	0.67	0.9
TEP-11-094	North	18.65	224.7	206.05	0.19	0.42	0.6
TEP-11-102	North	0	137	137	0.23	0.47	0.7
TEP-11-110	North	0	78	78	0.32	0.3	1.4
TEP-11-113	North	0	179.35	179.35	0.24	0.54	1.1
TEP-11-115	North	0	54.45	54.45	0.32	0.73	1.3
TEP-11-120	North	0	119.6	119.6	0.19	0.3	1.2
TEP-11-125	North	0	122.05	122.05	0.25	0.6	0.9
TEP-11-128	South	316	437.4	121.4	0.18	0.72	2.1
	including	318	401	83	0.2	0.89	2.3

Hole No.	Zone	From (m)	To (m)	Length (m)	Cu (%)	Au (g/t)	Ag (g/t)
TEP-11-130	South	149.75	253.7	103.95	0.12	0.22	2.5
	and	284.25	439.2	154.95	0.24	0.41	1.2
TIZ-11-003	Tizate	25.9	154	128.1	0.2	0.13	3.2
TIZ-11-006	Tizate	182	255	73	0.2	0.13	2.9
TIZ-11-007	Tizate	0	41	41	0.15	0.08	3.3
TIZ-11-011	Tizate	5.25	100.95	95.7	0.13	0.21	1.4
TIZ-11-013	Tizate	76.8	173.4	96.6	0.16	0.13	2.4
	and	218	320	102	0.22	0.14	4
TIZ-11-017	Tizate	60.4	301.04	240.65	0.2	0.18	2.3
TIZ-11-019	Tizate	87	148.55	61.55	0.18	0.15	1.3
TIZ-11-021	Tizate	123.9	229	105.1	0.2	0.16	1.5
TIZ-11-023	Tizate	0	97.75	97.75	0.2	0.17	1.4
TIZ-11-025	Tizate	6	106.8	100.8	0.19	0.08	1.2
TIZ-11-027	Tizate	0	42	42	0.16	0.15	1.4
TIZ-11-035	Tizate	0	63	63	0.24	0.27	5.1
TIZ-11-037	Tizate	0	63.1	63.1	0.2	0.23	3.9
TIZ-11-050	Tizate	0	85	85	0.18	0.34	1.7
TIZ-11-056	Tizate	0	92.15	92.15	0.31	0.21	1.8
TIZ-11-057	Tizate	0	107.9	107.9	0.17	0.21	2.5
TIZ-11-061	Tizate	0	140.65	140.65	0.19	0.26	1.9
TIZ-11-062	Tizate	4	230.05	226.05	0.15	0.32	1
TIZ-11-063	Tizate	52.2	193.6	141.4	0.21	0.19	2
TIZ-11-065	Tizate	5.15	238	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

No information is known regarding the sample preparation, analysis and security methods employed by INCO nor is it known whether INCO employed a quality control/quality assurance program for their drill programs.

11.2 Teck

No information is available regarding 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 30 g 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 sulphuric 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

No information is available regarding sample preparation, analysis and security methods during the Hecla drill programs. It is also not 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 2 m sample intervals within the mineralized zones apart from where broken ground and/or specific geological conditions determine otherwise.

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

Core was transported from site to the processing facility, in Tepalcatepec, 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 ten 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 laboratories in Durango, Durango State, Mexico for sample preparation and the pulps were then shipped to Inspectorate laboratories in Reno, Nevada for analysis.

Sampling issues were identified by ACA Howe. Certified reference material (CRM) that was assayed at Inspectorate Labs using the three acid digestion and ICP finish method returned copper results that were generally erratic and higher than expected.

To address this, a full review of Inspectorate analytical techniques was undertaken. It was recognized through this review that sample preparation for the three 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 2 mm followed by the pulverization of a 150 g 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 3 ppm Au used an aqua regia digestion with an AAS finish (Detection range was 0.005 to 10 ppm Au). Samples over 3 ppm Au used the fire assay method with a gravimetric finish (Detection range was 0.005 to 100 ppm Au).

Copper analysis used an aqua regia digestion with an AAS finish (Detection range was 0.2 to 10,000 ppm 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 were then air freighted to the ALS analytical laboratories in Vancouver, Canada, for analysis.

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

Copper analysis for samples below 10,000 ppm Cu used a three acid digestion with an ICP analysis (Detection range was 0.2 to 10,000 ppm Cu). Samples over 10,000 ppm 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 believed that the appropriate steps were taken to identify and re-assay the samples. Micon deemed the resulting Arian assays presented by Geologix appropriate for use in a Mineral Resource estimate.

11.5 Geologix

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

In 2010, core was transported from site to the processing facility, located in the grounds of the building that the Company currently occupies in Tepalcatepec, 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 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 when the rice bags were sealed, 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, BC.

All samples were assayed for gold by aqua regia digest with AAS finish on a 30 g sample and by ICP-AES for 33 elements, including copper, using a four acid "near total" digestion. High-grade gold (>10.0 g/t) samples were re-analyzed using fire assay with a gravimetric finish. High-grade (>10,000 ppm) copper samples were re-analyzed on a single element basis using an mineralized material grade four acid digestion with Inductively Coupled Plasma atomic emission spectroscopy (ICP-AES) finish.

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

ALS Chemex is an ISO 9001 and ISO 17025:2005 accredited facility. Micon believed that the sampling, transportation, preparation and analysis were considered consistent with exploration best practices for this type of deposit and were 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 Verification by 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 (four per 100 samples).
- The inclusion of field blanks and pulp blanks to assess laboratory sample preparation and analytical accuracy (three per 100 samples).
- The inclusion of field duplicates and externally assayed pulp duplicates to assess sample preparation and precision (three 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, BC 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	N/A	0.057	N/A

Source: Micon (2012)

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 three results returned values marginally greater than the lower limit of detection 0.5 ppm Au and were well within tolerance limits, returning values of up to 0.009 ppm Au. Copper grades in field blanks were on the whole acceptable with 67% returning values below one 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 re-analysis. Gold grades for pulp blanks showed that 67% of returned grades were below the limit of detection. Of the remaining samples eight returned values greater than 0.01 ppm Au, including one outlier, sample 145521 at 0.08 ppm Au. Copper values were much more variable with only 52% returning values below one 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 g 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 coordinates 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 coordinates 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 Verification by 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 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 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. 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, BC and Ore Research and Exploration Pty Ltd. of Australia.

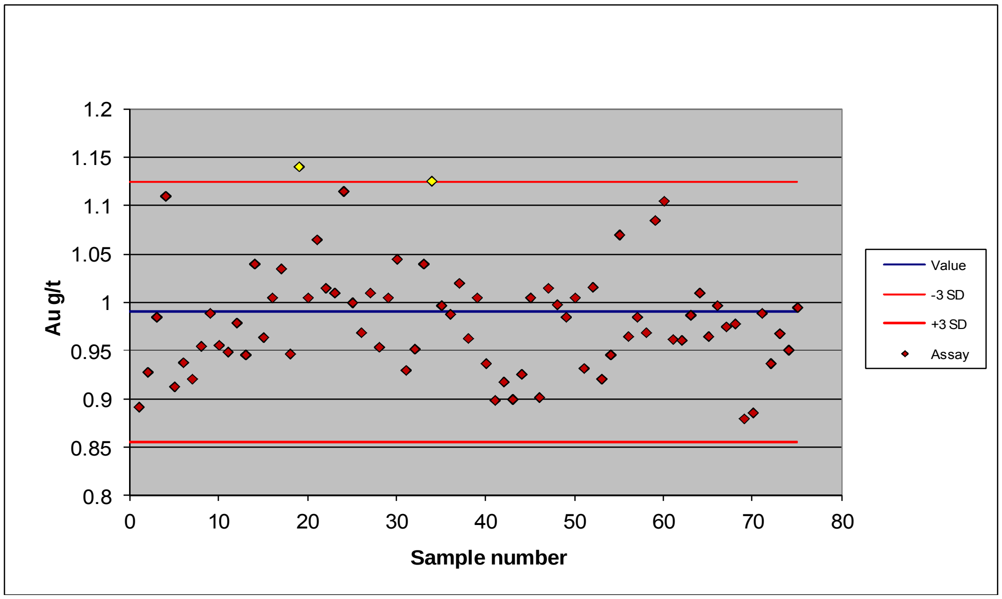
Table 12.2: Geologix CRM Statistics

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	0.1	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

Source: Geologix (2016)

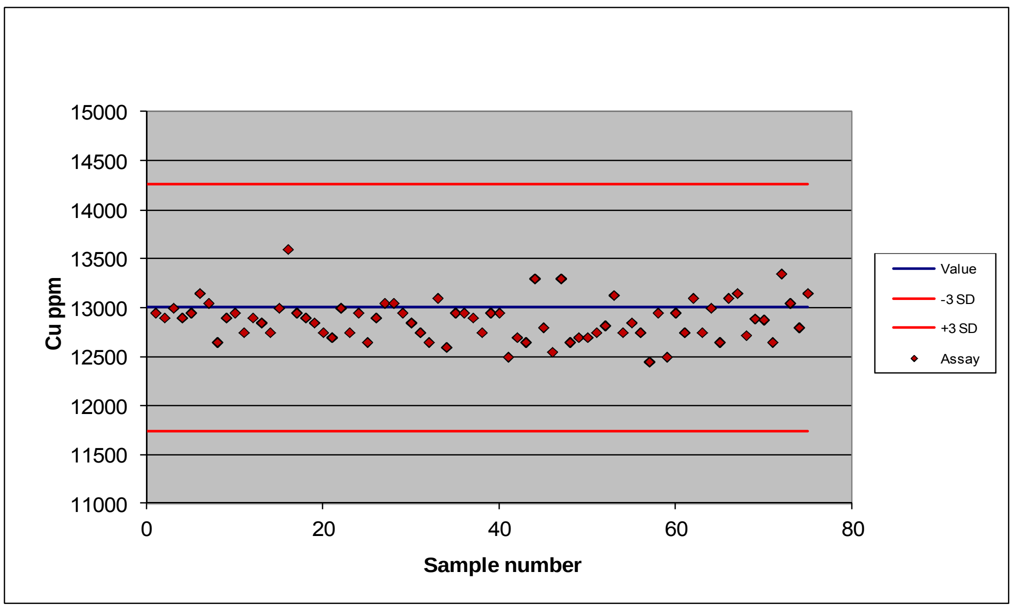
CRM samples were routinely submitted for assaying with core at a ratio of up to 1:30, totalling 4% of all samples. Initial drilling utilized CDNCGS-21, CDNCGS-23, 50pb and 52pb while the 2011 used 52c, 151a, 152a and 153a. Error plots for each CRM for gold and copper are presented in the following pages (Figures 12.1 to 12.18). Failures are identified as yellow squares in each plot.

Figure 12.1: CRM - CDN-CGS-21 - Au Values



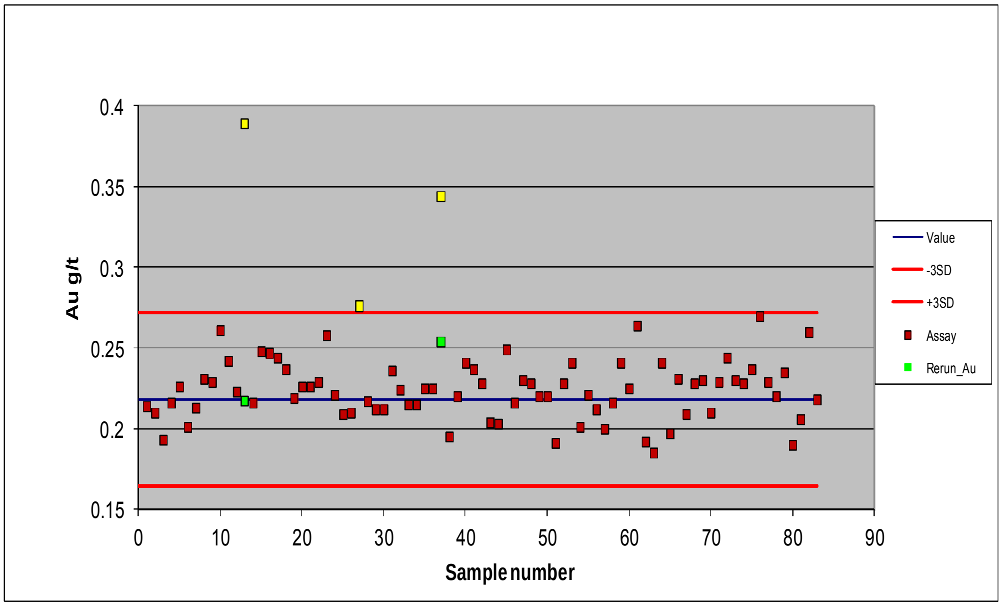
Source: Micon (2012)

Figure 12.2: CRM - CDN-CGS-21 - Cu Values



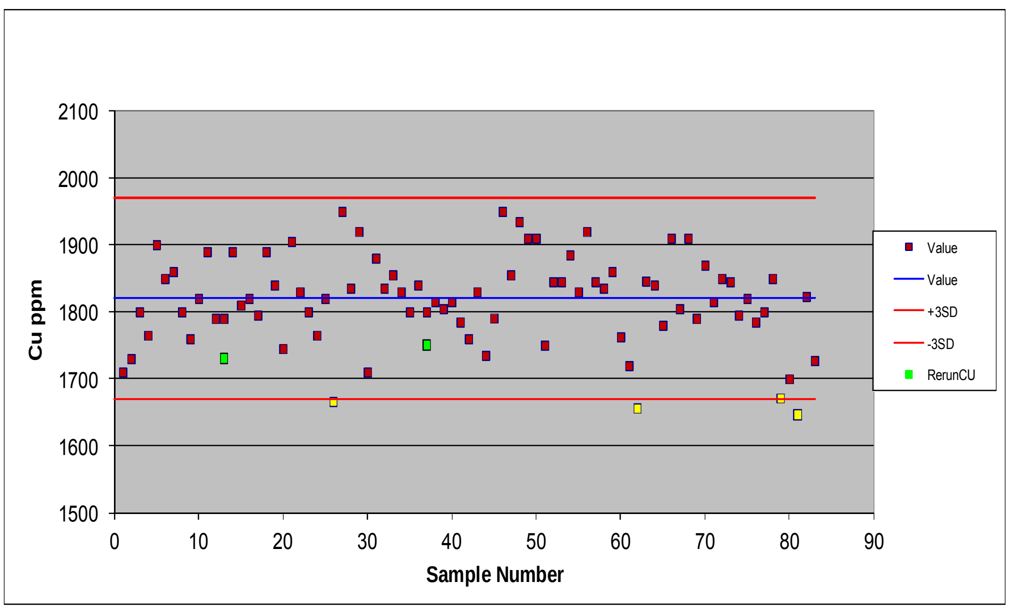
Source: Micon (2012)

Figure 12.3: CRM - CDN-CGS-23 - Au Values



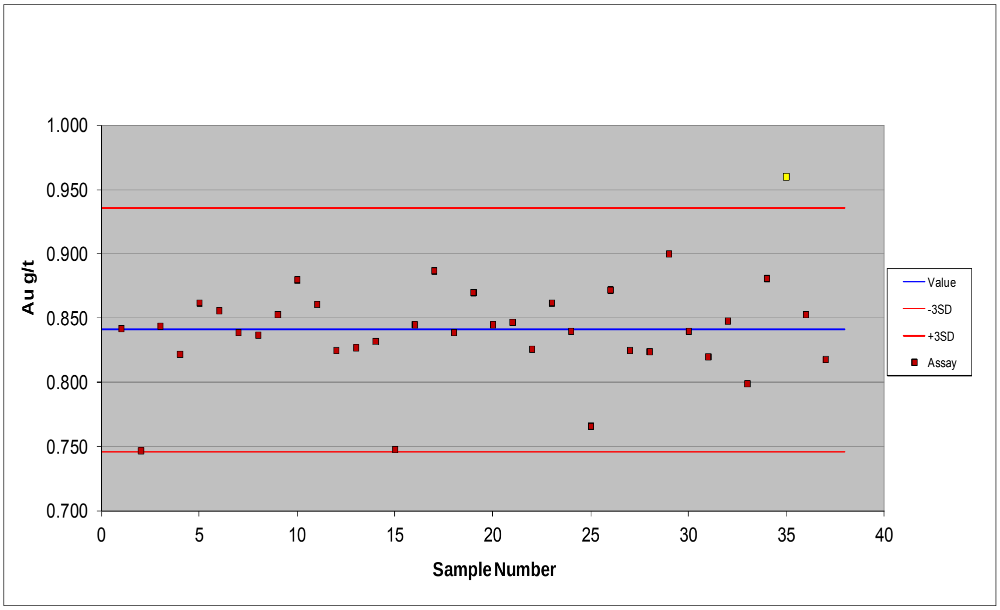
Source: Micon (2012)

Figure 12.4: CRM - CDN-CGS-23 - Cu Values



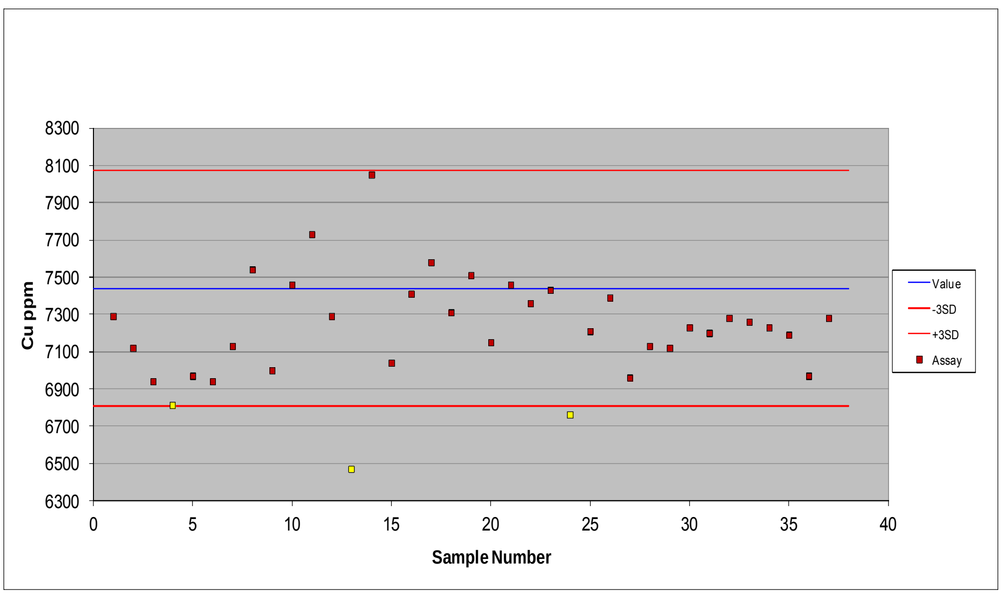
Source: Micon (2012)

Figure 12.5: CRM - Oreas-50Pb - Au Values



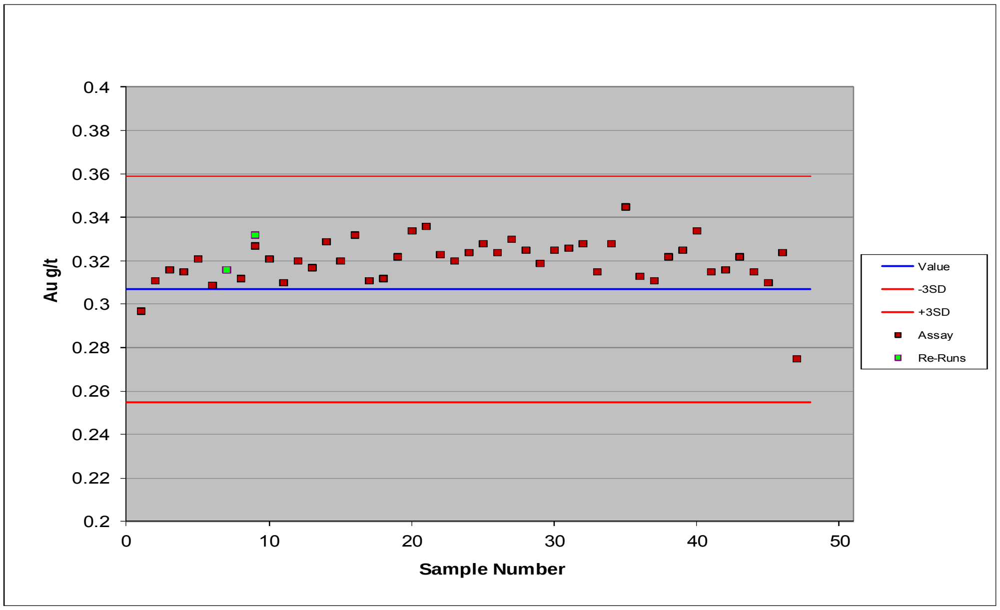
Source: Micon (2012)

Figure 12.6: CRM - Oreas-50Pb - Cu Values



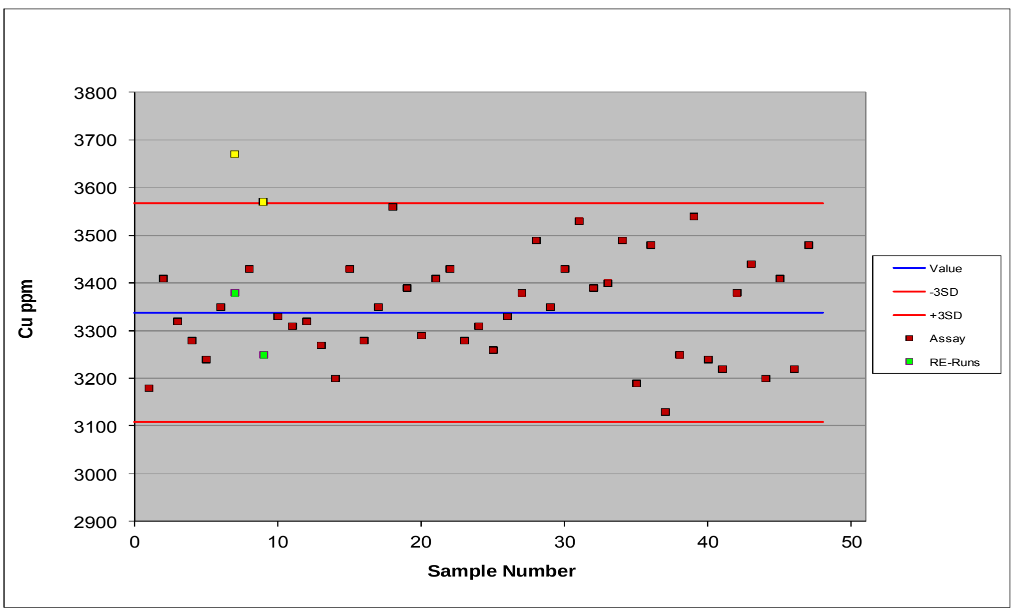
Source: Micon (2012)

Figure 12.7: CRM - Oreas-52Pb - Au Values



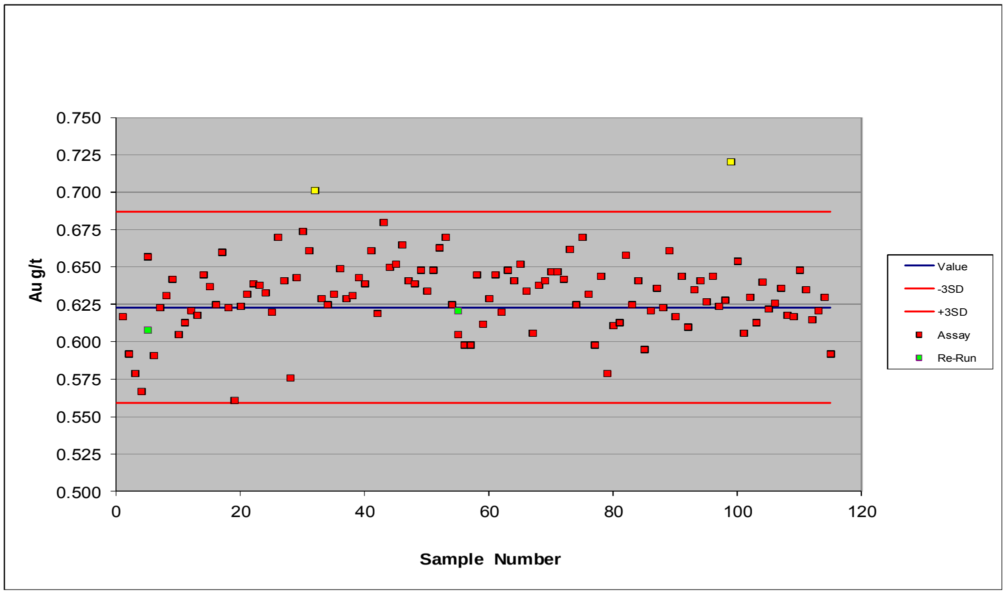
Source: Micon (2012)

Figure 12.8: CRM - Oreas-52Pb - Cu Values



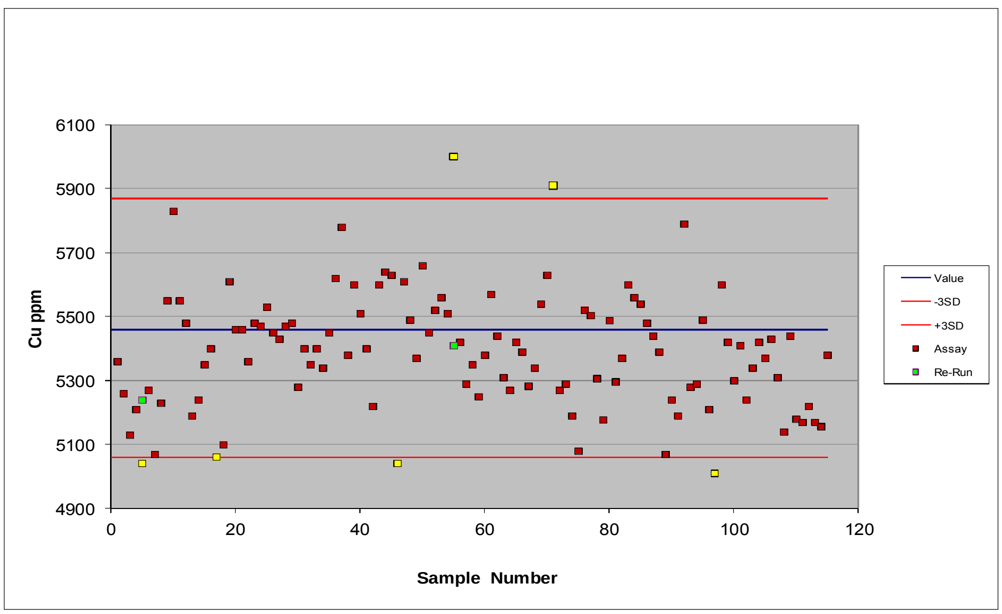
Source: Micon (2012)

Figure 12.9: CRM - Oreas-53Pb - Au Values



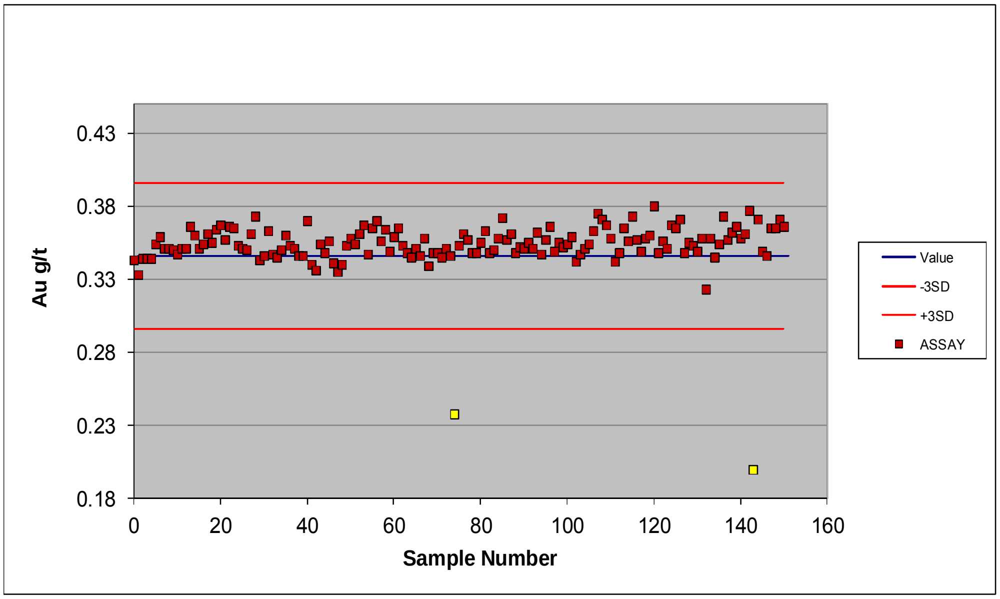
Source: Micon (2012)

Figure 12.10: CRM - Oreas-53Pb - Cu Values



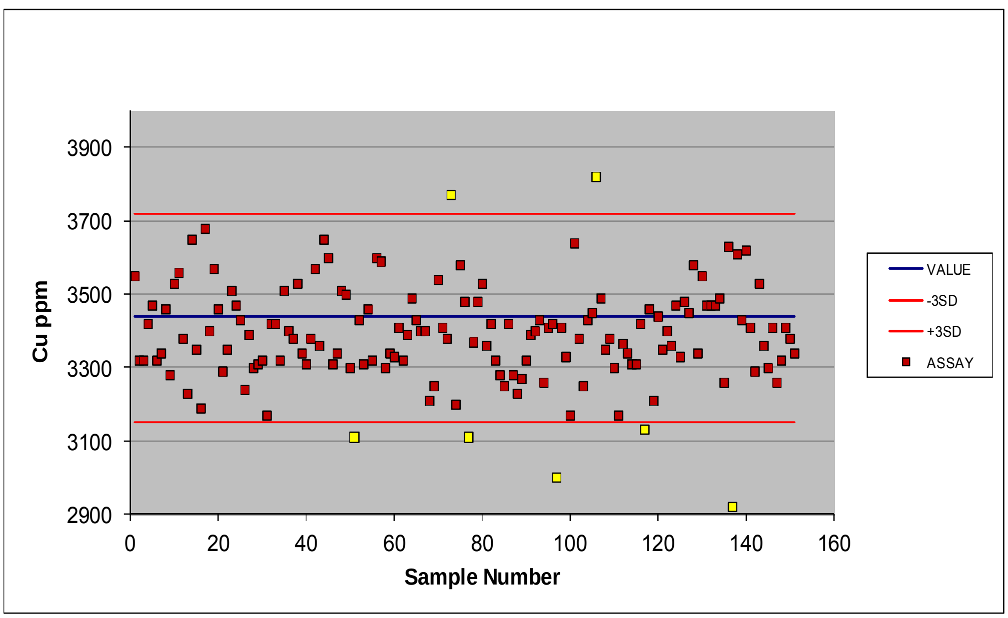
Source: Micon (2012)

Figure 12.11: CRM - Oreas-52c - Au Values



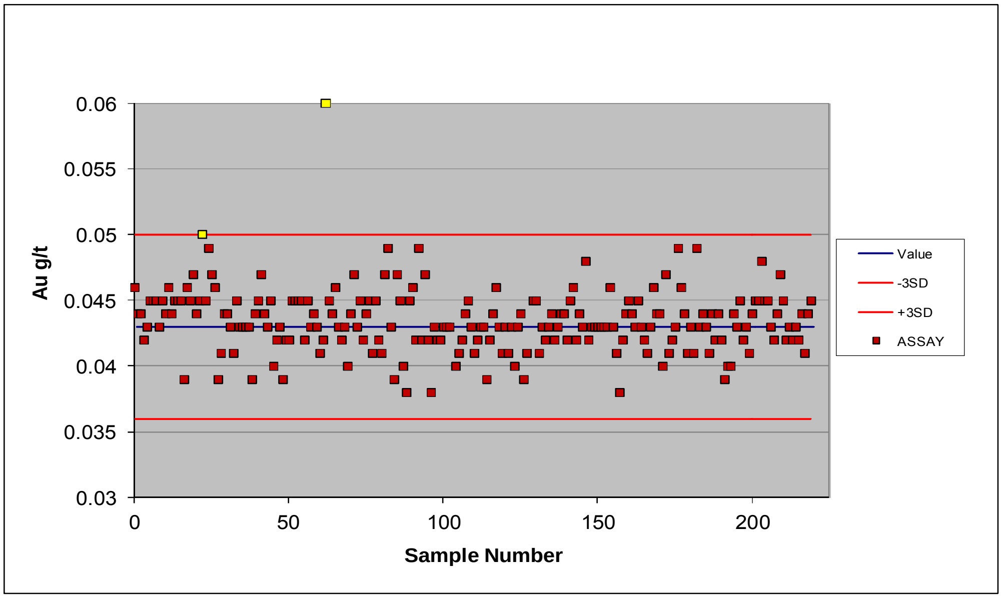
Source: Micon (2012)

Figure 12.12: CRM - Oreas-52c - Cu Values



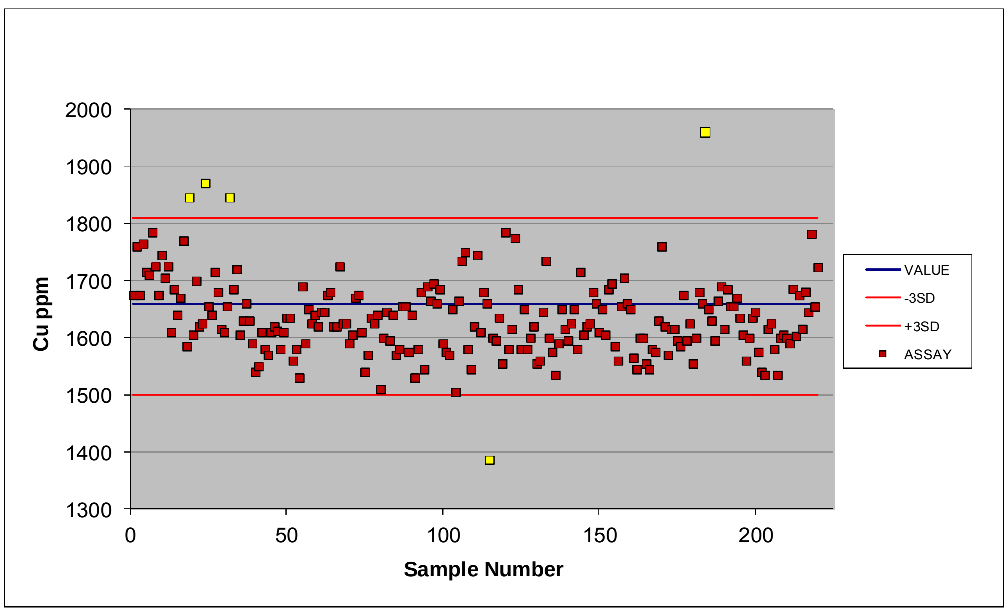
Source: Micon (2012)

Figure 12.13: CRM - Oreas-151a - Au Values



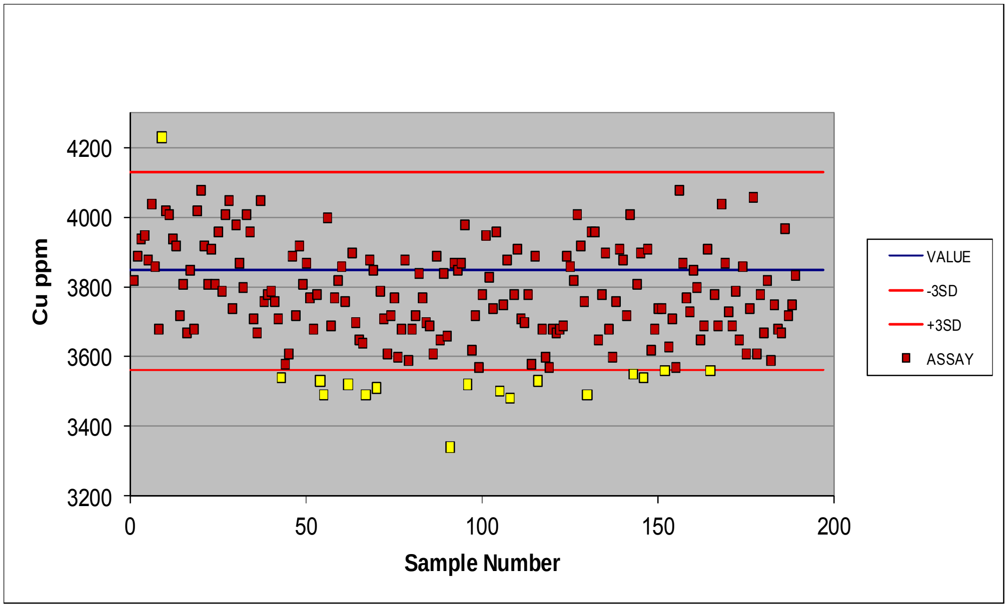
Source: Micon (2012)

Figure 12.14: CRM - Oreas-151a - Cu Values



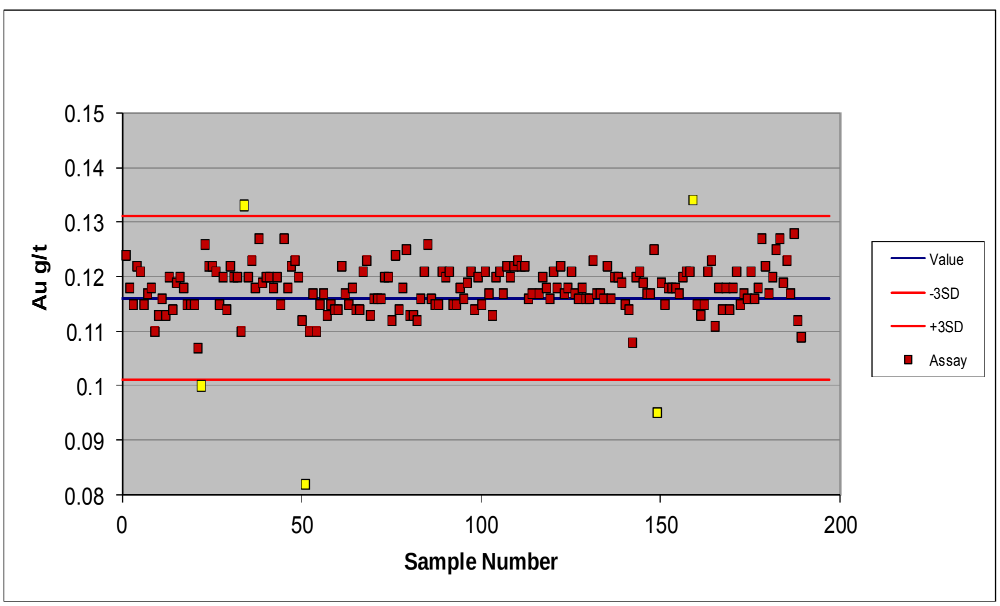
Source: Micon (2012)

Figure 12.15: CRM - Oreas-152a - Cu Values



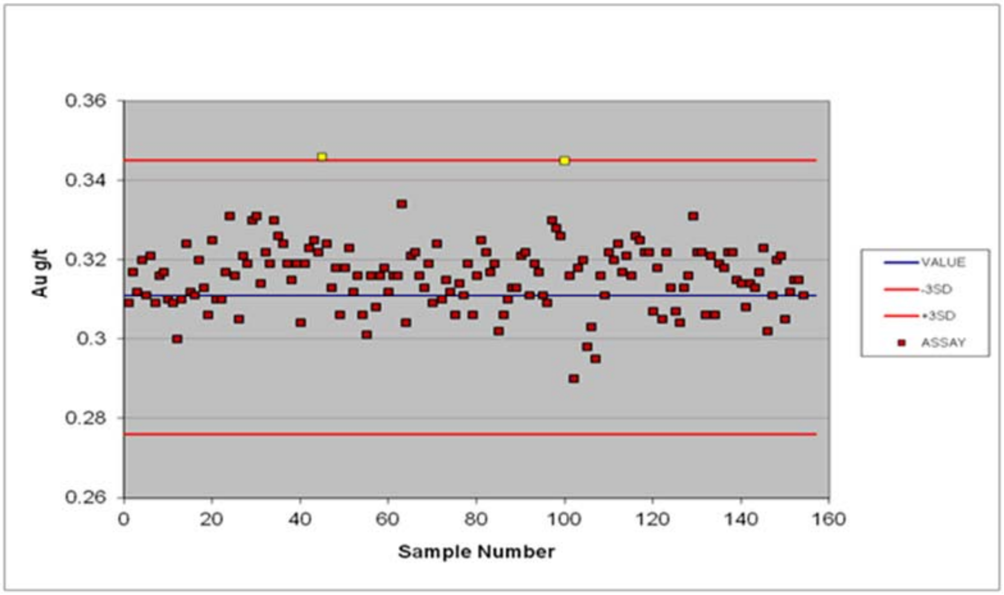
Source: Micon (2012)

Figure 12.16: CRM - Oreas-152a - Au Values



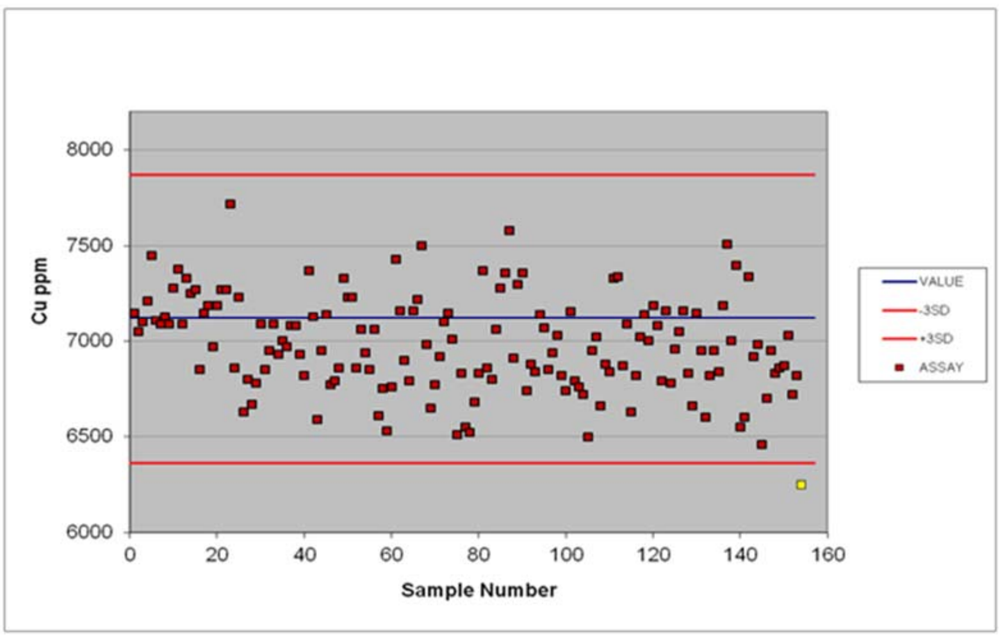
Source: Micon (2012)

Figure 12.17: CRM - Oreas-153a - Au Values



Source: Micon (2012)

Figure 12.18: CRM - Oreas-153a - Cu Values



Source: Micon (2012)

Most of the CRM for both gold and copper fall well within the ±2 standard deviations of the expected value. Of the failed CRMs (±3 standard deviations), 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.

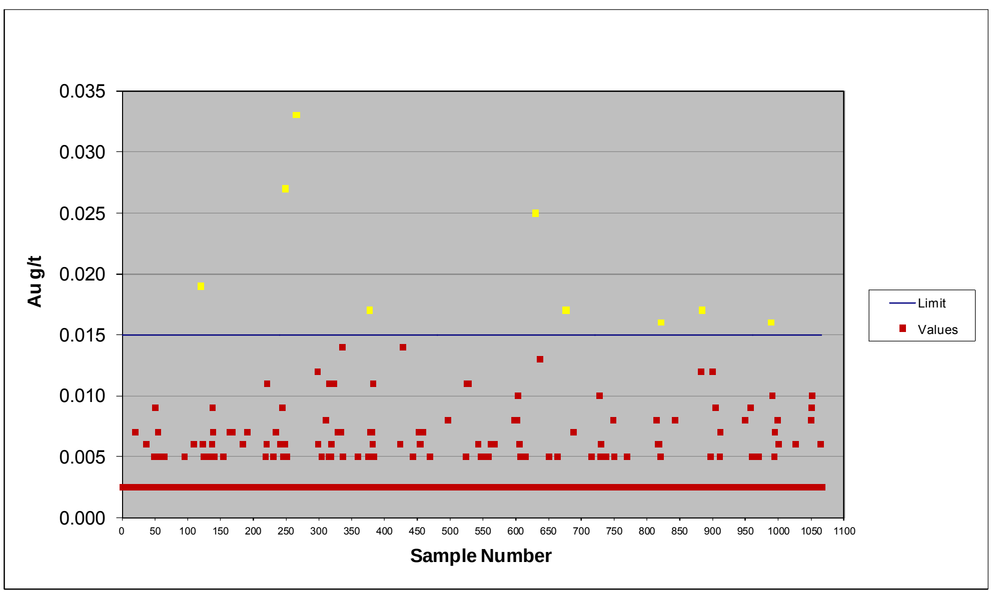
New or fresh CRMs may alleviate the random but minor failed CRM assays. Micon believes that the procedures in place for CRM are to industry standards and that the resultant assays reflect the mineralization within the deposits.

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.

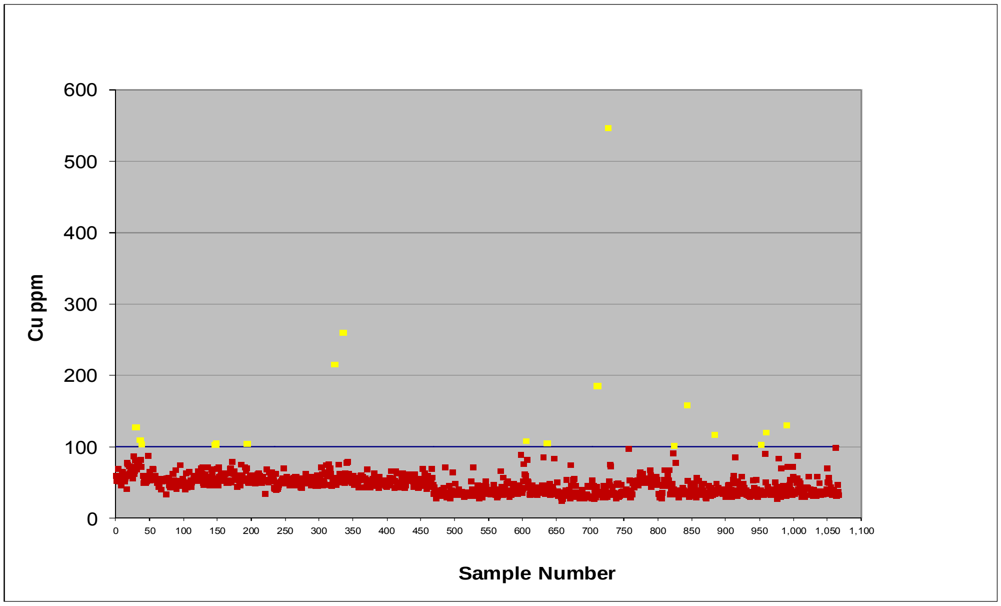
Blanks were obtained from two locations within the concessions but away from the known deposits (Location 1: 720954 E, 2115284 N and Location 2: 719423 E, 2115012 N). The blanks were identified as non-mineralized porphyritic andesite and non-mineralized granodiorite.

Figure 12.19 : Blank – Analyses Au (g/t)



Source: Micon (2012)

Figure 12.20: Blank - Analyses Cu (ppm)



Source: Micon (2012)

There were 1,067 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

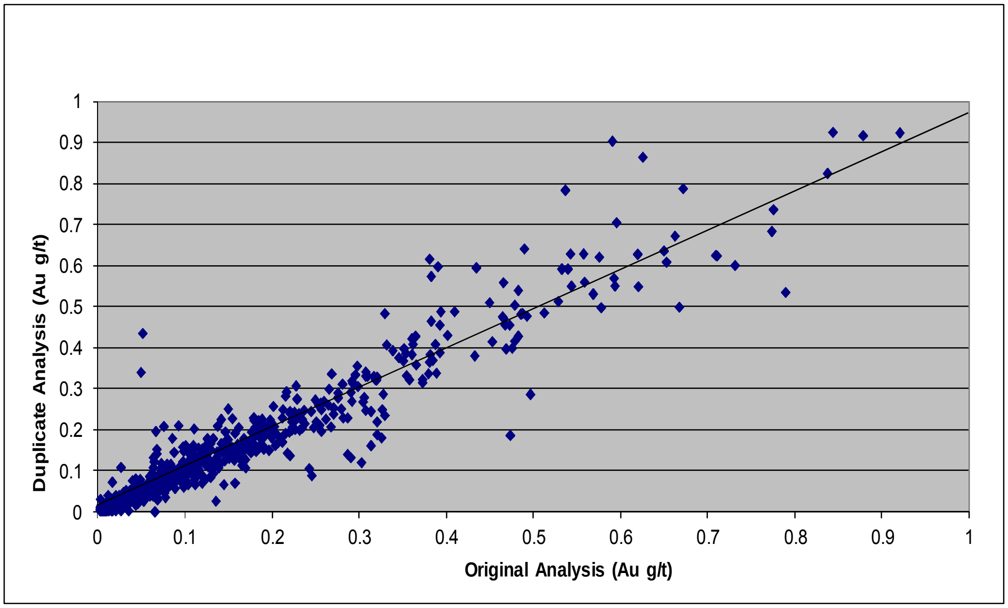
Source: Micon (2012)

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

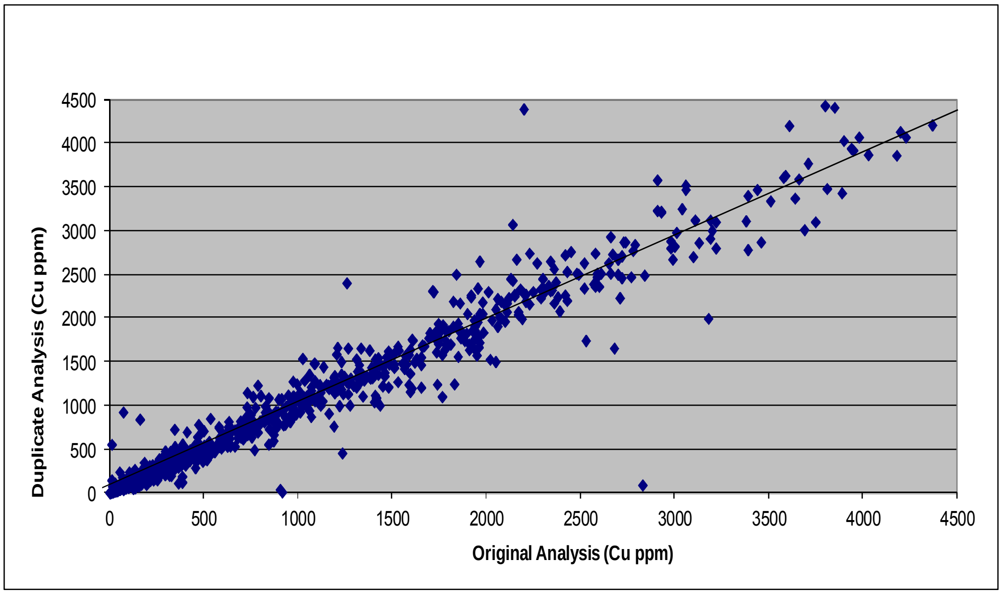
There were 1,048 duplicate core samples assayed in the sample stream. Duplicates samples were prepared by sawing the core in half and sending both halves of the core for assay. Assays were part of the ALS sample stream. There is a very good correlation for both gold and copper for the duplicate assays 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



Source: Micon (2012)

Figure 12.22: Tepal Core Duplicates - Cu



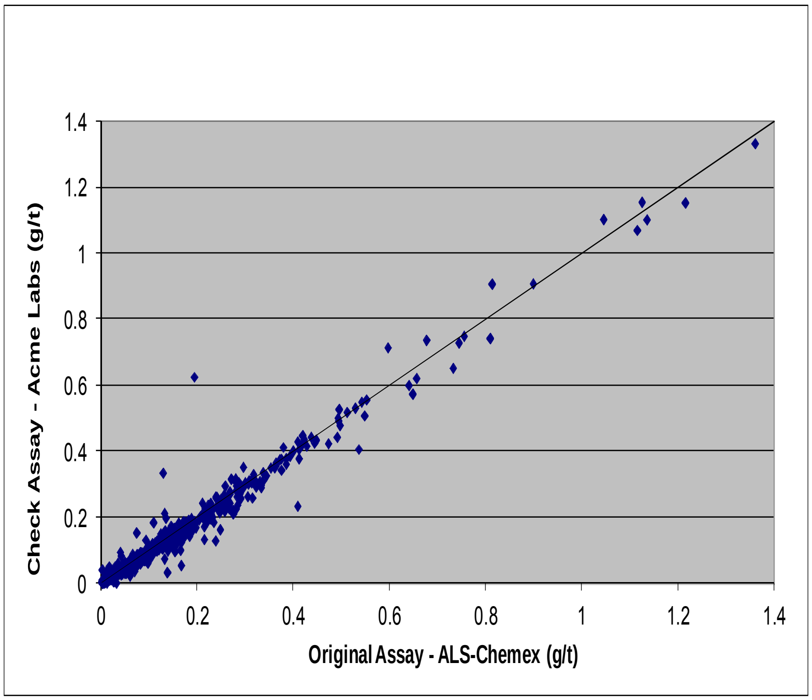
Source: Micon (2012)

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-recognized laboratory based in Vancouver. The laboratory maintains ISO 9001:2000 and has been approved for ISO/IEC 17025:2005 accreditation.

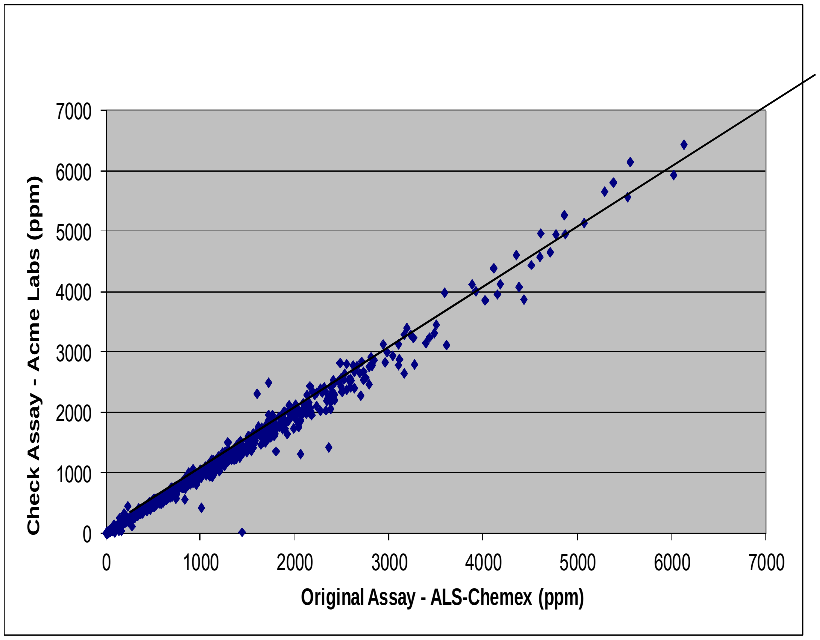
The results from the pulp re-assay program for gold, copper, silver and molybdenum are illustrated in Figures 12.23 to 12.26 respectively. The results seem to indicate that ALS is reporting slightly higher than Acme for silver. Values for gold, copper and molybdenum appear to correlate very well between the original lab and Acme labs.

Figure 12.23: Gold Check Assays



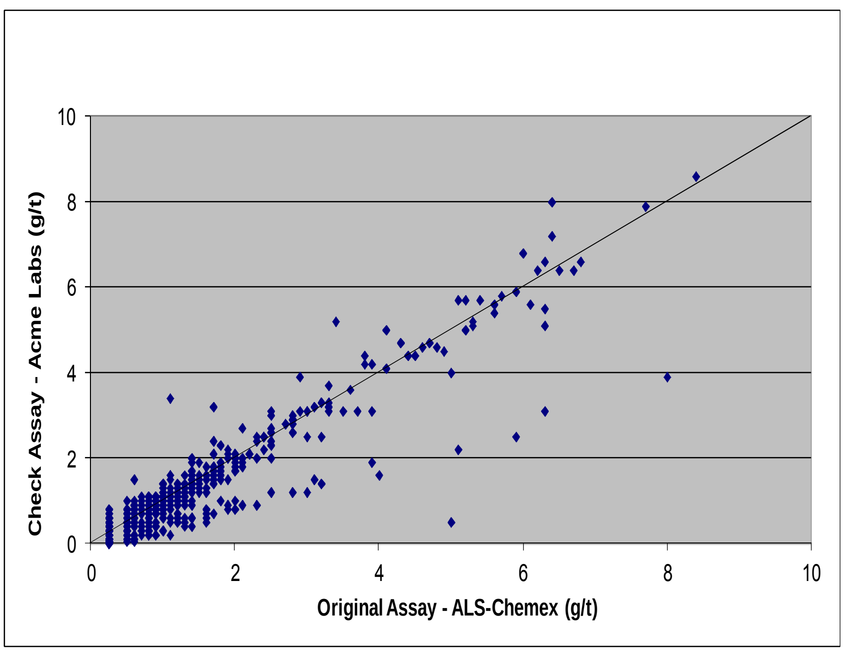
Source: Micon (2012)

Figure 12.24: Copper Check Assays



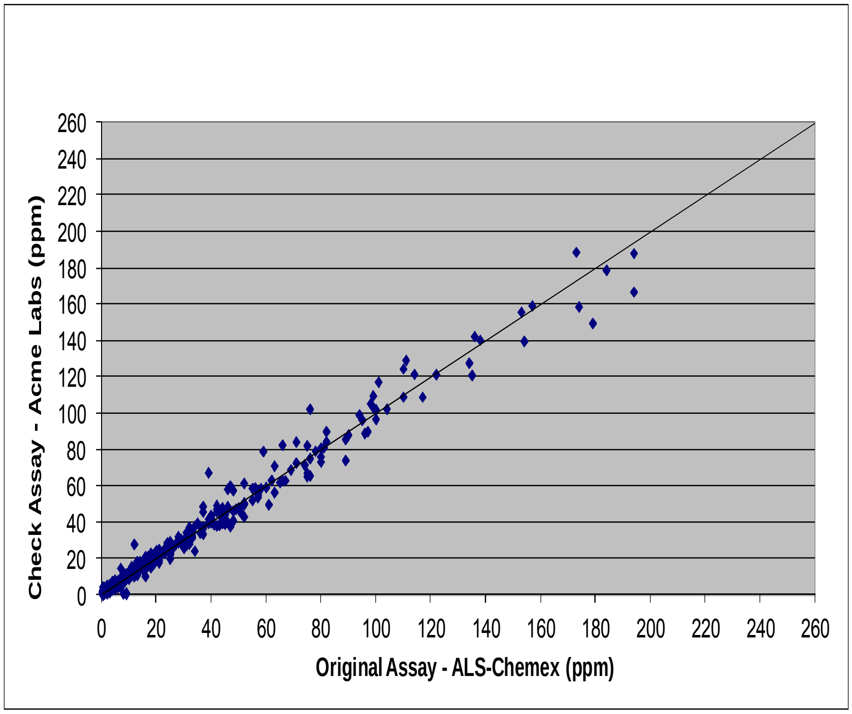
Source: Micon (2012)

Figure 12.25: Silver Check Assays



Source: Micon (2012)

Figure 12.26: Molybdenum Check Assays



Source: Micon (2012)

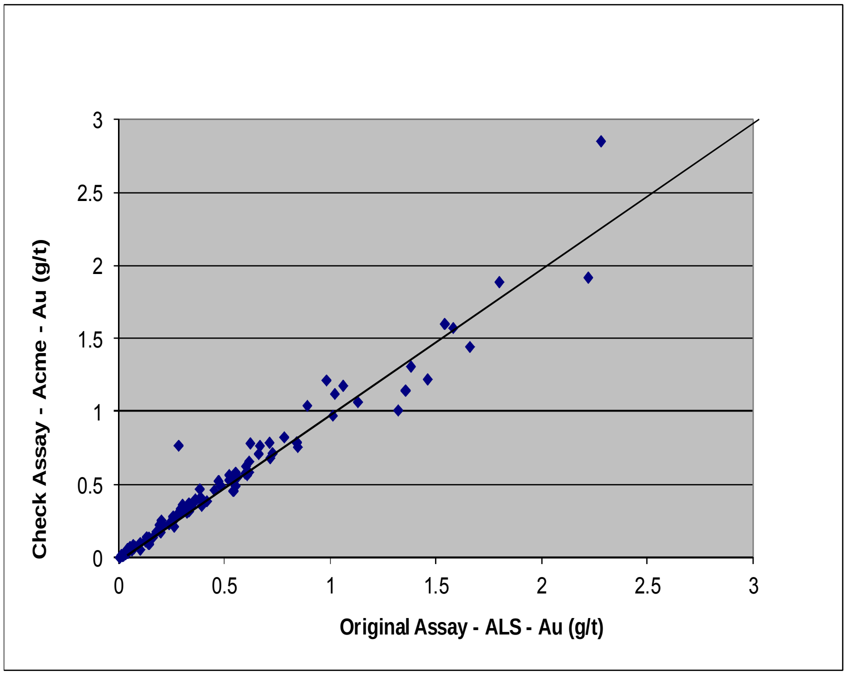
12.6 Historic Check Assays

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.

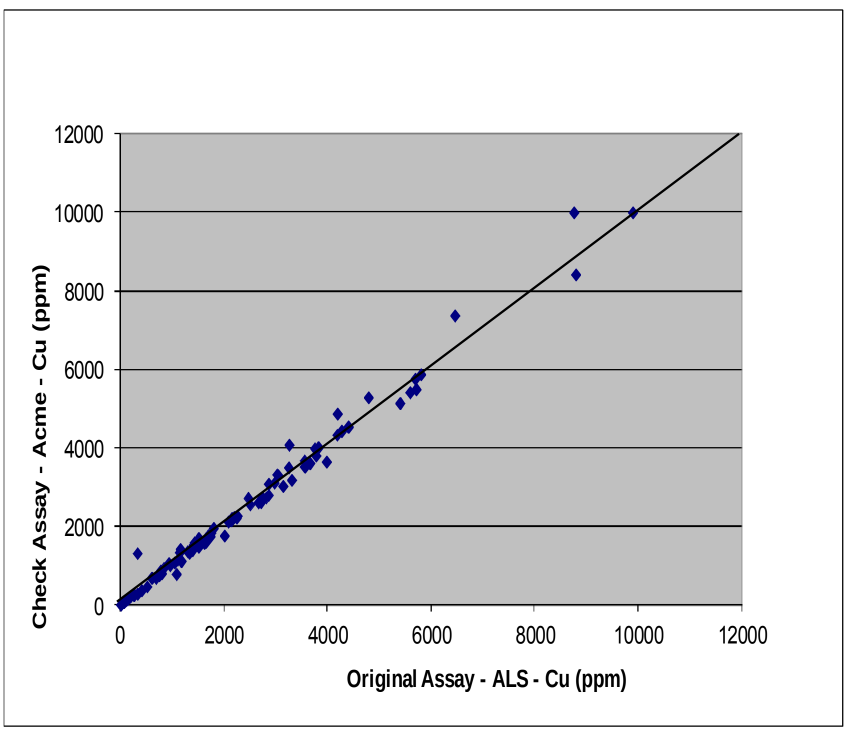
Results of the re-assay program returned very similar results to the original data entered in the database for the historical drill holes in most cases. There was a wider scatter of Teck gold values than Teck copper values. As the grades increased especially for gold there was some scatter of data, but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

Figure 12.27: Historic Hecla Gold Check Assays



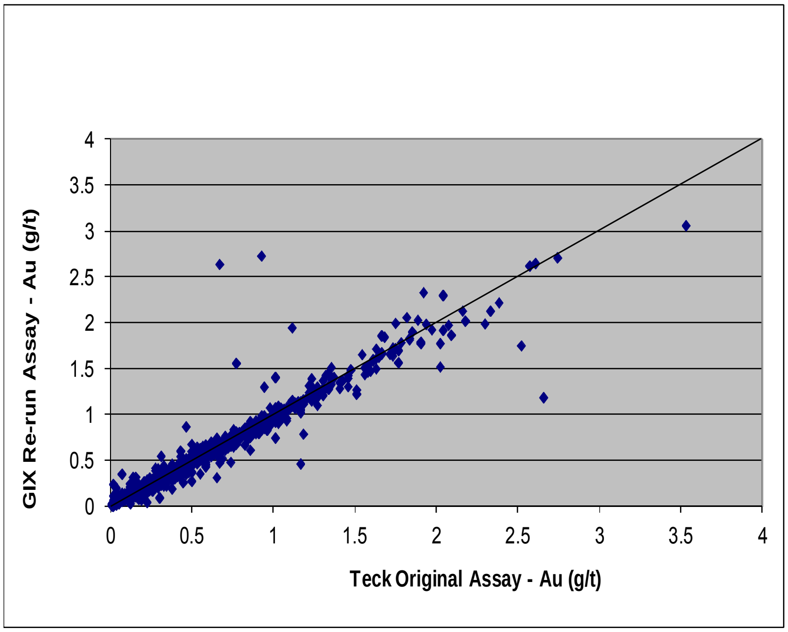
Source: Micon (2012)

Figure 12.28: Historic Hecla Copper Check Assays



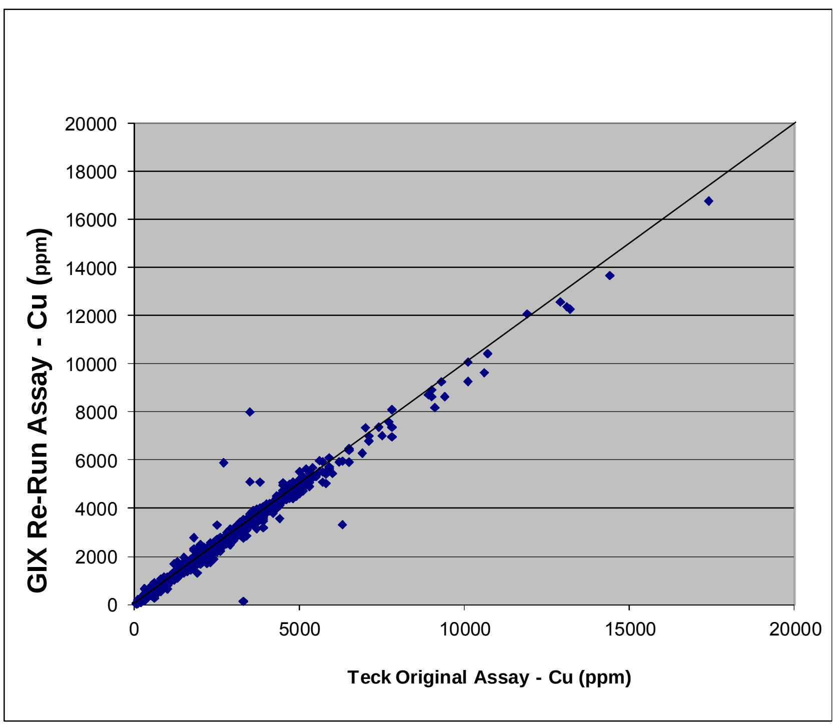
Source: Micon (2012)

Figure 12.29: Historic Teck Gold Check Assays



Source: Micon (2012)

Figure 12.30: Historic Teck Copper Check Assays



Source: Micon (2012)

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 coordinates 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 included the Hecla drill holes in the drill hole database and the 2012 Mineral Resource estimate.

12.8 Micon Database Validation

Micon obtained the digital scans of 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 deemed the digital database was clean of errors and was 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 No.	Geologix			Micon			Difference		
		N (m)	E (m)	El. (m)	N (m)	E (m)	El. (m)	N (m)	E (m)	El. (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

Source: Micon (2012)

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 was confident that the locations documented for the drilling were accurate.

12.9 Validation Summary

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

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

13 Mineral Processing and Metallurgical Testing

13.1 Background

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 have been considered similar since they contain similar copper and gold grades (see variability results in Table 13.5) and are of similar rock hardness. Tizate has been considered slightly different since it is of lower grade and is harder than the NZ/SZ mineralized material.

There has been no additional test work since the Tepal 2013 PFS was completed. The design criteria for the sulphide circuit flowsheet are the same utilizing the identical product grind size, retention times and reagents. The PFS design was for 35,000 t/d and this PEA has reduced the throughput to 22,000 t/d with modifications to equipment size only. The oxide circuit has two modifications from the 2013 PFS. Firstly, the PFS oxide grind circuit was batch processed through the sulphide SAG and ball mill and stored in a pond for reclamation and processing through the CIL/ADR circuit. This circuit has been removed and replaced with a stand-alone dedicated oxide secondary and tertiary crushing plant and grinding circuit with a single ball mill operating in closed circuit with cyclones. Secondly, the oxide CIL retention time has been increased from eight to 24 hours.

Sulphide feed hardness will be variable in the three pits, with the NZ and SZ being the moderately hard and Tizate being hard. Over 42 variability tests were completed with Bond Work index (BWi) hardness ranging from a low of 11.0 kWh/t to a high of 20.0 kWh/t, (SRK, 2012, Grinding and Crushing Circuit Equipment Sizing). Due to this variation, the milling circuit is designed using the 80% hardness BWi of 17.5 kWh/t for NZ and SZ and 20 kWh/t for the harder Tizate material. Additional power will be required to process at the target tonnage for Tizate. The oxide feed is soft from all three areas and will process material at a design capacity of 5,500 t/d.

The saleable products for this PEA will be a copper concentrate with gold and silver values obtained from a sulphide flotation, and a gold/silver doré bar from the site refinery. Molybdenum will be contained in the concentrate but is not considered payable in this study. A molybdenum separating flotation step will be 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 remaining gold and silver in the combined pyrite concentrate & 1st cleaner tailings will be extracted in the CIL and ADR circuits.

The surface oxides contain copper, gold and silver values; however, only the gold and silver is designed to be recovered for this PEA in a CIL circuit, carbon plant and refinery.

PEA average sulphide and oxide grades are approximately identical to the PFS and are presented in Table 13.1.

Table 13.1: Average Mill Feed Sulphide and Oxide Grades

Product	Cu (%)	Au (g/t)	Ag (g/t)
2013 PFS LOM Sulphide Head Grade	0.20	0.30	1.49
2016 PEA LOM Sulphide Head Grade	0.21	0.33	1.47
2013 PFS LOM Oxide Head Grade		0.42	1.25
2016 PEA LOM Oxide Head Grade		0.45	1.11

Source: JDS (2017)

The QP of this section confirms that, to the extent known, the test samples are representative of the various types and styles of mineralization and the mineral deposit as a whole, and that no extraordinary processing factors or deleterious elements exist that could have a significant effect on potential economic extraction.

13.2 Historical Metallurgical Testing

Metallurgical testing was first performed on the NZ and SZ in 1973 by INCO Ltd, and in the mid-1990's by Teck-Cominco Corporation. Further tests were performed in 2009 and 2010 to support a Preliminary Assessment (PA), following the NI 43-101 guidelines, done by SRK Consulting Inc. (SRK) on October 8, 2010 which was updated in another NI 43-101 report on April 29, 2011 with an increase in throughput from 25,000 to 35,000 t/d.

Data from the locked cycle flotation tests performed in 2010, and used in the PA and PEA reports, were used in this current PEA report. Only the NZ and SZ oxide feed had cyanide leach column tests performed for the April 2011 PEA report. Additional tests carried out since April 2011 included leach column tests and cyanide bottle roll tests on the Tizate oxide feed and variability sampling and flotation tests on the NZ, SZ, and Tizate sulphides, which are included in this PEA.

From the review of all test results, covellite was the only detrimental mineral found in the oxide feed that leaches copper to solution along with the desired gold. Additional test work is recommended to determine the effect on carbon loading.

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 continued until 2009 when further float and leach tests were commissioned by Geologix Explorations Inc. to support a PA.

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 mineralized material, whose results are presented in Table 13.2. The concentrate grade was poor but recoveries were reasonable, with recommendations to regrind the concentrate to achieve a saleable copper concentrate grade above 25%.

Table 13.2: 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

Source: Micon (2012)

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 1.07 to 1.36 g/t Au. Recoveries were good at 84% Au recovered to solution with a medium cyanide consumption of 0.75 kg/t.

These tests concluded that the mineralized feed 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 Programs - 2009 to 2013

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

The flotation results are summarized in G&T's August 2010 report "Metallurgical Assessment of the Tepal Project", 2012 "Metallurgical Assessment of the Tepal and Tizate Zones", 2012 "Variability Metallurgical Assessment", 2013 Further Metallurgical Testing of the Tepal Oxide Zones", and 2013 "Metallurgical Testing of Tepal Sulphide Ore".

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 2009-2013 NZ/SZ flotation and leach tests are summarized in Table 13.3. These results were used as the design criteria for the 2013 PFS and this PEA.

The concentrates had minor element assays performed to determine if any deleterious elements would diminish the value when calculating a NSR for this resource. The results in Table 13.4 are the minor element assays from the two locked cycle tests completed by G&T in 2010 used to calculate the NSR for this PEA.

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 cyanidation 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 oxide mineralized material. G&T Metallurgical Services also performed variability tests on 42 core samples from the NZ, SZ and Tizate Zones as well as further flotation tests on sulphide feed and oxide grind CN bottle roll tests in 2012 and 2013. The flotation results are shown in Table 13.5.

Table 13.3: 2013 PFS Metallurgical Design Criteria Summary

Product	Unit	Flotation	Oxide Leach
Resource Grade			
Tepal Grade			
Copper	%	0.22	N/A
Gold	g/t	0.37	0.47
Silver	g/t	1.02	0.99
Tizate Grade			
Copper	%	0.17	N/A
Gold	g/t	0.19	0.26
Silver	g/t	2.23	2.20
Recovery			
Tepal Recovery			
Copper	%	88.2	N/A
Gold	%	62.4	82.3
Silver	%	27.4	63.3
Tizate Recovery			
Copper	%	85.9	N/A
Gold	%	58.0	75.2
Silver	%	59.6	55.9
Concentrate Grade			
Concentrate Grade - Tepal			
Copper	%	25.7	N/A
Gold	g/t	32.8	as doré

Product	Unit	Flotation	Oxide Leach
Silver	g/t	42.9	as doré
Concentrate Grade - Tizate			
Copper	%	26.9	N/A
Gold	g/t	15.0	as doré
Silver	g/t	267.6	as doré

Source: Micon (2012)

Table 13.4: Concentrate Minor Element Assays

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

Source: G&T (2010)

No deleterious elements were found in the metallurgical tests reviewed.

Table 13.5: 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

Source : G&T KM2944 2012

G&T's KM2944 Variability Metallurgical Assessment concluded that the mineral composition of the sulphide material in all 42 samples was relatively consistent across all zones (reported 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.

Work indices are shown in Table 13.6 to Table 13.8.. The flowsheet treating both NZ/SZ and Tizate is described in Section 17 and consists of conventional crushing, grinding, flotation, cyanidation and dewatering.

Table 13.6: Sulphide Hardness Variability Test Results, North Zone

Sample Designation	BBWI (kWh/t)	SMC Test Data								
		DWi (kWh/m^3)	DWi (%)	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)	A	b	SG	ta
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

Source : G&T KM2944 2012

Table 13.7: Sulphide Hardness Variability Test Results, South Zone

Sample Designation	BBWI kWh/t	SMC Test Data								
		DWi kWh/m^3	DWi %	Mia kWh/t	Mih kWh/t	Mic kWh/t	A	b	SG	ta
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

Source : G&T KM2944 2012

Table 13.8: Sulphide Hardness Variability Test Results, Tizate

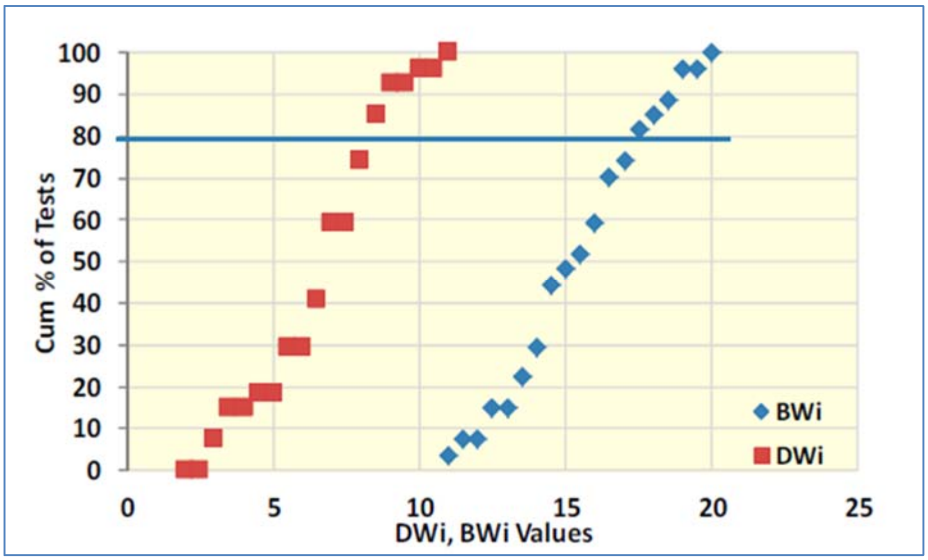
Sample Designation	BBWI kWh/t	SMC Test Data								
		DWi kWh/m3	DWi %	Mia kWh/t	Mih kWh/t	Mic kWh/t	A	b	SG	ta
Composite 2	-	7.29	71	21.2	15.9	8.2	71.6	0.51	2.66	0.36
Composite 3	-	6.99	67	20.3	15.1	7.8	60.5	0.63	2.68	0.37
Composite 4	-	9.6	89	25.5	20.3	10.5	100	0.29	2.75	0.27
Composite 5	-	8.54	83	23.4	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

Source: G&T KM2944 2012

For the variability tests, G&T concluded that, "mineralogically, the remaining sulphides are very similar across the three zones," and mineralized material hardness results for all variability samples had "Bond ball mill work indices [BWi] that ranged between 10.1 and 18.4 kWh/t." The crusher work indices were 6.5 kWh/m3 (1.9 Std. Dev.), 6.7 kWh/m3 (2.3 Std. Dev.) and 8.7 kWh/m3 (1.6 Std. Dev.) for the NZ, SZ, and Tizate Zones, respectively.

SMC and BWi tests were carried out on all sulphides, while and only the SMC tests were performed on the oxides. The 80% hardest values chosen for mill sizing included a DWi of 8.3 kWh/m3 and a BWi of 17.5 kWh/t, as shown in Figure 13.1.

Figure 13.1: Cumulative Drop Weight Index & Bond Work Index Values (Tepal)



Source: SRK Grinding Circuit Design 2012

Work indices varied greatly with the hardest mineralized material appearing at greater depth in the Tizate mineralized zone. The Tizate mineralized material was approximately 17% harder than the Tepal mineralized material.

13.4 Metallurgical Test Programs – 2013 to Present

An initial economic study was completed by Geologix 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 tailings. This would involve a simple pyrite flotation step on the chalcopyrite flotation tailings 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 alternative 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 157 µm K80;
- Cyanide leaching was conducted on whole mineralized material 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; and
- 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 flowsheets are tabulated below in Table 13.9 to Table 13.12.

Table 13.9: Oxide Flotation Flowsheet

Test Number	Composite	Mass (%)	Assay				Distribution (%)			
			Cu (%)	S (%)	Ag (g/t)	Au (g/t)	Cu	S	Ag	Au
1	NSOX	11.4	0.70	2.00	2.0	2.00	37	83	29	59
7	NSOX	10.2	0.74	2.54	3.0	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	0.30	0.42	5	0.87	17	43	41	41
Average		**9.1**	**0.32**	**0.38**	**7**	**1.09**	**15**	**40**	**33**	**39**

NOSX= North South Oxide
TOX= Tizate Oxide
Source: G&T KM3568 (2013)

Table 13.10: Gravity Flowsheet

Test Number	Composite	Grind Size (µm K80)	Knelson Concentrate		Pan Concentrate	
			Grade (g/t)	Recovery (%)	Grade (g/t)	Recovery (%)
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**

Source: G&T KM3568 (2013)

Table 13.11: Cyanide Leach Flowsheet

Test Number	Composite	Grind Size (µm K80)	Extraction (%)		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**

Source: G&T KM3568 (2013)

Table 13.12: Acid Flowsheet

Test Number	Composite	Grind Size (µm K_{80})	Liquor Assay (g/t)			Extraction (%)		
			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

Source: G&T KM3568 (2013)

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% recovered to the flotation concentrate respectively. The Tizate oxide (TOX) sample results were much poorer, with only 15% and 39 % of the copper and gold in the feed recovered to the concentrate respectively. Due to the disappointing results of the samples, flotation would not be a suitable process.

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

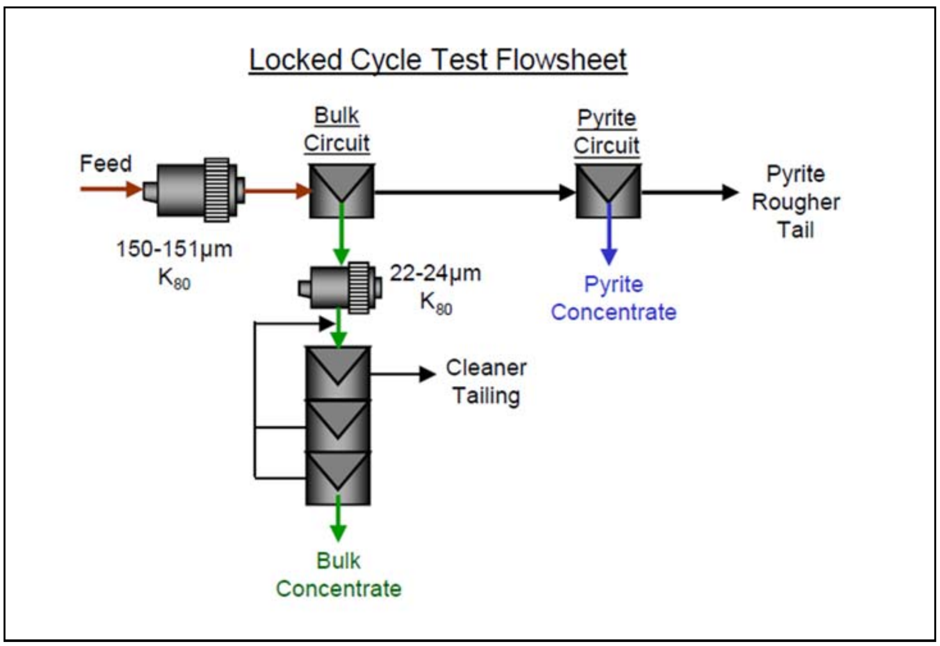
The best overall gold and silver extraction performance was achieved by direct cyanidation. The average gold leach performed was about 89% and 83 % the NSOX and TOX Composites, respectively. The best results were achieved at the finer primary grind size to a P_{80} of 95 µm. Average cyanide consumptions levels were 1.5 and 0.5 kg/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.

13.4.1 G&T KM3578 - March 6, 2013 Sulphide Report

Sulphide samples from Tizate and North/South Zone were sent to G&T laboratories for flowsheet optimization using previous test work results. The program consisted of mill feed material characterization and flowsheet optimization, including bench scale rougher and cleaner flotation tests, locked cycle tests and cyanide leaching of first cleaner tailings and pyrite concentrate. The locked cycle test flowsheet is presented in Figure 13.2 and test conditions and results in Figure 13.3.

Figure 13.2: Locked Cycle Test Flowsheet



Source: G&T KM3578 (2013)

Figure 13.3: Locked Cycle Test Conditions and Results

Test Conditions

Stage	pH	Reagent Dosage - g/tonne				
		Lime	3418A	PAX	Fuel Oil	NaCN
Primary Grind	11.3-11.5	2100-2300	-	-	10	-
Rougher	11.0-11.4	√	20	-	-	-
Regrind	11.1-11.5	200-350	-	-	10	10
Cleaner	11.0	√	32	-	-	-
Pyrite Circuit	10.8-11.4	-	-	80	-	-

Locked Cycle Test Results

Product	Weight %	Assay - percent or g/tonne						Distribution - percent					
		Cu	Mo	Fe	S	Ag	Au	Cu	Mo	Fe	S	Ag	Au
Test 17: N/S Composite (RG 22μm K80)													
Flotation Feed	100	0.22	0.002	4.2	2.33	1	0.35	100	100	100	100	100	100
Bulk Con	0.6	29.3	0.044	27.6	31.9	91	22.0	81.6	14.3	4.0	8.4	40.8	38.2
Bulk 1st Clnr Tail	8.6	0.16	0.012	7.8	5.61	3	1.01	6.1	52.6	15.8	20.6	15.9	24.8
Pyrite Ro Con	6.6	0.10	0.002	16.3	19.3	3	1.14	3.0	6.5	25.7	54.6	12.4	21.4
Pyrite Ro Tail	84.2	0.02	0.001	2.7	0.46	1	0.06	9.4	26.6	54.4	16.5	30.9	15.6
Test 18: Tizate Composite (RG 24μm K80)													
Flotation Feed	100	0.18	0.006	4.5	2.16	2	0.21	100	100	100	100	100	100
Bulk Con	0.6	24.1	0.499	24.3	30.2	180	11.4	77.2	50.0	3.1	8.0	46.5	31.0
Bulk 1st Clnr Tail	7.7	0.15	0.021	5.8	3.08	5	0.41	6.7	27.9	10.1	11.0	15.8	15.2
Pyrite Ro Con	8.3	0.12	0.006	15.0	18.1	5	0.80	5.6	9.3	27.8	69.7	18.8	31.8
Pyrite Ro Tail	83.4	0.02	0.001	3.2	0.29	1	0.05	10.5	12.8	59.0	11.4	18.9	22.0

Source: G&T KM3578 (2013)

The flowsheet selected is a conventional sulphide copper flotation to a saleable chalcopyrite concentrate followed by a pyrite flotation. The pyrite concentrate and first cleaner tailings will be combined to feed a CIL circuit to recover gold and silver to a doré bar.

13.4.1.1 Concentrate Quality

Minor element analysis was completed on the concentrates produced from KM3578 locked cycle tests 17 and 18. The analysis indicates “the bulk concentrates generally contained acceptable levels of common penalty elements.” The analysis can be found in Figure 13.4.

Figure 13.4: Quality of Flotation Concentrates

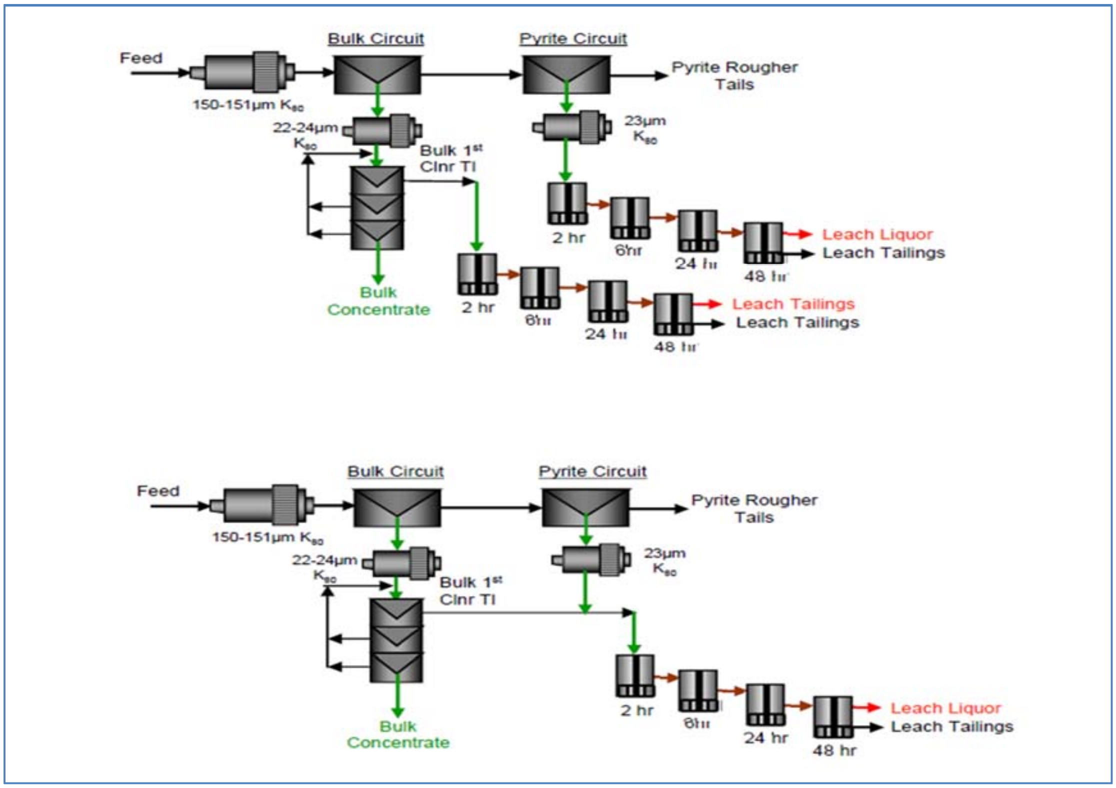
Element	Symbol	Unit	Test Method	N/S Zone, Test 17 Bulk Con III-V	Tizate, Test 18 Bulk Con III-V
Aluminum	Al	%	Fusion ICP-OES	0.90	1.66
Antimony	Sb	%	2 Acid ICP-OES	0.004	0.006
Arsenic	As	g/t	2 Acid ICP-OES	0.007	0.064
Bismuth	Bi	g/t	2 Acid ICP-OES	<2	<2
Cadmium	Cd	g/t	AR FAAS	10	178
Calcium	Ca	%	Fusion ICP-OES	0.44	1.06
Carbon	C	%	Leco	0.31	0.48
Cobalt	Co	g/t	AR FAAS	38	44
Copper	Cu	%	Titr	29.5	24.5
Fluorine	F	g/t	Fusion ISE	130	190
Gold	Au	g/t	FA FAAS	23.7	11.9
Iron	Fe	%	AR FAAS	27	24
Lead	Pb	%	AR FAAS	0.28	0.79
Magnesium	Mg	%	Fusion ICP-OES	0.24	0.39
Manganese	Mn	%	Fusion ICP-OES	0.02	0.02
Mercury	Hg	g/t	LeForte rt CV-AAS	<1	<1
Molybdenum	Mo	%	2 Acid ICP-OES*	0.04	0.57
Nickel	Ni	g/t	AR FAAS	250	118
Palladium	Pd	g/t	FA ICP-OES	0.06	0.41
Phosphorus	P	g/t	3 Acid ICP-OES	131	183
Platinum	Pt	g/t	FA ICP-OES	0.65	1.05
Selenium	Se	g/t	ESHKA ICP-OES	110	97
Silicon	Si	%	Fusion ICP-OES	2.04	3.56
Sulphur	S	%	Leco	32.1	30.2
Silver	Ag	g/t	AR FAAS	89	178
Zinc	Zn	%	AR FAAS	0.14	0.11

Source: G&T KM3578 (2013)

13.4.1.2 Pyrite and First Cleaner Cyanidation

Pyrite concentrate from locked cycle tests 17 and 18 was reground to a P_{80} of 23 µm for N//S and 30 µm for Tizate. The pyrite concentrate and first cleaner tailings were then leached for 48 hours both separately and as a combined process stream. The flowsheets and leach resultsare shown in Figure 13.5 and Figure 13.6.

Figure 13.5: Pyrite Concentrate and First Cleaner Tailings Cyanidation Flowsheets (Tepal)



Source: G&T KM3578 (2013)

Figure 13.6: Tepal KM3578 Cyanidation 6 Hour Leach Results

Test Number	Distribution, %	
	Au	Ag
19 – Pyrite Rougher Concentrate	47.4	35.4
20 – First Cleaner Tailings	70.4	51.1
25 – Combined Streams	63.5	52.4
Calculated Combined Streams 19 and 20	56.4	42.0

Source: G&T KM3578 2013

Pyrite concentrate from locked cycle tests 17 and 18 was reground to a P_{80} of 23 µm for N//S and 30 µm for Tizate. The pyrite concentrate and first cleaner tailings were then leached for 48 hours both separately and as a combined process stream. The flowsheets and leach resultsare shown in Figure 13.5 and Figure 13.6.

13.5 Flotation and Leach Recovery Predictions

13.5.1 Sulphide Feed Flotation Copper Concentrate Recovery Prediction

Table 13.13 is a summary of sulphide copper concentrate recovery predictions used in the design criteria for this PEA.

Table 13.13: PEA Copper Concentrate Design Criteria Summary

Product Resource Grade	Unit	Value
Tepal Grade		
Copper	%	0.24
Gold	g/t	0.42
Silver	g/t	0.93
Tizate Grade		
Copper	%	0.17
Gold	g/t	0.20
Silver	g/t	2.17
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

Source: 2013 PFS

13.5.2 Pyrite Flotation and Leach Results

G&T test KM3578-25/26CN leach was completed on the combined reground pyrite concentrate and copper first cleaner tailings. Gold and silver circuit extractions were 63.5% and 52.4% for N/S and 57.0% and 77.9% for Tizate based on a 6-hour leach time. Cyanide and lime consumption is expected to be between 2.5 kg/t and 1.4 kg/t for Tepal and 2.8 kg/t and 1.5 kg/t for Tizate. It is expected that overall recovery to a doré bar in the CIL circuit

Table 13.14 summarizes the extraction of gold and silver from the pyrite flotation concentrate and copper first cleaner tailings leach streams.

G&T test KM3578-25/26CN leach was completed on the combined reground pyrite concentrate and copper first cleaner tailings. Gold and silver circuit extractions were 63.5% and 52.4% for N/S and 57.0% and 77.9% for Tizate based on a 6-hour leach time. Cyanide and lime consumption is expected to be between 2.5 kg/t and 1.4 kg/t for Tepal and 2.8 kg/t and 1.5 kg/t for Tizate. It is expected that overall recovery to a doré bar in the CIL circuit

Table 13.14: Pyrite Concentrate and Copper First Cleaner Tailings Leach Overall Extraction

Product	Unit	Extraction
Tepal		
Gold	%	16.5
Silver	%	15.5
Tizate		
Gold	%	16.0
Silver	%	18.5

Source: G&T KM3577-25/26CN

13.5.3 Oxide Leach Results

Error! Not a valid bookmark self-reference. below summarizes the recovery of gold and silver from the oxide leach. The oxide recovery was based on results from G&T test KM3568-03/04CN. The oxide material will be leached for 24 hours at a P_{80} of 143µm. Cyanide and lime consumption is expected to average 1.4 kg/t and 2.4 kg/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 approximately 2% for both the sulphide and oxide material. It is anticipated that all the carbon fines would either be treated off-site for precious metal recovery or sold outright.

Table 13.15: Oxide Leach Recovery

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

Source: G&T KM3568-03/04CN

13.6 Relevant Results

Based on the test work summarized above, the selected flowsheet consists of sequential copper and pyrite flotation. The rougher copper concentrate is reground to a P_{80} of 22 microns and then undergoes three stages of cleaner flotation to produce a marketable copper concentrate. The pyrite flotation concentrate is reground to a P_{80} of 23 microns and combined with the copper first cleaner tailings process stream to be leached in the CIL circuit.

A trade-off study for the 2013 PFS was completed and based on the results a (CIL) process was selected to recover the gold and silver from the oxide feed.

An acid wash, carbon strip and refinery circuit will be installed to recover the gold and silver from the respective oxide and sulphide CIL circuits.

The variability tests indicated that the same flowsheet can be applied to NZ, SZ and Tizate feed materials.

Preliminary process design criteria and estimated reagent requirements are summarized in Table 13.16 and Table 13.17.

Table 13.16: Process Design Criteria Derived from Test Work

Description	Unit	Sulphide Resource (Tepal/Tizate)	Oxide Resource (Tepal/Tizate)
Design Criteria			
Crusher Work Index	kWh/t	6.6/8.7	-
Bond Ball Mill Work Index	kWh/t	17.5/20	9.0
Abrasion Index	g	0.2*	0.025
Primary Grind Size, P_{80}	µm	150	143
Copper Regrind Grind Size, P_{80}	µm	22 to 24	
Pyrite Regrind Grind Size, P_{80}	µm	23	
Pre-leach Thickener Loading Rate	$t/h/m^2$	0.3	
Leach Time	h	8	
Carbon Loading	g metal/t carbon	3,000	
Carbon Plant	t	4	
Head Grade			
Head Grade (Average LOM)	% Cu	0.21	
Head Grade (Average LOM)	g/t Au	0.33	0.45
Head Grade (Average LOM)	g/t Ag	1.47	1.11
Metal Recovery (Tepal/Tizate)			
Copper Grade	%	26.2	
Copper Flotation Recovery	%	88.2/85.9	
Gold Recovery to Copper Concentrate	%	62.4/58	
Silver Recovery to Copper Concentrate	%	27.4/59.6	
Gold Leach Extraction	%	16.5/16.0	83.2/75.2
Silver Leach Extraction	%	15.5/18.5	63.3/55.9
Copper Recovery			
Copper Rougher Mass Pull	%	9.2	

Description	Unit	Sulphide Resource (Tepal/Tizate)	Oxide Resource (Tepal/Tizate)
Copper Rougher Laboratory Flotation Time	min	15	
Copper Third Cleaner Mass Pull	%	1.4	
Copper Cleaner Total Laboratory Flotation Time	min	17	
Pyrite Rougher Mass Pull	%	6.6	
Pyrite Rougher Laboratory Flotation Time	min	15	
Copper Concentrate Loading Rate	$t/h/m^2$	0.25*	
Filtration – Copper Concentrate			
Filter Filtration Rate	$kg/m^2/hr$	350*	
Filter Cake Moisture Content	%	8	
Gold/Silver Leach			
Pre-leach Thickener Loading Rate	$t/h/m^2$	0.3*	0.3*
Process Selected	-	CIL	CIL
Leach Time	hr	8	24
Leach Density	%	45	45
Gold/Silver Recovery			
Process Selected	-	ADR	
Circuit Capacity	t Carbon	6	
Carbon Loading	g Au / t Carbon	3,000*	3,000*
Cyanide Destruction			
Destruction Retention Time	hr	2*	2*
$CN_{(WAD)}$ Concentration	mg/l $CN_{(WAD)}$	150*	150*
SMBS Consumption	$SO_2:CN_{(WAD)}$	4*	4*
Lime Consumption	$Lime:CN_{(WAD)}$	4*	4*
Copper Sulphate Consumption	mg/l	30*	30*

* Vendor Recommended, no test work available

Source: JDS (2017), 2013 PFS

Table 13.17: Reagent Consumption Derived from Test Work

Description	Unit	Value (Sulphide)	Value (Oxide)
MIBC	g/t	61	
PAX	g/t	125	
Lime	g/t	2,615	2,300
Lime (Detox)	t/a	2,646	1,974
Cyanide	g/t	295	1,370
Flocculant	g/t	40	20
Antiscalant	g/t	20	20
SMBS	t/a	2,646	1,974
$CuSO_4H_2O$	t/a	624	103
HCL	t/a	80	121
NaOH	t/a	72	109
Carbon	t/a	25	30

Source: JDS (2017), 2013 PFS

14 Mineral Resource Estimate

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

Three Mineral Resource estimates, one following NI 43-101 guidelines, 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 data from 353 drill holes. 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 Category	Tonnage (kt)	Average Grade				Contained Metal	
			Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Au (koz)	Cu (Mlb)
Tepal North	Measured	14,000	0.50	0.29	0.78	0.002	228	89
	Indicated	55,000	0.30	0.21	1.01	0.002	533	252
	M + I	69,000	0.34	0.22	0.96	0.002	761	341
Tepal South	Measured	20,000	0.47	0.22	1.07	0.002	300	96
	Indicated	21,000	0.45	0.20	1.17	0.002	305	91
	M + I	41,000	0.46	0.21	1.12	0.002	605	187
Tizate	Measured	-	-	-	-	-	-	-
	Indicated	77,000	0.18	0.17	2.29	0.006	438	285
	M + I	77,000	0.18	0.17	2.29	0.006	438	285
Total	**Measured**	**34,000**	**0.48**	**0.25**	**0.95**	**0.002**	**528**	**185**
	Indicated	**153,000**	**0.26**	**0.19**	**1.67**	**0.004**	**1,276**	**628**
	M + I	**187,000**	**0.30**	**0.20**	**1.54**	**0.004**	**1,804**	**813**

*Assumptions used to calculate the soft pit constraint: Au Price US$ 1,300/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.**

Source: Micon (2012)

Table 14.2 summarizes the Inferred Mineral Resources of the three deposits above the same US$ 5.00/t equivalent value NSR cut-off.

Table 14.2: Inferred Mineral Resources at US$5/t Equivalent Value Cut-Off

Deposit	Resource Category	Tonnage (kt)	Average Grade				Contained Metal	
			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	34,000	0.15	0.15	1.70	0.007	169.8	114.8
Total	**Inferred**	**36,000**	**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%
Source: Micon (2012)

The following sub-sections outline the parameters and assumptions made to complete this estimate.

14.1.1 Mineralogical Model

Geologix generated a 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/t 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/t limit was included for continuity of the model. Blocks inside the mineralogical models were classified as “Resource” 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 completed 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	49	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	**420**		**353**	**68,759**

*DD = diamond drilling, RC= reverse circulation drilling

Source: Micon (2012)

The locations of the INCO holes could not be confirmed, therefore these were omitted 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 2 m which was chosen as the composite length. Therefore, the samples were composited to 2 m, honouring domain contacts. The minimum composite length was 1 m with remnants and less than 1 m 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
Coefficient. 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

Source: Micon (2012)

Table 14.5: Tepal North Zone Oxide 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	0.39	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
Coefficient. 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

Source: Micon (2012)

Table 14.6: Tepal South 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	0.48	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
Coefficient 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

Source: Micon (2012)

Table 14.7: Tepal South Zone Oxide 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	0.42	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
Coefficient 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

Source: Micon (2012)

Table 14.8: Tizate 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	0.18	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
Coefficient. 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

Source: Micon (2012)

Table 14.9: Tizate Zone Oxide 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	0.19	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
Coefficient. 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

Source: Micon (2012)

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 2 m composites so that the capping was less severe.

Table 14.10: Tepal Property Capping Summary for Gold

Zone	Domain	Threshold Au (g/t)	Data Capped		
			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

Source: Micon (2012)

Table 14.11: Tepal Property Capping Summary for Copper

Zone	Domain	Threshold Cu (%)	Data Capped		
			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

Source: Micon (2012)

Table 14.12: Tepal Property Capping Summary for Silver

Zone	Domain	Threshold Ag (g/t)	Data Capped		
			Number	Proportion (%)	Metal Loss (%)
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

Source: Micon (2012)

Table 14.13: Tepal Property Capping Summary for Molybdenum

Zone	Domain	Threshold Mo (ppm)	Data Capped		
			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.
Source: Micon (2012)

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 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 creates sufficient sample density to be able to generate variograms for gold and copper, especially in the sulphide zones. Average ranges from gold and copper are used so every block is estimated with the same search distance. There is insufficient data 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 C_0	Sill $C_1/C_2/C_3$	Rotation			Ranges		
				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

Source: Micon (2012)

Table 14.15: Variogram Parameters, South Tepal

Zone	Metal	Nugget C_0	Sill $C_1/C_2/C_3$	Rotation			Ranges		
				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

Source: Micon (2012)

Table 14.16: Variogram Parameters, Tizate

Zone	Metal	Nugget C_0	Sill $C_1/C_2/C_3$	Rotation			Ranges		
				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
Tizate 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	0.10	0.30	-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

Source: Micon (2012)

14.1.7 Specific Gravity

Specific gravity (SG) samples were collected approximately every 50 m 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. feed 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 in 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. 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	Mineralized Material	2.42	13
	Sulphide	Mineralized Material	2.70	86
	Oxide	Waste	2.45	14
	Sulphide	Waste	2.73	229
South	Oxide	Mineralized Material	2.46	4
	Sulphide	Mineralized Material	2.72	81
	Oxide	Waste	2.45	16
	Sulphide	Waste	2.73	109
Tizate	Oxide	Mineralized Material	2.49	4
	Sulphide	Mineralized Material	2.74	169
	Oxide	Waste	2.39	10
	Sulphide	Waste	2.73	318
Total				**1,053**

Source: Micon (2012)

The number of oxide feed sample determinations is low compared to sulphide determinations. Micon's recommendation was that additional oxide feed 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

Source: Micon (2012)

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

Source: Micon (2012)

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 Mineralized Material
102	Tepal North Sulphide Mineralized Material
129	Tepal North Oxide Waste
130	Tepal North Sulphide Waste
201	Tepal South Oxide Mineralized Material
202	Tepal South Sulphide Mineralized Material
229	Tepal South Oxide Waste
230	Tepal South Sulphide Waste
301	Tizate Oxide Mineralized Material
302	Tizate Sulphide Mineralized Material
329	Tizate Oxide Waste
330	Tizate Sulphide Waste

Source: Micon (2012)

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: Block Model Search Parameters

Domain	Metal	Search Pass	Rotation			Range			Composites		Max. Samples per Hole
			Z (°)	Y (°)	X (°)	X (m)	Y (m)	Z (m)	Min	Max	
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

Source: Micon (2012)

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 (ID^2) 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 Metal Loss (%)	Copper Metal Loss (%)	Gold Metal Loss (%)	Copper 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	-0.8	1.4	1.3

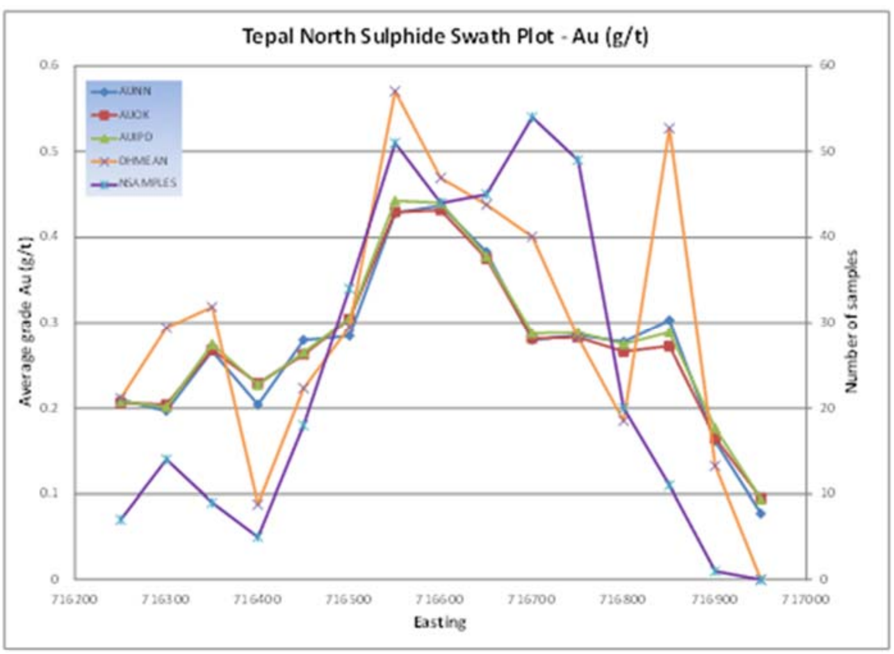
Note: Based on US$ 5 /t equivalent
Source: Micon (2012)

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 (ID^2 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 ID^2 method. The data in Table 14.22 corroborates these relationships.

Swath plots were generated on each deposit for gold and copper. The plots include declustered composite sulphide grades compared to OK, ID^2 and NN sulphide block grades in west-east, south-north and vertical directions through each deposit.

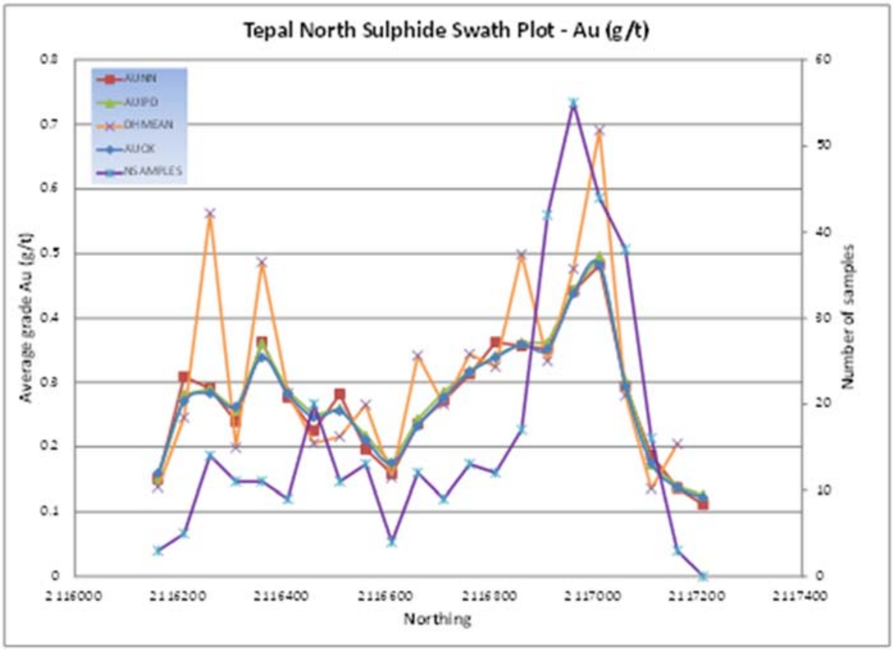
Figure 14.1: Tepal North Sulphide Gold W-E Swath Plot



Source: Micon (2012)

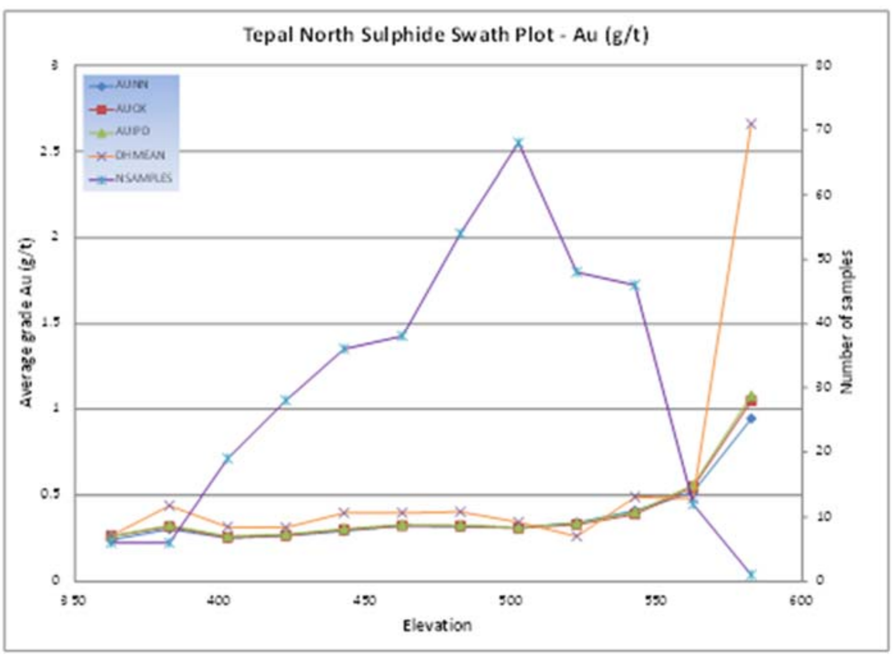
Figures 14.2 and 14.5 illustrate a potential starter pit located at approximately 2117000 m N.

Figure 14.2: Tepal North Sulphide Gold S-N Swath Plot



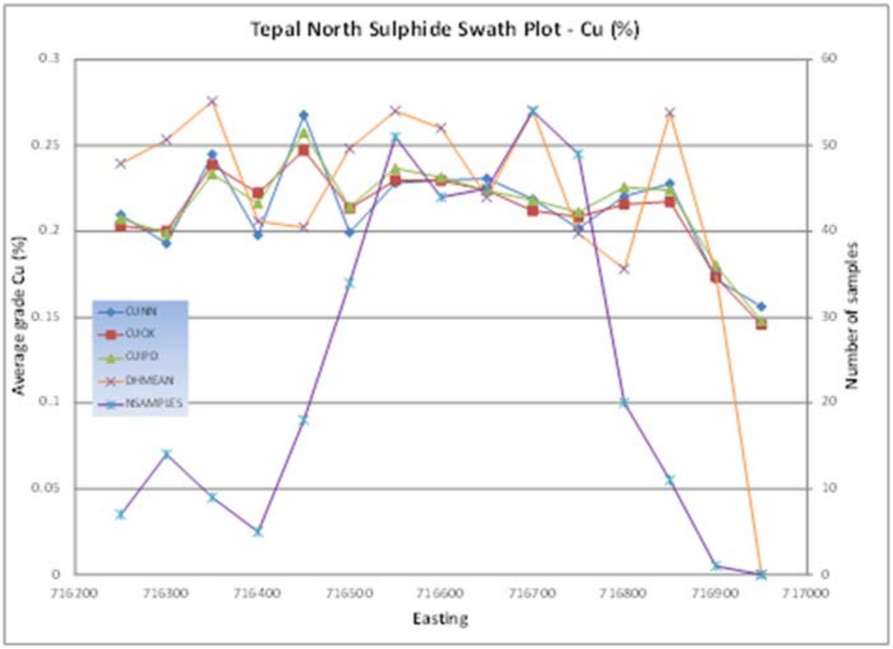
Source: Micon (2012)

Figure 14.3: Tepal North Sulphide Gold Elevation Swath Plot



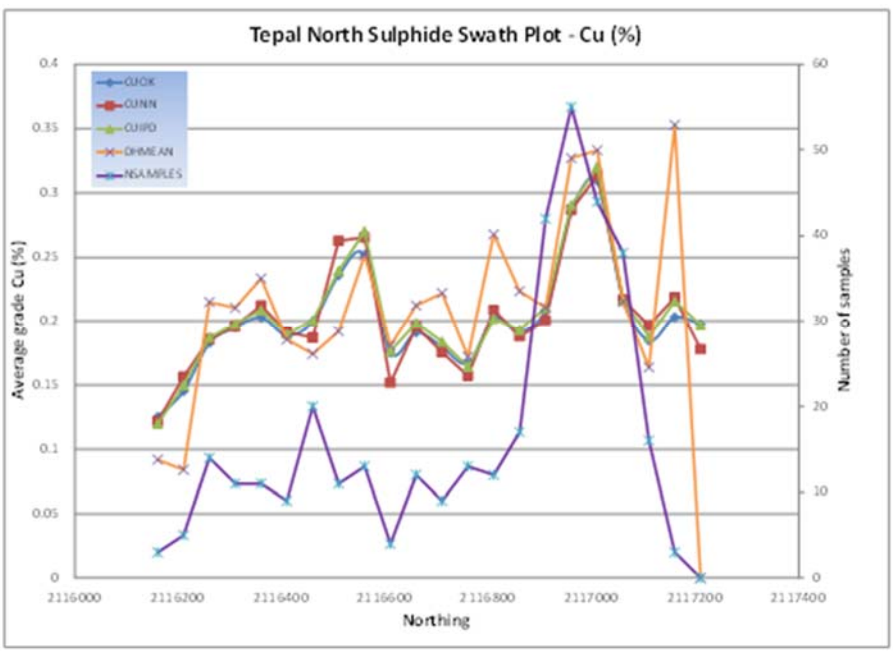
Source: Micon (2012)

Figure 14.4: Tepal North Sulphide Copper W-E Swath Plot



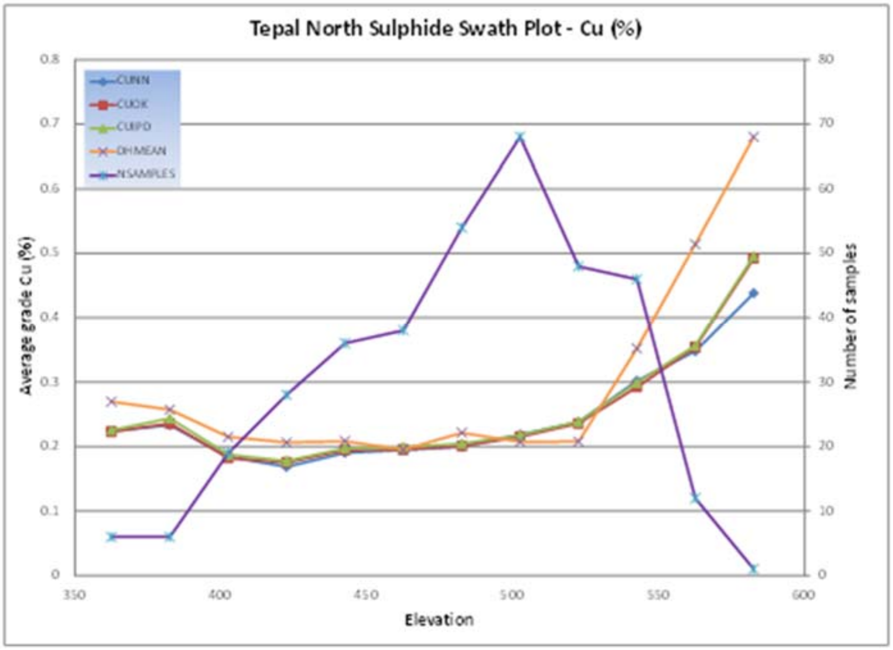
Source: Micon (2012)

Figure 14.5: Tepal North Sulphide Copper S-N Swath Plot



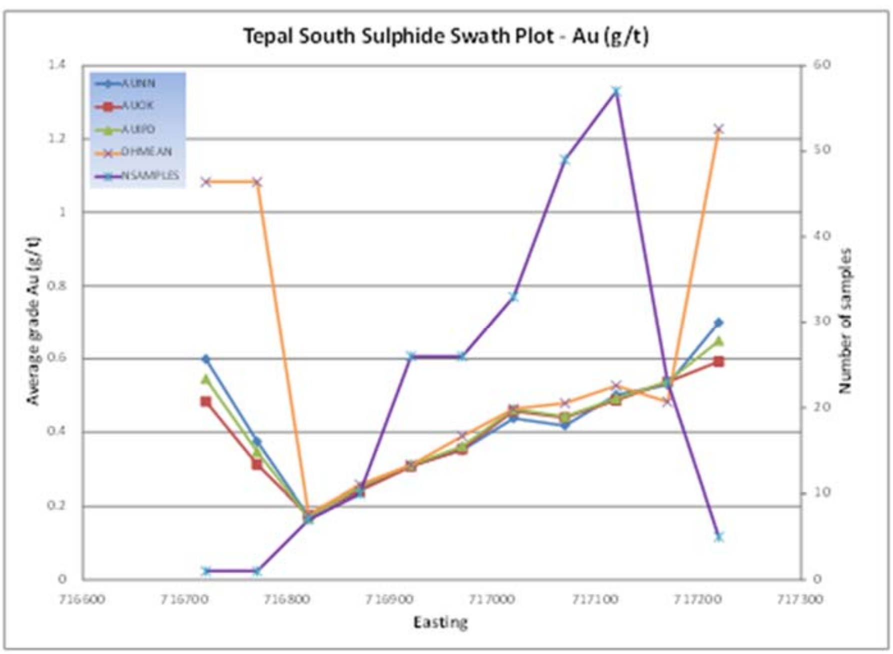
Source: Micon (2012)

Figure 14.6: Tepal North Sulphide Copper Elevation Swath Plot



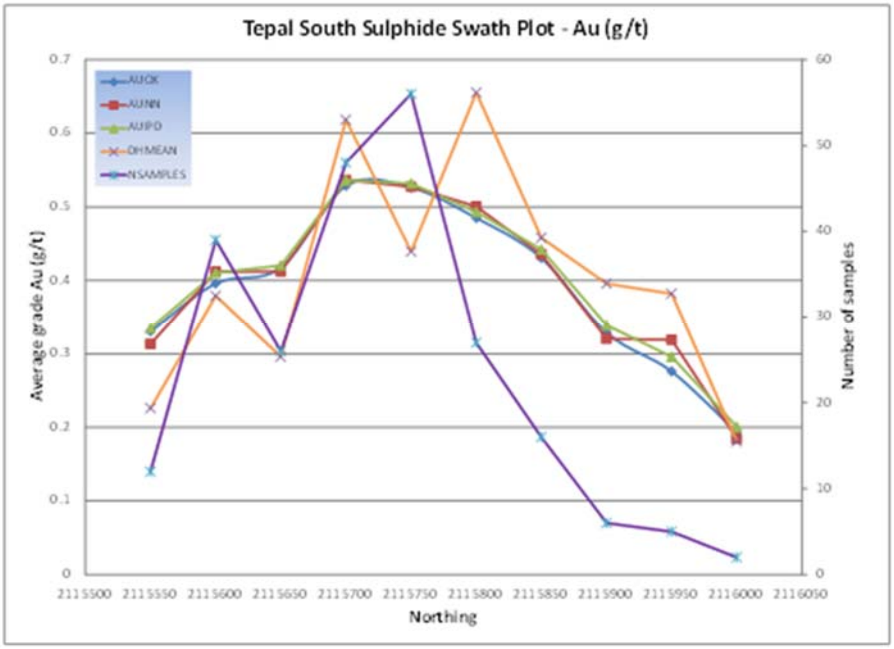
Source: Micon (2012)

Figure 14.7: Tepal South Sulphide Gold W-E Swath Plot



Source: Micon (2012)

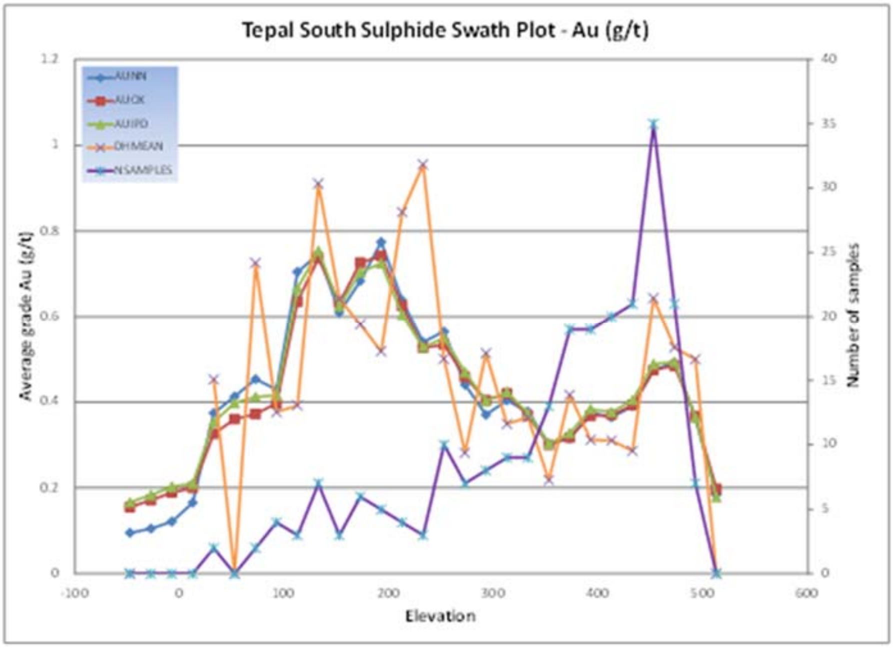
Figure 14.8: Tepal South Sulphide Gold S-N Swath Plot



Source: Micon (2012)

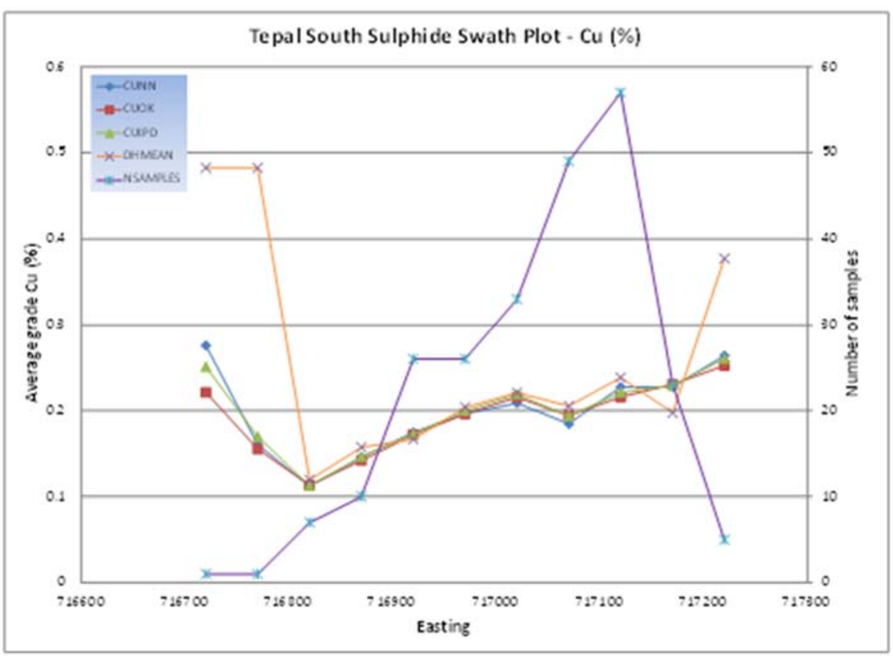
Figures 14.9 and 14.12 illustrate the high-grade mineralization below the South Zone optimized soft pit.

Figure 14.9: Tepal South Sulphide Gold Elevation Swath Plot



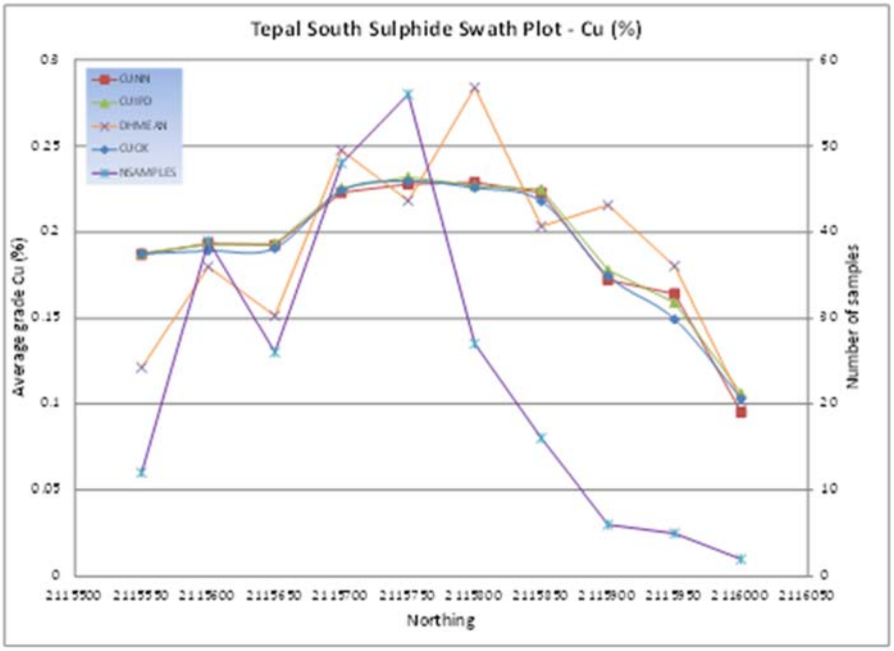
Source: Micon (2012)

Figure 14.10: Tepal South Sulphide Copper W-E Swath Plot



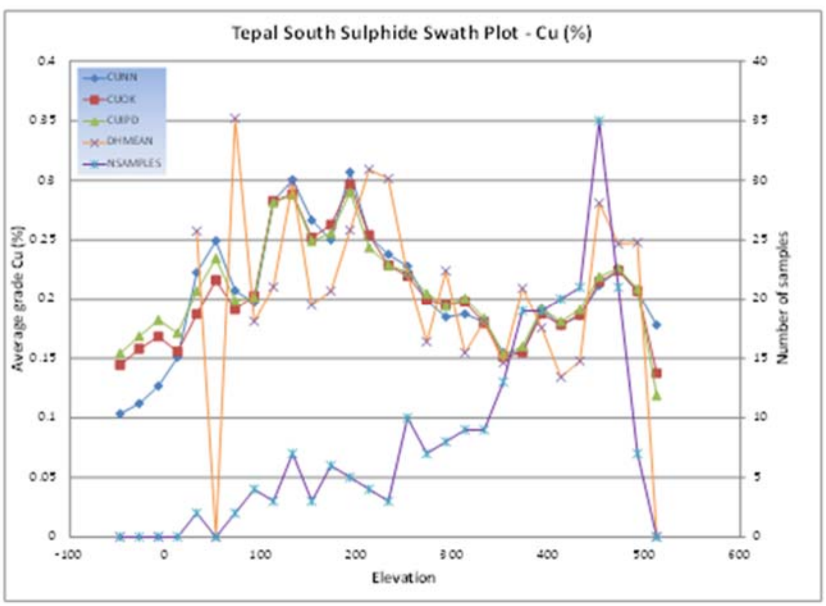
Source: Micon (2012)

Figure 14.11: Tepal South Sulphide Copper S-N Swath Plot



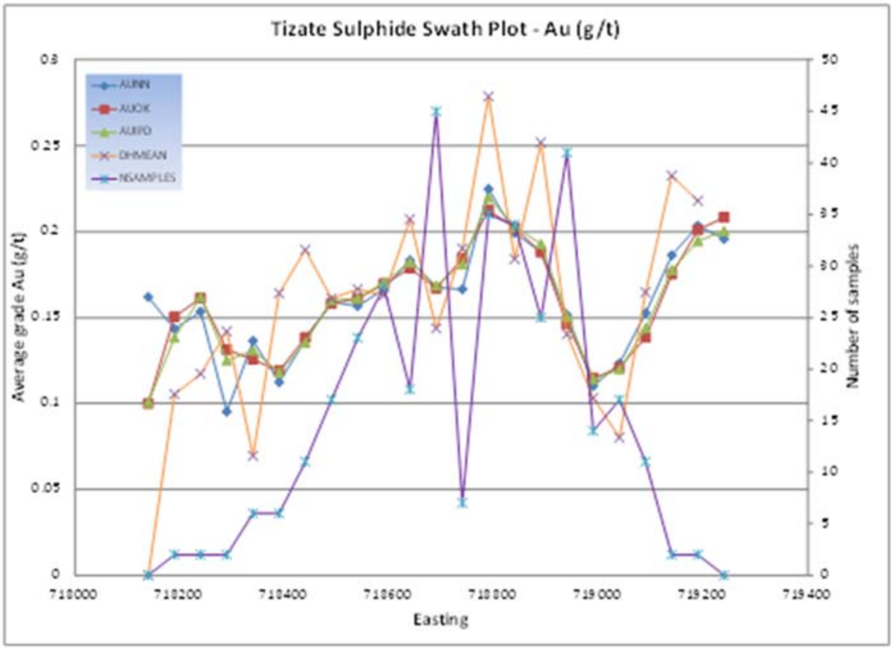
Source: Micon (2012)

Figure 14.12: Tepal South Sulphide Copper Elevation Swath Plot



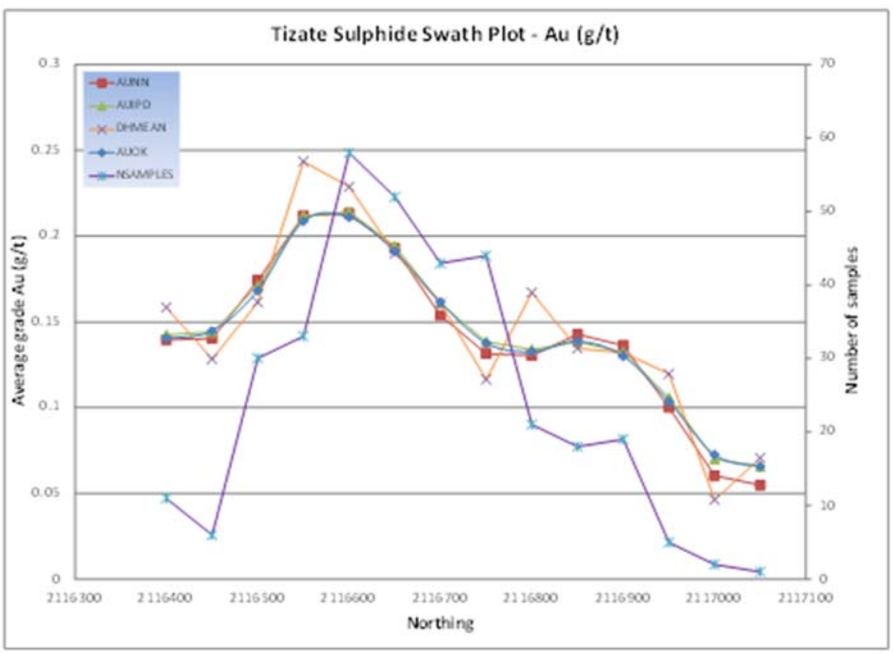
Source: Micon (2012)

Figure 14.13: Tizate Sulphide Gold W-E Swath Plot



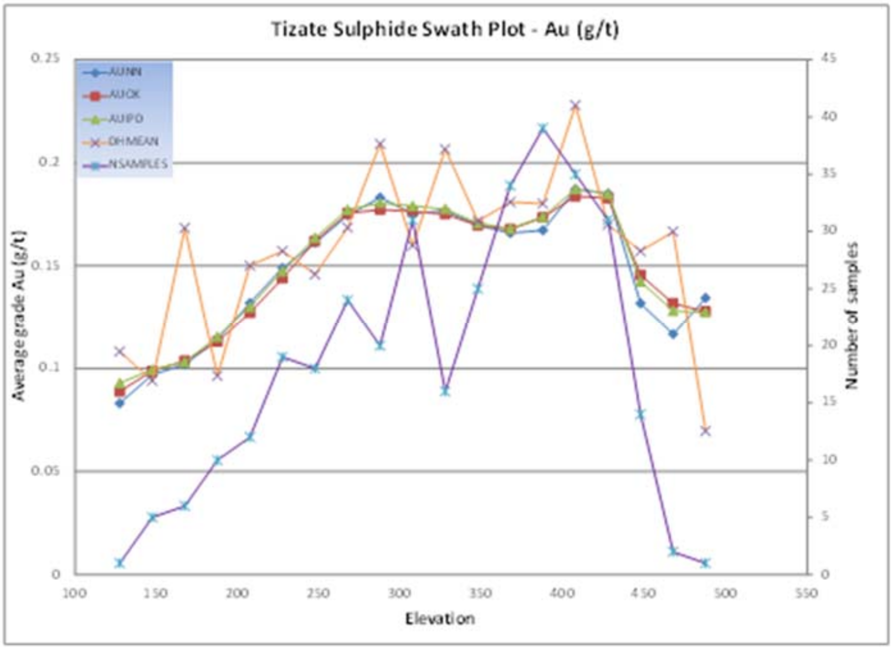
Source: Micon (2012)

Figure 14.14: Tizate Sulphide Gold S-N Swath Plot



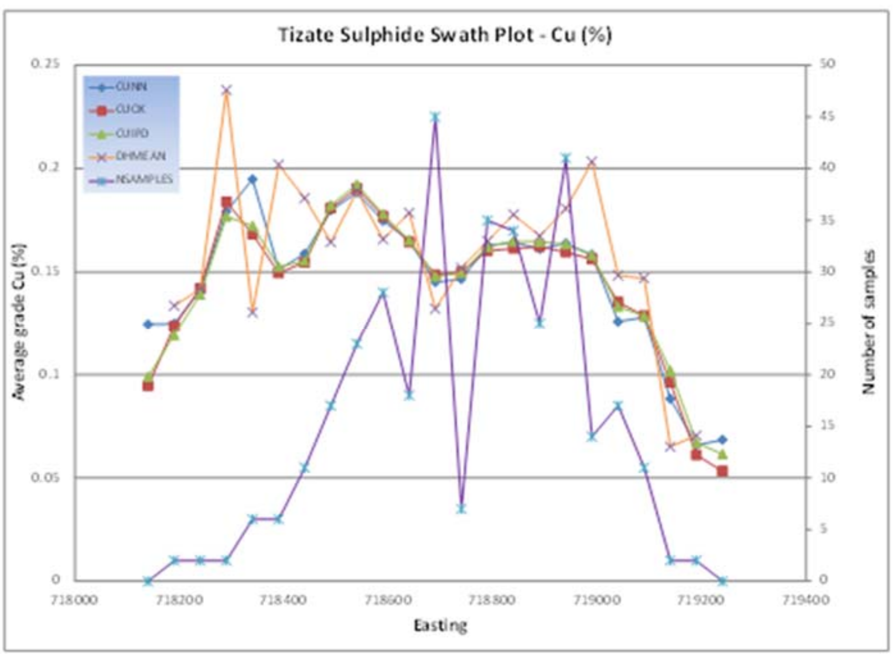
Source: Micon (2012)

Figure 14.15: Tizate Sulphide Gold Elevation Swath Plot



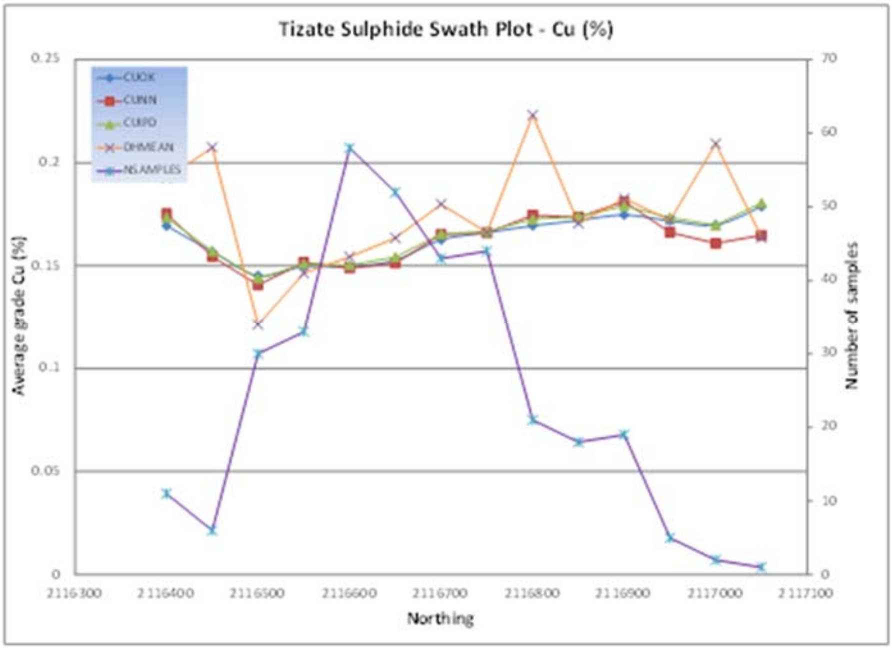
Source: Micon (2012)

Figure 14.16: Tizate Sulphide Copper W-E Swath Plot



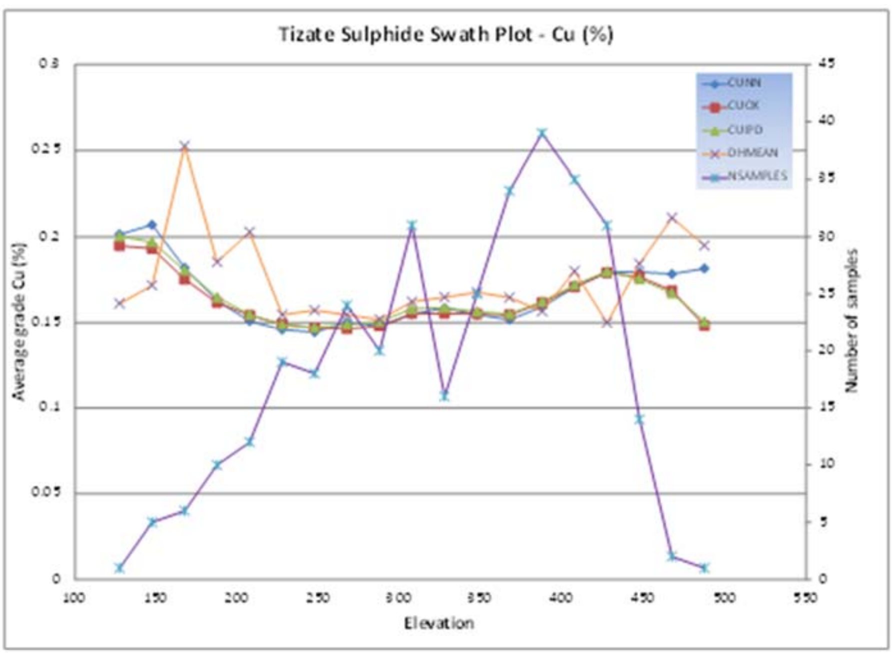
Source: Micon (2012)

Figure 14.17: Tizate Sulphide Copper S-N Swath Plot



Source: Micon (2012)

Figure 14.18: Tizate Sulphide Copper Elevation Swath Plot



Source: Micon (2012)

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 for instance 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
Coefficient of Variation	1.05	0.71	0.85	0.54
Number of Samples	4,135	44,445	4,135	44,445

Source: Micon (2012)

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
Coefficient of Variation	1.00	0.68	1.12	0.68
Number of Samples	1,097	12,681	1,097	12,681

Source: Micon (2012)

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
Coefficient of Variation	0.81	0.60	0.62	0.45
Number of Samples	2,855	35,541	2,855	35,541

Source: Micon (2012)

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
Coefficient of Variation	0.66	0.45	0.58	0.32
Number of Samples	253	3,227	253	3,227

Source: Micon (2012)

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
Coefficient of Variation	0.74	0.48	0.48	0.29
Number of Samples	3,932	82,837	3,932	82,837

Source: Micon (2012)

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
Coefficient of Variation	0.68	0.44	0.56	0.28
Number of Samples	255	7,396	255	7,396

Source: Micon (2012)

The statistics indicate that the degree of smoothing has been reduced due to the infill 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 believed that the block model results portrayed 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 (NI) 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 and revised May 10, 2014).

Under these definitions:

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.
The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

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 Modifying 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. 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 below.

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 are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply, but not verify geological and grade continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

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 are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

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 estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

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	1,300	1,300	Kitco 3 yr trailing avg. 02/2012
Copper Price	US$/lb	3.30	3.30	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

Source: Micon (2012)

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 is of the opinion that US $5/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 35,000 t/d of mineralized material which is anticipated by Geologix.

The Mineral Resource classification was based on variography and the resulting search passes generated from this variography work. For Tepal North and South, 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 were no Measured blocks in Tizate.

Both Measured and Indicated categories were forced to look for two drill holes (maximum four composites per hole) and five composites total (Table 14.21). The Inferred category required one drill hole (maximum four composites per hole) and four composites total (Table 14.21).

Tepal Project Mineral Resources are summarized in Tables 14.30 to 14.35.

Table 14.30: Tepal North Zone Oxide Mineral Resources

Resource Class	Cut-off Eq. V. ($/t)	Tonnes (kt)	Average Grade				Metal	
			Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Au (koz)	Cu (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	0.80	0.002	0.2	0.1
Inferred	9	15	0.34	0.26	0.73	0.002	0.2	0.1

Source: Micon (2012)

Table 14.31: Tepal North Zone Sulphide Mineral Resource s

Resource Class	Cut-off Eq. V. ($/t)	Tonnes (kt)	Average Grade				Metal	
			Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Au (koz)	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

Source: Micon (2012)

Table 14.32: Tepal South Zone Oxide Mineral Resource

Resource Class	Cut-off Eq. V. ($/t)	Tonnes (kt)	Average Grade				Metal	
			Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	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	0.20	1.07	0.001	46	14
M + I	9	2,871	0.48	0.20	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

Source: Micon (2012)

Table 14.33: Tepal South Zone Sulphide Mineral Resource

Resource Class	Cut-off Eq. V. ($/t)	Tonnes (kt)	Average Grade				Metal	
			Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Au (koz)	Cu (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	1.00	0.002	5	1

Source: Micon (2012)

Table 14.34: Tizate Zone Oxide Mineral Resource

Resource Class	Cut-off Eq. V. ($/t)	Tonnes (kt)	Average Grade				Metal	
			Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Au (koz)	Cu (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

Source: Micon (2012)

Table 14.35 Tizate Zone Sulphide Mineral Resources

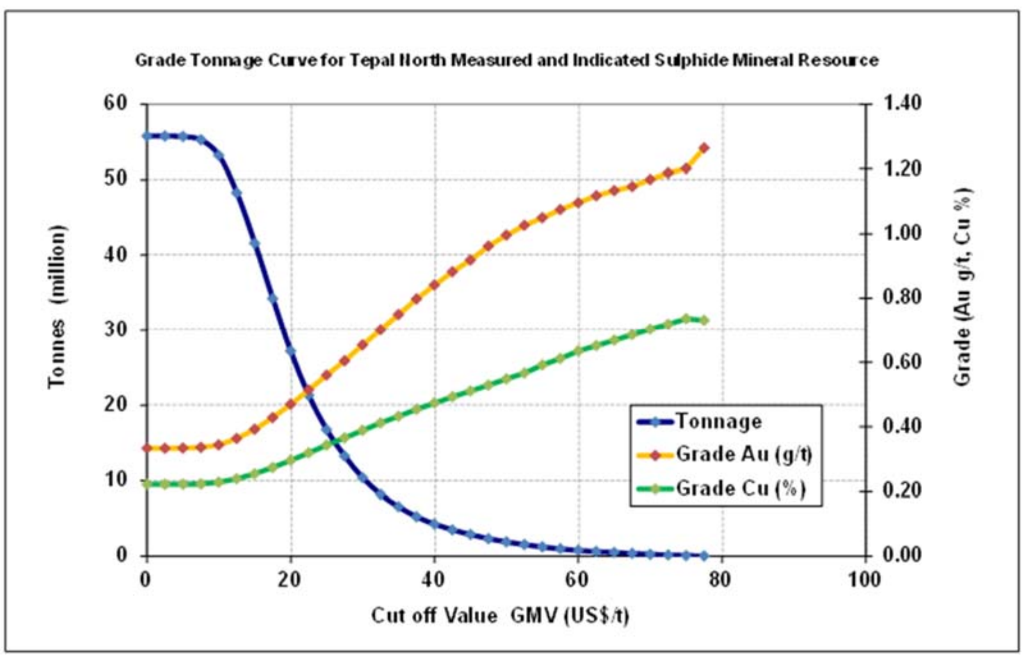
Resource Class	Cut-off Eq. V. ($/t)	Tonnes (kt)	Average Grade				Metal	
			Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Au (koz)	Cu (Mlb)
Indicated	1	73,335	0.17	0.17	2.28	0.007	407	267
Indicated	3	73,334	0.17	0.17	2.28	0.007	407	267
Indicated	5	73,194	0.17	0.17	2.29	0.007	406	267
Indicated	7	72,516	0.17	0.17	2.3	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

Source: Micon (2012)

14.1.12 Cut-off Grade Sensitivity

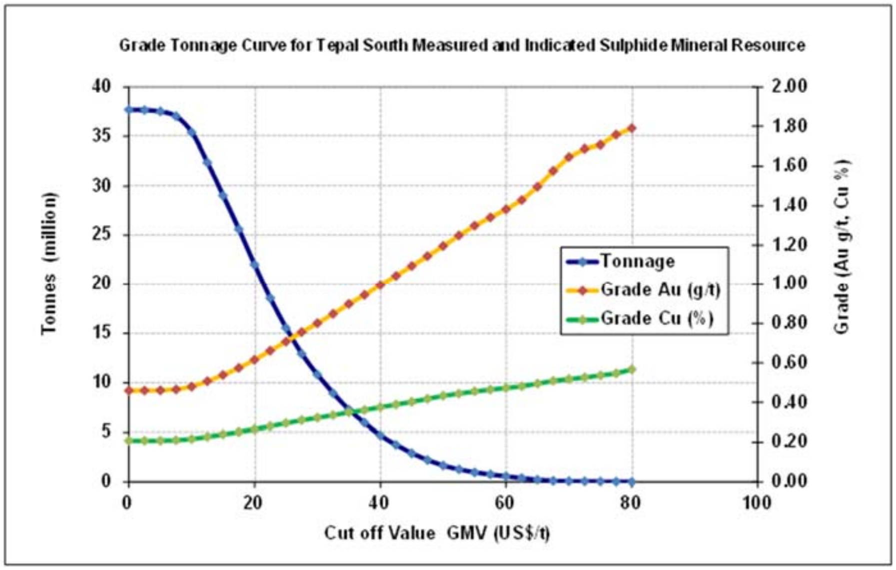
The following graphs illustrate the Tepal North, Tepal South and Tizate Zones sensitivities of tonnage and grade to cut-off values.

Figure 14.19: Grade/Tonnage Curve for Tepal North Measured & Indicated Sulphide Mineral Resource



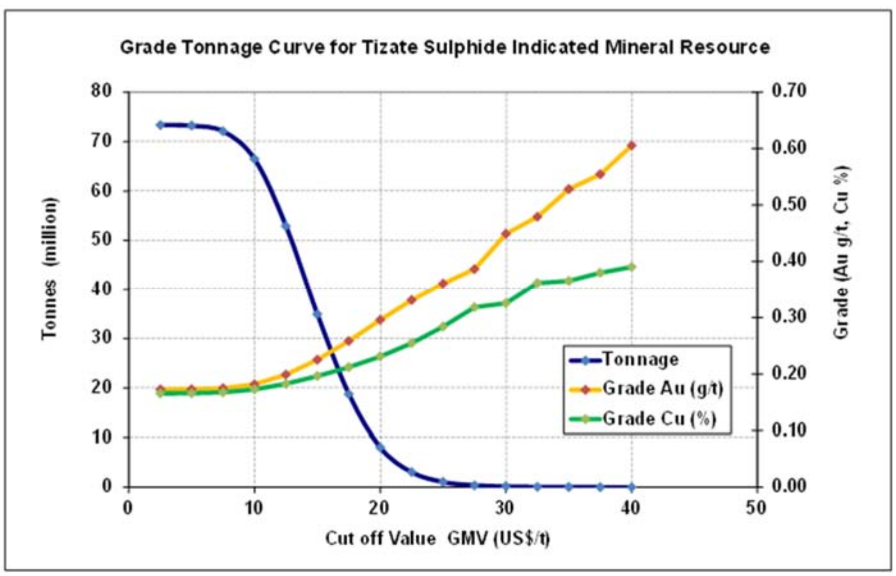
Source: Micon (2012)

Figure 14.20: Grade/Tonnage Curve for Tepal South Measured & Indicated Sulphide Mineral Resource



Source: Micon (2012)

Figure 14.21: Grade/Tonnage Curve for Tizate Indicated Sulphide Mineral Resource



Source: Micon (2012)

In general, 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 located 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. 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. A study is needed to determine the economic viability of this high-grade zone and assess the possibility of extraction.

Table 14.36 lists the tonnage and grade at a variety of cut-off equivalents (US$ 1,000/oz Au and US$ 2.75/lb 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 eventual 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 Eq. V. ($/t)	Tonnes (kt)	Average Grade Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)	Metal Au (koz)	Cu (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

Source: Micon (2012)

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 300 m to the southwest and 150 m to the northeast.

Infill drilling in all three deposits has increased the confidence in the continuity of mineralization and hence the upgrading 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 additional drilling is required.

The mineral resource estimate remains valid today since Geologix has not undertaken any additional drilling to further define the three mineralized zones or undertaken mine production since the time of of the 2012 mineral resource estimate.

15 Mineral Reserve Estimates

This PEA report does not state a Mineral Reserve.

16 Mining Methods

16.1 Open Pit Mine Design

Open Pit mine design was conducted using a combination of software packages, including Hexagon MineSight™, and GEOVIA's GEMS™ and Whittle™. For the Tepal deposits, the ultimate shell limits, along with the associated phasing, were based on the pit optimization analysis described in this report (Section 16.2.2).

The resultant pit shape for Tepal North will contain 38.9 Mt of mineralized material (including both sulphide and oxide feed) with an average gold grade of 0.42 g/t and average copper grade of 0.25% (for all deposits the copper recovered is from sulphide feed only). The total waste tonnage in the pit will be 25.1 Mt for a strip ratio of 0.6:1. The Tepal South shell will contain 16.2 Mt of mineralized material with an average gold grade of 0.47 g/t and 0.21% copper and a total waste tonnage of 10.9 Mt for a strip ratio of 0.7:1. Tizate contains a total of 35.4 Mt of sulphide and oxide mineralized material with a gold grade of 0.21 g/t and 0.17% copper. The strip ratio is 0.5 with 16.5 Mt of waste.

Table 16.7 summarizes the combined tonnages and grades contained within the planned ultimate pit shells for the Tepal North and South and Tizate deposits (using the incremental cut-off value of $8.19/t for sulphide and $10.13/t for oxide). Figure 16.5, Figure 16.6, Figure 16.7 and Figure 16.8 represent the plan and section views of the planned ultimate pit shapes.

Table 16.1: Pit Shell Summary

Parameter	Units	Deposit			Total
		Tepal North	Tepal South	Tizate	
Sulphide Mill Feed*	Mt	31.1	13.4	33.8	**78.3**
Gold grade	g/t	0.40	0.47	0.20	**0.33**
Copper grade	%	0.25	0.22	0.17	**0.21**
Silver grade	g/t	0.86	1.10	2.17	**1.47**
Oxide CIL Feed*	Mt	7.8	2.8	1.5	**12.2**
Gold grade	g/t	0.47	0.47	0.29	**0.45**
Silver grade	g/t	0.94	1.05	2.07	**1.11**
Total Feed	Mt	38.9	16.2	35.4	**90.5**
Gold grade	g/t	0.42	0.47	0.21	**0.34**
Copper grade**	%	0.25	0.22	0.17	**0.21**
Silver grade	g/t	0.88	1.09	2.16	**1.42**
Contained gold	koz	521	244	236	**1,000**
Contained copper**	Mlbs	174	64	128	**366**
Contained silver	koz	1101	565	2459	**4,125**
Waste	Mt	25.1	10.9	16.5	**52.5**
Strip Ratio	w:o	0.6	0.7	0.5	**0.6**
Total Material	**Mt**	**64.0**	**27.1**	**51.8**	**142.9**

Note: *diluted tonnes and grade @ 5% external dilution and 100% mining recovery

** Copper recovered only from sulphide mill feed

Source: JDS (2017)

16.2 Optimization

16.2.1 Input Parameters

The 3D Mineral Resource block models for Tepal (containing both the North and South Zones) and Tizate, as developed by Micon International Ltd in 2012, were used as the basis for deriving the economic shell limits for the Tepal Project.

Estimates were made for metal prices, mining dilution, process recovery, off-site costs and royalties. Mining, processing, and general administration operating cost estimates (OPEX) were also calculated based on calculated processing throughput and, along with geotechnical parameters, formed the basis for OP optimization. The OP mining costs were estimated for both plant feed material and waste mining, where variations in haulage profiles and equipment selection were taken into account in the cost estimate.

Sulphide block value is based on the net smelter return (NSR) of copper concentrate produced by sulphide flotation and revenue from doré bar produced by sulphide cyanidation. The value of oxide blocks was based on revenue from doré bar produced by the oxide carbon-in-leach (CIL) process. Table 16.2 summarizes the NSR inputs and optimization parameters.

Table 16.2: Input Parameters Used in the LOM OP Optimization

Parameter	Unit	Sulphide Flotation	Sulphide Cyanidation	Oxide CIL
Metal Prices				
Copper	US$/lb	2.25		
Gold	US$/oz	1,250		
Silver	US$/oz	20.00		
Tepal - Recovery				
Cu Recovery	%	88.2	-	-
Au Recovery	%	62.4	17.2	88.5
Ag Recovery	%	27.4	13.6	73.4
Tizate - Recovery				
Cu Recovery	%	85.9	-	-
Au Recovery	%	58.0	22.0	82.5
Ag Recovery	%	59.6	12.2	70.7
Copper Concentrate				
Cu - Tepal	%	25.7	-	-
Cu - Tizate	%	26.9	-	-
Au[1]	g/t	variable	-	-
Ag[1]	g/t	variable	-	-
Moisture Content	%	8%	-	-
Smelter Payables				
Cu Payable	%	96.5	-	-
Cu Deduction	%	1.0	-	-
Au Payable	%	97.0	99.9	
Ag Payable	%	90.00	97.0	
Ag Deduction	g/t	30.0	-	-
Treatment & Refining Costs				
Cu Concentrate Transport Charge	US$/dmt	90.00	-	-
Cu Refining Charge	US$/payable lb	0.09	-	-
Au Refining Charge	US$/payable oz	5.00	7.50	
Ag Refining Charge	US$/payable oz	0.50	1.40	
Transport Costs				
Ocean Freight	US$/wmt	60.00	-	-

<table>
<tr><th>Parameter</th><th>Unit</th><th>Sulphide Flotation</th><th>Sulphide Cyanidation</th><th>Oxide CIL</th></tr>
<tr><td>Truck freight to Port</td><td>US$/wmt</td><td>36.73</td><td>-</td><td>-</td></tr>
<tr><td>Representation at Port</td><td>US$/wmt</td><td>1.00</td><td>-</td><td>-</td></tr>
<tr><td>Port Charges</td><td>US$/wmt</td><td>10.50</td><td>-</td><td>-</td></tr>
<tr><td>Insurance</td><td>US$/wmt</td><td>1.93</td><td>-</td><td>-</td></tr>
<tr><td>Losses</td><td>US$/wmt</td><td>7.50</td><td>-</td><td>-</td></tr>
<tr><td colspan="5">Operating Costs</td></tr>
<tr><td>Mining Cost - waste</td><td>US$/tonne mined</td><td colspan="3" rowspan="2">1.80</td></tr>
<tr><td>Mining Cost – mineralized material</td><td>US$/tonne mined</td></tr>
<tr><td>Processing Cost</td><td>US$/tonne milled</td><td>5.79</td><td>1.06</td><td>8.70</td></tr>
<tr><td>G&A</td><td>US$/tonne milled</td><td>0.90</td><td>-</td><td>0.90</td></tr>
<tr><td>Tailings Cost</td><td>US$/tonne milled</td><td>0.05</td><td>-</td><td>0.05</td></tr>
<tr><td>Royalties – Cu</td><td>%</td><td colspan="3" rowspan="3">2.5</td></tr>
<tr><td>Royalties - Au</td><td>%</td></tr>
<tr><td>Royalties - Ag</td><td>%</td></tr>
<tr><td colspan="5">Other Parameters</td></tr>
<tr><td>Processing Rate</td><td>t/d milled</td><td>22,000</td><td rowspan="2">12% of Sulp Flot. Rate</td><td>8,000</td></tr>
<tr><td>Processing Rate</td><td>t/a milled</td><td>8.0</td><td>2.92</td></tr>
<tr><td>External Mining Dilution</td><td>%</td><td colspan="3">5.0</td></tr>
<tr><td>Mining Recovery</td><td>%</td><td colspan="3">100</td></tr>
<tr><td>Discount Rate</td><td>%</td><td colspan="3">5</td></tr>
<tr><td>Slope Angles (overall)</td><td>Deg.</td><td colspan="3">varies</td></tr>
</table>

*The values in this table vary slightly from those used in the economic model as parameters were further refined in the economic model. The differences are not considered material to the pit shape definition.

Source: JDS (2017)

The mineral inventory block models for the Tepal and Tizate deposits were then used with OP optimization software to determine optimal mining shells. This evaluation included the aforementioned parameters. The economic shell limits for both Tepal and Tizate also include Inferred Mineral Resources.

Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves, and there is no certainty that the Inferred Resources would be upgraded to a higher resource category.

16.2.1.1 Cut-off Grade

Table 16.4 summarizes the parameters used to determine the incremental (or mill) cut-off grade (COG) (based on NSR). The incremental (or mill) COG incorporates all OPEX except mining and incorporates dilution. This incremental cut-off is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the NPVS optimization.

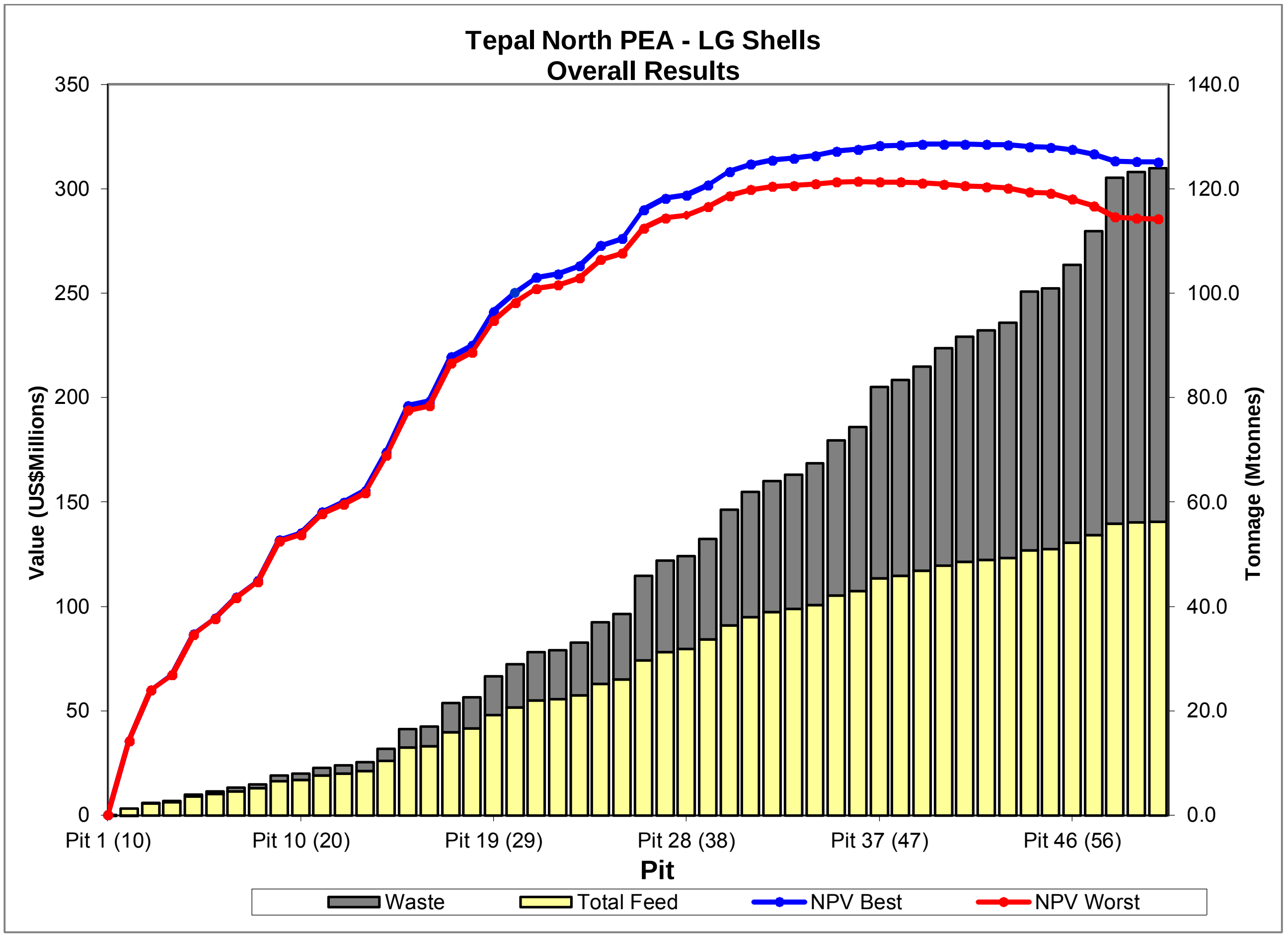
This mill cut-off was applied to all of the estimates that follow. For the sulphide material the NSR cut-off is $8.19/t milled, while for oxide it is $10.13/t milled.

16.2.2 Optimization Results

A series of optimized shells were generated for the Tepal deposits based on varying revenue factors. The results were analyzed with shells chosen as the basis for ultimate limits and preliminary pit stage selection. Refer to Table 16.3 to Table 16.5 and Figure 16.1 to Figure 16.3 for a summary of the optimization results for Tepal North, Tepal South and Tizate. The total diluted tonnes and grade are a combination of sulphide and oxide feed.

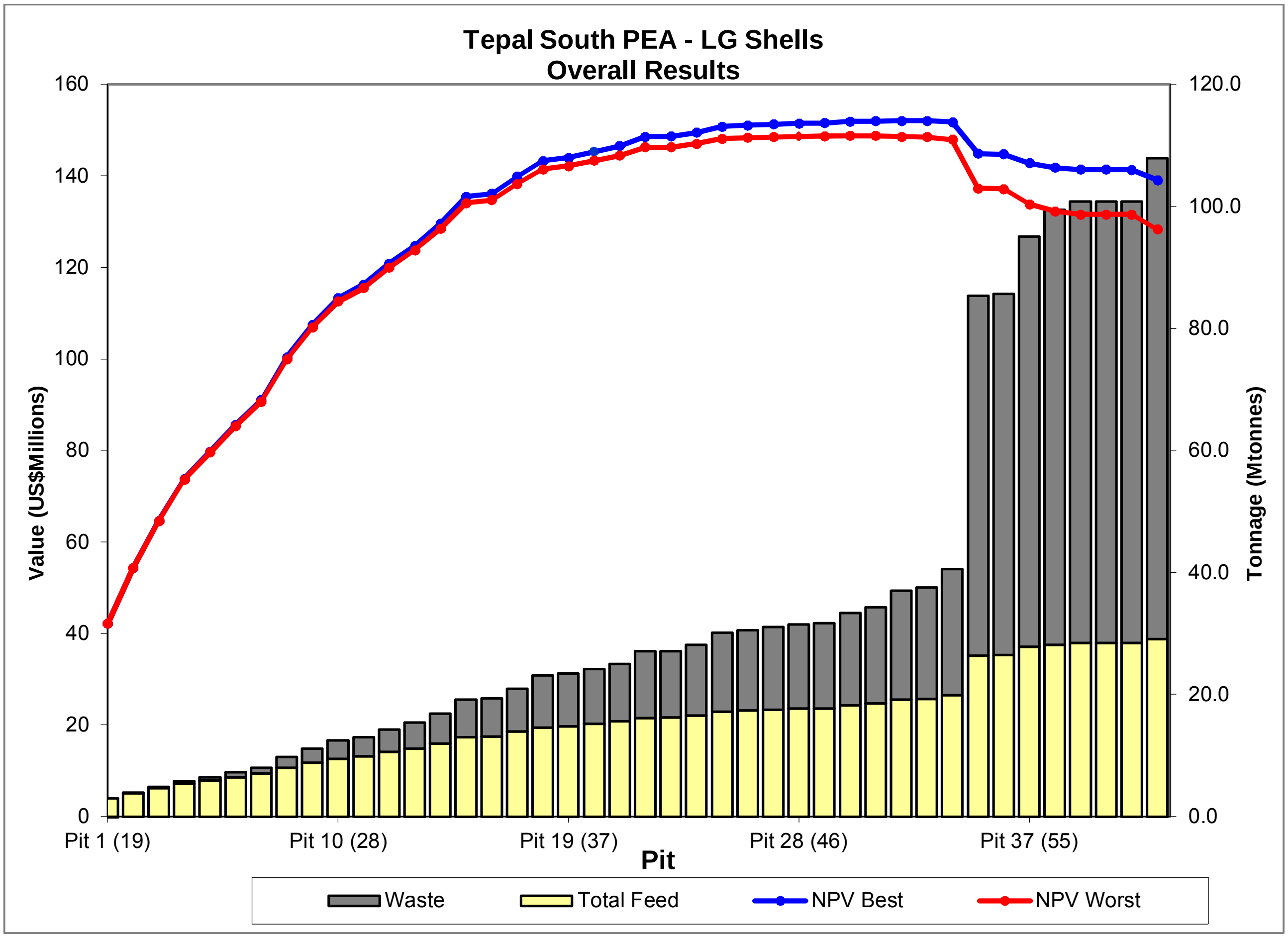
Whittle produces both “best case” (i.e. mine out shell 1, the smallest shell, and then mine out each subsequent shell from the top down, before starting the next shell) and “worst case” (mine each bench completely to final limits before starting next bench) scenarios. These two scenarios provide a bracket for the range of possible outcomes. The shells were produced based on varying revenue factors to produce a series of nested shells with the NPV results shown. Note the NPV values noted here do not include capital cost estimates (CAPEX) and were used only to determine the basic mining shapes. The actual NPV of the Project is summarized in Section 23 of this report.

Figure 16.1: Tepal North Pit Optimization Results



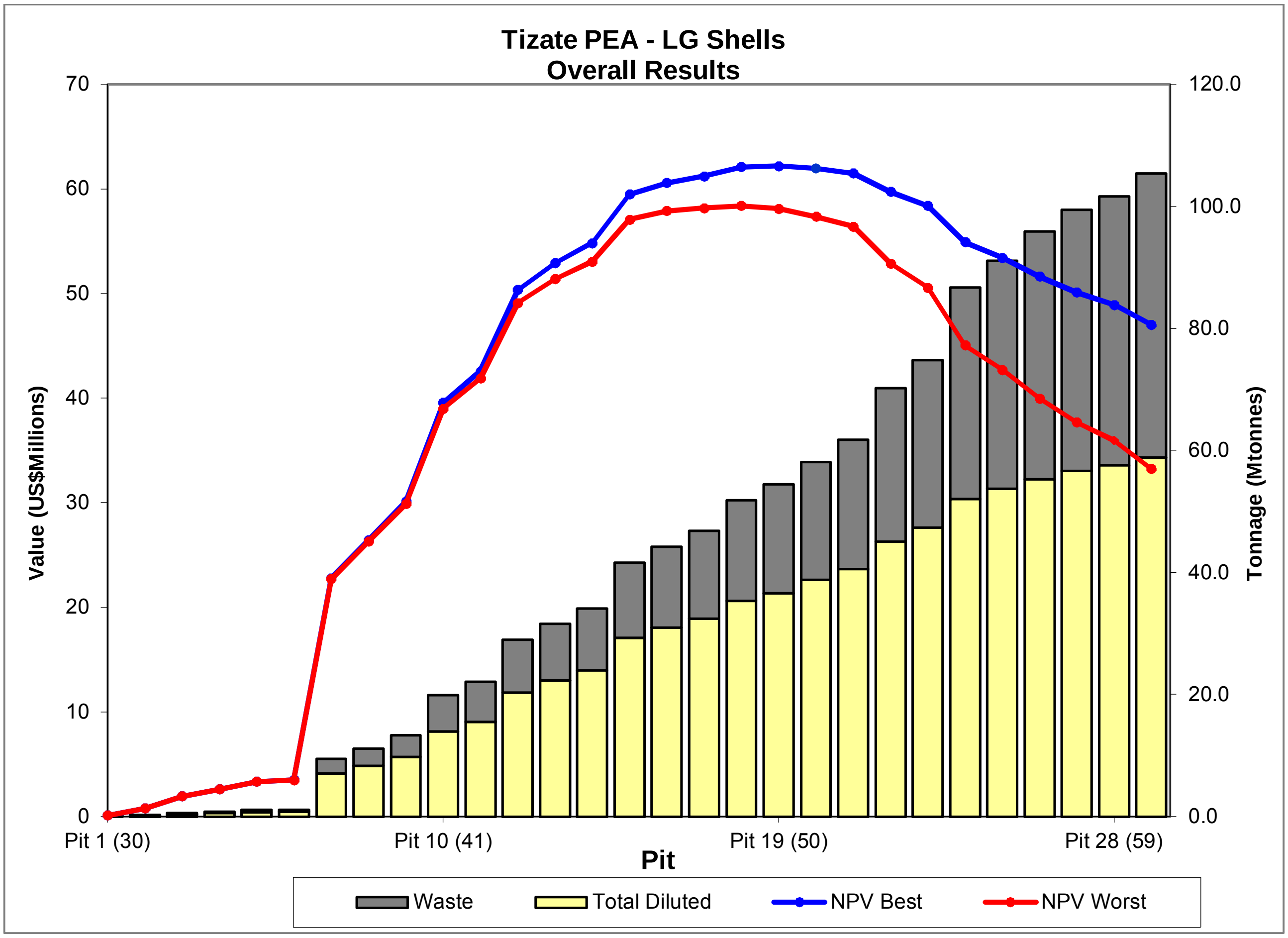
Source: JDS (2017)

Figure 16.2: Tepal South Pit Optimization Results



Source: JDS (2017)

Figure 16.3: Tizate Pit Optimization Results



Source: JDS (2017)

To better determine the optimum shell on which to base the detailed pit and pit stage designs and scheduling, and to gain a better understanding of the various deposits, the shells were analyzed in a preliminary schedule. The schedule assumed a maximum processing rate of 22 kt/d sulphide feed and 8 kt/d of oxide feed. No stockpiles were used in the analysis and no CAPEX was added. Ultimate shells were chosen based on a review of mineralized rock and waste tonnages, incremental strip ratios and impact on the NPV.

Based on the analysis of the shells and preliminary schedule, pit shell 32 was chosen as the base case shell for Tepal North, while pit shell 22 was selected at Tepal South and for Tizate, pit shell 18 for detailed pit designs and scheduling.

Table 16.3: Overall Optimization Results Tepal North (excluding Capital Costs)

Pit (#)	RevFac (value)	Life (yrs)	Total Feed (Mt)	Total Diluted Grades			Waste	Strip Ratio	Total Material	Total CF (US$ M)	NPV Best (US$ M)	NPV Worst (US$ M)
				Au (g/t)	Cu (%)	Ag (g/t)	(Mt)	(t:t)	(Mt)			
Pit 1 (10)	0.20	0.0	0.0	1.55	0.43	0.93	0	-0.04	0	0	0	0
Pit 2 (12)	0.24	0.2	1.3	0.97	0.45	0.87	0	-0.02	1	36	36	36
Pit 3 (13)	0.26	0.3	2.2	0.94	0.43	0.88	0	0.03	2	60	60	60
Pit 4 (14)	0.28	0.3	2.5	0.93	0.42	0.90	0	0.10	3	68	67	67
Pit 5 (15)	0.30	0.4	3.6	0.84	0.41	0.89	0	0.09	4	88	87	87
Pit 6 (16)	0.32	0.5	4.0	0.82	0.41	0.89	0	0.12	5	95	94	94
Pit 7 (17)	0.34	0.6	4.6	0.80	0.40	0.90	1	0.14	5	106	104	104
Pit 8 (18)	0.36	0.6	5.1	0.78	0.39	0.91	1	0.15	6	114	112	112
Pit 9 (19)	0.38	0.8	6.5	0.74	0.36	0.92	1	0.17	8	134	132	131
Pit 10 (20)	0.40	0.8	6.8	0.73	0.36	0.90	1	0.17	8	138	135	134
Pit 11 (21)	0.42	0.9	7.6	0.70	0.35	0.89	1	0.19	9	148	145	144
Pit 12 (22)	0.44	1.0	8.0	0.70	0.35	0.90	2	0.20	10	153	150	149
Pit 13 (23)	0.46	1.1	8.5	0.68	0.35	0.91	2	0.19	10	159	155	154
Pit 14 (24)	0.48	1.3	10.5	0.63	0.34	0.90	2	0.21	13	179	174	172
Pit 15 (25)	0.50	1.6	13.0	0.58	0.32	0.90	4	0.27	17	203	196	194
Pit 16 (26)	0.52	1.7	13.3	0.58	0.32	0.90	4	0.28	17	205	198	196
Pit 17 (27)	0.54	2.0	15.9	0.55	0.30	0.88	6	0.35	21	228	220	216
Pit 18 (28)	0.56	2.1	16.6	0.55	0.30	0.87	6	0.36	23	234	225	222
Pit 19 (29)	0.58	2.4	19.2	0.52	0.29	0.92	7	0.39	27	253	241	237
Pit 20 (30)	0.60	2.6	20.7	0.51	0.29	0.91	8	0.40	29	263	250	245
Pit 21 (31)	0.62	2.7	21.9	0.50	0.28	0.90	9	0.42	31	271	258	252
Pit 22 (32)	0.64	2.8	22.3	0.50	0.28	0.90	9	0.42	32	273	259	254
Pit 23 (33)	0.66	2.9	23.0	0.50	0.28	0.90	10	0.44	33	277	263	257
Pit 24 (34)	0.68	3.1	25.2	0.48	0.27	0.88	12	0.47	37	289	273	266
Pit 25 (35)	0.70	3.2	26.0	0.48	0.27	0.88	13	0.48	39	293	276	269
Pit 26 (36)	0.72	3.7	29.7	0.46	0.27	0.90	16	0.55	46	309	290	281

Pit (#)	RevFac (value)	Life (yrs)	Total Feed (Mt)	Total Diluted Grades			Waste	Strip Ratio	Total Material	Total CF (US$ M)	NPV Best (US$ M)	NPV Worst (US$ M)
				Au (g/t)	Cu (%)	Ag (g/t)	(Mt)	(t:t)	(Mt)			
Pit 27 (37)	0.74	3.9	31.3	0.45	0.26	0.88	18	0.56	49	316	295	286
Pit 28 (38)	0.76	4.0	31.9	0.45	0.26	0.87	18	0.56	50	318	297	287
Pit 29 (39)	0.78	4.2	33.6	0.44	0.26	0.87	19	0.58	53	324	302	291
Pit 30 (40)	0.80	4.5	36.4	0.42	0.25	0.87	22	0.61	58	332	308	297
Pit 31 (41)	0.82	4.7	38.0	0.42	0.25	0.88	24	0.63	62	336	312	300
Pit 32 (42)	0.84	4.8	38.9	0.42	0.25	0.88	25	0.64	64	339	314	301
Pit 33 (43)	0.86	4.9	39.5	0.41	0.25	0.88	26	0.65	65	340	315	302
Pit 34 (44)	0.88	5.0	40.3	0.41	0.24	0.89	27	0.67	67	341	316	302
Pit 35 (45)	0.90	5.2	42.1	0.40	0.24	0.88	30	0.70	72	344	318	303
Pit 36 (46)	0.92	5.4	43.0	0.40	0.24	0.88	31	0.73	74	345	319	304
Pit 37 (47)	0.94	5.6	45.3	0.40	0.24	0.87	37	0.81	82	348	321	303
Pit 38 (48)	0.96	5.7	45.8	0.40	0.24	0.87	38	0.82	83	348	321	303
Pit 39 (49)	0.98	5.8	46.8	0.39	0.24	0.88	39	0.84	86	349	321	303
Pit 40 (50)	1.00	6.0	47.8	0.39	0.23	0.88	42	0.87	89	349	321	302
Pit 41 (51)	1.02	6.0	48.6	0.39	0.23	0.89	43	0.89	92	349	321	301
Pit 42 (52)	1.04	6.1	48.9	0.39	0.23	0.89	44	0.90	93	349	321	301
Pit 43 (53)	1.06	6.1	49.3	0.39	0.23	0.90	45	0.91	94	348	321	300
Pit 44 (54)	1.08	6.3	50.8	0.38	0.23	0.90	50	0.98	100	347	320	298
Pit 45 (55)	1.10	6.3	51.0	0.38	0.23	0.90	50	0.98	101	347	320	298
Pit 46 (56)	1.12	6.5	52.2	0.38	0.23	0.91	53	1.02	105	345	319	295
Pit 47 (57)	1.14	6.7	53.7	0.37	0.23	0.91	58	1.08	112	342	317	292
Pit 48 (58)	1.16	7.0	55.9	0.37	0.23	0.93	66	1.19	122	338	313	286
Pit 49 (59)	1.18	7.0	56.1	0.37	0.23	0.93	67	1.19	123	337	313	286
Pit 50 (60)	1.20	7.0	56.2	0.37	0.23	0.93	68	1.20	124	337	313	286

Note: Green shading denotes planned optimized pit selected for the PEA.
Source: JDS (2017)

Table 16.4: Overall Optimization Results Tepal South (excluding Capital Costs)

Pit (#)	RevFac (value)	Life (yrs)	Total Feed (Mt)	Total Diluted Grades			Waste (Mt)	Strip Ratio (t:t)	Total Material (Mt)	Total CF (US$ M)	NPV Best	NPV Worst
				Au (g/t)	Cu (%)	Ag (g/t)					(US$ M discounted)	(US$ M discounted)
Pit 1 (19)	0.38	0.4	3.0	0.59	0.24	1.05	0	0.00	3	43	42	42
Pit 2 (20)	0.40	0.5	3.8	0.59	0.25	1.05	0	0.02	4	55	54	54
Pit 3 (21)	0.42	0.6	4.6	0.58	0.25	1.06	0	0.06	5	66	65	65
Pit 4 (22)	0.44	0.7	5.4	0.57	0.25	1.06	0	0.08	6	75	74	74
Pit 5 (23)	0.46	0.7	5.9	0.57	0.25	1.06	1	0.09	6	81	80	80
Pit 6 (24)	0.48	0.8	6.4	0.56	0.24	1.07	1	0.13	7	87	86	85
Pit 7 (25)	0.50	0.9	7.0	0.55	0.24	1.07	1	0.14	8	93	91	91
Pit 8 (26)	0.52	1.0	8.0	0.55	0.24	1.09	2	0.22	10	103	100	100
Pit 9 (27)	0.54	1.1	8.8	0.54	0.23	1.09	2	0.26	11	110	107	107
Pit 10 (28)	0.56	1.2	9.5	0.53	0.23	1.09	3	0.31	12	116	113	113
Pit 11 (29)	0.58	1.2	9.9	0.53	0.23	1.09	3	0.32	13	120	116	115
Pit 12 (30)	0.60	1.3	10.5	0.52	0.23	1.09	4	0.35	14	125	121	120
Pit 13 (31)	0.62	1.4	11.1	0.51	0.23	1.10	4	0.39	15	129	125	124
Pit 14 (32)	0.64	1.5	12.0	0.50	0.22	1.10	5	0.41	17	134	130	128
Pit 15 (33)	0.66	1.6	13.0	0.50	0.22	1.11	6	0.48	19	140	135	134
Pit 16 (34)	0.68	1.6	13.1	0.49	0.22	1.11	6	0.48	19	141	136	135
Pit 17 (35)	0.70	1.7	13.9	0.49	0.22	1.10	7	0.50	21	145	140	138
Pit 18 (36)	0.72	1.8	14.6	0.48	0.22	1.10	9	0.58	23	149	143	141
Pit 19 (37)	0.74	1.8	14.8	0.48	0.22	1.10	9	0.59	23	150	144	142
Pit 20 (38)	0.76	1.9	15.2	0.48	0.22	1.10	9	0.59	24	151	145	143
Pit 21 (39)	0.78	1.9	15.6	0.47	0.21	1.09	9	0.60	25	152	147	144
Pit 22 (40)	0.80	2.0	16.2	0.47	0.21	1.09	11	0.67	27	155	149	146
Pit 23 (41)	0.82	2.0	16.2	0.47	0.21	1.09	11	0.67	27	155	149	146
Pit 24 (42)	0.84	2.1	16.6	0.47	0.21	1.09	12	0.70	28	156	149	147
Pit 25 (43)	0.86	2.1	17.2	0.46	0.21	1.08	13	0.75	30	157	151	148
Pit 26 (44)	0.88	2.2	17.4	0.46	0.21	1.08	13	0.75	30	157	151	148

Pit (#)	RevFac (value)	Life (yrs)	Total Feed (Mt)	Total Diluted Grades			Waste (Mt)	Strip Ratio (t:t)	Total Material (Mt)	Total CF (US$ M)	NPV Best	NPV Worst
				Au (g/t)	Cu (%)	Ag (g/t)					(US$ M discounted)	(US$ M discounted)
Pit 27 (45)	0.90	2.2	17.5	0.46	0.21	1.08	14	0.78	31	158	151	149
Pit 28 (46)	0.92	2.2	17.7	0.46	0.21	1.08	14	0.78	32	158	151	149
Pit 29 (47)	0.94	2.2	17.7	0.46	0.21	1.08	14	0.78	32	158	152	149
Pit 30 (48)	0.96	2.3	18.2	0.45	0.21	1.07	15	0.83	33	158	152	149
Pit 31 (49)	0.98	2.3	18.5	0.45	0.21	1.07	16	0.85	34	158	152	149
Pit 32 (50)	1.00	2.4	19.1	0.45	0.21	1.07	18	0.93	37	159	152	149
Pit 33 (51)	1.02	2.4	19.3	0.45	0.21	1.06	18	0.95	38	159	152	148
Pit 34 (52)	1.04	2.5	19.9	0.44	0.21	1.06	21	1.03	41	158	152	148
Pit 35 (53)	1.06	3.3	26.3	0.44	0.20	1.10	59	2.24	85	152	145	137
Pit 36 (54)	1.08	3.3	26.4	0.44	0.20	1.10	59	2.24	86	152	145	137
Pit 37 (55)	1.10	3.5	27.8	0.44	0.20	1.10	67	2.42	95	149	143	134
Pit 38 (56)	1.12	3.5	28.1	0.44	0.20	1.10	71	2.53	99	148	142	132
Pit 39 (57)	1.14	3.5	28.4	0.44	0.20	1.10	72	2.55	101	148	141	132
Pit 40 (58)	1.16	3.5	28.4	0.44	0.20	1.10	72	2.55	101	148	141	132
Pit 41 (59)	1.18	3.5	28.4	0.44	0.20	1.10	72	2.55	101	148	141	132
Pit 42 (60)	1.20	3.6	29.0	0.44	0.20	1.10	79	2.72	108	145	139	128

Note: Green shading denotes planned optimized pit selected for the PEA.

Source: JDS (2017)

Table 16.5: Overall Optimization Results Tizate (excluding Capital Costs)

Pit (#)	RevFac (value)	Life (yrs)	Total Diluted (Mt)	Total Diluted Grades			Waste (Mt)	Strip Ratio (t:t)	Total Material (Mt)	Total CF (US$ M)	NPV Best (US$ M discounted)	NPV Worst (US$ M discounted)
				Au (g/t)	Cu (%)	Ag (g/t)						
Pit 1 (30)	0.60	0.0	0.0	0.41	0.17	1.81	0	0.04	0	0	0	0
Pit 2 (32)	0.64	0.0	0.2	0.38	0.16	1.59	0	0.16	0	1	1	1
Pit 3 (33)	0.66	0.0	0.4	0.36	0.17	2.06	0	0.42	1	2	2	2
Pit 4 (35)	0.70	0.1	0.5	0.35	0.17	2.02	0	0.39	1	3	3	3
Pit 5 (36)	0.72	0.1	0.7	0.33	0.17	2.24	0	0.46	1	3	3	3
Pit 6 (37)	0.74	0.1	0.8	0.33	0.17	2.27	0	0.47	1	4	4	4
Pit 7 (38)	0.76	0.9	7.1	0.26	0.18	2.57	2	0.33	9	23	23	23
Pit 8 (39)	0.78	1.0	8.3	0.25	0.18	2.51	3	0.33	11	27	26	26
Pit 9 (40)	0.80	1.2	9.8	0.25	0.18	2.48	4	0.36	13	31	30	30
Pit 10 (41)	0.82	1.7	14.0	0.24	0.18	2.39	6	0.42	20	41	40	39
Pit 11 (42)	0.84	1.9	15.5	0.23	0.18	2.35	6	0.42	22	45	43	42
Pit 12 (43)	0.86	2.5	20.4	0.22	0.18	2.25	9	0.43	29	53	50	49
Pit 13 (44)	0.88	2.8	22.2	0.22	0.18	2.26	9	0.42	32	56	53	51
Pit 14 (45)	0.90	3.0	24.0	0.22	0.18	2.24	10	0.42	34	59	55	53
Pit 15 (46)	0.92	3.6	29.3	0.21	0.18	2.14	12	0.42	42	64	59	57
Pit 16 (47)	0.94	3.9	31.0	0.21	0.17	2.13	13	0.43	44	65	61	58
Pit 17 (48)	0.96	4.0	32.4	0.21	0.17	2.14	14	0.45	47	66	61	58
Pit 18 (49)	0.98	4.4	35.4	0.21	0.17	2.16	16	0.47	52	67	62	58
Pit 19 (50)	1.00	4.6	36.6	0.21	0.17	2.14	18	0.49	54	68	62	58
Pit 20 (51)	1.02	4.8	38.8	0.20	0.17	2.15	19	0.50	58	67	62	57
Pit 21 (52)	1.04	5.0	40.5	0.20	0.17	2.13	21	0.52	62	67	61	56
Pit 22 (53)	1.06	5.6	45.1	0.20	0.17	2.16	25	0.56	70	64	60	53
Pit 23 (54)	1.08	5.9	47.4	0.20	0.17	2.13	27	0.58	75	63	58	51
Pit 24 (55)	1.10	6.5	52.0	0.20	0.17	2.09	35	0.67	87	58	55	45
Pit 25 (56)	1.12	6.7	53.7	0.20	0.17	2.08	37	0.70	91	56	53	43
Pit 26 (57)	1.14	6.9	55.3	0.20	0.17	2.06	41	0.73	96	54	52	40
Pit 27 (58)	1.16	7.0	56.6	0.20	0.17	2.05	43	0.76	99	51	50	38
Pit 28 (59)	1.18	7.2	57.6	0.19	0.17	2.08	44	0.76	102	50	49	36
Pit 29 (60)	1.20	7.3	58.9	0.19	0.17	2.08	47	0.79	105	47	47	33

Note: Green shading denotes planned optimized pit selected for the PEA.

Source: JDS (2017)

16.3 Open Pit Stages

The pit shapes for Tepal were further analyzed and optimizations were conducted in order to better define the possible stage shapes within the ultimate pit limits. It was decided to divide the pit sequence into three stages at Tepal North (TPN1, TPN2 and TPN3), one stage at Tepal South (TS1) and two stages at Tizate (TIZ1 and TIZ2) for the mine plan development to maximize the grade in the early years, reduce the pre-stripping requirements, and to maintain the process facility at full production capacity. The pit tonnages, grades, and contained metal of the stages for all three deposits are summarized in Table 16.6.

Table 16.6: Tepal Pit Stage Tonnages and Grades

Parameter	Units	Deposit/Stage								Total
		Tepal North				Tepal South	Tizate			
		TPN1	TPN2	TPN3	TPN Total	TPS Total	TIZ1	TIZ2	TIZ Total	
Sulphide Mill Feed*	Mt	11.0	14.1	6.0	**31.1**	**13.4**	18.5	15.3	**33.8**	**78.3**
Gold grade	g/t	0.53	0.35	0.29	**0.4**	**0.47**	0.21	0.19	**0.20**	**0.33**
Copper grade	%	0.32	0.22	0.21	**0.25**	**0.22**	0.17	0.18	**0.17**	**0.21**
Silver grade	g/t	0.80	0.85	1.01	**0.86**	**1.10**	2.38	1.91	**2.17**	**1.47**
Oxide CIL Feed*	Mt	4.8	1.5	1.5	**7.8**	**2.8**	1.4	0.1	**1.5**	**12.2**
Gold grade	g/t	0.52	0.37	0.39	**0.47**	**0.47**	0.30	0.27	**0.29**	**0.45**
Silver grade	g/t	0.95	1.01	0.86	**0.94**	**1.05**	2.05	2.26	**2.07**	**1.11**
Total Feed	Mt	15.8	15.7	7.4	**38.9**	**16.2**	19.9	15.4	**35.4**	**90.5**
Gold grade	g/t	0.52	0.36	0.31	**0.42**	**0.47**	0.22	0.19	**0.21**	**0.34**
Copper grade**	%	0.32	0.22	0.21	**0.25**	**0.22**	0.17	0.18	**0.17**	**0.21**
Silver grade	g/t	0.84	0.87	0.98	**0.88**	**1.09**	2.36	1.91	**2.16**	**1.42**
Contained gold	koz	266	180	75	**521**	**244**	141	95	**236**	**1,000**
Contained copper**	Mlbs	77	69	28	**174**	**64**	68	60	**128**	**366**
Contained silver	koz	429	438	235	**1101**	**565**	1509	949	**2459**	**4,125**
Waste	Mt	5.9	13.6	5.6	**25.1**	**10.9**	10.0	6.4	**16.5**	**52.5**
Strip Ratio	w:o	0.4	0.9	0.7	**0.6**	**0.7**	0.5	0.4	**0.5**	**0.6**
Total Material	**Mt**	**21.7**	**29.3**	**13.0**	**64.0**	**27.1**	**30.0**	**21.9**	**51.8**	**142.9**

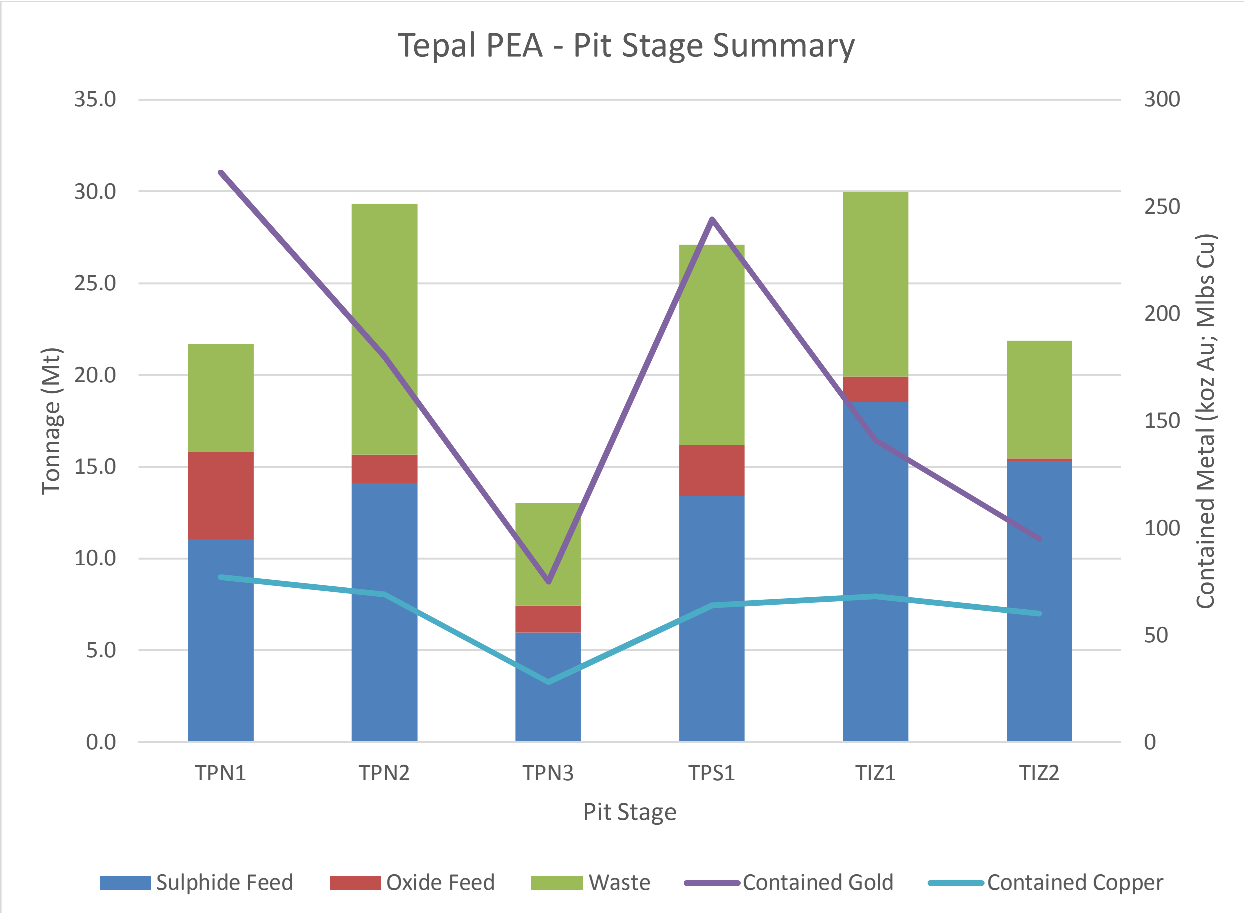
Note: *diluted tonnes and grade @ 5% external dilution and 100% mining recovery

** Copper recovered only from sulphide mill feed

Source: JDS (2017)

Figure 16.4 illustrates the planned stage designs for Tepal, with tonnes, grades, and strip ratios shown. The planned stage designs selected provide reasonable pushback widths with mining starting in the higher grade mineralized zones and progressing outwards from the initial stages.

Figure 16.4: Tepal Project – Pit Stage Summary



Source: JDS (2017)

16.4 Mine Production Schedule

The production schedule for the Tepal deposits incorporates the various pits and pit stages mentioned above. The majority of the feed tonnes at Tepal are comprised of Measured and Indicated resources. Only 2% of the total feed tonnes are based on Inferred Resources.

Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves, and there is no certainty that the Inferred Resources would be upgraded to a higher resource category.

The mining schedule is based on achieving the target 8 Mt/a (22 kt/d) of sulphide feed through the mill. The maximum oxide feed is set at 2.0 Mt/a (5.5 kt/d). Any additional oxide material mined over the maximum feed rate (occurs in the early part of the schedule) is placed in a ROM stockpile and then fed into the mill in a later period as required. Note that the oxide doré production is expected to commence during the second year of mine construction in order to offset a portion of the initial capital cost.

The Tepal deposits are planned to produce a total of 90.5 Mt of mineralized feed (78.3 Mt of sulphide and 12.2 Mt of oxide) along with 52.5 Mt of waste (0.6:1 overall strip ratio) over a 10-year mine operating life (including pre-production). The current LOM plan focuses on achieving consistent processing feed production rates, mining of higher value material early in the schedule, balancing grade and strip ratios, while trying to maximize NPV. Mining is planned to commence at Tepal North and then will move onto Tepal South, with Tizate production planned later in the mine life.

The average mining rate over the 10-year LOM is planned to be 36,000 t/d, reaching a maximum of 46,000 t/d during Year 2.

Table 16.8 is a summary of total material movement by year for the LOM production schedule (both as totals, as well as by each stage and feed type). Table 16.9 illustrates the proposed processing schedule.

Table 16.7: Proposed LOM Production Schedule

Description	Unit	YEAR											
		Pre-prod.	1	2	3	4	5	6	7	8	9	10	Total
TPN1													
Sulphide Resource	Mt		8.0	3.0									**11.0**
Sulphide Cu grade	%		0.35	0.24									**0.32**
Sulphide Au grade	g/t		0.55	0.47									**0.53**
Sulphide Ag grade	g/t		0.84	0.70									**0.80**
Oxide	Mt	1.1	3.7										**4.8**
Oxide Au grade	g/t	0.64	0.48										**0.52**
Oxide Ag grade	g/t	0.80	0.99										**0.95**
Waste	Mt	3.8	1.9	0.2									**5.9**
Rock	Mt	4.9	13.6	3.1									**21.7**
Strip Ratio	w:o	3.4	0.2	0.1									**0.4**
TPN2													
Sulphide	Mt			0.3		7.4	6.4						**14.1**
Sulphide Cu grade	%			0.26		0.21	0.23						**0.22**
Sulphide Au grade	g/t			0.22		0.30	0.43						**0.35**
Sulphide Ag grade	g/t			1.36		0.91	0.77						**0.85**
Oxide	Mt	0.3			0.5	0.7							**1.5**
Oxide Au grade	g/t	0.36			0.41	0.36							**0.37**
Oxide Ag grade	g/t	1.29			1.16	0.78							**1.01**
Waste	Mt	7.8				4.8	1.0						**13.7**
Rock	Mt	8.2		0.3	0.5	12.9	7.5						**29.3**
Strip Ratio	w:o	23.4				0.6	0.2						**0.9**
TPN3													
Sulphide	Mt						1.6	4.3					**6.0**
Sulphide Cu grade	%						0.28	0.19					**0.21**
Sulphide Au grade	g/t						0.26	0.30					**0.29**

Description	Unit	YEAR											
		Pre-prod.	1	2	3	4	5	6	7	8	9	10	Total
Sulphide Ag grade	g/t						1.05	1.00					**1.01**
Oxide	Mt	0.0					1.5						**1.5**
Oxide Au grade	g/t	0.31					0.39						**0.39**
Oxide Ag grade	g/t	0.69					0.86						**0.86**
Waste	Mt	0.4					5.1	0.1					**5.6**
Rock	Mt	0.4					8.2	4.5					**13.0**
Strip Ratio	w:o	29.9					1.6	0.0					**0.7**
TPS1													
Sulphide	Mt			4.7	8.0	0.7							**13.4**
Sulphide Cu grade	%			0.22	0.22	0.21							**0.22**
Sulphide Au grade	g/t			0.46	0.48	0.43							**0.47**
Sulphide Ag grade	g/t			1.03	1.14	1.07							**1.10**
Oxide	Mt		0.5	2.3									**2.8**
Oxide Au grade	g/t		0.44	0.48									**0.47**
Oxide Ag grade	g/t		0.86	1.09									**1.05**
Waste	Mt		1.1	6.5	3.3	0.0							**10.9**
Rock	Mt		1.6	13.5	11.3	0.6							**27.1**
Strip Ratio	w:o		2.5	0.9	0.4	0.0							**0.7**
TIZ1													
Sulphide	Mt							3.7	8.0	6.8			**18.5**
Sulphide Cu grade	%							0.17	0.17	0.16			**0.17**
Sulphide Au grade	g/t							0.21	0.21	0.22			**0.21**
Sulphide Ag grade	g/t							2.94	2.50	1.93			**2.38**
Oxide	Mt							1.4					**1.4**
Oxide Au grade	g/t							0.30					**0.30**
Oxide Ag grade	g/t							2.05					**2.05**
Waste	Mt							6.7	2.8	0.5			**10.0**
Rock	Mt							11.8	10.8	7.3			**30.0**
Strip Ratio	w:o							1.3	0.4	0.1			**0.5**

Description	Unit	YEAR											
		Pre-prod.	1	2	3	4	5	6	7	8	9	10	Total
TIZ2													
Sulphide	Mt									1.2	8.0	6.0	**15.3**
Sulphide Cu grade	%									0.20	0.19	0.16	**0.18**
Sulphide Au grade	g/t									0.15	0.19	0.20	**0.19**
Sulphide Ag grade	g/t									1.90	1.87	1.96	**1.91**
Oxide	Mt								0.0	0.1	0.0		**0.1**
Oxide Au grade	g/t								0.27	0.27	0.28		**0.27**
Oxide Ag grade	g/t								2.03	2.35	2.10		**2.26**
Waste	Mt								1.0	3.4	1.7	0.2	**6.4**
Rock	Mt								1.1	4.8	9.8	6.3	**21.9**
Strip Ratio	w:o								40.6	2.6	0.2	0.0	**0.4**
Total Sulphide	Mt		8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	6.0	**78.3**
Total Sulphide Cu grade	%		0.35	0.23	0.22	0.21	0.24	0.18	0.17	0.17	0.19	0.16	**0.21**
Total Sulphide Au grade	g/t		0.55	0.45	0.48	0.31	0.40	0.26	0.21	0.21	0.19	0.20	**0.33**
Total Sulphide Ag grade	g/t		0.84	0.92	1.14	0.92	0.82	1.89	2.50	1.92	1.87	1.96	**1.47**
Total Oxide	Mt	1.5	4.1	2.3	0.5	0.7	1.5	1.4	0.0	0.1	0.0		**12.2**
Total Oxide Au grade	g/t	0.58	0.48	0.48	0.41	0.36	0.39	0.30	0.27	0.27	0.28		**0.45**
Total Oxide Ag grade	g/t	0.91	0.98	1.09	1.16	0.78	0.86	2.05	2.03	2.35	2.10		**1.11**
Total Waste t	Mt	12.0	3.0	6.7	3.3	4.7	6.1	6.9	3.8	3.9	1.7	0.2	**52.5**
Total Rock t	Mt	13.5	15.2	17.0	11.8	13.5	15.6	16.3	11.9	12.1	9.8	6.3	**142.9**
Total Strip Ratio	w:o	8.2	0.2	0.6	0.4	0.5	0.6	0.7	0.5	0.5	0.2	0.0	**0.6**
Total Material Mined	**t/day**	**36,991**	**41,617**	**46,641**	**32,325**	**36,942**	**42,814**	**44,689**	**32,611**	**33,041**	**26,756**	**17,198**	

Source JDS (2017)

17 Recovery Methods

17.1 Introduction

The process selected is based on test work described in Section 13 and consists of copper flotation of sulphides to produce a saleable concentrate, cyanide leach of the first cleaner tailings and pyrite flotation concentrate and cyanide leach of oxide material. Two plants have been designed; one for the sulphides and one for the oxides.

The sulphide circuit consists of crushing, grinding, conventional copper flotation with regrind, concentrate dewatering, filtering and load-out. The copper rougher tailings will feed a pyrite rougher flotation circuit to produce a concentrate that will be reground and combined with the copper first cleaner tailings to be leached.

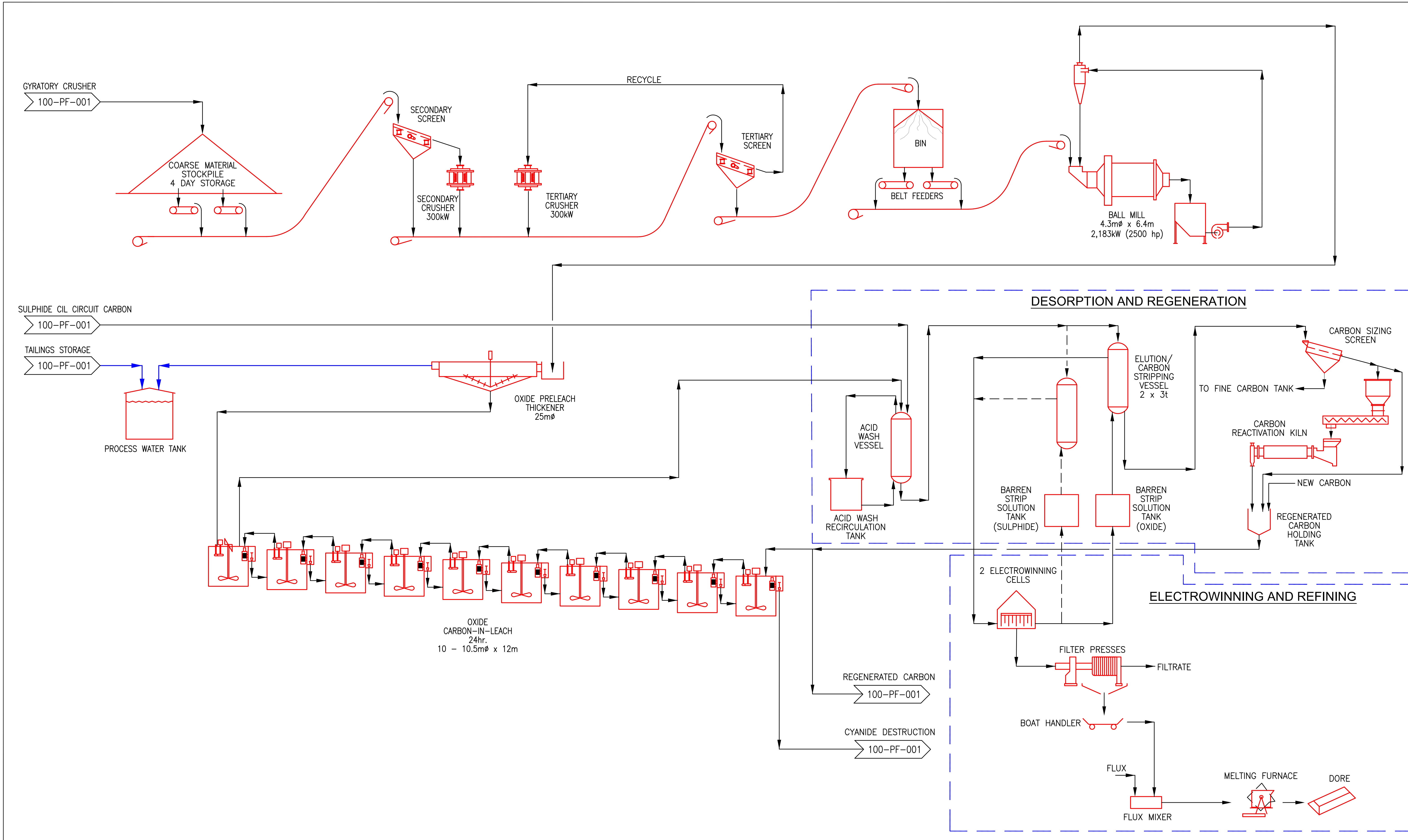
The oxide circuit includes crushing, grinding, carbon-in-leach (CIL) and an adsorption, desorption and refining (ADR) plant. The carbon from the oxide and sulphide CIL circuits will be processed in the ADR plant to produce doré bars. The CIL tailings will feed a cyanide destruction circuit before being recombined with the pyrite rougher tailings to be pumped to the TMF.

The sulphide mill process plant is designed with a nominal capacity of 22,000 t/d and the oxide circuit at 5,500 t/d. The crushing circuit will operate 18 hours per day at a utilization of 75%. The milling and leaching circuits will operate 24 hours per day, 365 days per year at an availability of 92%.

17.2 Oxide Plant Design

Oxide feed will be processed through the gyratory crusher one day out of five to provide 27,500 t to the crushed mineralized material stockpile. The crushed mineralized material stockpile will feed the secondary and tertiary crushing circuits at a rate of 5,500 t/d to produce a final product 80% passing (P_{80}) of 9.5 mm. The crusher product will feed one ball mill for further reduction to P_{80} of 143 microns. From the grinding circuit the cyclone overflow will feed the pre-leach thickener followed by CIL. The loaded carbon from the CIL circuit will be pumped to the ADR plant to remove the precious metals. The CIL tailings will combine with the sulphide CIL tailings at the cyanide destruction circuit. The final tailings will be pumped to the TMF.

Figure 17.1 shows the oxide overall process flowsheet.



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE	REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE	REFERENCE DRAWING	REF. DWG No.

SCALE:	None	DATE:
DESIGNED:	J.D.S.	Sept 23 16
DRAWN:	B. Wong	Sept 23 16
CHECKED:		
APPROVED:		

JDS Energy & Mining Inc.

TITLE
GEOLOGIX EXPLORATION INC.
TEPAL MINE PEA
OXIDE SIMPLIFIED FLOWSHEET, 5,500tpd

PROJECT No.	DRAWING No.	REV No.
	100–PF–002	

17.2.1 Oxide Design Criteria

The proposed grinding circuit will process oxide feed at a nominal rate of 5,500 t/d. The major criteria used in the design are summarized in Table 17.1.

Table 17.1: Major Design Criteria for Oxide Material

Criteria	Units	Value	Source
Primary Daily Process Rate (1 day out of 5)	t/d	27,500	JDS
Crushing Availability	%	75	JDS
Primary Crushing Rate	t/h	1,528	JDS
Secondary Crushing Rate	t/h	306	JDS
Grinding Availability	%	92	JDS
Grinding Process Rate	t/h	249	JDS
Ball Mill Product Size, 80% passing	µm	143	G&TKM3568-03CN
Solids Specific Gravity	N/A	2.45	SRK Grinding Circuit Design, Average of SG Table 2A, 2012
Drop Weight Index, DWi (80% hardest)	kWh/m^3	4.1	SRK Grinding Circuit Design2012
Bond Ball Mill Index, BWi	kWh/t	9.0	Micron PFS 2012
Abrasion Index	g	0.025	Oct. 2010 PEA Tech. Report SRK Section 16.5.2
Pre-leach Thickener Settling Rate	$t/h/m^2$	0.75*	Vendor Recommended
Leach Time	h	24	G&T KM3568-03CN

*No test work available

Source: JDS (2017)

17.2.2 Oxide Coarse Feed Stockpile and Reclaim

The coarse mineralized material storage facility will consist of a stockpile with two in-line belt feeders located within a corrugated pipe reclaim tunnel. The belt feeders will transfer mineralized material to the conveyor feeding a double deck screen. The stockpile will have a 30,000 t capacity that can support process plant operations for five days when the primary gyratory crusher is processing sulphide material.

17.2.3 Oxide Crushing and Screening

The oxide stockpile will feed a double deck vibrating screen with the top deck feeding the secondary crusher and the second deck feeding the tertiary crusher. The screen undersize, with a target P_{80} of 9.5 mm, will feed the fine material surge bin.

17.2.4 Oxide Grinding Circuit

The grinding circuit will consist of a ball mill operating in closed circuit with a hydrocyclone cluster. Material from the 6 hour surge bin will be fed to the ball mill via the ball mill feed conveyor. The grinding circuit will operate at a nominal throughput of 249 t/h (fresh feed), and produce a final particle size P_{80} of 143 µm.

The preliminary ball mill is sized to be 4.3 m in diameter by 6.4 m in length driven by a 1,865 kW motor. Additional test work is recommended in the next stage of engineering to confirm the BWi.

Water will be added to the ball mill to maintain the mineralized feed charge in the mill at a constant slurry density of 70%. Slurry will overflow from the mill to a trommel screen, attached to the mill discharge end. The mill trommel screen oversize will overflow into a trash bin for removal from the system.

The ball mill hydrocyclone cluster will classify the feed slurry into coarse and fine fractions. The coarse underflow will flow back to the ball mill feed end for additional grinding. The overflow will flow by gravity to the trash screen ahead of the pre-leach thickener.

17.2.5 Oxide Pre-leach Thickener

The rougher concentrate will be pumped to a vibrating trash screen for removal of trash material and then feed the concentrate thickener. Flocculant solution (anionic polyacrylamide) will be added to the thickener feed to promote settling of the solids. The thickener will have a diameter of 25 m and produce a thickened product of 45% solids in the underflow. The thickener overflow will report to the process water tank.

The underflow slurry from the thickener will be pumped to the CIL circuit.

17.2.6 Oxide Leaching

The thickener underflow will be pumped to the first of 10 to 10.5 m diameter by 12 m high CIL tanks. The CIL circuit is designed to provide 24 hours of residence time. Each tank includes an agitator, carbon transfer pump and interstage screen. All leach tanks will be located outside and adjacent to the main process building.

As the slurry flows through the 10 CIL tanks, it will be leached and the dissolved gold and silver will be adsorbed onto activated carbon. The loaded carbon will be transferred to the same ADR plant as the loaded carbon from the sulphide process plant.

The average carbon concentration in the CIL circuit is expected to be approximately 25 g/L. As the slurry proceeds through the circuit, metal values in the solids and solution will progressively decrease. Carbon will leave the first CIL tank once metal loading reaches its maximum. The carbon is transferred countercurrent to the slurry flow to maximize precious metal recovery. Loaded carbon will be collected and transferred to the acid wash tank at a rate of 3 t/d. The tailings stream from the CIL circuit will flow by gravity onto a stationary safety screen to capture any carbon particles that may have escaped from the final CIL tank. Captured carbon particles will be collected in bins. Safety screen undersize will then be pumped to the CIL tailings CN destruction circuit.

Lime slurry will be added to the first and second leach tanks to maintain protective alkalinity at a design pH of 10 to prevent evolution of hydrogen cyanide gas (HCN). Cyanide will be added to the circuit and oxygen/air will be sparged from the bottom of the leach tanks.

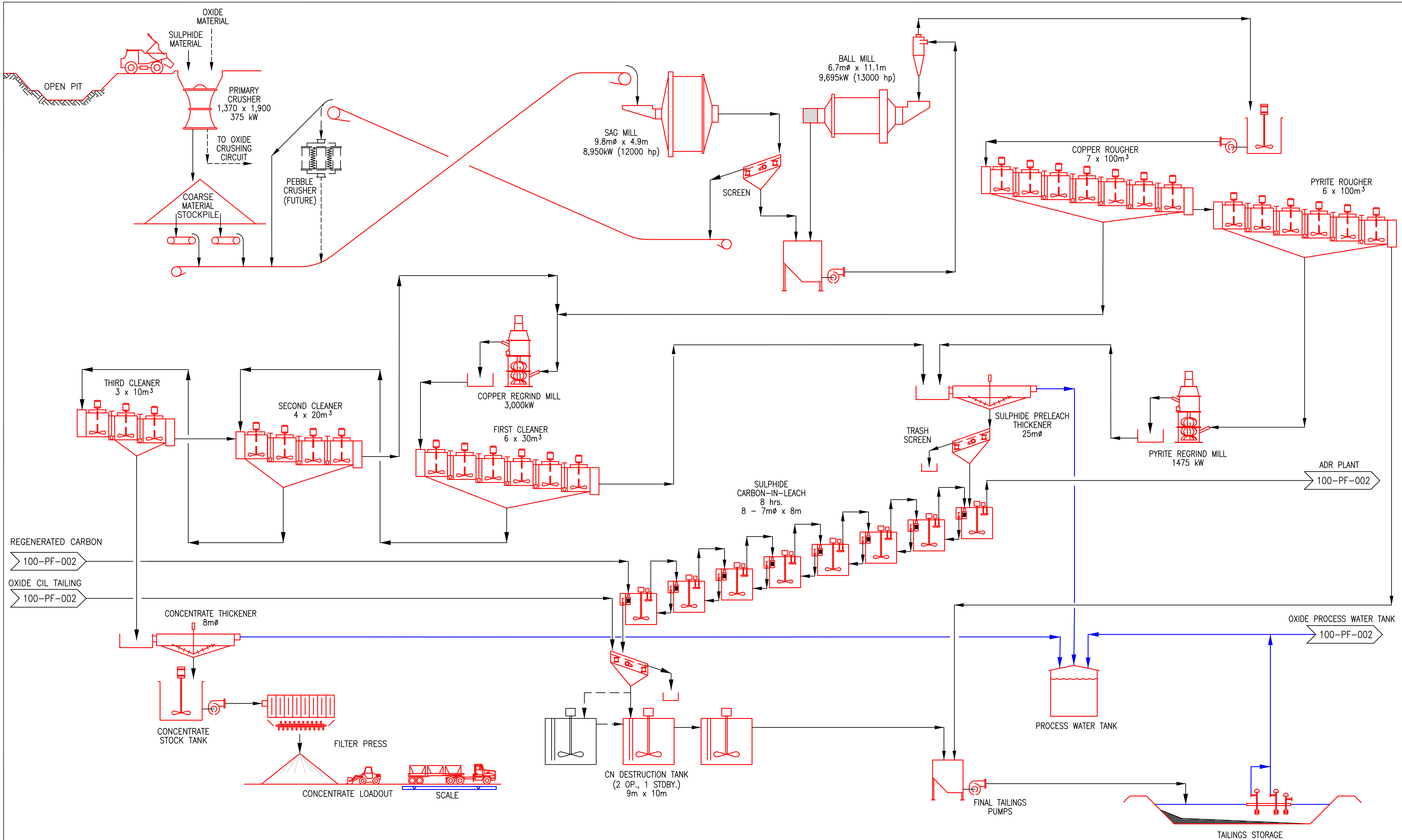
17.3 Sulphide Plant Design

The sulphide concentrator was designed to process 22,000 t/d sulphide mineralized feed. The ROM will be reduced through three stages of comminution, then the copper minerals, along with some gold and silver, will be recovered by flotation.

The copper rougher/scavenger concentrates will be reground and cleaned to a final commercial concentrate grade and then dewatered. The produced copper-gold concentrate will be trucked off-site to a copper smelter.

Copper rougher/scavenger tailings will be sent to pyrite flotation. Pyrite concentrate and the first copper cleaner tailings will be combined and thickened for feed to a sulphide CIL circuit. Loaded carbon will be sent to the common oxide/sulphide ADR plant where doré bars will be produced. The combined flotation tailings and cyanide destruction tailings will be pumped to the TMF. A reclaim barge will recover water from the TMF for re-use in the process plant as make-up water.

Figure 17.2 below shows the sulphide overall process flowsheet.



SCALE:	None
DESIGNED:	J.D.S. — Sept 23 16
DRAWN:	B. Wong — Sept 23 16
CHECKED:	
APPROVED:	

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JDS Energy & Mining Inc.

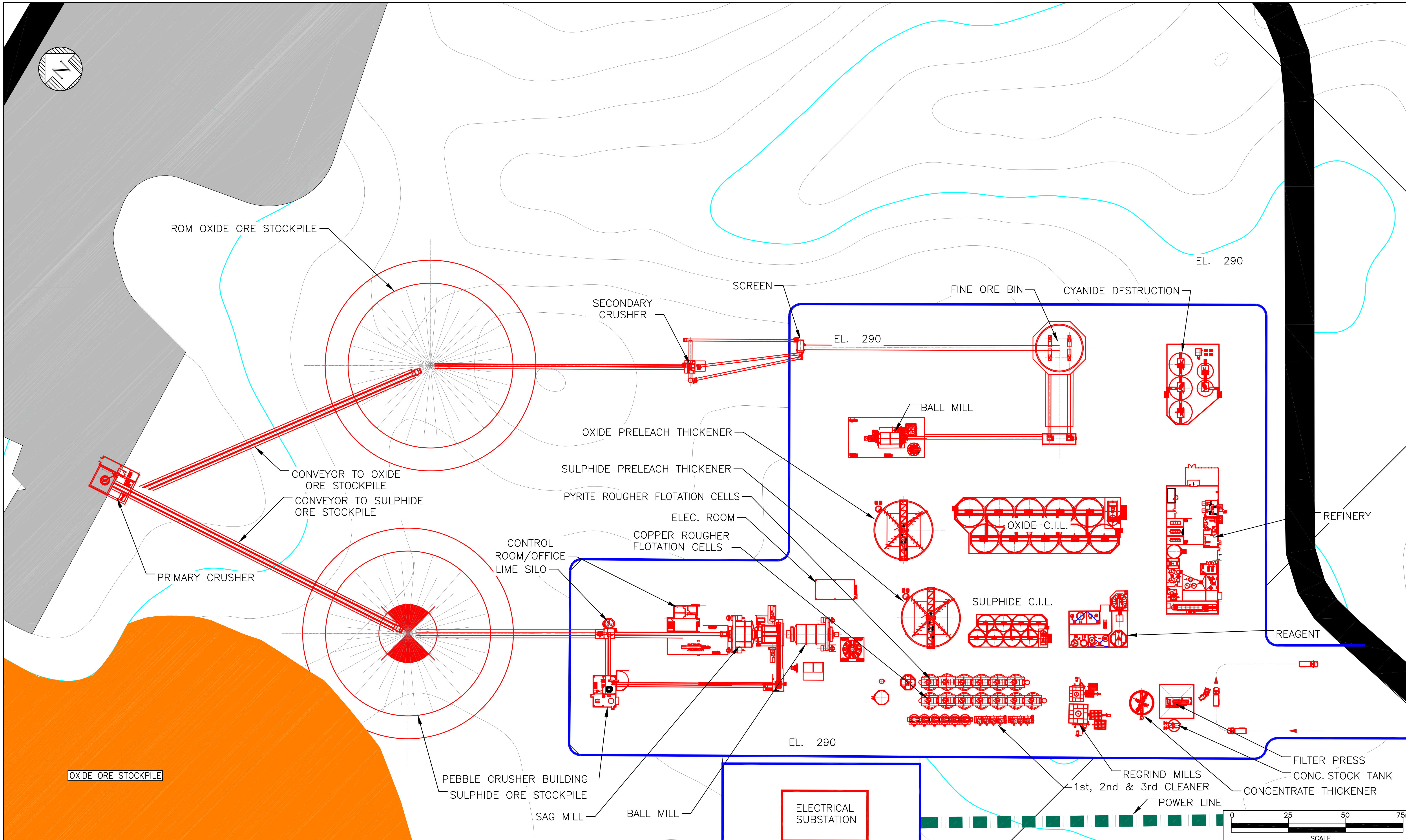
TITLE
GEOLOGIX EXPLORATION INC.
TEPAL MINE PEA
SULPHIDE SIMPLIFIED FLOWSHEET, 22,000tpd

PROJECT No. | DRAWING No. 100-PF-001 | REV No.

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

- Primary crushing;
- Conveying to coarse mineralized feed stockpile;
- Coarse mineralized feed reclaim;
- Primary grinding circuit;
- Copper rougher/scavenger flotation;
- Rougher/scavenger concentrate regrind;
- Three-stage copper cleaner flotation;
- Copper concentrate thickening, pressure filtration and load-out;
- Pyrite rougher flotation;
- Pyrite concentrate and first copper cleaner tailings leaching; and
- Tailings disposal to the tailings management facility.

Figure 17.3 below shows the process plant layout.



ROM OXIDE ORE STOCKPILE
SCREEN
SECONDARY CRUSHER
FINE ORE BIN
CYANIDE DESTRUCTION
EL. 290
EL. 290
BALL MILL
OXIDE PRELEACH THICKENER
SULPHIDE PRELEACH THICKENER
PYRITE ROUGHER FLOTATION CELLS
ELEC. ROOM
COPPER ROUGHER FLOTATION CELLS
CONTROL ROOM/OFFICE
LIME SILO
CONVEYOR TO OXIDE ORE STOCKPILE
CONVEYOR TO SULPHIDE ORE STOCKPILE
PRIMARY CRUSHER
OXIDE C.I.L.
SULPHIDE C.I.L.
REFINERY
REAGENT
FILTER PRESS
CONC. STOCK TANK
CONCENTRATE THICKENER
REGRIND MILLS
1st, 2nd & 3rd CLEANER
POWER LINE
EL. 290
OXIDE ORE STOCKPILE
PEBBLE CRUSHER BUILDING
SULPHIDE ORE STOCKPILE
SAG MILL
BALL MILL
ELECTRICAL SUBSTATION
0
25
50
75m
SCALE

17.3.1 Sulphide Design Criteria

The concentrator is planned to process sulphide mineralized feed at a nominal rate of 22,000 t/d. The major criteria used in the design are summarized in Table 17.2.

Table 17.2: Major Design Criteria for Sulphide Material

Criteria	Unit	Value	Source
Operating Days	d	365	JDS
Operating Hours	h/d	24	JDS
Daily Process Rate	t/d	22,000	JDS
Crushing Availability	%	75	JDS
Primary Crushing Rate	t/h	1,528	JDS
Grinding & Flotation Availability	%	92	JDS
Grinding & Flotation Process Rate	t/h	996	JDS
SAG Mill Feed Size, 80% passing	mm	150	Vendor Simulation
Ball Mill Product Size, 80% passing	µm	150	G&T KM3578
Concentrate Regrind Size, 80% passing	µm	22	G&T KM3578
Solids Specific Gravity	-	2.74	SRK Grinding Circuit Design, Average of SG Table 2A, 2012
Drop Weight Index, 80% hardest, Tepal N & S	kWh/m^3	8.3	SRK Grinding Circuit Design, 2012
Bond Ball Mill Work Index, 80% hardest, Tepal N & S	kWh/t	17.5	SRK Grinding Circuit Design, Average of SG Table 2A, 2012
Drop Weight Index, 80% hardest, Tizate	kWh/m^3	10.3	SRK Grinding Circuit Design, Average of SG Table 2A, 2012
Bond Ball Mill Work Index, 80% hardest, Tizate	kWh/t	20.0	SRK Grinding Circuit Design, Average of SG Table 2A, 2012

Source: JDS (2017)

The SAG mill and ball mills were sized based on the drop weight index and the Bond ball mill work index for the Tepal North and South mineralized material deposits and Tizate deposit.

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

17.3.2 Primary Crushing

The gyratory crusher is proposed as a permanent installation that will take ROM mineralized feed and produce a product of 80% passing 150 mm. Haul trucks are planned to supply ROM material to the primary crusher dump pocket, where they will unload from one of two dump aprons. The dump pocket will have a hydraulic rock breaker to reduce any oversize rocks that may clog the crusher feed. The gyratory crusher will process the sulphide and oxide ROM mineralized feed at a rate of 1,528 t/h.

The coarse material will discharge from the underside of the crusher into a hopper from which an apron feeder will metre the flow onto the sacrificial primary crusher discharge belt conveyor. The material will then be conveyed to either the sulphide or oxide coarse mill feed stockpile. The crusher will process 27,500 t/d for four days to supply the sulphide stockpile with a 5-day supply. On the fifth day, 27,500 t of oxide mineralized material will be processed for the oxide plant.

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

17.3.3 Stockpile and Reclaim

The coarse mill feed storage facility will consist of a stockpile with two in-line belt feeders located within a corrugated pipe reclaim tunnel. The belt feeders will transfer mill feed rock to the conveyor feeding the SAG mill in the plant. The stockpile will have a 30,000 t capacity that can support process plant operations for the one day when the primary gyratory crusher is processing oxide material. Apron feeders will reclaim the material and metre the flow onto the SAG mill feed conveyor. The SAG feed conveyor will be equipped with a belt scale.

A dust collection and suppression system will 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 incorporates a SAG mill and ball mill at a processing rate of 996 t/h or 22,000 t/d. For the harder Tizate mill feed material, the production rate will need to be decreased. The mine plan has been adjusted to feed Tepal mineralized material in the first few years. This has the benefit of higher throughput and higher head grades.

The SAG mill will be fed at a controlled rate by the reclaim apron feeders located under the coarse mill feed stockpile. Lime will be added to the SAG mill feed belt conveyor to raise the pH of the slurry to 10.5, to aid copper flotation. A SAG mill ball bin and feeder will 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, will pass over a screen to remove over-size pebbles. The pebbles will be conveyed by a series of three conveyors back to the SAG mill feed conveyor. A pebble crusher will be added to the circuit to handle pebbles from the harder Tizate mill feed material or earlier if required

The SAG mill screen underflow will combine with both ball mill discharges into one common pump box. The ball mill will operate in closed circuit with cyclone clusters. The overflow slurry stream will feed the copper rougher flotation circuit at a target P_{80} of 150 µm at approximately 35% solids by weight. Cyclone underflow will flow by gravity to the ball mill at approximately 70% solids by weight. The circulating load will be approximately 300% of new mill feed. A ball charge system will add grinding media as required for maintaining grinding charge.

17.3.5 Copper Rougher Flotation

The overflow slurry from the cyclone clusters will flow by gravity to the flotation conditioning tank. Reagents will be added to the conditioning tank to prepare the slurry as feed to the first of seven 100 m_3 mechanically agitated copper rougher flotation cells. The total retention time of the rougher circuit will be 16 minutes with a total mass pull to the concentrate of approximately 9.2%. The copper rougher concentrate will be pumped to the regrind circuit and the tailings will feed the pyrite flotation circuit.

Flotation reagents include lime for pH control, 3418A and PAX as collectors, and methyl isobutyl carbinol (MIBC) as the frother.

17.3.6 Regrind Circuit

The concentrate from the copper rougher circuit will be pumped to the regrind circuit for further size reduction to a target P_{80} of 22 µm. The regrind cyclone feed will be pumped to the regrind cyclone cluster. The cyclone overflow will bypass the regrind circuit and feed the regrind product pump box. The cyclone underflow, containing approximately 70% of the feed to the cyclone, will be approximately 50% solids by weight and will report to the regrind mill feed pump box.

17.3.7 Cleaner Flotation

Regrind product will be pumped to the first cleaner flotation cells. Concentrate from the first cleaner cells will be pumped to the second cleaner flotation cells while second cleaner concentrate will be pumped to the third cleaner flotation cells. Concentrate from the third cleaner flotation cells will be the final copper concentrate. Tailings from stage three will flow by gravity into the feed of the second cleaner cells, and the tailings from the second cleaner cell will flow into the first cleaner flotation cells. Tailings from the first cleaner will flow by gravity into the sulphide pre-leach thickener feed pump box.

17.3.8 Concentrate Dewatering, Filtering and Load-out

The concentrate from the third cleaner will be pumped to an 8 m diameter, high rate thickener. Flocculant will be added to the thickener feed to accelerate the settling process. The underflow will be thickened to 60% solids and then pumped to the 8 hour concentrate stock tank. The overflow will be sent to the process water tank to be used as make-up water in the plant.

Concentrate from the stock tank will be pumped to the pressure filter to reduce the moisture content to approximately 8%. The filtered concentrate will drop to the floor below and a front-end loader will be used to load the concentrate into trucks and transported to the nearest port to await shipment to the smelter.

Additional test work is recommended in the next stage of engineering to determine the settling rate and filtration rate of the concentrate.

17.3.9 Pyrite Flotation and Regrind

The copper rougher tailings will feed six 100 m^3 mechanically agitated pyrite rougher flotation cells, providing a total retention time of 12 minutes and a total mass pull to the concentrate of approximately 6.6%. The pryite rougher concentrate is pumped to the regrind circuit and the tailings reports to the final tailings pump box.

The pyrite concentrate will be sent to the regrind circuit to reduce the feed size to a P_{80} of 23 µm. The regrind cyclone feed will be pumped to the regrind cyclone cluster. The cyclone overflow will bypass the regrind circuit and feed the regrind product pump box. The cyclone underflow, containing approximately 70% of the feed to the cyclone, will be approximately 50% solids by weight and will report to the regrind mill feed pump box.

17.3.10 Sulphide (Pyrite and First Cleaner Tailings) Pre-leach Thickener and Leaching

Pyrite concentrate combined with the first copper cleaner tailings will be pumped to a dedicated thickener and thickened to about 45% solids. The thickener underflow will be pumped to the CIL circuit. The thickener overflow will be pumped to the process water tank to be used as make-up water in the process plant.

The CIL circuit includes eight 7 m diameter by 8 m high tanks. The CIL tanks will provide eight hours of residence time. All tanks will be the same design as those used in the oxide leach, arranged in the same hexagonal pattern to minimize the footprint, and will sit on a series of descending steps.

Loaded carbon from the sulphide leach circuit will be sent to the sulphide/oxide ADR plant to recover gold and silver. The sulphide CIL tailings will be pumped to the cyanide destruction circuit where it will be combined and treated with the oxide CIL tailings.

17.3.11 ADR Plant – Oxide and Sulphide

The carbon from both the oxide and sulphide CIL circuits will be processed in the same ADR plant. The plant will be sized to process 3 t of carbon from each circuit. The overall capacity of the plant is 6 t/d. Additional test work is recommended to confirm carbon loading and design parameters.

17.3.11.1 Carbon Acid Wash

Loaded carbon will be treated with a 3% hydrochloric acid solution in the acid wash tank to remove calcium deposits, magnesium, sodium salts, silica, and fine iron particles. Organic foulants such as oils and fats are unaffected by the acid and will be removed after the elution step by thermal reactivation utilizing a kiln.

The carbon will first be rinsed with fresh water. Acid will then be pumped from the dilute acid tank to the acid wash vessel. Acid will be pumped upward through the acid wash vessel and overflow back to the dilute acid tank. The carbon will then be rinsed and neutralized with fresh water to remove the acid and any mineral impurities.

A recessed impeller pump will transfer acid washed carbon from the acid wash vessel into the elution vessel. Carbon slurry will discharge directly into the top of the elution vessel. Under normal operation, only one elution will take place each day.

17.3.11.2 Carbon Stripping (Elution)

The carbon stripping (elution) process will utilize barren solution to strip the carbon to create a pregnant solution, which will be pumped through electrowinning and back to the strip column.

The strip column will be a carbon steel tank.. During the strip cycle, solution containing approximately 1% sodium hydroxide and 0.1% sodium cyanide at a temperature of 140°C (280°F) and 450 kPa (65 psi) will be circulated through the strip vessel. Solution exiting the top of the elution vessel will be cooled below its boiling point by the heat recovery heat exchanger. Heat from the outgoing solution will be transferred to the incoming cold solution, prior to the cold solution passing through the solution heater. An electric boiler will be used as the primary solution heater.

17.3.11.3 Carbon Regeneration

A recessed impeller pump will transfer the stripped carbon from the elution vessel to the kiln feed dewatering screen. The kiln feed screen doubles as a dewatering screen and a carbon sizing screen, where fine carbon particles will be removed. Oversize carbon from the screen will discharge by gravity to the carbon regeneration kiln feed hopper. Screen undersize carbon, containing carbon fines and water, will drain by gravity into the carbon fines tank. Subsequently, the carbon fines will be collected into bags for disposal. An electric fired horizontal kiln with residual heat dryer will be utilized to regenerate 100% of the stripped carbon. The regeneration kiln discharge will be transferred to the carbon quench tank by gravity, cooled by fresh water and/or carbon fines water prior to being pumped back into the processing circuit. The carbon regeneration will use residual heat from the kiln to heat the pre-dryer.

To compensate for carbon losses by attrition, new carbon is added to the carbon attrition tank along with fresh water, mixed and pumped to the kiln discharge screen. The fresh carbon will be combined with the regenerated carbon in the quench tank.

17.3.11.4 Gold Electrowinning and Refining

Pregnant solution from the strip vessel will be pumped to the refinery for electrowinning to produce a gold sludge. Pregnant solution will be pumped through the electrowinning cell and the resulting barren solution will be pumped back into the barren solution tank for reuse, with periodic bleeding to the CIL circuit.

Gold-rich sludge will then be washed off the steel cathodes in the electrowinning cell (using high-pressure water) into the sludge holding tank. Periodically, the sludge will be drained, filtered, dried, mixed with fluxes and smelted in an electric direct-fire induction furnace to produce gold doré. This process will take place within a secure and supervised area. The gold doré will be stored in a vault to await shipment.

17.3.12 Cyanide Destruction

The cyanide destruction of CIL tailings thickener underflow will consist of three (1 on standby) mechanically agitated tanks, each with a capacity of 620 m^3. Cyanide will be destroyed using the SO_2/Air process. Treated slurry from the cyanide destruction circuit will flow by gravity to the tailings pump box.

Process air will be sparged near the bottom of two 9 m diameter by 10 m high cyanide destruction tanks, under the agitator impeller, for two hours. Lime slurry will be added to maintain the optimum pH of 8.0 – 8.5 – 9.0 and copper sulphate ($CuSO_4$) will be added as a catalyst. Sodium metabisulphite (SMBS) will be dosed into the system as a solution as the source of SO_2. This system has been designed to reduce the total cyanide concentration to less than 5 ppm CN_{WAD} prior to transfer to the final tailings pumpbox. No test work has been complete for cyanide destruction and is recommended in the next stage of engineering to confirm the design parameters.

17.3.13 Tailings

Combined oxide and sulphide leach tailings from the cyanide destruction circuit and pyrite rougher tailings will report to the final tailings pump box. The final tailings will be pumped to the tailings pond. Water from the tailings pond will be pumped back to the process water tank for use in the plant.

17.4 Reagents Handling and Storage

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

Reagents used in the process will include:

- Lime;
- Aero 3418;
- Frother (MIBC);
- Potassium Amyl Xanthate (PAX);
- Flocculant(s);
- Caustic Soda;
- Sodium Cyanide;
- Metabisulphite (SMBS);
- Sodium SMBS;
- Copper Sulphate ($CuSO_4$); and
- 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 will be used for reagent preparation. The mixing and holding tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. The reagent preparation areas will 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 will be delivered in bulk and pneumatically unloaded into a silo. The lime silo will have seven days of storage. Some quicklime will be slaked on-site, and the milk of lime will be pumped to the points of addition using a closed loop system.

MIBC and HCL will be supplied in 1 t totes. Metering pumps will be connected to the bulk containers, and pumped directly to the points of addition.

17.4.1 Water Supply

The water for the mill operation will be supplied from the tailings pond reclaim water system. Reclaimed water from the tailings pond will comprise most of the process water, with the balance supplied as fresh water from the fresh water tank and thickener overflow.

17.4.2 Process Communication System

The process communication system will be based on an ethernet fibre-optic network. This network will provide communication between the process controllers in the electrical rooms at primary crushing, coarse mineralized feed, SAG mill, grinding, flotation, oxide processing, process water pond, main substation and server room.

Radio-based (MODEM) process control communication will 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 will also be provided between the controller at the main substation and the controller at the reclaim barge.

Cable-based ethernet links will 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 will 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 will connect the process communication system with the business communication system.

18 Project Infrastructure and Services

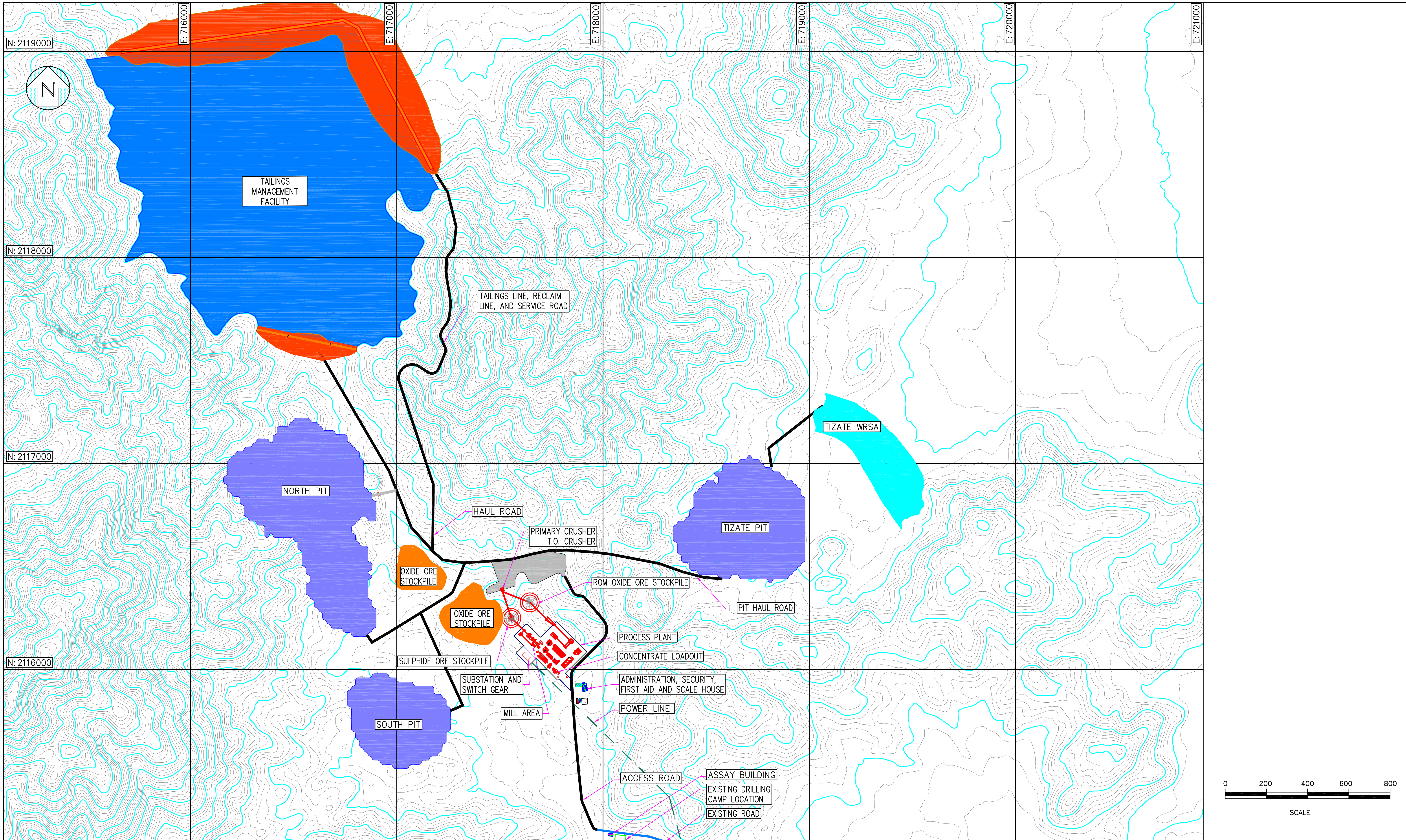
18.1 Summary

The Project envisions the upgrading or construction of the following key infrastructure items:

- Approximately 8 km plant site access road;
- Power supply from the Comisión Federal de Electricidad (CFE) grid, transmission to site, and Project site distribution;
- Tailings management facility (TMF);
- Site haul roads and service roads;
- Waste rock storage area (WRSA);
- Crushing and grinding circuits;
- Oxide plant with CIL;
- Sulphide plant with copper flotation and CIL;
- Adsorption, desorption and refining (ADR) plant;
- Assay laboratory;
- Security, scale house, administration and first aid facilities;
- Fresh water supply, fire/fresh water storage and distribution, sewage collection and treatment and drainage and run-off settling ponds; and
- Permanent accommodation complex.

18.2 General Site Arrangement

An overall site plan for the Project area is shown in Figure 18.1.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E: 721000
N
TAILINGS MANAGEMENT FACILITY
TAILINGS LINE, RECLAIM LINE, AND SERVICE ROAD
TIZATE WRSA
NORTH PIT
HAUL ROAD
PRIMARY CRUSHER T.O. CRUSHER
TIZATE PIT
OXIDE ORE STOCKPILE
ROM OXIDE ORE STOCKPILE
OXIDE ORE STOCKPILE
PIT HAUL ROAD
PROCESS PLANT
CONCENTRATE LOADOUT
SULPHIDE ORE STOCKPILE
SUBSTATION AND SWITCH GEAR
ADMINISTRATION, SECURITY, FIRST AID AND SCALE HOUSE
MILL AREA
POWER LINE
SOUTH PIT
ACCESS ROAD
ASSAY BUILDING
EXISTING DRILLING CAMP LOCATION
EXISTING ROAD
0
200
400
600
800
SCALE

18.3 Site Access

The proposed site access road will be an 8 km basic one-lane “farm-road” with two stream crossings and no major elevation changes. The site access road will be upgraded prior to construction of the mine to facilitate travel between the main road and the Tepal site. There will be a small crossing over the irrigation channel at the T-junction of the main road and the site access road that will have to be temporarily expanded in order to accommodate 40’ long trucks. The site access road will be a “Type D” road as defined by Mexican standards, with a width of 10 m, shoulders, fore-slopes, and storm water run-off ditches on both sides of the road. The existing road will be widened, raised with a sub-base, and resurfaced with gravel. Two “Arizona-type” stream crossings will be required. These crossings will be constructed of reinforced concrete within the stream channel.

18.4 Power Supply

Electrical power for the Project will be provided by CFE, the federal power authority in Mexico. A new 20 km long, 115kV overhead power line will connect the substation at the tap off point in Tepalcatepec to the mine site substation. Power will be distributed to the tailings reclaim area via a 25kV overhead line.

The total Project electrical peak load is estimated to be approximately 28 MW with the total average annual power consumption estimated to be approximately 174,941,000 kWh/a.

The permanent emergency power distribution system is planned from the emergency diesel powered generators at the main substation. All loads that require emergency power will be fed from the emergency generators through their normal feeding paths.

18.5 Ancillary Facilities

It is anticipated that the mining contractor will provide the following facilities:

- Truck shop and truck wash facilities;
- Detonator and explosive storage;
- Fuel storage; and
- Mine dry.

18.5.1 Security and Scale House

The security and scale house is planned to be located near the south-west corner of the mill site. This will be the location from which all persons and vehicles entering and leaving site will be monitored and controlled. The building will include an ambulance and first aid room, workplace monitoring office and security/scale house facilities. The building will be constructed with masonry walls on top of a concrete slab on grade.

Closed circuit cameras will provide feeds to a screen monitor located inside the building. The structure will be equipped with a telephone system facilitating communications both on and off-site (for emergency purposes).

As well, security personnel will be equipped with base station radios. The offices are also planned to be equipped with desk top computers linked to the site computer network.

18.5.2 Camp

The location of the camp is planned at the south-west corner of the plant site near the security building. The camp will be constructed using single story pre-fabricated modular trailer units. Each unit will be joined together and supported on concrete cinder blocking and enclosed with plywood skirting to finished grade. The camp will be comprised of single-occupancy rooms with central washrooms. It will be used during the construction stage and throughout the operations stage. The camp is design to accommodate approximately 50 management and support staff.

18.5.3 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 will contain offices for up to 15 management and support staff employees. Washrooms, meeting and lunchrooms will also be included. The building will be constructed using single story pre-fabricated modular trailer units. Each unit will be joined together and supported on concrete cinder blocking and enclosed with plywood skirting to finished grade.

18.6 Water Management

18.6.1 Site Water Balance

A monthly mine site water balance has been developed for each phase of the mine life. The modelling was based on the estimated mean monthly hydrometeorological conditions and the proposed mine and mill production schedule.

The water balance model tracked inflows and outflows on a monthly basis from the following facilities:

- TMF;
- Process plant (mill);
- The open pits; and
- 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; and
- Fresh water make-up, which was assumed to be obtained from groundwater.

The major losses of water include evaporation from ponds and wetted surfaces, and water lost in the tailings voids.

The site is predicted to be in a water deficit condition under average climatic conditions, with the deficit increasing during dry years; therefore, make-up water will likely be required during all phases of the mine life.

The water balance indicates that the TMF will operate in a water deficit on an mean annual basis. As expected, the deficit will increase under dry conditions. The estimated average annual make-up water requirements range from 1 Mm^3 towards the end of the mine life to 3.6 Mm^3 in the earlier years of the mine life. The corresponding annual make-up water requirements under 1-in-20 year dry conditions range from approximately 2 to 4.6 Mm^3.

18.6.2 Seepage Collection Pond Design

Six seepage collection and water management ponds are proposed at the Project site to collect surface run-off and seepage from the TMF embankment drains. Water collected in the seepage collection ponds will be recycled to the TMF.

The seepage collection ponds downstream of the WRSA will provide a collection point for surface run-off and seepage. Water from these ponds will be incorporated in the water balance and used beneficially at the site.

18.6.3 Site-Wide Stormwater Pond

A site-wide stormwater pond located at the southeast of the site will 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 site indirect catchment area;
- Plant site direct catchment area; and
- Site-wide stormwater pond direct catchment area.

The water in the pond will 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 is approximately 250,000 m^3.

18.6.4 Water Well Fields and Storage Pond

A well field will be required 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, will range from 1 Mm^3 to 3.6 Mm^3. The well field will likely be located east of the Tizate Dump shown on Figure 18.1 and may require approximately seven 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).

The 250,000 m^3 site-wide stormwater pond would be used to store water during the wet season and to help buffer the demand on the well field during the dry season.

The maximum well field pumping rate has been estimated based on the greatest water requirement during the Project life, which will be approximately 3.6 Mm^3 per year. This represents the largest annual water shortfall during the Project life.

Pumping from the well field is planned on a 24 hours per day, seven days per week basis during the dry season and for 12 hours per day, seven days per week during the wet season. Less water will be required for make-up during the wet season; therefore, excess water withdrawal beyond the process requirements will be stored in the water storage pond.

Additional hydrogeological studies are required to determine the actual location of the well field(s) and the approximate number of wells and depths of completion. A suitable aquifer will need to be identified and field tested.

18.7 Waste Rock Storage Area

The majority of the waste rock from the open pits is used to construct the TMF embankments. The WRSA has been sited north-east of the Tizate open pit and is designed for a maximum capacity to 2.0 Mt.

Overburden will consist primarily of weathered oxide material. Waste rock will generally consist of hard oxide and sulphide rock types such as tonalite, altered volcanics, and volcanics.

The overall final slope of the waste storage sites will be established at 2H:1V to facilitate reclamation.

18.8 Tailings Management Facility

The Tepal mill is planned to operate at a nominal throughput of approximately 27,500 t/d over the 10-year mine life generating a total of approximately 90 Mt of tailings will be stored in the TMF.

The proposed TMF is located approximately 2 km northwest of the plant site, and was designed to store tailings, process water, surface run-off, and incident precipitation.

18.8.1 Tailings Management Facility Design

The location of the proposed TMF was selected based on an alternatives assessment that considered economic, environmental, and operational factors.

The TMF will comprise the two embankments shown on Figure 18.1:

- The main embankment to the north of the impoundment; and
- The saddle embankment at the south.

The starter embankment will be constructed with 3H:1V upstream and downstream slopes and will be underlain by a 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 will serve as a water retaining dam prior to the planned deposition of the first tailings in Year -1.

The embankments will be developed in stages throughout the life of the Project using the downstream construction method for Stage 2 and the centreline construction method for Stage 3. The initial embankment will 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 will be constructed using oxide waste rock from the open pits.

Seepage from the TMF will be intercepted by the vertical chimney drain within the embankment. Seepage will flow through the continuous filter and transition drains into a series of foundation drains at select low points in the embankment footprint. The foundation drain outlets will daylight into the seepage collection ponds constructed at topographic low points downstream of the embankment. Water in the seepage collection ponds will be monitored and recycled to the TMF by a system of pumps and pipes.

Construction will be staged to minimize capital expenditure and defer costs where possible. The starter facility will provide adequate capacity for start-up water collection. Additional stages of construction will occur at two- to five-year intervals over the approximately 10-year mine life. The TMF will be closed and reclaimed by capping the facility with oxide waste rock, topsoil, and revegetated to support the desired end land use.

18.8.2 Tailings Discharge and Reclaim

The tailings slurry will be pumped through a HDPE pipeline at approximately 30% solids (by weight) and discharged around the perimeter of the TMF.

The reclaim system is designed to deliver the process water requirements for the mill. The water will be pumped from the TMF supernatant pond to a process water tank at the mill for reuse in the process. The reclaim pumps will be mounted on a floating barge and a booster pump station will be located between the barge and the mill head tank. The average elevation of the tailings supernatant pond will increase steadily over the life of the Project resulting in lower pumping head requirements in the later years.

19 Market Studies and Contracts

19.1 Market Studies

A preliminary market study on the potential concentrate sales from the Tepal Project was completed by Cliveden Trading, 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 Project. 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 QP Gord Doerksen, P.Eng.

Concentrate transportation will be conducted using trucks from the mine site to Lazaro Cárdenas. Shipment and port handling costs were estimated based on Cliveden's recent work with other clients. The PEA 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 terms used in the economic analysis.

Table 19.1: NSR Assumptions Used in the Economic Analysis

Assumptions	Unit	Value
Copper Concentrate NSR Parameters		
Payable Copper	%	96.5
Copper Minimum Deduction	%	1
Payable Gold	%	97.5
Gold Minimum Deduction	g/t	0
Payable Silver	%	90.0
Silver Minimum Deduction	g/t	0
Copper Treatment Charge	US$/dmt concentrate	97.35
Copper Refining Charge	US$/payable lb	0.097
Gold Refining Charge	US$/payable oz	5.00
Silver Refining Charge	US$/payable oz	0.50
Concentrate Transportation	US$/dmt	90.04
Doré NSR Parameters		
Payable Gold	**%**	99.9
Gold Refining Charge	US$/payable oz	7.50
Payable Silver	%	97.0
Silver Refining Charge	US$/payable oz	1.40

Source: JDS (2017)

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 Royalties

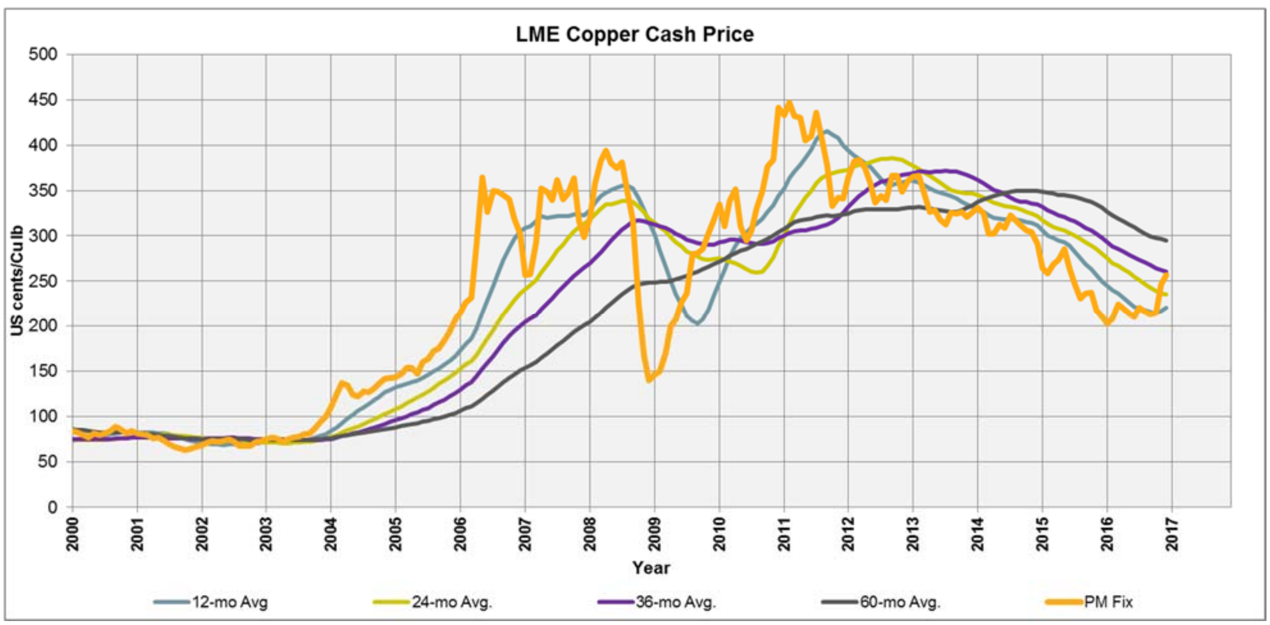
The Tepal Property is subject to a royalty payment of 2.5% NSR based on the sale of minerals and is payable to Minera Tepal. Geologix has a first-right-of-refusal on the Minera Tepal NSR royalty should Minera Tepal elect to sell the royalty.

This financial commitment is included in the cash flow. Total third party royalties for the Project amount to US$41M over the LOM.

19.4 Metal Prices

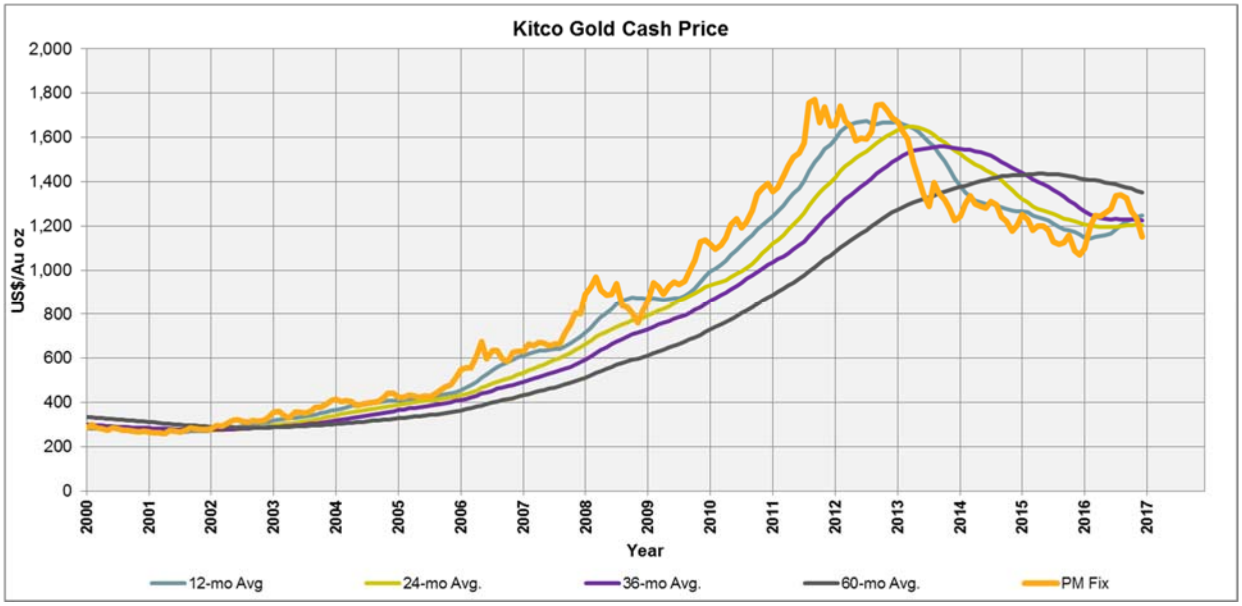
The precious metal markets are highly liquid and benefit from terminal markets around the world (London, New York, Tokyo, and Hong Kong). Historical copper, gold and silver prices are shown in Figure 19.1, Figure 19.2 and Figure 19.3.

Figure 19.1: Historical Copper Price (updated to December 2016)



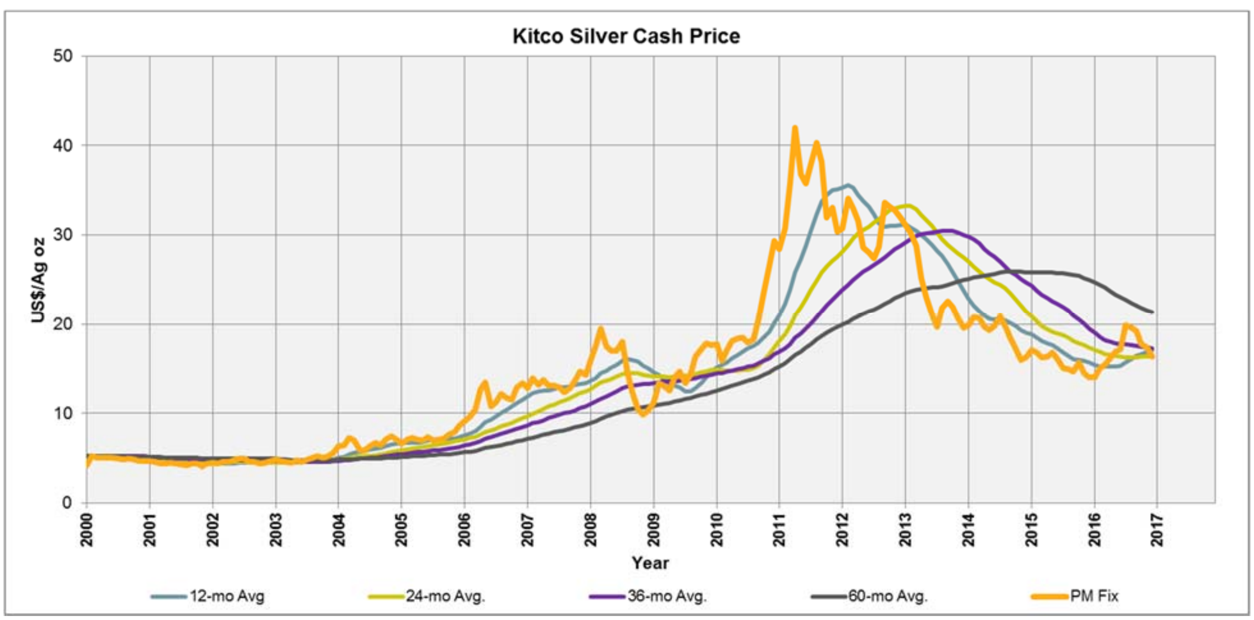
Source: London Metal Exchange (2016)

Figure 19.2: Historical Gold Price (updated to December 2016)



Source: Kitco (2016)

Figure 19.3: Historical Silver Price (updated to December 2016)



Source: Kitco (2016)

The gold price selected was based on the 12-month average as at December 2016 and copper and silver prices used in the economic analysis are based on recently released comparable technical reports. A sensitivity analysis on metal prices was completed as part of the overall economic analysis. The results of this are discussed in Section 23. Table 19.2 outlines the metal prices used in the economic analysis.

It must be noted that metal prices are highly variable and are driven by complex market forces and are extremely difficult to predict.

Table 19.2: Metal Price and Exchange Rate

Assumptions	Unit	Value
Cu Price	US$/lb	2.50
Au Price	US$/oz	1,250
Ag Price	US$/oz	18.00
FX Rate	MX$:US$	18

Source: JDS (2017)

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, Mexico. 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 800 mm of rain falls with the majority occurring in August and September. The dry season occurs from November to April when there is 0 to 25 mm of precipitation and only 1 to 29 days of rain. Annual evaporation ranges from 600 to 700 mm. Winds at the Project site are predominantly from the northeast.

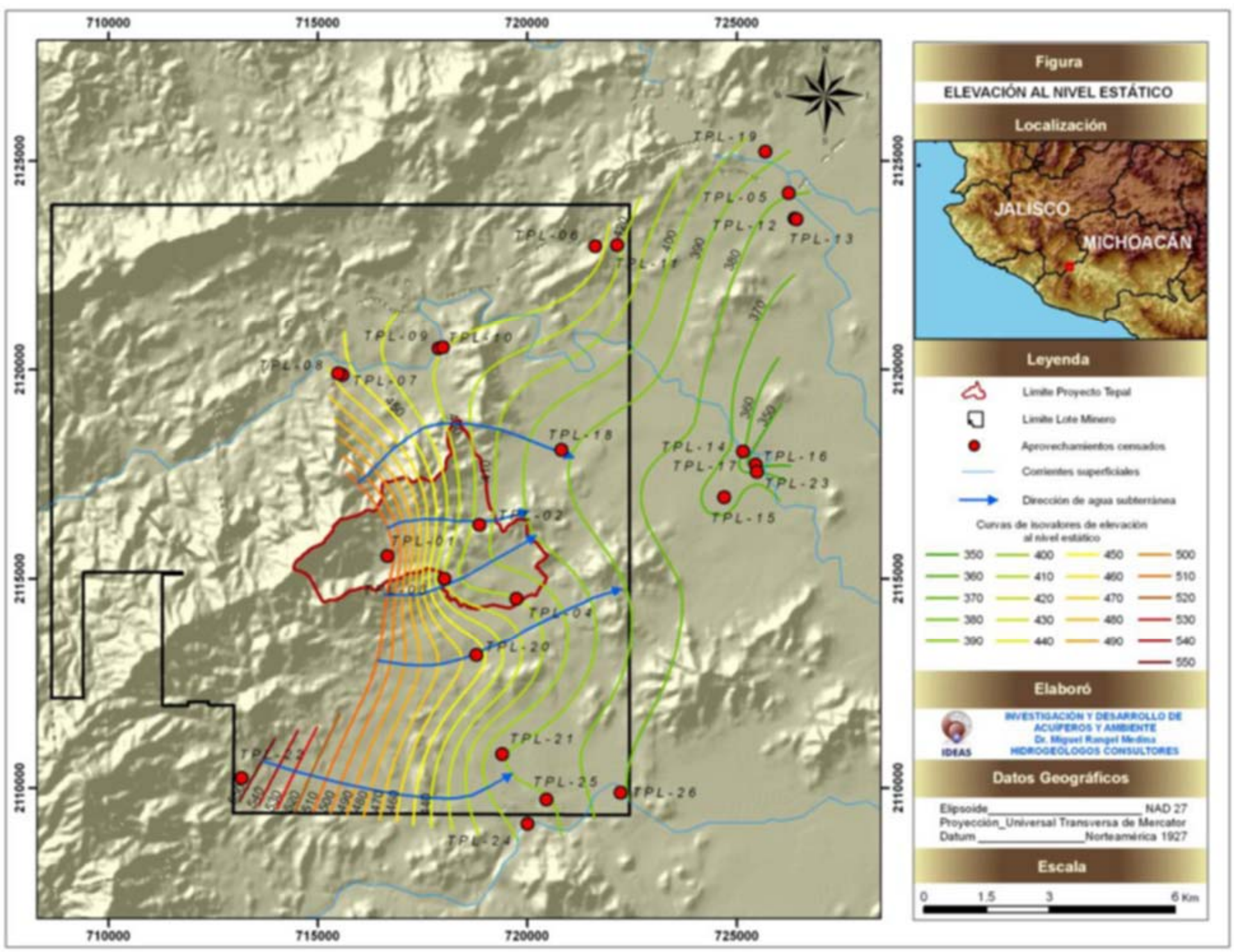
20.1.1 Aquatic Resources

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 11,860 km^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 35,046 km^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 to60 m deep on high grounds and 6 to15 m 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



Source: 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 10 m in height with densities of 2,104 to 3,308 individuals per hectare. The shrub layer ranges in height from 3 to 6 m 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).

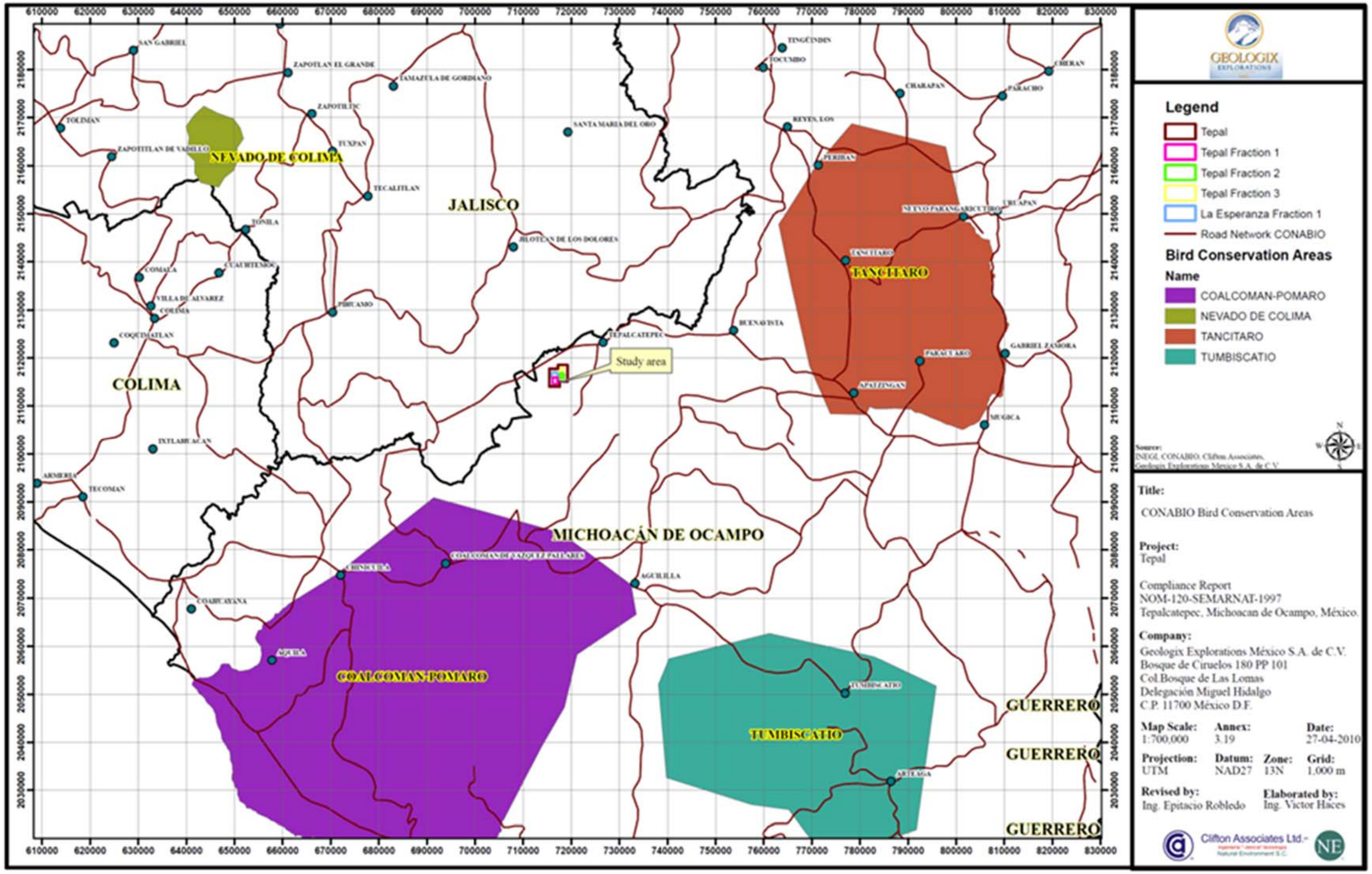
In the ranch lands to the northeast and southeast of the concessions, vegetation is dominated by spiny and xerophilic woody forest species.

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



Source: Clifton Associates Ltd (2012)

The Project is not in a protected area. The closest conservation areas are Important Bird Areas (IBAs), the Coalcomán-Pómaro (MX025) is located approximately 32 km 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, BC. 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 Test Work

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 will 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 was utilized to strategically place waste rock in locations which would facilitate a closure plan. Cut-offs used will require verification with ongoing test work, but provide a preliminary basis for this assessment.

The test work 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 will be further examined in the ongoing test program via leach extraction tests and planned kinetic test work to follow. It is expected that greater metal leaching potential will 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 Test Work

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. Test work 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 generated conclusions on 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 will have potential for acid generation and metal leaching.

Additional test work 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 test work should include synthetic precipitation leaching, meteoric water mobility leaching, and humidity cell tests with samples chosen based on current results. Tailings test work 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 will be segregated and strategically disposed of in waste rock dumps. PAG waste rock dumps will 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 will 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 water will be kept separate from water that comes in contact with tailings, pit walls, waste rock and/or mineralized material 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 will be collected and used in the process plant. Additional make-up water may be needed and will be obtained from groundwater wells. Any surplus water will 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 will be capped and revegetated to minimize infiltration and prevent acid generation and leaching over the long-term. Seepage will be collected, analyzed 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 will also be part of the Project 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 will 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 Mexico 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 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 2013, Geologix submitted an Environmental Impact Assessment (MIA-P) and it was approved based on the 2013 PFS study. 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. This was completed in 2013.

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 will need to include an on-site medical clinic, paramedics, doctors, ambulance and medical emergency evacuation plan.

Labour collective agreements are planned to 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 will include removal of the process plants, powerline and ancillary facilities. Pits will be closed out by constructing a perimeter berm and installing cautionary signs next to steep pit walls. The waste rock and tailings areas will be capped where necessary to minimize water infiltration on PAG material and to prepare the site for revegetation. Disturbed areas are planned to be revegetated with native species. Site roads that are not be required by the surrounding communities will 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 PEA, it is assumed that reclamation costs will 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 will 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 will be designed to avoid water treatment after closure to the extent possible.

21 Capital Cost Estimate

21.1 Summary & Estimate Results

LOM Project capital costs total $301 M, consisting of the following distinct phases:

- **Pre-production Capital Costs** – includes all costs to develop the Property for a 22,000 t/d sulphide circuit and a 5,500 t/d oxide circuit. Initial capital costs total $214 M (including $22 M contingency) and are expended over a 24-month pre-production construction and commissioning period; and
- **Sustaining Capital Costs** – includes all costs related to TMF expansion and the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $87 M (including $7 M in contingency) and are expended in operating Years 1 through 10, and
- **Closure Capital Costs** - includes all costs related to the closure, reclamation, and ongoing monitoring of the mine post operations. Closure costs total $23M (net of equipment salvage values), and are primarily incurred in Year 10, with costs extending into Year 15 for ongoing monitoring activities.

The capital cost estimate was compiled using a combination of quotations, database costs, and database factors.

Table 21.1 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q4 2016 dollars with no escalation. The estimate assumes that the mining equipment will be provided by the mining contractor.

Table 21.1: Capital Cost Summary

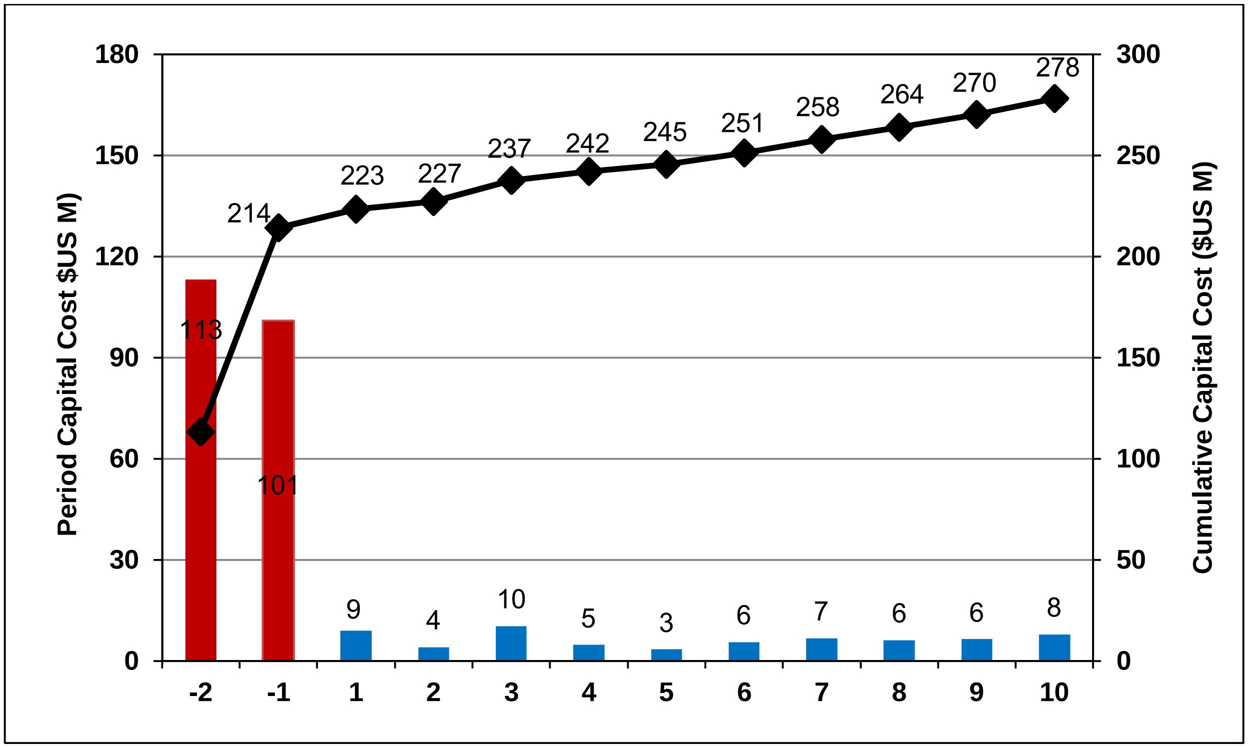
WBS	Area	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
1000	Mining	12.4	3.0	15.4
2000	Site Development/Earthworks	5.5	0.0	5.5
3000	Sulphide Processing Plant	77.7	5.7	83.4
4000	Oxide Processing Plant	29.9	0.0	29.9
5000	Tailings & Waste Rock Management	8.6	48.5	57.1
6000	Surface Infrastructure	25.2	0.0	25.2
7000	Project Indirects	10.5	0.0	10.5
8000	EPCM	15.3	0.0	15.3
9000	Owner Costs	6.9	0.0	6.9
C100	Closure (Net of Salvage)	0.0	22.9	22.9
	Subtotal Pre-Contingency	**191.9**	**80.1**	**272.0**
9900	Contingency	22.3	6.6	28.9
	Total Capital Costs	**214.2**	**86.7**	**300.9**

Source: JDS (2017)

21.2 Capital Cost Profile

All capital costs for the Project have been distributed against the development schedule in order to support the economic cash flow model. Figure 21.1presents an annual LOM capital cost profile (excluding closure years).

Figure 21.1: Capital Cost Profile (Excluding Closure Years)



Source: JDS (2017)

21.3 Key Estimate Assumptions

The following key assumptions were made during development of the capital estimate:

- Open pit mine development activities will be performed by a mining contractor; and
- All surface construction (civil, structural, architectural, mechanical, piping, electrical, and instrumentation) will be performed by contractors under the management of an EPCM contractor.

21.4 Key Estimate Parameters

The following key parameters apply to the capital estimates:

- Estimate Class: The capital cost estimates are considered Class 4 estimates (-20%/+30%). The overall project definition is estimated to be 10%;
- Estimate Base Date: The base date of the estimate is December 13th, 2016. No escalation has been applied to the capital cost estimate for costs occurring in the future;
- Units of Measure: The International System of Units (SI) is used throughout the capital estimate; and
- Currency: All capital costs are expressed in United States Dollars (US$). Portions of the estimate were estimated in other currencies and converted to US$ using the exchange rates shown in Table 21.2.

Table 21.2: Estimate Exchange Rates

Currency	Symbol	X : US$
United States Dollar	US$	1.00
Canadian Dollar	CA$	1.34
Mexican Peso	MXN	18.00

Source: JDS (2017)

21.5 Basis of Estimate

21.5.1 Labour Rates

The majority of installation costs within the estimate have been factored based on mechanical equipment costs. Where applicable within the estimate, an average all-in contractor crew labour-rate of $20/hr has been applied, based on buildups from other recent and similar studies.

Operational labour rates were built up from first principles. Base rates are based prevailing wages in the area, and legal premiums and benefits were built up to create all-in rates. Operational labour rates and staffing levels are described further within Section 22.

21.5.2 Fuel & Energy Supply

Where applicable, a delivered fuel price of $0.885/L and a grid power energy supply price of $0.089/kWh has been used throughout the estimate.

21.5.3 Mine Capital Costs

Mine capital cost estimates have been assembled using a mining contractor unit rate of $2.13/t from first principals, based on the mine production schedule.

21.5.3.1 Pre-Stripping

Pre-stripping costs for the removal of barren waste material at the open pit prior to mineralized material processing are established using a mining contractor unit rate of $3.46/t (in the capital phase, dropping to $2.13/t average LOM) and the mine schedule.

21.5.3.2 Mine Mobile Equipment

Open pit mining equipment is provided by the mining contractor and the cost of the equipment is included in their unit mining rate.

21.5.3.3 Fixed Mine Equipment

The fixed mine equipment sector includes costs for dewatering piping and accessories, survey equipment, computers, and engineering software.

21.5.4 Site Development & Road Works

Site development costs are generally based on high level material take-offs and database unit pricing.

21.5.4.1 Site Development

Material take-offs were developed from preliminary 3D models for earthen pads. Database unit costs were applied for excavations, fills, and surfacing materials. Allowances were made for settling ponds, surface water control, and temporary roads.

21.5.4.2 Site Roads

Costs for site roads (including both the main access road and on-site access roads) costs are based on road lengths from the general arrangement drawings and database $/km unit rates for gravel roads in similar ground conditions.

21.5.5 Process Plant

The process plant capital costs include all of the direct costs to construct the 22,000 t/d sulphide plant and 5,500 t/d oxide plant.

The process plant capital cost estimate was assembled form a combination of supplier quotations and database allowances. Table 21.3 presents a summary basis of estimate for the various commodity types within the process plant estimate.

Table 21.3: Process Plant Basis of Estimate

Commodity	Estimate Basis
Equipment	
Major Equipment	Budget quotations were solicited from qualified suppliers for the major equipment identified in the flow sheets and equipment register.
Minor Equipment	In-house data (firm and budgetary quotations from recent projects) was used for minor or low value equipment.
Installation (Labour & Materials)	
Concrete	High level take-off quantities were developed from general arrangement drawings and database concrete quantity ratios per facility area (m^3/m^2). Database unit rates were applied to the take-off quantities.
Internal Structural Steel	Factored based on mechanical equipment costs.
Process Plant Building	Database unit costs ($/m^2) applied to areas determined from the general arrangement drawings. Fabric walled buildings assumed for the process area buildings (ADR area). Lump sum allowances included for modular control and lunch rooms.
Mechanical Equipment Installation	Factored based on mechanical equipment costs.
Piping	Factored based on mechanical equipment costs.
Electrical & Instrumentation	Factored based on mechanical equipment costs.

Source: JDS (2017)

21.5.6 On-Site Infrastructure

The on-site infrastructure estimate at the Tepal Project includes an assay laboratory, permanent accommodation complex, administration complex, on-site power distribution, water, and waste handling infrastructure, the surface mobile support fleet and information technology (IT) and communications systems. Table 21.4 presents a summary basis of estimate for the various commodity types within the process plant estimate.

Table 21.4: Infrastructure Basis of Estimate

Component	Estimate Basis
Accommodations, administration Complex, and assay laboratory	Factored database costs, based on the accommodations, dry, and office requirements determined for the operations and construction phases.
Maintenance Facilities, Bulk Fuel Storage & Distribution	To be provided by the mining contractor.
Power Transmission Line	A factored database costs for the overhead power line (including right-of-way clearing and grubbing) and mobile substations have been included in the estimate.
Site Utilities	Site utilities include on-site power distribution, emergency power generation, a chlorinator water treatment plant, incinerator, and septic field. Lump sum allowances have been applied to these facilities based on experience at similar operations.
Surface Mobile Equipment	Surface equipment fleet requirements are determined based on material movement requirements and experience at similar operations, and considering site conditions specific to the Project. No equipment replacements are anticipated for the surface equipment fleet due to the short mine life and relatively low utilization of equipment.
	Database unit pricing has been applied to the surface equipment fleet quantities.
IT & Communications	Lump sum allowances based on experience at similar operations.

Source: JDS (2017)

21.6 Indirect Cost Estimate

Indirect costs are those that are not directly accountable to a specific cost object. Table 21.5 presents the basis of estimate for each of the indirect cost categories. The majority of indirect costs in the estimate are factors or allowances based on recently completed definitive estimates for similar projects.

Table 21.5: Indirect Cost Basis of Estimate

Commodity	Basis
Construction Support Services	Time based cost allowance for general construction site services (temporary power, heating & hoarding, contractor support, etc.) applied against the surface construction schedule
Temporary Facilities & Utilities	Allowance for construction offices and ablution facilities
	Allowance for diesel construction power
Contractor Mobilization	Factored allowance (2.0% of direct costs) for contractor mobilization and miscellaneous expenses; Note that contractor profit on labour and materials are included in the direct cost unit rates
Logistics & Freight	Factored allowance (7.0% of direct equipment costs) for all freight and logistics
Start-up and Commissioning	Factored allowance (2.0%) for spare parts
	Factored allowance (1.0%) for the provision of vendor services for commissioning support
Engineering & EPCM	Factored allowance (10%) of total direct construction costs (excluding mining) for detailed engineering and procurement

Source: JDS (2017)

21.7 Owner's Cost Estimate

Owner's costs are items that are included within the operating costs during production. These items are included in the initial capital costs during the construction phase and capitalized. The cost elements described below are described in more detail within Section 22.

- Pre-production processing: Costs of the Owner's processing labour, power, and consumables incurred before declaration of commercial production; and
- Pre-production general & administration: Costs of the Owner's labour and expenses (camp and catering, safety, finance, security, purchasing, support labour, maintenance, equipment usage, management, etc.) incurred prior to commercial production.

21.8 Closure Cost & Salvage Value

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for an open pit mine. Typical activities include:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the TMF;
- Access road closure;
- Power transmission line and substation removal;
- Revegetation and seeding; and
- Ongoing site monitoring.

A total lump sum closure cost of $23 M has been used for the estimate, based on factored costs from similar projects.

21.9 Cost Contingency

Contingency was evaluated by major work breakdown structure (WBS) area, based on the level of design and pricing confidence. The result was an overall blended contingency of 12% or $29 M LOM.

21.10 Capital Estimate Exclusions

The following items have been excluded from the capital cost estimate:

- Working capital (included in the financial model);
- Financing costs;
- Currency fluctuations;
- Lost time due to severe weather conditions beyond those expected in the region;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in project schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;
- Any project sunk costs (studies, exploration programs, etc.);
- Local taxes (e.g. VAT);
- Closure bonding; and
- Escalation cost.

22 Operating Cost Estimate

The OPEX estimate is based on a combination of experience, reference project, budgetary quotes and factors as appropriate with a preliminary study.

The operating cost estimate in this study includes the costs to mine, process the mineralized material to produce copper concentrate and doré and general and administrative expenses (G&A). These items total the Project operating costs and are summarized in Table 22.1. The estimate is based on contractor mining. The target accuracy of the operating cost is -25/+30%.

The operating cost estimate is broken into four major sections:

- Open pit mining;
- Processing – sulphide flotation/cyanidation;
- Processing – oxide CIL; and
- G&A.

The total operating unit cost is estimated to be US$9.65/t processed. Average annual, total LOM and unit operating cost estimates are summarized in Table 22.1.

Operating costs are expressed in US dollars. No allowance for inflation has been applied.

Table 22.1: Breakdown of Estimated Operating Costs

Operating Costs	Avg Annual (M$)	$/t processed	LOM (M$)
Mining*	31	3.30	299
Processing – Sulphide Flotation/Cyanidation	44	4.75	430
Processing – Oxide CIL	8	0.85	77
G&A	7	0.75	67
Total	**90**	**9.65**	**873**

*Average LOM Mining cost amounts to $2.16/t mined at a 0.6:1 strip ratio (excluding pre-production tonnes mined).
Totals may not add due to rounding
Source: JDS (2017)

The main operating cost component assumptions are shown in Table 22.2.

Table 22.2: Main OPEX Component Assumptions

Item	Unit	Value
Electrical Power Cost	$/kWh	0.089
Overall Power Consumption (all facilities)	kWh/t processed	17.2
Diesel Cost (delivered)	$/litre	0.885
LOM Average Manpower	employees	325

Source: JDS (2017)

22.1 Operations Labour

This section provides an overview of total workforce and the methods used to compile the labour rates.

Table 22.3 summarizes the total planned workforce during project operations.

Table 22.3: Summary of Personnel

Department	Total Persons Employed (Peak)
Mining - Contractor	141
Mining – Geologix Staff	19
Processing	101
G&A	74
Total Personnel - All Areas	**335**

Source: JDS (2017)

Labour is a significant portion of annual operating cost. Labour rates include base wage and allowances for overtime, insurance, tax, and benefits.

Labour burdens were assembled using first principles and range from 31 to 37%. The following items are included in the burdened labour rates:

- Monthly social security (IMSS) at 25%;
- Monthly payroll tax (ISN) at 2%;
- Yearly Christmas bonus at 15 days per year;
- Yearly vacation pay at 25%; and
- Monthly savings funds at 13%.

22.2 Basis of Estimate

22.2.1 Mine Operating Cost Estimate

The mine operating costs include all open pit mining activities such as pit and dump operations, road maintenance, mine supervision, technical services and mine support equipment. The average LOM mine operating costs (excludes pre-production) are estimated to be $3.30/t processed or $2.16/t mined. The majority of mining activities are to be performed by a mining contractor. Geologix will provide technical services and support equipment all other activities will be provided by the mining contractor.

22.2.2 Processing Operating Cost Estimate

Process operating costs were developed using labour rates based on operating mines in the area and sufficient personnel to operate the process plant, factored maintenance cost, budget quotes for consumables and a factored power requirement. Process operating costs are summarized below in Table 22.4. Costs are subdivided into operating categories.

Table 22.4: Processing Operating Cost by Category

Category	Sulphide $/t processed	Oxide $/t processed
Labour	0.11	0.26
Equipment Maintenance & Consumables (Reagents, Media, Liners and other Wear Parts)	3.70	5.13
Power & Fuel	1.68	0.95
Grand Total by Activity	**5.49**	**6.34**

Source: JDS (2017)

Process labour includes burden for salaried and hourly employees to account for in-country benefits, training, production bonus and potential ex-patriot benefits and costs.

Equipment maintenance was calculated by applying a factor of 4% to major process equipment cost. Costs for media were determined using engineering calculations based on mill power draw, abrasion index and vendor quotes for media as a cost per tonne. Reagent requirements from recent test work and budget quotes from vendors were used to calculate the cost of reagents. Mill liners and wear parts for major equipment were based on vendor recommended requirements and quotes.

Power costs were calculated from the total installed power assuming $0.089/kWh.

22.2.3 General and Administrative Operating Cost Estimate

The general and administration costs include all off-site and on-site activities including personnel transportation, camp catering and cleaning, surface support equipment, water management, environmental monitoring, facilities maintenance, insurance, and all associated labour. The G&A operating cost is estimated to be $0.75/t processed and can be attributed to two categories:

- Labour; and
- On-site items.

Table 22.5: Summary of G&A Costs

Cost Category	$/t processed	LOM (M$)
Labour	0.18	16
On-site Items	0.57	51
Total G&A Costs	**0.75**	**67**

Source: JDS (2017)

23 Economic Analysis

23.1 Summary of 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.

Univariate sensitivity analyses were performed for variations in metal prices, head 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 schedules and forecasts 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 and Section 22 of this report (presented in 2016 dollars). The economic analysis has been run with no inflation (constant dollar basis).

This PEA is preliminary in nature and includes the use of Inferred Mineral Resources (2 Mt of planned mill feed material) that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized.

23.2 Assumptions

The summary of the mine plan and payable metals produced is outlined in Table 23.1.

Table 23.1: LOM Plan Summary

Parameter	Unit	Value
Mine Life	Years	10
Resource Mined	kt	90,479
Sulphide Resource		
Throughput Rate	kt/d	22
Average Cu Head Grade	%	0.21
Average Au Head Grade	g/t	0.33
Average Ag Head Grade	g/t	1.47
Oxide Resource		
Oxide Resource	kt/d	5
Average Au Head Grade	g/t	0.45
Average Ag Head Grade	g/t	1.11
Cu Payable	Mlbs	308
	Mlbs/a	32
Au Payable	koz	766
	koz/a	79
Ag Payable	koz	2,458
	koz/a	252

Source: JDS (2017)

Other factors include the following:

- 2 Mt of mill feed material is in the Inferred category;
- Discount rate of 5% (sensitivities using other discount rates have been calculated);
- Closure cost of $23 M;
- Nominal 2016 dollars;
- 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 operating costs (mining, processing, and G&A) in Year 1;
- Tax pools totalling $22 M were used in the tax model for after-tax results;
- Results are presented on 100% ownership; and
- No management fees or financing costs (equity fund-raising was assumed).

The model excludes all pre-development and sunk costs up to the start of detailed engineering (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, financing costs, etc.).

Table 23.2 outlines the metal prices and MXN:US$ exchange rate assumptions used in the economic analysis. The gold price selected was based on the 12-month average as at December 2016 and copper and silver prices used in the economic analysis are based on recently released comparable technical reports.

The reader is cautioned that the metal prices and exchange rates used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized if the Project is taken into production. The metal prices are based on many complex factors and there are no reliable long-term predictive tools.

Table 23.2: Metal Price & Foreign Exchange Rates used in Economic Analysis

Parameter	Unit	Value
Copper Price	US$/lb	2.50
Gold Price	US$/oz	1,250
Silver Price	US$/oz	18.00
Exchange Rate	MXN:US$	18

Source: JDS (2017)

23.3 Revenues & NSR Parameters

Mine revenue is derived from the sale of copper concentrate and doré bars into the international marketplace. No contractual arrangements for refining exist at this time. Details regarding the terms used for the economic analysis can be found in the market studies (Section 19) of this report. Table 23.3 indicates the NSR parameters that were used in the economic analysis.

Table 23.3: NSR Parameters Used in Economic Analysis

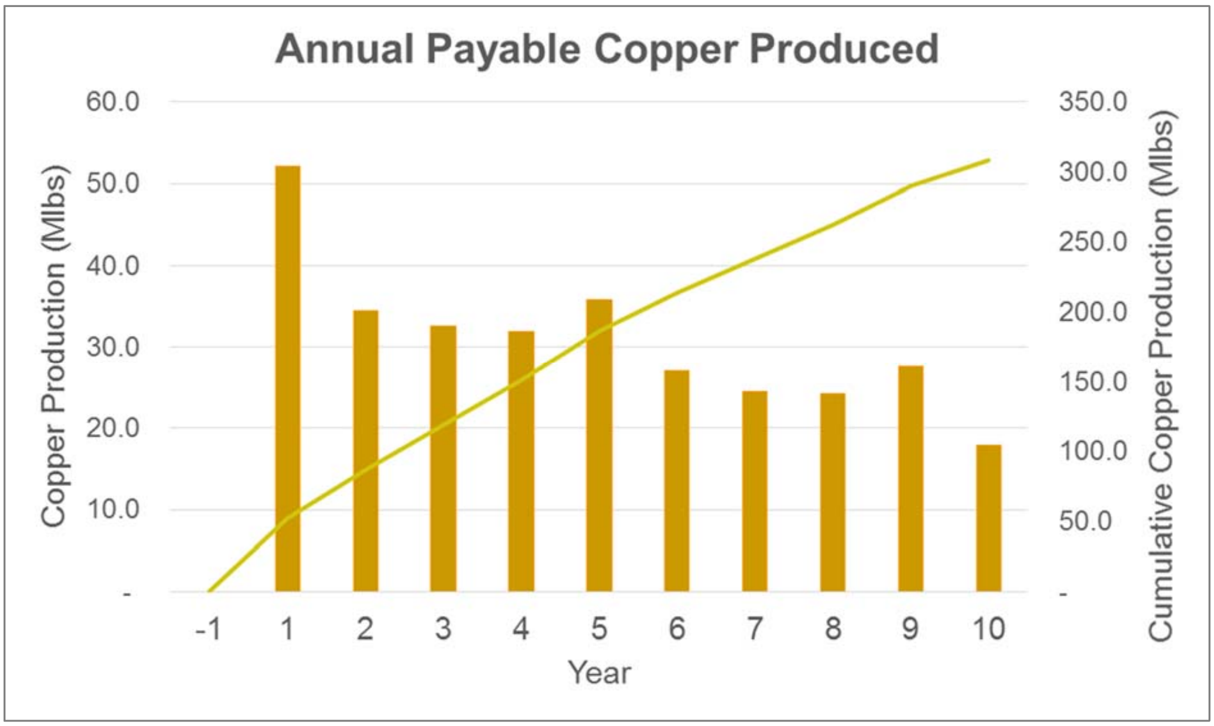
Assumptions & Inputs	Unit	Value
Mine Operating Days	days/a	365
Royalty	% NSR	2.5
Sulphide Recoveries		
Flotation		
Tepal		
Cu	%	88.2
Au	%	62.4
Ag	%	27.4
Tizate		
Cu	%	85.9
Au	%	58.0
Ag	%	59.6
Total		
Cu	%	87.4
Au	%	61.2
Ag	%	48.0
Cyanidation		
Tepal		
Au	%	16.5
Ag	%	15.5
Tizate		
Au	%	16.0
Ag	%	18.5
Total		
Au	%	16.0
Ag	%	17.1
Oxide Recoveries		
Tepal		
Au	%	83.2
Ag	%	63.3
Tizate		
Au	%	75.2
Ag	%	55.9
Total		
Au	%	80.9
Ag	%	60.3
Losses to Solution	%	2
NSR Parameters		
Copper Concentrate		
Cu Concentrate Payable	%	96.5
Cu Minimum Deduction	%	1
Au Payable	%	97.5
Au Minimum Deduction	g/t	0
Ag Payable	%	90.0
Ag Minimum Deduction	g/t	0.0
Cu Treatment Charge	US$/dmt concentrate	0.40
Cu Refining Charge	US$/pay lb	0.097
Au Refining Charge	US$/pay oz	5.00
Ag Refining Charge	US$/pay oz	0.50

Assumptions & Inputs	Unit	Value
Concentrate Transportation	US$/dmt	90.04
Doré NSR Parameters		
Au Payable	%	99.9
Ag Payable	%	97.0
Au Refining Charge	US$/pay oz	7.50
Ag Refining Charge	US$/pay oz	1.40

Source: JDS (2017)

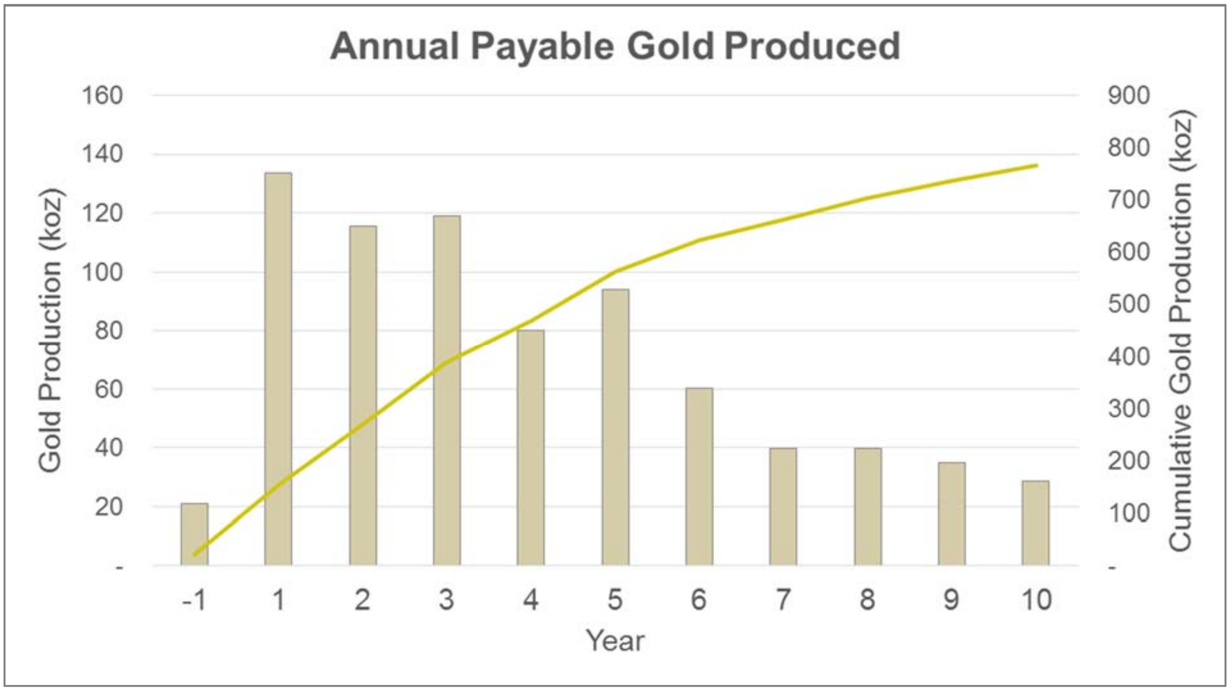
Figure 23.1, Figure 23.2, and Figure 23.3 show breakdowns of the amount of copper, gold and silver recovered during the mine life and the amount of payable metal for the Project. A total of 308 Mlbs of copper, 766 koz of gold, and 2,458 koz of silver are projected to be produced during the mine life. Gold accounts for about 54% of gross project revenues, copper for about 43% and silver accounting for 2%.

Figure 23.1: Payable Copper Production by Year



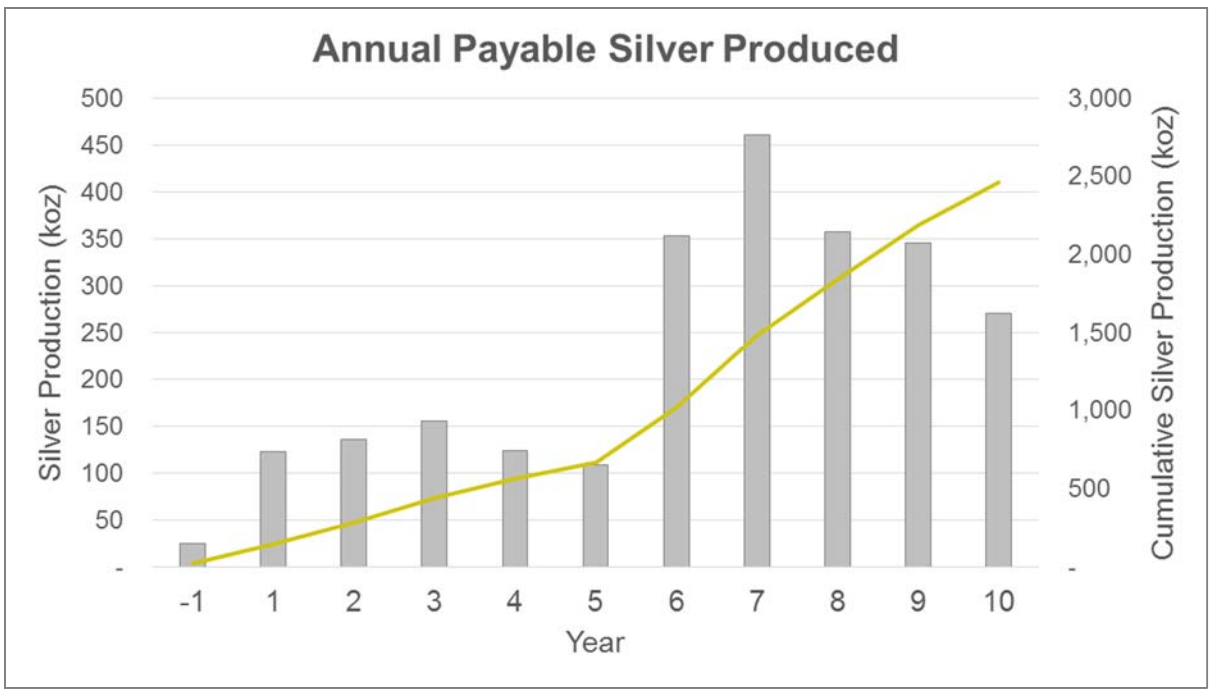
Source: JDS (2017)

Figure 23.2: Payable Au Production by Year



Source: JDS (2017)

Figure 23.3: Payable Ag Production by Year



Source: JDS (2017)

23.4 Taxes

The Project has been evaluated on an after-tax basis in order to provide a more indicative, but still approximate, value of the potential project economics. A tax model was prepared by JDS and reviewed by Victoria Viveash, CPA, CMA, as indicated in Section 3, who possesses applicable Mexican mineral tax regime experience. Current tax pools totalling $22 M were used in the analysis. The tax model contains the following assumptions:

- 30% income tax rate;
- Precious Metal Royalty (0.5% of Gross Revenues from Au and Ag); and
- A 7.5% Mining Royalty of Earnings before Interest, Taxes, Depreciation and Amortization (EBITDA).

Total taxes for the Project amount to $160 M.

23.5 Royalties

A 2.5% NSR royalty is payable to Minera Tepal as described in Section 19. Total third party royalties for the Project amount to $41 M over the LOM. More details on the structure of this royalty can be found in Section 19.

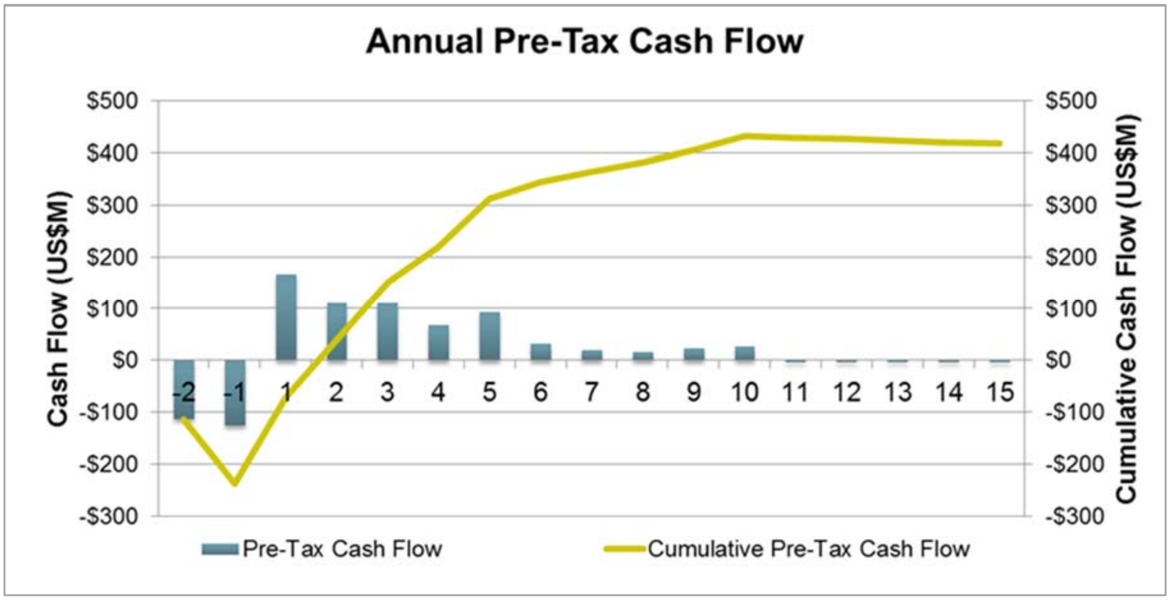
23.6 Results

At this preliminary stage, the Project has a pre-tax IRR of 36% and a net present value using a 5% discount rate ($NPV_{5\%}$) of $299 M using the metal prices described in Section 19.

Figure 23.4 shows the projected cash flows, and Table 23.4 summarizes the economic results of the Tepal Project.

The pre-tax break-even gold price for the Project is approximately US$737/oz, based on the LOM plan presented herein, a copper price of US$2.50/lb and a silver price of US$18.00/oz.

Figure 23.4: Pre-Tax Annual Cash Flows



Source: JDS (2017)

Table 23.4: Summary of Results

Summary of Results	Unit	Value
Cash Cost (Net of Byproduct)	US$/oz	313
Cash Cost (incl. Sustaining and Closure CAPEX)	US$/oz	396
Capital Costs		
Pre-Production Capital	M$	192
Pre-Production Contingency	M$	22
Total Pre-Production Capital	**M$**	**214**
Sustaining & Closure Capital	M$	80
Sustaining & Closure Contingency	M$	7
Total Sustaining & Closure Capital	M$	87
Total Capital Costs Incl. Contingency	**M$**	**301**
Working Capital	M$	23
Pre-Tax Cash Flow	LOM M$	417
	M$/a	43
Taxes	LOM M$	160
After-Tax Cash Flow	LOM M$	257
	M$/a	26
Pre-Tax $NPV_{5\%}$	**M$**	**299**
Pre-Tax IRR	**%**	**36**
Pre-Tax Payback	**Years**	**1.6**
After-Tax $NPV_{5\%}$	**M$**	**169**
After-Tax IRR	**%**	**24**
After-Tax Payback	**Years**	**2.3**

Source: JDS (2017)

23.7 Sensitivities

A univariate sensitivity analysis was performed to examine which factors most affect the Project economics when acting independently of all other cost and revenue factors. Each variable evaluated was tested using the same percentage range of variation, from -20% to +20%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the LOM. For instance, the metal prices were evaluated at a +/- 20% range to the base case, while the head grade and all other variables remained constant. This may not be truly representative of market scenarios, as metal prices may not fluctuate in a similar trend. The variables examined in this analysis are those commonly considered in similar studies – their selection for examination does not reflect any particular uncertainty.

Notwithstanding the above noted limitations to the sensitivity analysis, which are common to studies of this sort, the analysis revealed that the Project is most sensitive to metal prices, followed by head grades and operating costs. The Project showed the least sensitivity to capital costs. Table 23.5 and Figure 23.5 show the results of the sensitivity tests.

A sensitivity analysis was also performed on discount rate. The results of this test are demonstrated in Table 23.5 and Table 23.6.

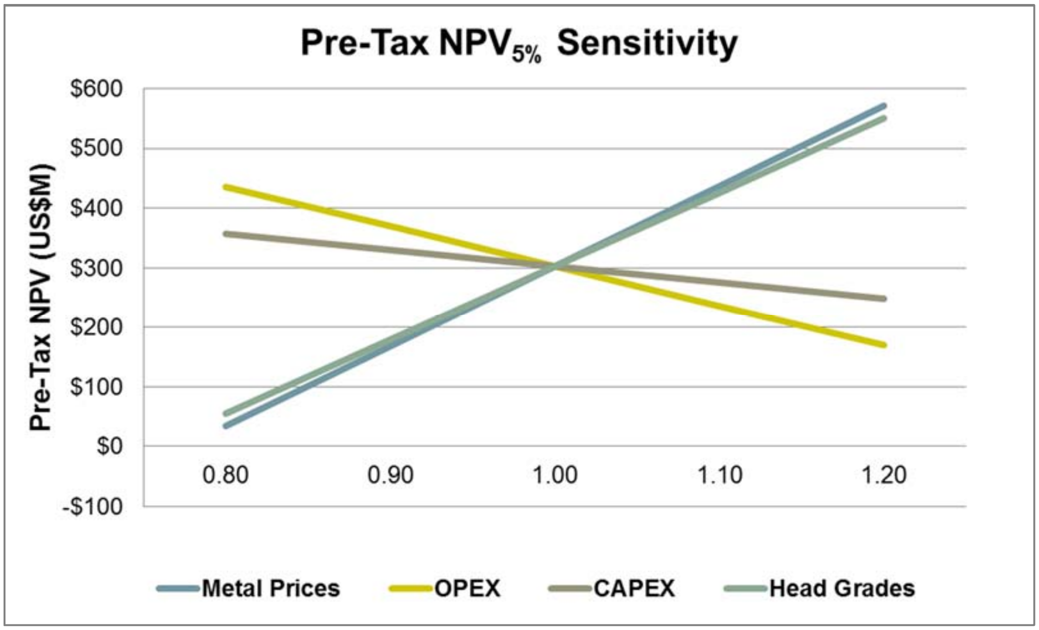
The economic cash flow model for the Tepal Project is illustrated in Figure 23.6.

Table 23.5: Pre-Tax and After-Tax Sensitivity Results on NPV@ 5%

Variable	Pre-Tax $NPV_{5\%}$ (M$)			After-Tax $NPV_{5\%}$ (M$)		
	-20% Variance	0% Variance	20% Variance	-20% Variance	0% Variance	20% Variance
Metal Prices	312	299	567	-19	169	345
Head Grade	53	299	546	-4	169	331
OPEX	433	299	166	258	169	74
CAPEX	353	299	246	223	169	116

Source: JDS (2017)

Figure 23.5: Sensitivity Results, Pre-Tax $NPV_{5\%}$



Source: JDS (2017)

Table 23.6: Base Case Scenario Discount Rate Sensitivity

Discount Rate (%)	Pre-Tax NPV (M$)	After-Tax NPV (M$)
0	417	257
5	299	169
8	245	129
10	213	105
12	185	85

Source: JDS (2017)

Parameter	Source	Unit	LOM Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
METAL PRICES																										
Cu	link	US$/lb	**2.50**	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50
Au	link	US$/oz	**1,250**	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Ag	link	US$/oz	**18.00**	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00	18.00
PRODUCTION SCHEDULE																										
Total Mined																										
Sulphide Resource	**calc**	**ktonnes**	**78,310**			**-**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**6,040**										
Cu	calc	%	**0.21%**			-	0.35%	0.23%	0.22%	0.21%	0.24%	0.18%	0.17%	0.17%	0.19%	0.16%										
Au	calc	g/t	**0.33**			-	0.55	0.45	0.48	0.31	0.40	0.26	0.21	0.21	0.19	0.20										
Ag	calc	g/t	**1.47**			-	0.84	0.92	1.14	0.92	0.82	1.89	2.50	1.92	1.87	1.96										
Oxide Resource	**calc**	**ktonnes**	**12,169**			**1,461**	**4,141**	**2,333**	**484**	**718**	**1,483**	**1,420**	**25**	**89**	**15**	**-**										
Au	calc	g/t	**0.45**			0.58	0.48	0.48	0.41	0.36	0.39	0.30	0.27	0.27	0.28	-										
Ag	calc	g/t	**1.11**			0.91	0.98	1.09	1.16	0.78	0.86	2.05	2.03	2.35	2.10	-										
Total Resource	calc	ktonnes	**90,479**	-	-	1,461	12,171	10,363	8,514	8,748	9,513	9,450	8,055	8,119	8,045	6,040	-	-	-	-	-	-	-	-	-	-
Total Waste	link	ktonnes	**52,464**			12,040	3,019	6,661	3,285	4,736	6,114	6,861	3,847	3,941	1,721	237	-	-	-	-	-	-	-	-	-	-
Total Mined	**calc**	**ktonnes**	**142,943**	**-**	**-**	**13,502**	**15,190**	**17,024**	**11,799**	**13,484**	**15,627**	**16,312**	**11,903**	**12,060**	**9,766**	**6,277**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Strip Ratio	calc	w:o	**0.6**	-	-	8.2	0.2	0.6	0.4	0.5	0.6	0.7	0.5	0.5	0.2	0.0	-	-	-	-	-	-	-	-	-	-
Mining Rate	calc	ktpd	**37.5**	-	-	37.0	41.6	46.6	32.3	36.9	42.8	44.7	32.6	33.0	26.8	17.2	-	-	-	-	-	-	-	-	-	-
CONTAINED METAL																										
Total Contained Metal in Sulphide Resource																										
Cu	calc	ktonnes	**166**	-	-	-	28	18	17	17	19	15	13	13	15	10	-	-	-	-	-	-	-	-	-	-
	calc	Mlbs	**366.4**	-	-	-	61.4	40.6	38.3	37.6	42.2	32.2	29.7	29.4	33.4	21.7	-	-	-	-	-	-	-	-	-	-
Au	calc	kg	**25,693**	-	-	-	4,383	3,647	3,821	2,462	3,177	2,086	1,702	1,688	1,501	1,225	-	-	-	-	-	-	-	-	-	-
	calc	koz	**826**	-	-	-	141	117	123	79	102	67	55	54	48	39	-	-	-	-	-	-	-	-	-	-
Ag	calc	kg	**114,785**	-	-	-	6,706	7,386	9,120	7,380	6,603	15,158	20,093	15,455	15,048	11,835	-	-	-	-	-	-	-	-	-	-
	calc	koz	**3,690**	-	-	-	216	237	293	237	212	487	646	497	484	380	-	-	-	-	-	-	-	-	-	-
Total Contained Metal in Oxide Resource																										
Au	calc	koz	**174**	-	-	26	31	31	30	24	19	13	0	1	0	-	-	-	-	-	-	-	-	-	-	-
Ag	calc	koz	**435**	-	-	41	63	70	67	50	41	94	2	7	1	-	-	-	-	-	-	-	-	-	-	-
MILLING SCHEDULE																										
Total Processed																										
Sulphide Resource	**calc**	**ktonnes**	**78,310**			**-**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**8,030**	**6,040**										
Cu	calc	%	**0.21%**			-	0.35%	0.23%	0.22%	0.21%	0.24%	0.18%	0.17%	0.17%	0.19%	0.16%										
Au	calc	g/t	**0.33**			-	0.55	0.45	0.48	0.31	0.40	0.26	0.21	0.21	0.19	0.20										
Ag	calc	g/t	**1.47**			-	0.84	0.92	1.14	0.92	0.82	1.89	2.50	1.92	1.87	1.96										
Mill Throughput Rate	calc	ktpd	**21.6**	-	-	-	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	16.5	-	-	-	-	-	-	-	-	-	-
Oxide Resource	**calc**	**ktonnes**	**12,169**			**1,400**	**2,008**	**2,008**	**2,008**	**1,714**	**1,483**	**1,420**	**25**	**89**	**15**	**-**										
Au	calc	g/t	**0.45**			0.58	0.48	0.48	0.46	0.43	0.39	0.30	0.27	0.27	0.28	-										
Ag	calc	g/t	**1.11**			0.91	0.98	1.09	1.03	0.90	0.86	2.05	2.03	2.35	2.10	-										
Mill Throughput Rate	calc	ktpd	**4.8**	-	-	3.8	5.5	5.5	5.5	4.7	4.1	3.9	0.1	0.2	0.0	-	-	-	-	-	-	-	-	-	-	-
Ore Re-Handled	link	ktonnes	**2,520**			-	-	-	1,524	996	-	-	-	-	-	-										
SULPHIDE PROCESSING																										
Tepal North & South																										
Cu	link	%	**88.2%**			-	88.2%	88.2%	88.2%	88.2%	88.2%	88.2%	-	-	-	-										
Au	link	%	**62.4%**			-	62.4%	62.4%	62.4%	62.4%	62.4%	62.4%	-	-	-	-										
Ag	link	%	**27.4%**			-	27.4%	27.4%	27.4%	27.4%	27.4%	27.4%	-	-	-	-										
Tizate																										
Cu	link	%	**85.9%**			-	-	-	-	-	-	85.9%	85.9%	85.9%	85.9%	85.9%										
Au	link	%	**58.0%**			-	-	-	-	-	-	58.0%	58.0%	58.0%	58.0%	58.0%										
Ag	link	%	**59.6%**			-	-	-	-	-	-	59.6%	59.6%	59.6%	59.6%	59.6%										
Total Flotation Recovery																										
Cu	link	%	**87.4%**			-	88.2%	88.2%	88.2%	88.2%	88.2%	87.2%	85.9%	85.9%	85.9%	85.9%										
Au	link	%	**61.2%**			-	62.4%	62.4%	62.4%	62.4%	62.4%	60.8%	58.0%	58.0%	58.0%	58.0%										
Ag	link	%	**48.0%**			-	27.4%	27.4%	27.4%	27.4%	27.4%	50.4%	59.6%	59.6%	59.6%	59.6%										
Cu Recovered	calc	Mlbs	**320.2**	-	-	-	54.1	35.8	33.8	33.1	37.2	28.1	25.5	25.2	28.7	18.6	-	-	-	-	-	-	-	-	-	-
Au Recovered	calc	koz	**506**	-	-	-	88	73	77	49	64	41	32	31	28	23	-	-	-	-	-	-	-	-	-	-
Ag Recovered	calc	koz	**1,770**	-	-	-	59	65	80	65	58	246	385	296	288	227	-	-	-	-	-	-	-	-	-	-
Concentrate																										
Copper Concentrate Grade	calc	%	**26.2%**	-	-	-	26.2%	26.2%	26.2%	26.2%	26.2%	26.2%	26.2%	26.2%	26.2%	26.2%	-	-	-	-	-	-	-	-	-	-
Au in Cu Concentrate	calc	g/t	**28.4**	-	-	-	29.2	36.7	40.8	26.8	30.8	26.1	22.4	22.4	17.5	22.1	-	-	-	-	-	-	-	-	-	-
Ag in Cu Concentrate	calc	g/t	**99.3**	-	-	-	19.6	32.6	42.7	35.3	28.1	157.2	271.3	210.8	180.6	218.8	-	-	-	-	-	-	-	-	-	-
Pull Factor	calc		**141**	-	-	-	86	129	137	140	125	165	182	184	162	187	-	-	-	-	-	-	-	-	-	-
Moisture Content	input	%	**8%**	-	-	-	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	-	-	-	-	-	-	-	-	-	-
Cu Concentrate Produced	**calc**	**dmt**	**554,374**	**-**	**-**	**-**	**93,735**	**62,052**	**58,505**	**57,362**	**64,369**	**48,624**	**44,135**	**43,689**	**49,671**	**32,232**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	**wmt**	**602,581**	**-**	**-**	**-**	**101,886**	**67,448**	**63,592**	**62,350**	**69,966**	**52,852**	**47,973**	**47,488**	**53,990**	**35,035**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Cu Payable	input	%	**96.5%**	-	-	-	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%	-	-	-	-	-	-	-	-	-	-
Min. Cu Deduction	input	%	**1.0%**	-	-	-	1.0%	1.0%	1.0%	1.0%	1.0%	1.0%	1.0%	1.0%	1.0%	1.0%	-	-	-	-	-	-	-	-	-	-
Cu Payable based on % Payable	calc	%	**25.3%**	-	-	-	25.3%	25.3%	25.3%	25.3%	25.3%	25.3%	25.3%	25.3%	25.3%	25.3%	-	-	-	-	-	-	-	-	-	-
Cu Payable based on min. deduction	calc	%	**25.2%**	-	-	-	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	-	-	-	-	-	-	-	-	-	-
Cu Payable in Cu Concentrate	**calc**	**Mlbs**	**308.0**	**-**	**-**	**-**	**52.1**	**34.5**	**32.5**	**31.9**	**35.8**	**27.0**	**24.5**	**24.3**	**27.6**	**17.9**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	**US$M**	**770.0**	**-**	**-**	**-**	**130.2**	**86.2**	**81.3**	**79.7**	**89.4**	**67.5**	**61.3**	**60.7**	**69.0**	**44.8**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Cu Treatment Charge	input	US$/dmt conc	**97.35**	-	-	-	97.35	97.35	97.35	97.35	97.35	97.35	97.35	97.35	97.35	97.35	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**54.0**	-	-	-	9.1	6.0	5.7	5.6	6.3	4.7	4.3	4.3	4.8	3.1	-	-	-	-	-	-	-	-	-	-
Cu Refining Charge	input	US$/pay lb	**0.10**	-	-	-	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**29.9**	-	-	-	5.1	3.3	3.2	3.1	3.5	2.6	2.4	2.4	2.7	1.7	-	-	-	-	-	-	-	-	-	-
Transportation, Insurance & Losses	input	US$/dmt conc	**90.04**	-	-	-	90.04	90.04	90.04	90.04	90.04	90.04	90.04	90.04	90.04	90.04	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**49.9**	-	-	-	8.4	5.6	5.3	5.2	5.8	4.4	4.0	3.9	4.5	2.9	-	-	-	-	-	-	-	-	-	-
Net Cu Value in Cu Concentrate	**calc**	**US$M**	**636.2**	**-**	**-**	**-**	**107.6**	**71.2**	**67.1**	**65.8**	**73.9**	**55.8**	**50.7**	**50.1**	**57.0**	**37.0**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Au Payable	input	%	**97.5%**	-	-	-	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	97.5%	-	-	-	-	-	-	-	-	-	-
Au Min. Deduction	input	g/t in conc	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Payable Au in Cu Conc	calc	koz	**493**	-	-	-	86	71	75	48	62	40	31	31	27	22	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**616.3**	-	-	-	107.2	89.2	93.4	60.2	77.7	49.7	38.7	38.4	34.1	27.9	-	-	-	-	-	-	-	-	-	-
Au Refining Charge	input	US$/pay oz	**5.00**	-	-	-	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	-	-	-	-	-	-	-	-	-	-
	calc	US$	**2.5**	-	-	-	0.4	0.4	0.4	0.2	0.3	0.2	0.2	0.2	0.1	0.1	-	-	-	-	-	-	-	-	-	-
Net Au Value in Cu Conc	**calc**	**US$M**	**613.9**	**-**	**-**	**-**	**106.7**	**88.8**	**93.1**	**60.0**	**77.4**	**49.5**	**38.5**	**38.2**	**34.0**	**27.7**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**

Parameter	Source	Unit	LOM Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Ag Payable	input	%	**90.0%**	-	-	-	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	-	-	-	-	-	-	-	-	-	-
Ag Min. Deduction	input	g/t in conc	**0.0**	-	-	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-	-	-	-	-	-	-	-	-	-
Payable Ag in Cu Conc	calc	koz	**1,593**	-	-	-	53	59	72	59	52	221	347	267	260	204	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**28.7**	-	-	-	1.0	1.1	1.3	1.1	0.9	4.0	6.2	4.8	4.7	3.7	-	-	-	-	-	-	-	-	-	-
Ag Refining Charge	input	US$/pay oz	**0.50**	-	-	-	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**0.8**	-	-	-	0.0	0.0	0.0	0.0	0.0	0.1	0.2	0.1	0.1	0.1	-	-	-	-	-	-	-	-	-	-
Net Ag Value in Cu Conc	**calc**	**US$M**	**27.9**	**-**	**-**	**-**	**0.9**	**1.0**	**1.3**	**1.0**	**0.9**	**3.9**	**6.1**	**4.7**	**4.5**	**3.6**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Net Cu Concentrate NSR (Pre-Royalty)	**calc**	**US$M**	**1,278.0**	**-**	**-**	**-**	**215.2**	**161.1**	**161.5**	**126.8**	**152.2**	**109.1**	**95.2**	**93.0**	**95.5**	**68.3**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Recovery to Dore - Sulphide Cyanidation																										
Total Cyanidation Recovery																										
Au Recovery	calc	%	**16.0%**			-	16.2%	16.2%	16.2%	16.2%	16.2%	16.0%	15.7%	15.7%	15.7%	15.7%										
Ag Recovery	calc	%	**17.1%**			-	15.2%	15.2%	15.2%	15.2%	15.2%	17.3%	18.1%	18.1%	18.1%	18.1%										
Au Recovered	calc	koz	**132**	-	-	-	23	19	20	13	17	11	9	9	8	6	-	-	-	-	-	-	-	-	-	-
Au Payable	input	%	**99.9%**	-	-	-	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	-	-	-	-	-	-	-	-	-	-
	calc	**koz**	**132**	**-**	**-**	**-**	**23**	**19**	**20**	**13**	**17**	**11**	**9**	**9**	**8**	**6**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	US$M	**165.4**	-	-	-	28.5	23.7	24.8	16.0	20.6	13.4	10.7	10.6	9.5	7.7	-	-	-	-	-	-	-	-	-	-
Au Refining Charge	input	US$/oz	**7.50**	-	-	-	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**1.0**	-	-	-	0.2	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	-	-	-	-	-	-	-	-	-	-
Net Au Value from Sulphide Cyanidation	**calc**	**US$M**	**164.4**	**-**	**-**	**-**	**28.3**	**23.5**	**24.7**	**15.9**	**20.5**	**13.3**	**10.6**	**10.6**	**9.4**	**7.7**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Ag Recovered	calc	koz	**630**	-	-	-	33	36	45	36	32	84	117	90	88	69	-	-	-	-	-	-	-	-	-	-
Ag Payable	input	%	**97.0%**	-	-	-	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	-	-	-	-	-	-	-	-	-	-
	calc	**koz**	**611**	**-**	**-**	**-**	**32**	**35**	**43**	**35**	**31**	**82**	**114**	**87**	**85**	**67**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	US$M	**11.0**	-	-	-	0.6	0.6	0.8	0.6	0.6	1.5	2.0	1.6	1.5	1.2	-	-	-	-	-	-	-	-	-	-
Ag Refining Charge	input	US$/oz	**1.40**	-	-	-	1.40	1.40	1.40	1.40	1.40	1.40	1.40	1.40	1.40	1.40	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**0.9**	-	-	-	0.0	0.0	0.1	0.0	0.0	0.1	0.2	0.1	0.1	0.1	-	-	-	-	-	-	-	-	-	-
Net Ag Value from Sulphide Cyanidation	**calc**	**US$M**	**10.1**	**-**	**-**	**-**	**0.5**	**0.6**	**0.7**	**0.6**	**0.5**	**1.4**	**1.9**	**1.5**	**1.4**	**1.1**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Total Sulphide Cyanidation NSR (Pre-Royalty)	**calc**	**US$M**	**174.6**	**-**	**-**	**-**	**28.8**	**24.1**	**25.4**	**16.5**	**21.0**	**14.7**	**12.5**	**12.0**	**10.8**	**8.8**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
OXIDE PROCESSING																										
Recovery to Dore																										
Tepal North & South																										
Au (before losses)	calc	%	**83.2%**			83.2%	83.2%	83.2%	83.2%	83.2%	83.2%															
Ag (before losses)	calc	%	**63.3%**			63.3%	63.3%	63.3%	63.3%	63.3%	63.3%															
Losses	*link*	*%*	***0.98***			*0.98*	*0.98*	*0.98*	*0.98*	*0.98*	*0.98*															
Au	link	%	**81.5%**			81.5%	81.5%	81.5%	81.5%	81.5%	81.5%	0.0%	-	-	-	-										
Ag	link	%	**62.0%**			62.0%	62.0%	62.0%	62.0%	62.0%	62.0%	0.0%	-	-	-	-										
Tizate																										
Au (before losses)	calc	%	**75.2%**									75.2%	75.2%	75.2%	75.2%											
Ag (before losses)	calc	%	**55.9%**									55.9%	55.9%	55.9%	55.9%											
Losses	*link*	*%*	***98.0%***									*0.98*	*0.98*	*0.98*	*0.98*											
Au	link	%	**73.7%**			-	-	-	-	-	-	73.7%	73.7%	73.7%	73.7%	-										
Ag	link	%	**54.8%**			-	-	-	-	-	-	54.8%	54.8%	54.8%	54.8%	-										
Total Oxide Recovery																										
Au Recovery	calc	%	**80.9%**			81.5%	81.5%	81.5%	81.5%	81.5%	81.5%	73.7%	73.7%	73.7%	73.7%											
Ag Recovery	calc	%	**60.3%**			62.0%	62.0%	62.0%	62.0%	62.0%	62.0%	54.8%	54.8%	54.8%	54.8%											
Au Recovered	calc	koz	**141**	-	-	21	25	25	24	19	15	10	0	1	0	-	-	-	-	-	-	-	-	-	-	-
Au Payable	input	%	**99.9%**	-	-	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	-	-	-	-	-	-	-	-	-	-	-
	calc	**koz**	**141**	**-**	**-**	**21**	**25**	**25**	**24**	**19**	**15**	**10**	**0**	**1**	**0**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	US$M	**176.0**	-	-	26.4	31.2	31.5	30.4	24.0	19.1	12.4	0.2	0.7	0.1	-	-	-	-	-	-	-	-	-	-	-
Au Refining Charge	input	US$/oz	**7.50**	-	-	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	-	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**1.1**	-	-	0.2	0.2	0.2	0.2	0.1	0.1	0.1	0.0	0.0	0.0	-	-	-	-	-	-	-	-	-	-	-
Net Au Value from Oxide CIL	**calc**	**US$M**	**175.0**	**-**	**-**	**26.2**	**31.0**	**31.3**	**30.2**	**23.9**	**19.0**	**12.3**	**0.2**	**0.7**	**0.1**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Ag Recovered	calc	koz	**262**	-	-	25	39	43	41	31	26	51	1	4	1	-	-	-	-	-	-	-	-	-	-	-
Ag Payable	input	%	**97.0%**	-	-	0.97	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	97.0%	-	-	-	-	-	-	-	-	-	-	-
	calc	**koz**	**254**	**-**	**-**	**25**	**38**	**42**	**40**	**30**	**25**	**50**	**1**	**4**	**1**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	US$M	**4.6**	-	-	0.4	0.7	0.8	0.7	0.5	0.4	0.9	0.0	0.1	0.0	-	-	-	-	-	-	-	-	-	-	-
Ag Refining Charge	input	US$/oz	**1.40**	-	-	1.40	1.40	1.40	1.40	1.40	1.40	1.40	1.40	1.40	1.40	-	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**0.4**	-	-	0.0	0.1	0.1	0.1	0.0	0.0	0.1	0.0	0.0	0.0	-	-	-	-	-	-	-	-	-	-	-
Net Ag Value from Oxide CIL	**calc**	**US$M**	**4.2**	**-**	**-**	**0.4**	**0.6**	**0.7**	**0.7**	**0.5**	**0.4**	**0.8**	**0.0**	**0.1**	**0.0**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Total Oxide CIL NSR (Pre-Royalty)	**calc**	**US$M**	**179.2**	**-**	**-**	**26.6**	**31.7**	**32.0**	**30.8**	**24.4**	**19.4**	**13.2**	**0.2**	**0.8**	**0.1**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
TOTAL REVENUES																										
Cu Concentrate																										
Cu	link	Mlbs	**308.0**	-	-	-	52.1	34.5	32.5	31.9	35.8	27.0	24.5	24.3	27.6	17.9	-	-	-	-	-	-	-	-	-	-
	link	US$M	**636.2**	-	-	-	107.6	71.2	67.1	65.8	73.9	55.8	50.7	50.1	57.0	37.0	-	-	-	-	-	-	-	-	-	-
Au	link	koz	**493**	-	-	-	86	71	75	48	62	40	31	31	27	22	-	-	-	-	-	-	-	-	-	-
	link	US$M	**613.9**	-	-	-	106.7	88.8	93.1	60.0	77.4	49.5	38.5	38.2	34.0	27.7	-	-	-	-	-	-	-	-	-	-
Ag	link	koz	**1,593**	-	-	-	53	59	72	59	52	221	347	267	260	204	-	-	-	-	-	-	-	-	-	-
	link	US$M	**27.9**	-	-	-	0.9	1.0	1.3	1.0	0.9	3.9	6.1	4.7	4.5	3.6	-	-	-	-	-	-	-	-	-	-
Cu Concentrate NSR (Pre-Royalty)	**link**	**US$M**	**1,278.0**	**-**	**-**	**-**	**215.2**	**161.1**	**161.5**	**126.8**	**152.2**	**109.1**	**95.2**	**93.0**	**95.5**	**68.3**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Sulphide to Dore																										
Au	link	koz	**132**	-	-	-	23	19	20	13	17	11	9	9	8	6	-	-	-	-	-	-	-	-	-	-
	link	US$M	**164.4**	-	-	-	28.3	23.5	24.7	15.9	20.5	13.3	10.6	10.6	9.4	7.7	-	-	-	-	-	-	-	-	-	-
Ag	link	koz	**611**	-	-	-	32	35	43	35	31	82	114	87	85	67	-	-	-	-	-	-	-	-	-	-
	link	US$M	**10.1**	-	-	-	0.5	0.6	0.7	0.6	0.5	1.4	1.9	1.5	1.4	1.1	-	-	-	-	-	-	-	-	-	-
Sulphide Cyanidation NSR (Pre-Royalty)	**link**	**US$M**	**174.6**	**-**	**-**	**-**	**28.8**	**24.1**	**25.4**	**16.5**	**21.0**	**14.7**	**12.5**	**12.0**	**10.8**	**8.8**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Oxide to Dore																										
Au	link	koz	**141**	-	-	21	25	25	24	19	15	10	0	1	0	-	-	-	-	-	-	-	-	-	-	-
	link	US$M	**175.0**	-	-	26.2	31.0	31.3	30.2	23.9	19.0	12.3	0.2	0.7	0.1	-	-	-	-	-	-	-	-	-	-	-
Ag	link	koz	**254**	-	-	25	38	42	40	30	25	50	1	4	1	-	-	-	-	-	-	-	-	-	-	-
	link	US$M	**4.2**	-	-	0.4	0.6	0.7	0.7	0.5	0.4	0.8	0.0	0.1	0.0	-	-	-	-	-	-	-	-	-	-	-
Oxide Ore - CIL NSR (Pre-Royalty)	**link**	**US$M**	**179.2**	**-**	**-**	**26.6**	**31.7**	**32.0**	**30.8**	**24.4**	**19.4**	**13.2**	**0.2**	**0.8**	**0.1**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
TOTAL METAL PRODUCED																										
Cu	calc	Mlbs	**308.0**	-	-	-	52.1	34.5	32.5	31.9	35.8	27.0	24.5	24.3	27.6	17.9	-	-	-	-	-	-	-	-	-	-
Au	calc	koz	**766**	-	-	21	133	115	119	80	94	60	40	40	35	28	-	-	-	-	-	-	-	-	-	-
Ag	calc	koz	**2,458**	-	-	25	123	136	156	123	108	353	461	357	345	271	-	-	-	-	-	-	-	-	-	-
NSR Pre-Royalty	**calc**	**US$M**	**1,631.7**	**-**	**-**	**26.6**	**275.7**	**217.2**	**217.7**	**167.6**	**192.6**	**137.0**	**108.0**	**105.8**	**106.5**	**77.1**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Royalties	input	%	**2.5%**	-	-	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**40.8**	-	-	0.7	6.9	5.4	5.4	4.2	4.8	3.4	2.7	2.6	2.7	1.9	-	-	-	-	-	-	-	-	-	-
NSR After Royalties	**calc**	**US$M**	**1,590.9**	**-**	**-**	**26.0**	**268.8**	**211.7**	**212.2**	**163.5**	**187.8**	**133.5**	**105.3**	**103.1**	**103.8**	**75.2**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**

Parameter	Source	Unit	LOM Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
STREAMING																										
Cu Stream	input	%	**0.00%**	-	-	-	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	-	-	-	-	-	-	-	-	-	-
Cu Revenue from Stream	input	US$/lb	**1.25**	-	-	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	-	-	-	-	-	-	-	-	-	-
		US$M	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Au Stream	input	%	**0.00%**	-	-	-	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	-	-	-	-	-	-	-	-	-	-
Au Revenue from Stream	input	US$/oz	**600**	-	-	600	600	600	600	600	600	600	600	600	600	600	-	-	-	-	-	-	-	-	-	-
		US$M	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ag Stream	input	%	**0.00%**	-	-	-	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	-	-	-	-	-	-	-	-	-	-
Ag Revenue from Stream	input	US$/oz	**9.00**	-	-	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	-	-	-	-	-	-	-	-	-	-
		US$M	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Revenues from Streaming Contract	calc	US$M	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Lost Project Revenues from Stream	calc	US$M	**0.0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Revenues After Streaming	**calc**	**US$M**	**1,590.9**	**-**	**-**	**26.0**	**268.8**	**211.7**	**212.2**	**163.5**	**187.8**	**133.5**	**105.3**	**103.1**	**103.8**	**75.2**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
OPERATING COSTS																										
Mining	calc	US$/t mined	**2.16**			1.27	1.95	1.97	2.38	2.11	2.02	2.16	2.36	2.39	2.36	2.42										
	link	US$M	**298.7**			17.1	29.7	33.5	28.1	28.4	31.5	35.2	28.1	28.8	23.0	15.2	-	-	-	-	-	-	-	-	-	-
Processing - Sulphide Flotation/Cyanidation	link	US$/t processed	**5.49**	-	-	5.49	5.49	5.49	5.49	5.49	5.49	5.49	5.49	5.49	5.49	5.49	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**429.7**	-	-	-	44.1	44.1	44.1	44.1	44.1	44.1	44.1	44.1	44.1	33.1	-	-	-	-	-	-	-	-	-	-
Processing - Oxide CIL	link	US$/t processed	**6.34**	-	-	6.34	6.34	6.34	6.34	6.34	6.34	6.34	6.34	6.34	6.34	6.34	-	-	-	-	-	-	-	-	-	-
	calc	US$M	**77.2**	-	-	8.9	12.7	12.7	12.7	10.9	9.4	9.0	0.2	0.6	0.1	-	-	-	-	-	-	-	-	-	-	-
G&A	calc	US$/t processed	**0.75**				0.67	0.68	0.68	0.70	0.69	0.72	0.83	0.84	0.85	1.10										
	link	US$M	**67.4**				6.7	6.8	6.8	6.8	6.6	6.8	6.7	6.8	6.8	6.7										
Total Operating Costs	**calc**	**US$M**	**873.1**	**-**	**-**	**26.0**	**93.2**	**97.0**	**91.7**	**90.2**	**91.5**	**95.1**	**79.0**	**80.3**	**74.0**	**55.0**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	**US$/t processed**	**9.65**	**-**	**-**	**18.58**	**9.28**	**9.67**	**9.13**	**9.25**	**9.62**	**10.07**	**9.81**	**9.89**	**9.20**	**9.11**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Gross Operating Income	**calc**	**US$M**	**717.9**	**-**	**-**	**(0.1)**	**175.7**	**114.7**	**120.6**	**73.3**	**96.3**	**38.4**	**26.3**	**22.9**	**29.8**	**20.1**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
CAPITAL COSTS																										
Mining	link	US$M	**15.4**	-	4.0	8.3	0.8	0.0	-	0.0	-	0.8	-	0.0	-	1.3										
Site Development/Earthworks	link	US$M	**5.5**	-	5.5	-	-	-	-	-	-	-	-	-	-	-										
Sulphide Processing Plant	link	US$M	**83.4**	-	23.3	54.4	-	-	5.1	-	-	0.1	0.6	-	-	-										
Oxide Processing Plant	link	US$M	**29.9**	-	29.9	-	-	-	-	-	-	-	-	-	-	-										
Tailings & Waste Rock Management	link	US$M	**57.1**	-	3.7	4.9	7.2	3.6	4.2	4.2	3.0	4.1	5.4	5.4	5.8	5.8										
Surface Infrastructure	link	US$M	**25.2**	-	16.8	8.4	-	-	-	-	-	-	-	-	-	-										
Project Indirects	link	US$M	**10.5**	-	4.9	5.6	-	-	-	-	-	-	-	-	-	-										
EPCM	link	US$M	**15.3**	-	11.6	3.7	-	-	-	-	-	-	-	-	-	-										
Owner Costs	link	US$M	**6.9**	-	1.6	5.2	-	-	-	-	-	-	-	-	-	-										
Closure (Net of Salvage)	link	US$M	**22.9**													7.9	3.0	3.0	3.0	3.0	3.0					
Subtotal	**calc**	**US$M**	**272.0**	**-**	**101.4**	**90.5**	**8.0**	**3.6**	**9.2**	**4.2**	**3.0**	**5.0**	**6.0**	**5.4**	**5.8**	**14.9**	**3.0**	**3.0**	**3.0**	**3.0**	**3.0**	**-**	**-**	**-**	**-**	**-**
Contingency	link	US$M	**28.9**	-	11.8	10.5	0.9	0.4	1.1	0.5	0.4	0.6	0.7	0.6	0.7	0.8										
Total Capital Costs	**calc**	**US$M**	**300.9**	**-**	**113.2**	**101.0**	**9.0**	**4.0**	**10.3**	**4.7**	**3.4**	**5.5**	**6.7**	**6.1**	**6.4**	**15.7**	**3.0**	**3.0**	**3.0**	**3.0**	**3.0**	**-**	**-**	**-**	**-**	**-**
Pre-Production	*calc*	*US$M*	*214.2*	-	*113.2*	*101.0*																				
Sustaining and Closure	*calc*	*US$M*	*86.7*				*9.0*	*4.0*	*10.3*	*4.7*	*3.4*	*5.5*	*6.7*	*6.1*	*6.4*	*15.7*	*3.0*	*3.0*	*3.0*	*3.0*	*3.0*	-	-	-	-	-
WORKING CAPITAL																										
Working Capital	calc	US$M	**0.0**			23.3	-	-	-	-	-	-	-	-	-	(23.3)	-	-	-	-	-	-	-	-		
Upfront Streaming Capital Received	input	US$M	**0.0**																							
TAXES																										
Total Taxes & Federal Mining Royalties	link	US$M	**160.2**	-	-	0.1	34.3	32.2	33.6	16.3	24.1	3.3	2.3	3.7	8.7	1.7	-	-	-	-	-	-	-	-	-	-
CASH FLOWS																										
Pre-Tax Results																										
Net Cash Flow	calc	US$M	**417.0**	-	(113.2)	(124.3)	166.7	110.7	110.3	68.6	92.9	32.9	19.6	16.8	23.4	27.7	(3.0)	(3.0)	(3.0)	(3.0)	(3.0)	-	-	-	-	-
Cumulative Cash Flow	calc	US$M		-	(113.2)	(237.5)	(70.8)	39.9	150.1	218.8	311.6	344.5	364.1	380.9	404.3	432.0	429.0	426.0	423.0	420.0	417.0	417.0	417.0	417.0	417.0	-
After-Tax Results																										
Net Cash Flow	calc	US$M	**256.7**	-	(113.2)	(124.5)	132.4	78.5	76.7	52.3	68.8	29.6	17.3	13.1	14.7	26.0	(3.0)	(3.0)	(3.0)	(3.0)	(3.0)	-	-	-	-	-
Cumulative Cash Flow	calc	US$M		-	(113.2)	(237.6)	(105.3)	(26.8)	49.9	102.2	171.0	200.6	217.9	231.0	245.7	271.7	268.7	265.7	262.7	259.7	256.7	256.7	256.7	256.7	256.7	-
ECONOMIC RESULTS																										
Pre-Tax Results																										
Pre-Tax NPV @ 5%	**calc**	**US$M**	**299.4**																							
Pre-Tax IRR	**calc**	**%**	**35.9%**																							
Pre-Tax Payback	**calc**	**Years**	**1.6**																							
After-Tax Results																										
After-Tax NPV @ 5%	**calc**	**US$M**	**169.4**																							
After-Tax IRR	**calc**	**%**	**23.6%**																							
After-Tax Payback	**calc**	**Years**	**2.3**																							

24 Adjacent Properties

JDS 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). La Verde has a Measured and Indicated Mineral Resource of 354 Mt grading 0.41% Cu, 0.043 g/t Au, and 2.3 g/t Ag at a cut-off of 0.2% Cu. There is an additional Inferred Mineral Resource of 168 Mt grading 0.41% Cu, 0.058 g/t Au and 2.3 g/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 115 km southeast of the Tepal Property. The Property was owned by Aquiline Resources Inc. in the 1990's. 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 200 m and extends to at least 300 m depth.

ASARCO (now Grupo Mexico or GMEXICO) mined several breccia bodies at Inguaran from 1971-1982 and extracted 7 Mt of ore grading 1.2% Cu (Osoria et al., 1991). Gold, silver and tungsten were by-products in the concentrates. The Property is presently owned by Rome Resources Ltd. of Surrey, British Columbia. The Inguaran Copper Mine is 140 km southeast of the Tepal Property.

25 Other Relevant Data and Information

There is no other relevant data or information related to the scope of this report.

26 Interpretations and Conclusions

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the positive economic result herein contained. The PEA proposes the use of industry standard equipment and operating practices. To date, the QPs are not aware of any fatal flaws for the Project.

Using the assumptions highlighted in this report, the Tepal Project offers sufficient economic potential to be advanced to the next stage of study (Pre- feasibility Study).

26.1 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 26.11 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 26.1: 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 Sustaining CAPEX costs would increase.	Additional testing and modelling 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 test work and modelling the costs could increase.	Further definition of PAG rock through additional testwork.
Resource Modelling	Resource volumes that were estimated using industry standard methods, but are still subject to some variation. Variability of grade and discontinuity of	Good grade control, careful mapping and regular resource model reconciliations can significantly reduce the risk of an un-

Risk	Explanation/Potential Impact	Possible Risk Mitigation
	mineralized zones can be the biggest issues of a resource model that is not representative of the deposit.	representative model.
Metallurgical Recoveries	The metallurgical recoveries in this study are based on numerous tests but results may vary when the actual mineralized material is mined. A drop in recoveries would have a direct impact on the project economics. Key tests required include abrasion index for sulphides, carbon loading, settling/loading rates, and cyanide destruction .	Continued test work 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.

Source: JDS (2017)

26.2 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 to the Project.

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 26.2: Project Opportunities

Opportunity	Explanation	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, PEA 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.
Metallurgy and Process Optimization	Further metallurgical test work to optimization the oxide leach circuit along with flotation testing to separate molybdenum minerals from the copper concentrate have the potential to increase recoveries and concentrate values. .	Detailed optimization of the new oxide process flow sheets could result in improved economics. Addition of a molybdenum circuit could add additional revenue not included in this PEA.
Potential to employ Good Used Equipment	There is considerable used equipment on the market that could be utilized in the process plant and/or the mine.	Capital cost reduction.

Source: JDS (2017)

27 Recommendations

In the opinion of JDS, financial analysis of this PEA demonstrates that the Tepal Project has positive economics and warrants consideration for advancement to pre-feasibility level engineering by Geologix. This more advanced study will further detail:

- Revised Mineral Reserve estimate;
- Processing engineering based on the PEA flowsheets;
- Project scheduling;
- Capital and operating cost estimation; and
- Economic results.

The study will improve the confidence in the Project design and execution and will result in an improved accuracy of project economics.

It is estimated that a PFS and its supporting work programs will cost approximately $800 K. A breakdown of the key components of the next study phase is summarized in Table 27.1. Major components of future work recommended are:

- Carry out further hydrogeological testing including drilling of test wells and performing draw down tests to better constrain the groundwater supply well field location and design;
- Additional metallurgical and process studies including the following:
    - 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 and thickener settling rates;
    - 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 will be combined with the pyrite leach solution. The ratio of each will need to be determined. It is recommended a vendor be contacted for input when developing the tests;
    - Investigate and test cyanide destruction; and
    - Flotation testing to separate molybdenum minerals from the chalcopyrite concentrate.

Table 27.1: Preliminary Feasibility Study Cost Estimate

Item	Cost (US$)
Hydrogeology Studies (test wells and draw down testing)	50,000
Processing and Metallurgy	250,000
Pre-feasibility Study	500,000
TOTAL	**800,000**

Source: JDS (2017)

28 References

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*.

Booth, D. (2007b), San Jose – Sampling Methodology and QAQC. Arian Silver Corp. internal memo, Dec 2007.

Booth, M. (2007a),San Jose Exploration by Arian, Arian Silver Corp. internal memo, Dec 2007.

Canadian Institute of Mining, Metallurgy and Petroleum, 2000, CIM Definition Standards on Mineral Resources and Reserves, November 14, 2004, 10 p.

Clifton Associates Ltd. 2011, Programa de Monitoreo de Calidad del Agua Campañas de Muestreo 1, y 2 (Febrero, Julio y Octubre 2011), Proyecto Tepal Mpio. Tepalcatepec, Michoacán, Prepared for Geologix, Text

Clifton Associates Ltd., 2011, Environmental Baseline, Draft. Prepared for Geologix Resources Ltd., Text.

Clifton Associates Ltd., 2011A. Environmental Baseline. Draft. Prepared for Geologix Resources Ltd.

Clifton Associates Ltd., 2011B. Programa de Monitoreo de Calidad del Agua Campañas de Muestreo 1, y 2 (Febrero, Julio y Octubre 2011) Proyecto Tepal Mpio. Tepalcatepec, Michoacán. Prepared for Geologix.

Cooke, D.R., Wilson, A.J. and Davies, A.G.S, 2004, Characteristics and genesis of porphyry copper gold deposits, in Cooke, D.R, Deyell, C. and Pongratz, eds., 24 ct Au Workshop, CODES Special Publication 5, Centre for Ore Deposit Research, University of Tasmania, p. 17 – 34.

Copper Cliff, 1974, Tepal Property, Michoacán State, Mexico: Assessment of and Recommendations on Mineralized zones; 9 p.

Cruymingin, V., 1973. Tepalcuatita Copper Prospect, Borehole 57002 Mill Testing. INCO Memorandum.

Duesing, C., July 3 1973. Tepalcuatita Copper Prospect. INCO Memorandum.

Eliott, M., 1993. The Extraction of Gold and Copper from the Tepalcatepec Samples. Teck Corp. Progress Report.

Exen Consulting Services 2012, Copper Gold Concentrate, NSR Estimate (Project # 1224), July 11, 2012, Text.

G&T's August 2010 report Metallurgical Assessment of the Tepal Project.

Gómez-Tagle, Valdes, A., 1997, Levantamiento Geológico de Alteración-Mineralización del Tepal, prepared for Minera Hecla S.A. de C.V.

GSHAP – Global Seismic Hazard Assessment Program, (1999), A Demonstration Project of the UN/International Decade of Natural Disaster Reduction, GSHAP (http://seismo.ethz.ch/GSHAP/.)

Hogg J., 2008; Resource Estimation Study on the San Jose Silver-Lead-Zinc Prospect, Zacatecas, Mexico, April 2008; Technical Report by ACA Howe International Ltd.

IDEAS, 2011. Hydrological and Hydrogeological Characterization Study for the Tepal Mining Project, Tepalcatepec Municipality, Michoacán. Prepared for Geologix Explorations Inc. 52 pp.

IDEAS, 2011. Hydrological and Hydrogeological Characterization Study for the Tepal Mining Project, Tepalcatepec Municipality, Michoacán. Prepared for Geologix Explorations Inc. 52 pp.

JDS Energy & Mining Inc., March 2013, Technical Report on the Pre-feasibility Study of the Tepal Project Michoacán, Mexico”

Knight Piésold Ltd., April 2012, 2011 Geomechanical Site Investigation Data Report (Ref. No. VA201-366/1-2)

Knight Piésold Ltd., August 2012, Pre-Feasibility Mine Site Water Balance File No.:VA201-366/1-A.01 Cont. No.:VA12-01490

Knight Piésold Ltd., May 2012, Pre-Feasibility Pit Slope Design Report (Ref. No. VA201-366/1-3)

Knight Piésold Ltd., November 2012, DRAFT Pre-Feasibility Tailings Storage Facility Design Report (REF. NO. VA201-366/1-5 Rev A)

Micon International Ltd., 2012, Technical Report on the Mineral Resources of the Tepal Gold-Copper Project, Michoacan State, Mexico for Geologix Explorations Inc., March 29, 2012.

Micon International Ltd., 2012, Geologix Explorations Inc., Tepal Project: Copper-Gold Concentrate NSR Estimates, Project #1224, July 11, 2012.

Murphy, B., Pilotto, D., Arseneau, G., Doersken, G. and Schlitt, W.J., 2011, “Revised Tepal Project, Preliminary Assessment Technical Report - Tepal and Tizate Deposits”, by SRK Consulting (Canada) Inc., for Geologix Explorations Inc.

Murphy, B., Pilotto, D., Robelo, E., White, G., Doersen, G. and Schlitt, W.J., 2010, “Tepal Project, Preliminary Economic Assessment Technical Report”, by SRK Consulting (Canada) Inc., for Geologix Explorations Inc.

Osoria H., A., Leija V., N., Esquivel, R., 1991, Economic Geology of the Inguaran Mining District, Michoacán; in Salas, G.P., editor, Economic Geology, Mexico: Boulder, Colorado, GSA, The Geology of North America, v. P-3.

Panteleyev, A. 1995, Porphyry Cu+/-Mo+/-Au, in Selected British Columbia Mineral Deposit Profiles, Volume 1 - Metallics and Coal, Lefebure, D.V. and Ray, G.E., Editors, British Columbia Ministry of Energy of Employment and Investment, Open File 1995-20, pages 87-92.

pHase Geochemistry, 2012. Geochemical Characterization for the Tepal Gold-Copper Project, Mexico.

Priesmeyer, S. Technical Report on the Tepal Property, Michoacán, Mexico ACA Howe International Limited (2007)

Proyecto “Tepal”, Municipio de Tepalcatepec, Mich.; report prepared on behalf of Minera Hecla S.A. de C.V., 15 p.

Sanchez Mejorada, Velasco y Ribe, 2012, Mining Rights Title Reort on Tepal Project Concessions, May 8,2012.

Shonk, K. N.,1994, The Tepal Gold-Copper Property, Michoacán, Mexico: report prepared on behalf of Teck Resources Inc., 26 p.

Shonk, K., 1994. The Tepal Gold-Copper Property. Teck Corp Technical Report.

Singer, D.A., Berger, V.I. and Moring B.C., 2005, Porphyry Copper deposits of the World: Database, Map, and Grade and Tonnage Models; United States Geological Survey Open File Report 2005-1060, 9 p.

SRK Consulting 2012, Griding & Crushing Circuit Equipment Sizing for Tepal Pre-feasibility Project, Memorandum, January 22, 2013, Text.

SRK Cosulting 2012, Geologix Tepal Comminution Circuit Options. Memorandum, December 18, 2012, Text.

Tanner, J.G. and Shedlock, K.M., (2004) “Seismic Hazard Maps of Mexico, the Caribbean, and Central and South America”, Tectonophysics 390, p. 159-175.

Vidal, R. M., H. Berlanga, and M. del Coro Arizmendi, 2009. (Mexico) Pp. 269-280 in C. Devenish, D.F. Díaz Fernández, R.P. Clay, I. Davidson & Yépez Zabala Eds. Important Bird Areas Americas – Priority sites for biodiversity conservation. Quito, Ecuador: BirdLife International (BirdLife Conservation Series No. 16), Text.

Von Cruyningen, R. and Thorndyke, J., 1973, Tepalcuatita Copper Prospect, Mexico, Borehole 57002, 0-88 metres, Mill Testing: INCO intercompany memorandum dated July 26, 1973, 1 p.

White, G.R., Resource Estimation Update Revised for the Tepal Gold-Copper Prospect, Michoacán, Mexico ACA Howe International Ltd. (Berkhamsted, Hertfordshire, UK, November 4, 2009).

29 Units of Measure, Abbreviations and Acronyms

Symbol/Abbreviation	Description
'	minute (plane angle)
"	second (plane angle) or inches
°	degree
°C	degrees Celsius
3D	three-dimensions
A	ampere
a	annum (year)
ABA	Acid-base accounting
ac	acre
Acfm	actual cubic feet per minute
ACK	apparent coherent kimberlite
ADR	Adsorption, desorption and refining
ALT	active layer thickness
ALT	active layer thickness
amsl	above mean sea level
AN	Qilaq mineral tenure
AN	ammonium nitrate
ARD	acid rock drainage
Au	gold
AWR	all-weather road
B	billion
BD	bulk density
Bt	billion tonnes
BTU	British thermal unit
BV/h	bed volumes per hour
bya	billion years ago
C$	dollar (Canadian)
Ca	calcium
CFE	Comisión Federal de Electricidad
cfm	cubic feet per minute
CIL	Carbon-in-leach
CIM	Canadian institute of mining and metallurgy
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
COG	Cut-off grade
cP	centipoise
Cr	chromium

Symbol/Abbreviation	Description
CRM	Certified reference material
Cu	copper
CV	Coefficient of variation
d	day
d/a	days per year (annum)
d/wk	days per week
dB	decibel
dBa	decibel adjusted
DMS	dense media separation
dmt	dry metric ton
DTM	Digital terrain model
DWT	dead weight tonnes
EA	environmental assessment
EBITDA	Earnings before Interest, Taxes, Depreciation and Amortization
EIS	environmental impact statement
ELC	ecological land classification
ERD	explosives regulatory division
FEL	front-end loader
ft	foot
ft^2	square foot
ft^3	cubic foot
ft^3/s	cubic feet per second
g	gram
G&A	General & administrative
g/cm^3	grams per cubic metre
g/L	grams per litre
g/t	grams per tonne
Ga	billion years
gal	gallon (us)
GJ	gigajoule
GMV	Gross metal value
GPa	gigapascal
gpm	gallons per minute (us)
GSC	geological survey of Canada
GTZ	glacial terrain zone
GW	gigawatt
h	hour
h/a	hours per year
h/d	hours per day
h/wk	hours per week
ha	hectare (10,000 m^2)
ha	hectare

Symbol/Abbreviation	Description
HG	high-grade
HLEM	horizontal loop electro-magnetic
hp	horsepower
HPGR	high-pressure grinding rolls
HQ	drill core diameter of 63.5 mm
Hz	hertz
ICP	Inductively Coupled Plasma
ICP-MS	Inductively Coupled Plasma mass spectrometry
in	inch
in^2	square inch
in^3	cubic inch
IP	Induced Polarization
IRR	internal rate of return
ISSSTE	Instituto de Seguridad Social al Servicio de Trabajadores del Estado
IT	Information technology
JDS	JDS Energy & Mining Inc.
K	hydraulic conductivity
k	kilo (thousand)
kg	kilogram
kg/h	kilograms per hour
kg/m^2	kilograms per square metre
kg/m^3	kilograms per cubic metre
km	kilometre
km/h	kilometres per hour
km^2	square kilometre
kPa	kilopascal
kt	kilotonne
kV	kilovolt
kVA	kilovolt-ampere
kW	kilowatt
kWh	kilowatt hour
kWh/a	kilowatt hours per year
kWh/t	kilowatt hours per tonne
L	litre
L/min	litres per minute
L/s	litres per second
LDD	large-diameter drill
LG	low-grade
LGM	last glacial maximum
LOM	Life of mine
m	metre
M	million

Symbol/Abbreviation	Description
m/min	metres per minute
m/s	metres per second
m^2	square metre
m^3	cubic metre
m^3/h	cubic metres per hour
m^3/s	cubic metres per second
Ma	million years
MAAT	mean annual air temperature
MAE	mean annual evaporation
MAGT	mean annual ground temperature
mamsl	metres above mean sea level
MAP	mean annual precipitation
masl	metres above mean sea level
Mb/s	megabytes per second
mbgs	metres below ground surface
Mbm^3	million bank cubic metres
Mbm^3/a	million bank cubic metres per annum
MBP	melt-bearing pyroclasts
mbs	metres below surface
mbsl	metres below sea level
Mct	million carats
mg	milligram
mg/L	milligrams per litre
MIBC	Methyl isobutyl carbinol
MIDA	microdiamond
min	minute (time)
mL	millilitre
mm	millimetre
Mm^3	million cubic metres
MMER	metal mining effluent regulations
MMSIM	metamorphosed massive sulphide indicator minerals
mo	month
MPa	megapascal
MSS	Mexicano del Seguro Social
Mt	million metric tonnes
MVA	megavolt-ampere
MW	megawatt
NAD	North American datum
NAG	Non-potentially acid generating
NG	normal grade
Ni	nickel
NI 43-101	national instrument 43-101
Nm^3/h	normal cubic metres per hour

Symbol/Abbreviation	Description
NN	Nearest Neighbour
NQ	drill core diameter of 47.6 mm
NSOX	North, South oxide
NSR	Net smelter return
NZ	North Zone
OP	Open pit
OP	open pit
OSA	overall slope angles
oz	troy ounce
P.Geo.	professional geoscientist
PA	Preliminary Assessment
Pa	Pascal
PAG	potentially acid generating
PAX	Potassium Amyl Xanthate
PEA	preliminary economic assessment
PFS	Pre- feasibility study
PGE	platinum group elements
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
psi	pounds per square inch
QA/QC	quality assurance/quality control
QP	qualified person
RC	Reverse circulation
RC	reverse circulation
RMR	rock mass rating
ROM	run-of-mine
rpm	revolutions per minute
RQD	rock quality designation
s	second (time)
S.G.	specific gravity
Scfm	standard cubic feet per minute
SEDEX	sedimentary exhalative
SFD	size frequency distribution
SG	specific gravity
SRK	SRK consulting services Inc.
st/kg	stones per kilogram
st/t	stones per metric tonne
SZ	South Zone
t	tonne (1,000 kg) (metric ton)
t	metric tonne
t/a	tonnes per year
t/d	tonnes per day

Symbol/Abbreviation	Description
t/h	tonnes per hour
TCR	total core recovery
TFFE	target for further exploration
TMF	tailings management facility
t/h	tonnes per hour
ts/hm^3	tonnes seconds per hour metre cubed
US	united states
US$	dollar (American)
UTM	universal transverse mercator
V	volt
VEC	valued ecosystem components
VK	volcaniclastic kimberlite
VMS	volcanic massive sulphide
VSEC	valued socio-economic components
w/w	weight/weight
WBS	Work breakdown structure
wk	week
wmt	wet metric ton
WRSA	Waste rock storage area
WRSF	waste rock storage facility
µm	microns
µm	micrometre

APPENDIX A

QP CERTIFICATES

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300

jdsmining.ca

CERTIFICATE OF AUTHOR

I, Gordon Doerksen, P.Eng., do hereby certify that:

1. I am currently employed as V.P. Technical Services with JDS Energy & Mining Inc. with an office at Suite 900-999 West Hastings Street, Vancouver, BC, V6C 2W2;

2. This certificate applies to the technical report titled "NI 43-101 Preliminary Economic Assessment Technical Report for the Tepal Project, Michoacán, Mexico", with an effective date of January 19, 2017, (the "Technical Report") prepared for Geologix Explorations Inc. ("the Issuer");

3. I am a Professional Mining Engineer (P.Eng. #32273) registered with the Association of Professional Engineers, Geologists of British Columbia. I am also a registered Professional Mining Engineer in Yukon Territory. I am a Member of the Canadian Institute of Mining and Metallurgy and a Registered Member of the Society of Mining Engineers of the AIME.

   I am a graduate of Montana Tech with a B.Sc. in Mining Engineering (1990). I have been involved in mining since 1985 and have practiced my profession continuously since 1990. I have held senior mine production and mine technical positions in mining operations in Canada, the US and in Africa. I have worked as a consultant for over eight years and have performed mine planning, project management, cost estimation, scheduling and economic analysis work, as a Qualified Person, for a significant number of engineering studies and technical reports many of which were located in Latin America.
   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.

4. I have not visited the Tepal Project site;

5. I am responsible for Section numbers 1, 2, 3, 4, 5, 6, 18 (except 18.6, 18.8), 19, 20, 21, 22, 23, 24, 25, 26, 27, 28, 29 of the Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had prior involvement with the property and was QP on the March 2013 JDS report titled: "Technical Report on the Prefeasibility Study of the Tepal Project Michoacán, Mexico"

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;

9. As of the effective date of the Technical Report and 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;

Effective Date: January 19, 2017
Signing Date: February 24, 2017

(original signed and sealed) "Gordon Doerksen, P.Eng."

Gordon Doerksen, P.Eng.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300

jdsmining.ca

CERTIFICATE OF AUTHOR

I, Dino Pilotto, P.Eng., do hereby certify that:

1. I am currently employed as Engineering Manager with JDS Energy & Mining Inc. with an office at Suite 900-999 West Hastings Street, Vancouver, BC, V6C 2W2;

2. This certificate applies to the technical report titled "NI 43-101 Preliminary Economic Assessment Technical Report for the Tepal Project, Michoacán, Mexico", with an effective date of January 19, 2017, (the "Technical Report") prepared for Geologix Explorations Inc. ("the Issuer");

3. I am a Professional Mining Engineer (P.Eng. #198167) registered with the Association of Professional Engineers of British Columbia. I am also a registered Professional Mining Engineer in Alberta, Northwest Territories and Nunavut, and the Yukon. I am a graduate of the University of British Columbia with a B.Sc. in Mining and Mineral Process Engineering (1987). I have practiced my profession continuously since June 1987. I have been involved with mining operations, mine engineering and consulting covering a variety of commodities at locations in North America, South America, Africa, and Eastern Europe.

4. 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.

5. I have visited the Tepal Project site On July 9-10, 2010;

6. I am responsible for Section numbers 1.8, 1.13 (open pit mining aspects), 15, 16, 21.5.3, 22.2.1 of the Technical Report;

7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

8. I have had prior involvement with the property and was QP on the April 29, 2011 SRK report titled: "Revised Tepal Project, Preliminary Assessment Technical Report – Tepal and Tizate Deposits"

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;

10. As of the effective date of the Technical Report and 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;

Effective Date: January 19, 2017
Signing Date: February 24, 2017

(original signed and sealed) "Dino Pilotto, P.Eng."

Dino Pilotto, P.Eng.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF QUALIFIED PERSON

I, Kelly S. McLeod, P. Eng., do hereby certify that:

1. I am currently employed as a Senior Engineer, Metallurgy, with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;
2. This certificate applies to the technical report titled "NI 43-101 Preliminary Economic Assessment Technical Report for the Tepal Project, Michoacán, Mexico", with an effective date of January 19, 2017, (the "Technical Report") prepared for Geologix Explorations Inc. ("the Issuer");
3. I am a Professional Metallurgical Engineer registered with the APEGBC, P.Eng. #15868. I am a graduate of McMaster University with a Bachelors of Engineering, Metallurgy, 1984. I have practiced my profession intermittently since 1984;
4. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43 101F1;
5. I have not personally visited the site;
6. I accept professional responsibility for sections 13 (except 13.5.1) and 17 of this technical report;
7. As a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
8. I have not had prior involvement with the subject property;
9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;
10. As of the effective date of the Technical Report and 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 misleadin.

Effective Date: January 19, 2017

Signing Date: February 24, 2017

Original signed and sealed

Kelly S. McLeod, P.Eng.

CERTIFICATE OF AUTHOR

I, Daniel Friedman, P.Eng., do hereby certify that:

1. I am a Specialist Civil Engineer with Knight Piésold Ltd. with a business address at Suite 1400, 750 West Pender Street, Vancouver, British Columbia Canada V6C 2T8;

2. This certificate applies to the technical report titled "NI 43-101 Preliminary Economic Assessment Technical Report for the Tepal Project, Michoacán, Mexico", with an effective date of January 19, 2017, (the "Technical Report") prepared for Geologix Explorations Inc. ("the Issuer");

3. I am a graduate of McGill University, (B.Eng. - Civil, 2003). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License No. 32571. My relevant experience with respect to mine waste and water management includes over twelve years of continuous work in the discipline. 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.

4. I have not visited the Tepal Project site;

5. I am responsible for Section numbers 18.6 and 18.8 of the Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had no prior involvement with the property that is the subject of the Technical Report;

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;

9. As of the effective date of the Technical Report and 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;

Effective Date: January 19, 2017
Signing Date: February 24, 2017

(original signed and sealed) "Daniel Friedman, P.Eng."

Daniel Friedman, P.Eng.

CERTIFICATE OF AUTHOR

I, Michael Godard, P.Eng., do hereby certify that:

1. I am currently employed as President, Godard Mineral Processing Ltd, 5399 Patrick Street, Bunaby, BC, V5J 3B2.

2. This certificate applies to the technical report titled "NI 43-101 Preliminary Economic Assessment Technical Report for the Tepal Project, Michoacán, Mexico", with an effective date of January 19, 2017, (the "Technical Report") prepared for Geologix Explorations Inc. ("the Issuer");

3. I am a Professional Metallurgical Engineer (P.Eng. #33114) registered with the Association of Professional Engineers, Geologists of British Columbia. I am a Member of the Canadian Institute of Mining and Metallurgy.

4. I am a graduate of University of British Columbia with a BaSc in Metallurgical Engineering, (1985). I have practiced my profession continuously since then.
   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.

5. I have visited the Tepal Project site in January 8 to 13, 2012.

6. I am responsible for Section numbers 13.5.1 of the Technical Report;

7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

8. I have had prior involvement with the property and was QP on the March 2013 JDS report titled: "Technical Report on the Prefeasibility Study of the Tepal Project Michoacán, Mexico"

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;

10. As of the effective date of the Technical Report and 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;

Effective Date: January 19, 2017
Signing Date: February 24, 2017

(original signed and sealed) "Michael Godard, P.Eng."

Michael Godard, P.Eng.

David K. Makepeace

I am one of the authors of this report entitled “NI 43-101 Technical Report Preliminary Economic Assessment on the Tepal Project, Michoacán, Mexico” dated effective January 19, 2017, (the Technical Report).

I, David Makepeace, M.Eng., P.Eng., do hereby certify that:

1. I am employed as a Senior Geologist – Environmental Engineer by:

    Micon International Limited,
    Suite 205 – 700 West Pender Street,
    Vancouver, British Columbia, V6C 1G8, Canada.
    Telephone : (604) 647-6463

2. I hold the following academic qualifications:
    - Bachelor of Applied Science - Geological Engineering, Queen’s University at Kingston, Ontario, 1976,
    - Master of Engineering - Environmental Engineering, University of Alberta, 1994.

3. I am a registered member of the:
    - Association of Professional Engineers and Geoscientists of British Columbia, licence - 14912.
    - Association of Professional Engineers, Geologists and Geophysicists of Alberta, licence - 29367.

4. I have worked as a geological engineer for a total of 37 years since my graduation from university.

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 professional associations (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for this report for the purposes of NI 43-101. My relevant experience in precious metal deposits includes mineral exploration, geological modeling, mineral resource estimates and operations of numerous properties in Canada, the USA, South America and Africa.

6. I am the author of sections 7, 8, 9, 10, 11, 12 and 14 of this Technical Report.

7. I visited the property from January 8 to 13, 2012.

8. I am the author of “Technical Report on the Mineral Resources of the Tepal Gold-Copper Project, Michoacán State, Mexico” dated March 29, 2012 (Micon) and co-author of “Technical Report on the Pre-feasibility Study of the Tepal Project. Michoacán, Mexico” dated March 2013 (JDS Energy & Mining).

9. As of the date of this certificate, I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

10. I am independent of the issuer applying all the tests in section 1.5 of NI 43-101 other than providing consulting services.

11. I have read NI 43-101, Companion Policy 43-101CP and Form 43-101FI, and the Technical Report has been prepared in compliance with that instrument, companion policy and form.

Dated at Vancouver, B.C. this 23rd day of February, 2017.

(Signed by) "*David K. Makepeace"* *(Sealed)*

David K. Makepeace, M.Eng., P.Eng. Professional Engineering Stamp

Senior Geologist – Environmental Engineer
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