Revised Tepal Project Preliminary Assessment Technical Report - Tepal and Tizate Deposits

Report Prepared for

Geologix Explorations Inc.

Report Prepared by

SRK Consulting (Canada) Inc.
2CG020.001
April 29, 2011

Revised Tepal Project

Preliminary Assessment Technical Report

Tepal and Tizate Deposits

Project Location: Michoacán, Mexico
Approximate UTM Coordinates: 2,117,000N, 717,000E

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April 29, 2011

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Executive Summary

This Technical Report was compiled by SRK Consulting (Canada) Inc. (“SRK”) for Geologix Explorations Inc. (“Geologix”). In addition to SRK, W. Joseph Schlitt of Hydrometal, Inc. is the Qualified Person (“QP”) for metallurgy and processing .

The purpose of the Technical Report is to describe the results of an updated Preliminary Assessment (“PA”) conducted on the Geologix Tepal gold-copper project (“Tepal” or “the property”) , located in Mexico. The updated PA incorporates drilling results to December 2010 for both the Tepal and Tizate deposits.

The reader is advised that the preliminary economic assessment summarized in this technical report is only intended to provide an initial, high-level review of the project potential. The PA mine plan and economic model include the use of a significant portion of inferred resources which are considered to be too speculative to be used in an economic analysis except as permitted by NI43-101 for use in PEA’s. There is no guarantee that inferred resources can be converted to indicated or measured resources and, as such, there is no guarantee that the project economics described herein will be achieved.

Location

The project is located in the State of Michoacán, Mexico near the town of Tepalcatepec. The property is 170 km south of Guadalajara, one of the largest cities in Mexico. The centre of the property is located at approximately 2,117,000N and 716,600E (UTM grid coordinates) at an average elevation of 550 masl. The climate is generally hot and arid with about 500 mm of precipitation per annum. The property consists of six contiguous concessions covering an area of about 13,843 ha (Priesmeyer, 2007).

The property has been explored intermittently by various companies for almost thirty years, commencing with INCO in 1972 and followed by Teck, Hecla and Arian.

The property is located within the Coastal Ranges of south-western Mexico south of the Neogene Trans-Mexican Volcanic Belt. Basement rocks consist of Cretaceous to early Tertiary (?) intermediate plutons, stocks and plugs intruding weakly metamorphosed sedimentary and volcanic rocks of probable Jurassic to Cretaceous age. The Jurassic to Cretaceous sedimentary and volcanic rocks are part of an accreted Mesozoic island arc volcanosedimentary assemblage. At least some of the intrusive rocks are probably coeval with the volcanic units. Neogene basalts locally overly basement rocks and represent outliers of the Trans- Mexican Volcanic Belt (Priesmeyer, 2007).

Geology and Resources

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. Mineralization occurs along a line of three small tonalite stocks just west of the north-northwest-trending fault that passes through the centre of the property.

All three stocks are composed of multiple intrusive phases with tonalite porphyry and tonalite porphyry intrusion breccia phases hosting the highest grade mineralization. Most of the resource is hosted by these lithologies in the northern and southernmost stocks (North Zone and South Zone, respectively). Both the North and South zones are crudely formed from a gold-rich core with the highest gold and copper values and highest Au:Cu ratios to a copper dominant periphery with lower Au:Cu ratio to a barren pyritic halo (Shonk, 1994).

The Tepal and Tizate deposits are dominantly a copper-gold (Cu-Au) resource. The bulk of the resource (85% to 90%) is sulphidic, but is overlain by a distinct oxide zone. The sulphide responds well to conventional milling, with production of a good quality Cu-Au flotation concentrate. The oxide material is a candidate for cyanide leaching, either in crushed ore heaps or coarse ore dumps. This would produce gold and some cyanide soluble copper. The latter would be removed from the gold circuit as a sulphide and combined with the concentrate using SART (“sulphidation-acidification-recycling-thickening”) technology. The SART process has been commercially used with success at other operations, including Telfer in Australia and Maricunga in Chile.

Table 1: Metallurgical Recovery Assumptions

		North and South		Tizate	
Item	**Unit**	**Sulphide Flotation**	**Oxide Heap Leach/SART**	**Sulphide Flotation**	**Oxide Heap Leach/SART**
Recovery					
Copper	%	87.4	14.3	85.3	6.8
Gold	%	60.7	78.4	66.2	68.8
Silver	%	0.0	0.0	55.5	38.9
Cu Concentrate Grade (Flotation and SART concentrate)					
Copper	%	25.1	70.0	24.2	70.0
Gold	g/t	variable with Cu	variable with Cu	variable with Cu	variable with Cu
Silver	g/t	variable with Cu	variable with Cu	variable with Cu	variable with Cu

The March 2011 resource estimate for gold and copper at Tepal and Tizate deposits is compliant with the requirements of CIM (Table 2 and Table 3).

Table 2: Mineral resource statement*, Tepal deposit, Tepal Property, SRK Consulting (Canada) Inc., March 15, 2011

Class	Tonnes	Au (g/t)	Cu (%)
Indicated	46,500,000	0.470	0.260
Inferred	47,500,000	0.350	0.220

Table 3: Mineral resource statement*, Tizate deposit, Tepal Property, SRK Consulting (Canada) Inc., March 15, 2011

Class	Tonnes	Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)
Indicated	11,300,000	0.230	0.200	2.240	0.007
Inferred	45,700,000	0.202	0.180	2.330	0.006

*Note: * Reported at a cut-off grade of $5.00 equivalent for open pit scenario. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All values have been rounded to reflect the relative accuracy of the estimates.*

Mining and Reserves

It is proposed that the Tepal and Tizate deposits be developed as open pit mines. Mining of the two deposits will produce a total of 14.3 Mt of oxide heap leach feed, 130.2 Mt of mill sulphide feed and 165.3 Mt of waste (1.14:1 overall strip ratio) over a 19 year mine operating life. The current life of mine ("LOM") plan focuses on achieving consistent heap leach and mill feed production rates, mining of higher grade material early in schedule, and balancing grade and strip ratios.

Mine design for the Tepal and Tizate open pits commenced with the development of Net Smelter Return ("NSR") models. The models included estimates of metal prices, exchange rate, mining dilution, mill and heap leach recovery, concentrate grade, smelting and refining payables and costs, freight and marketing costs and royalties. The NSR models were based on a 10 m x 10 m x 5 m block size for both Tepal and Tizate. Gemcom Whittle™ - Strategic Mine Planning™ ("Whittle™") software was then used to determine the optimal mining shell. Preliminary pit phase designs were selected and preliminary mine planning and scheduling was then conducted on the optimal pit shells. The mineral resources within the pit shells are summarized by category and type, in Table 3 using an internal NSR cut-off grade of $5.23/t at 5% dilution.

The Tepal deposit is divided into a North and South Pit. The mining sequence for both Tepal and Tizate was further divided into a number of pit phases designed to maximize grade; reduce pre-stripping requirements in the early years; provide required oxide production for the heap leach process; and keep and maintain the process plant at full production capacity. The LOM mine production schedule is shown in Table 5.

Waste Management

Waste rock from the Tepal pits would be deposited in engineered waste rock facilities ("WRF") adjacent to both the North and South Pits. Waste from the Tizate pit is planned for a WRF to the south east of the pit. The North WRF, would be located immediately north of the North Pit and is designed to contain 108 Mt of waste. The West WRF, would be located on the west side of the South Pit and has a design capacity of 14 Mt. The Tizate WRF would have a capacity of 52 Mt.

The tailings management facility ("TMF") is envisioned to be about 4 km east of the plant and will be a valley fill impoundment using cycloned tailings. The TMF was designed to hold up to 130 Mt of tailings.

Table 4: PA Tepal Project LOM Resource (@ $5.23/t NSR cut-off)

Category	Oxide							Sulphide							Total						
	Quantity (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Contained Au (koz)	Contained Cu (Mlbs)	Contained Ag (koz)	Quantity (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Contained Au (koz)	Contained Cu (Mlbs)	Contained Ag (koz)	Quantity (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Contained Au (koz)	Contained Cu (Mlbs)	Contained Ag (koz)
Tepal																					
Indicated	4.6	0.51	0.24		75	25		41.0	0.46	0.25		602	226		45.6	0.46	0.25		677	251	
Inferred	7.9	0.35	0.20		89	36		38.6	0.36	0.22		442	188		46.5	0.35	0.22		530	223	
Total Tepal	**12.5**	**0.41**	**0.22**		**164**	**61**		**79.6**	**0.41**	**0.24**		**1,043**	**413**		**92.1**	**0.41**	**0.23**		**1,207**	**474**	
Tizate																					
Indicated	0.3	0.30	0.20	2.32	3	1	25	11.3	0.22	0.19	2.12	81	46	771	11.6	0.22	0.19	2.13	84	48	796
Inferred	1.4	0.32	0.21	2.66	15	6	119	39.2	0.20	0.17	2.16	249	150	2717	40.6	0.20	0.17	2.17	263	156	2,837
Total Tizate	**1.7**	**0.32**	**0.21**	**2.59**	**18**	**8**	**145**	**50.5**	**0.20**	**0.18**	**2.15**	**329**	**196**	**3488**	**52.2**	**0.21**	**0.18**	**2.16**	**347**	**204**	**3,633**

Table 5: LOM Mine Production Schedule – Tepal and Tizate Project

Section	Item	Unit	Years1 - 19 Total	YEAR																		
				1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
MATERIAL SCHEDULE																						
Mining Total	Operating Days	*days*	**6,617**	365	365	366	365	365	365	366	365	365	365	365	365	365	365	365	365	365	365	45
Mining Total	Waste	Mt	**165.3**	1.91	2.53	9.54	9.36	11.82	16.26	12.90	16.00	14.00	14.61	15.94	4.92	14.30	6.83	4.89	3.39	3.66	2.50	-
	Oxide Ore	Mt	**14.3**	0.75	3.00	3.00	3.00	2.18	2.35	-	-	-	-	-	-	-	-	-	-	-	-	-
	Sulphide Ore	Mt	**130.2**	-	0.99	7.97	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	1.28
	Total Mining	**Mt**	**309.9**	**2.7**	**6.5**	**20.5**	**20.4**	**22.0**	**26.6**	**20.9**	**24.0**	**22.0**	**22.6**	**23.9**	**12.9**	**22.3**	**14.8**	**12.9**	**11.4**	**11.7**	**10.5**	**1.3**
	Strip Ratio	waste:ore	**1.14**	2.5	0.6	0.9	0.9	1.2	1.6	1.6	2.0	1.8	1.8	2.0	0.6	1.8	0.9	0.6	0.4	0.5	0.3	-
	Daily Production	t/day	**46,827**	7,279	17,863	56,030	55,767	60,274	72,893	57,112	65,734	60,274	61,948	65,586	35,384	61,093	40,625	35,307	31,195	31,953	28,773	28,489
North and South Pit Flotation	Waste	Mt	**119.7**	1.9	2.5	9.5	9.4	11.8	6.2	12.9	16.0	14.0	14.6	15.9	4.9	0.0						
	Flotation Circuit Feed	Mt	**79.7**		1.0	8.0	8.0	8.0	4.7	8.0	8.0	8.0	8.0	8.0	8.0	2.1						
	Cu head grade	%Cu	**0.23**		0.46	0.34	0.25	0.24	0.22	0.22	0.21	0.21	0.20	0.20	0.22	0.25						
	Au head grade	g/t Au	**0.41**		0.76	0.53	0.41	0.35	0.36	0.38	0.39	0.38	0.35	0.35	0.43	0.70						
	Ag head grade	g/t Ag	**-**																			
North and South Pit Heap Leach	HL Feed	Mt	**12.55**	0.7	3.0	3.0	3.0	2.2	0.6													
	Cu head grade	%Cu	**0.22**	0.28	0.27	0.22	0.20	0.17	0.18													
	Au head grade	g/t Au	**0.41**	0.72	0.49	0.36	0.40	0.29	0.28													
	Ag head grade	g/t Ag	**-**																			
Tizate Pit Flotation	Waste	Mt	**45.6**						10.1							14.3	6.8	4.9	3.4	3.7	2.5	
	Flotation Circuit Feed	Mt	**50.5**						3.3							5.9	8.0	8.0	8.0	8.0	8.0	1.3
	Cu head grade	%Cu	**0.18**						0.19							0.18	0.18	0.17	0.17	0.17	0.18	0.18
	Au head grade	g/t Au	**0.20**						0.23							0.19	0.22	0.21	0.21	0.21	0.18	0.11
	Ag head grade	g/t Ag	**2.15**						2.21							1.83	1.97	1.87	1.82	2.23	2.79	3.75
Tizate Heap Leach	HL Feed	Mt	**1.73**						1.7													
	Cu head grade	%Cu	**0.20**						0.20													
	Au head grade	g/t Au	**0.32**						0.32													
	Ag head grade	g/t Ag	**2.60**						2.60													

Capital and Operating Costs

Operating costs for the project are summarized in Table 6. All costs are in $US dollars.

Table 6: Operating Cost Estimate

Area	Unit	Estimate
Open pit mining	$/t mined	$1.36
	$/t processed (HL + Flot.)	$2.92
Heap Leach/SART Processing	$/t to HL	$4.31
Flotation	$/t to Flotation	$4.30
General and Administrative	$/t processed (HL + Flot.)	$0.68
Tailings Deposition	$/t to Flotation	$0.28
Total OPEX	$/t processed (HL + Flotation)	$8.05
Unit OPEX per Cu equivalent	$/lb Eq. Cu payable	1.31
Unit OPEX per Au equivalent	$/oz Eq. Au payable	478

Capital costs for the project were developed from a mix of first principles, reference projects, and experience. The annual capital costs by major category are shown in Table 7.

Table 7: Capital Cost Estimate Summary

Category	Unit	Total	Year					
			-2	-1	1	2	3 to 18	19
Mining Equipment	M$	77.6		24.0	5.2	23.0	25.4	
Roads and General Infrastructure	M$	15.7		15.7				
Electrical Power Line and Generators	M$	14.2		14.2				
Flotation Process Plant	M$	124.0		24.0	100.0			
Heap Leach Pad and Facility	M$	17.3		17.3				
Tailings Management Facility	M$	19.9		10.0	9.9			
Owners Costs	M$	9.6		4.1	5.5			
EPCM	M$	28.2		12.2	16.0			
Closure	M$	9.0						9.0
Contingency (10%)	M$	31.5		12.1	13.7	2.3	2.5	0.9
Working Capital	M$	0			3.4			-3.4
Total Capital Cost	**M$**	**346.7**		**133.5**	**153.7**	**25.3**	**28.0**	**6.5**

Technical Economic Analysis

Simplified earnings before interest, taxation, depreciation and amortization ("EBITDA") analyses were compiled for three cases using varying copper, gold and silver prices. For each case the mill feed tonnes were held constant and the metal prices were varied only in the economic model. The base case (Case B) metal prices were used for Whittle optimization and mine planning were $2.75/lb Cu, $1,000/oz Au and $16.00/oz Ag. The metal prices used in the economic model for the three cases are shown in Table 8.

Table 8: Metal Prices by Case

Case	Copper Price ($/lb)	Gold Price ($/oz)	Silver Price ($/oz)
Case A	2.75	900	16.00
Case B (Base Case used for mine design)	2.75	1,000	16.00
Case C	3.50	1,200	16.00

Common assumptions to all cases included:

- 5% discount rate ("DR") for net present value ("NPV") calculation;
- 100% equity financing as per guidance by Geologix;
- Exclusion of all pre-development costs as per guidance by Geologix (e.g., exploration and resource definition costs, engineering field work and studies costs, environmental baseline studies costs, etc.);
- Exclusion of all duties and taxes (a brief description of Mexican taxes is included in Section 19.7);
- 2.5% royalty on net smelter return;
- All 2011 costs were assumed to be sunk costs with analysis beginning in 2012 (Year 0).

The results of the economic analysis indicate that the project is economic for the assumptions made as shown in Table 9.

Table 9: Case B (Base Case) LOM Key Economic Results

Parameter	Unit	Results
Case A		
EBITDA $NPV_{0\%}$	M$	653
EBITDA $NPV_{5\%}$	M$	347
EBITDA IRR	%	20
EBITDA payback period	Production years	4.5
Case B		
EBITDA $NPV_{0\%}$	M$	749
EBITDA $NPV_{5\%}$	M$	412
EBITDA IRR	%	22
EBITDA payback period	Production years	4
Case C		
EBITDA $NPV_{0\%}$	M$	1,320
EBITDA $NPV_{5\%}$	M$	786
EBITDA IRR	%	34
EBITDA payback period	Production years	3

Ranges of gold and copper prices that, when combined, result in a break-even situation or an $NPV_{5\%}$ of $0 are shown in Table 10.

Table 10: Combined Break-even ($0 $NPV_{5\%}$) Copper and Gold Prices

Copper Price ($/lb)	Gold Price ($/oz)
1.49	1,000
1.69	900
1.89	800
2.08	700
2.00	740
2.25	615
2.50	485
2.75	360
3.00	235

Environment & Permitting

The present environmental baseline has expanded to 3,217 ha as a reference inventory, covering the 1,406 ha of the Tepal mining concessions and micro/nano basins of direct influence. The following are the general environmental conclusions:

- The project is located in the vicinity of land routes suitable for the operation of a mining project, however, the local road system is rudimentary and requires an important work of access in the event of major mining related activities.
- The Tepal concession are located on surface land belonging to the Tepalcuatita Ranch, private land and ejido lands, implying potential displacement of productive activities (cattle ranching and seasonal agriculture) and closing rural roads recently used by the local community (travel to and forth the highway and La Estanzuela).

- The Environmental Baseline (“LBA”) covers 200% more surface area than the footprint of the project (3,217 ha studied versus 1,406 ha of the current Tepal mining concession), this allows for a better understanding of the local environmental system and future consideration for the preliminary mine development plan.
- Following the completion of a conceptual mining development plan, new areas for the expansion of environmental inventories should be contemplated in order to include potential new sites of interest (mining infrastructure).
- The main components that have been considered for the establishment of this area of study correspond to the area of geological interest (mining concessions), the possible development of open pit mining, areas suitable for the establishment of a process plant, associated infrastructure and the construction of an access road dedicated to the mining unit, that connects the project to the East (towards the state highway); as well and the hydrological micro / nano basins of direct influence from the project.
- Water quality at the Tepal Project is considered a key item in regard to potential areas of opportunity for community support and consideration of pre-mining parameters (cyanide, metals, etc.).
- Currently, the additional environmental monitoring activities are, at this moment, focused towards pre-mine stages, development and gap analysis in regard to specific infrastructure and environmental design/management.

Geologix is required to prepare and submit to SEMARNAT different environmental reports (MIA, ETJ, ER) for environmental impact authorizations prior to site preparation and construction for operation permits, land use modification, risk assessment, among others. Overall environmental permitting in Michoacán can take from six months to one year with land tenure usually being the most sensitive issue in delaying the permitting process.

The current environmental baseline information indicates that there are no environmental “fatal flaws” identified for the proposed Tepal Project. The extent of habitat degradation in the area as well as the surrounding conservation status (heterogeneous mosaic), current land use and local trends do suggest the need for an integrated and careful environmental management policy and program in order to ensure that the mine site activities can coexist with the local communities.

Conclusions

Industry standard mining, process design, construction methods and economic evaluation practices have been used to assess the Tepal Project. SRK has concluded that there is adequate geological and other pertinent data available to generate a PA.

Based on current knowledge and assumptions, the results of this study show that the project is positive economics (within the very preliminary parameters of a PEA) and should be advanced to the next level of study by conducting the work indicated in the recommendations section of this report.

As with almost all mining ventures, there are a large number of risks and opportunities that can affect the outcome of the Tepal project. Most of these risks and opportunities are based on uncertainty, such as lack of scientific information (test results, drill results, etc.) or the lack of control over external drivers (metal price, exchange rates, etc.).

Subsequent higher-level engineering studies will be required to further refine these risks and opportunities, identify new risks and opportunities and define strategies for risk mitigation or opportunity implementation.

While a significant amount of information is still required to do a complete assessment, at this point there do not appear to be any fatal flaws for the project.

The study has achieved its original objective of providing a preliminary review of the potential economic viability of the Tepal project.

Recommendations

It is recommended that the project be advanced to the preliminary feasibility study stage, following a definition drilling program that will attempt to convert inferred resources into indicated or measured resources. The cost of the definition drilling program, pre-feasibility study and associated field and lab work is estimated to be $6M.

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Disclaimer

The opinions expressed in this Report have been based on the information supplied to SRK Consulting (Canada) Inc. (SRK) by Geologix Explorations Ltd. (Geologix). These opinions are provided in response to a specific request from Geologix to do so, and are subject to the contractual terms between SRK and Geologix. SRK has exercised all due care in reviewing the supplied information. Whilst SRK has compared key supplied data with expected values, the accuracy of the results and conclusions from the review are entirely reliant on the accuracy and completeness of the supplied data. Opinions presented in this report apply to the site conditions and features as they existed at the time of SRK's investigations, and those reasonably foreseeable. These opinions do not necessarily apply to conditions and features that may arise after the date of this Report.

1 Introduction

This Technical Report was compiled by SRK Consulting (Canada) Inc. ("SRK") for Geologix Explorations Inc. ("Geologix").

The purpose of the Technical Report is to describe the results of an updated preliminary economic assessment ("PEA") conducted on the Geologix's Tepal gold-copper project located in Michoacán, Mexico.

The reader is advised that the preliminary economic assessment summarized in this technical report is only intended to provide an initial, high-level review of the project potential. The PA mine plan and economic model include the use of a significant portion of inferred resources which are considered to be too speculative to be used in an economic analysis except as allowed for in PA studies. There is no guarantee that inferred resources can be converted to indicated or measured resources and, as such, there is no guarantee that the project economics described herein will be achieved.

Several sections of this report are taken from the two preceding technical reports written by ACA Howe International Ltd. (ACA) titled "Resource Estimation Update for the Tepal Gold-Copper Prospect, Michoacán, Mexico" dated Sept. 24, 2008 for Arian Silver Corporation and "Resource Estimation Update Revised for the Tepal Gold-Copper Prospect, Michoacán, Mexico" by dated November 4, 2009 for Geologix. The previous ACA report information is referenced as appropriate. Other references can be found in Section 23.

The qualified persons ("QP's") responsible for this report are shown in Table 1.1 along with their responsibilities and site visit dates and descriptions. Each QP in this report takes sole responsibility for their work as outlined in their QP Certificates.

All units in this report are based on the International System of Units ("SI"), except industry standard units, such as troy ounces for the mass of precious metals. All currency values are United States Dollars ("US$" or "$") unless otherwise stated.

This report uses abbreviations and acronyms common within the minerals industry, and are explained in Section 23 of this report.

Table 1.1: Qualified Persons and Site Visit Information

<table>
<tr><th>Qualified Person</th><th>Responsibility</th><th>Site Visit Date</th><th>Scope of Site Visit</th></tr>
<tr><td>Dino Pilotto, P.Eng.
SRK</td><td>Mining, Infrastructure and Waste Management</td><td>July 8-11, 2010</td><td rowspan="3">Drive from Guadalajara (the largest city in the region) to the project site.

Tour of the project area to inspect potential locations for the open pit, tailings management facility, waste dump and plant site.

Review of representative diamond drill for geologic and geotechnical characteristics.

Visited the adjacent town, Tepalcatepec, to view the local infrastructure including the regional electrical substation.

Traveled to Ixtapa to inspect road conditions and view the facilities at the port of Lázaro Cárdenas.</td></tr>
<tr><td>Bruce Murphy, FSAIMM
SRK</td><td>Geotechnical Considerations</td><td>July 8-11, 2010</td></tr>
<tr><td>W. Joseph Schlitt, P.Eng.
Hydrometallurgy</td><td>Metallurgy and Mineral Processing</td><td>July 8-11, 2010</td></tr>
<tr><td>Gilles Arseneau,
P.Geo
SRK</td><td>Geology and Mineral Resource Estimation</td><td>March 12-13, 2011</td><td>Tour of the project site and review of drill program.

Review of data collection methodologies

Review of sampling techniques and assay QA/QC protocols.

Review and verify project data.</td></tr>
<tr><td>Gordon Doerksen, P.Eng.
SRK</td><td>Economic model,
Environmental Considerations and report compilation</td><td>n/a</td><td>Mr. Doerksen is responsible for the economic and environmental aspects of this report and relied on the site inspection done by Dino Pilotto and Bruce Murphy of SRK.</td></tr>
</table>

2 Reliance on Other Experts

Preparation of this report is based upon public and private information provided by Geologix and information provided in various previous Technical Reports listed in Section 23 of this report.

This report also relies upon the work and opinions of and data from some non-QP experts. The following list outlines the information provided by other experts, who are independent to the authors:

- Flotation and comminution test work by G&T Metallurgical Services Ltd.;
- Heap leaching test work completed by McClelland Laboratories, Inc.; and
- Epitacio Robledo of Clifton Associates Ltd. for the Environmental Considerations.

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

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

Neither SRK nor the authors of this technical report are qualified to provide extensive comment on legal issues associated with the Tepal property. As such, portions of Section 3 dealing with the types and numbers of mineral tenures and licenses, the nature and extent of Geologix's title and interest in the Tepal property, the terms of any royalties, back-in rights, payments or other agreements and encumbrances to which the property is subject are descriptive in nature and are provided exclusive of a legal opinion.

3 Property Description and Location

The following sections are excerpted from Priesmeyer, 2007.

3.1 Property Description and Location

The Tepal Property is located in the municipality of Tepalcatepec, Michoacán state in south-western Mexico near the town of Tepalcatepec (Figure 3.1). The property is 70 km west of Apatzingán and 170 km south of Guadalajara, one of the largest cities in Mexico. The property is centered at the approximate UTM grid coordinates of 2,116,995N and 716,594 E at an average elevation of 550 metres ("m").

The property consists of seven contiguous concessions totalling 17,237.20 hectares ("ha") (Figure 3.3, Table 3.1). Arian acquired a concession called Tepal 2, which was permitted over free ground and completely surrounded the five smaller concessions. The area of the Tepal 2 concession is 12,437.2 ha. Geologix in late 2010 was granted the right to acquire 100% of the mineral rights to the Division Tepal 1 subject to an underlying option agreement with Mineral Tepal S.A. de C.V..

Table 3.1: Concession Titles for Tepal

Name of Concession	Title number	Area (ha)	Date of Title	Expiration Date	Owner
La Esperanza Fracción 1	216873	120.00	5 June 2002	4 June 2052	Minera Tepal S.A. de C.V.
Tepal	219924	986.00	7 May 2003	6 May 2053	Minera Tepal S.A. de C.V.
Tepal Fracción 1	216874	140.00	5 June 2002	4 June 2052	Minera Tepal S.A. de C.V.
Tepal Fracción 2	216875	70.00	5 June 2002	4 June 2052	Minera Tepal S.A. de C.V.
Tepal Fracción 3	216876	90.00	5 June 2002	4 June 2052	Minera Tepal S.A. de C.V.
Tepal 2	229354	12,437.2	12 Apr 2007	12 Apr 2057	Arian Silver de Mexico S.A. de C.V
Tepal 1	**230299**	**3,394.00**	3 August 2007	27 June 2055	Minera Tepal S.A. de C.V.
Total		**17,237.20**			

The seven concessions listed in Table 3.1 would have been surveyed in order for the titles to be issued as this is a requirement under Mexican law. Arian has not surveyed the concessions independently.

3.2 Mineral Rights

Arian signed an agreement with Minera Tepal S.A. de C.V. ("Minera Tepal") for the rights to the concessions described in Table 3.2. Under the agreement, Arian must pay a total of US $5,000,000 over a five year period for a 100% interest in the property. Arian can exercise the option or terminate the agreement at any time. The payment schedule is outlined in Table 3.3.

On September 24, 2009 the Company signed an agreement (the “Arian Letter Agreement”) with Arian Silver Corp. (“Arian”) whereby the Company was granted the exclusive rights to purchase Arian’s 100% interest in the Tepal Gold-Copper Project in the state of Michoacán, Mexico. Under the terms of the agreement, the Company has the option to complete the purchase of 100% of the property, subject to a 2.5% net smelter return (“NSR”) royalty, by delivering to Arian US$3.0 million in staged payments before February 23, 2011 and assuming the balance of Arian’s obligations under the terms of the underlying property option agreement.

On December 29th, 2010, the Company signed an option agreement to acquire additional claims totalling 34 square kilometers at its Tepal Project in Michoacán State, Mexico. The definitive option agreement with Minera Tepal SA de CV ("Minera Tepal") grants the Company the right to acquire 100% of the Division Tepal 1 mineral claim ("Tepal 1") subject to a 2% net smelter return ("NSR") payable to Minera Tepal.

In accordance with the Company's Option to Purchase Agreement (the "Agreement") with Arian for the purchase of a 100% interest in the Tepal Gold-Copper Project in Michoacán, Mexico, the Company delivered on February 24th, 2011, a total of US$1,023,000 cash (including IVA) and 1,089,318 common shares of the Company to Arian as consideration for the final payment obligations under the terms of the Agreement between Geologix and Arian. The shares issued to Arian are subject to a four month restricted sale period.

On April 4, 2011, the Company has accelerated delivery of the final US$2.3 million property option payment due to the underlying vendor (Minera Tepal, SA de CV) and earned a 100% interest (subject to a 2.5% net smelter return) in the 138 square kilometre land package in Michoacán state, Mexico.

The principal terms of the Arian Letter Agreement are as follows:

The Company advanced to Arian the sum of US$517,500 which was used by Arian to complete an outstanding underlying option payment due to Minera Tepal S.A. de C.V. (“Minera Tepal”) (US$450,000 plus the applicable 15% value-added tax of US$67,500). The advance was made by the Company to Arian as an interest free loan and was due for repayment on April 23, 2010 unless the Company elected to proceed with the option to purchase the Tepal Property, in which case the sum of the loan would be applied against the eventual purchase price.

In consideration for the loan, Arian granted the Company a five month exclusivity period to permit the Company to undertake due diligence of the Tepal Property. Following completion of the due diligence review of the property, the Company had the option to elect, at any time within the five-month exclusivity period, to acquire the Tepal Property from Arian on an option basis for a total consideration of US$3.0 million, payable to Arian in two instalments:

- An initial payment of US$1.0 million, plus forgiveness of the interest free loan of US$450,000, for a total of US$1.45 million on or before February 23, 2010 (paid); and
- A payment of US$1.55 million on or before February 23, 2011.

At the Company's election, each such payment may be made in cash, or up to 50% in the Company's Common Shares valued at the 10-day average closing price of the Common Shares immediately prior to the time of each payment.

The Company also assumed the balance of Arian's obligations under the terms of an underlying property option agreement subject to a 2.5% NSR and is responsible for completing staged payments to the underlying property vendor as follows:

Table 3.2: Staged Payment Requirements

Date	Payment amount
6-Jun-10	US$ 900,000 (paid)
6-Jun-11	US$ 2,300,000 (paid)
Total	**US$ 3,200,000**

On January 11, 2010 the Company notified Arian that it elected to proceed with the acquisition of the Tepal Property.

On January 26, 2010 the Company and Arian entered into a definitive agreement confirming the terms of the Arian Letter Agreement. The Company and Arian subsequently agreed to modify the initial option payment payable on February 23, 2010. Pursuant to a letter agreement dated February 17, 2010 the parties agreed that the Company would pay US$725,000 in cash on or before February 23, 2010 (paid) and US$725,000 on or before March 4, 2010 in cash or Common Shares (issued). The payment due on March 4, 2010 was paid through the issuance of 3,434,193 Common Shares at a value of $0.22 per share.

Arian's agreement with Minera Tepal has a first-right-of-refusal on this royalty should Minera Tepal elect to sell the royalty. A 15% value-added tax ("IVA") is to be paid by Arian, now Geologix for each option and royalty payment. In December 2007, Arian located an additional concession (Tepal 2) totalling 12,437.2 ha, for Mx$30,000 which has been included in the Property.

Table 3.3: Payment Schedule for Tepal Property

Amount	Due Date
$100,000	Paid upon signing
$150,000	Paid December 6, 2006
$250,000	Paid June 6, 2007
$300,000	Paid December 6, 2007
$500,000	Paid June 6, 2008
$500,000	Paid June 6, 2009
$900,000	Paid June 6, 2010
$2,300,000	Paid Feb 24, 2011

The principal terms of the 2nd Minera Tepal S.A. de C.V. Letter Agreement on the Division Tepal 1 concession are as follows:

To complete the option, the agreement requires the Company to make cash payments to Minera Tepal, totalling $2,990,000 (all amounts expressed in $US) as outlined in Table 3.4.

The definitive option agreement with Minera Tepal SA de CV ("Minera Tepal") grants the Company the right to acquire 100% of the Division Tepal 1 mineral claim ("Tepal 1") subject to a 2% net smelter return ("NSR") payable to Minera Tepal.

Payments are subject to Mexican Value Added Tax which will be paid by the Company and applied for reimbursement .

A 2.0% NSR based on the sale of all minerals is payable to Minera Tepal. Geologix retains a right of first refusal on any sale or assignment of royalties. Geologix may purchase at any time all or part of the NSR for $1,100,000 plus Value Added Tax for each 1% of the royalty.

Table 3.4: Payment Schedule for Tepal 1 Property

Amount	Due Date
$57,500	Paid upon signing
$57,500	June 1, 2011
$115,000	December 1, 2011
$172,000	June 1, 2012
$287,500	December 1, 2012
$862,500	December 1, 2013
$1,437,500	December 1, 2014

The majority of surface rights for the property are owned by three individuals. However, other minor portions of the property outside of the main resource areas and proposed infrastructure are owned by several ejidos. While Arian did not have a formal agreement with the ejido owners, they have negotiated a verbal access agreement allowing them access on to those portions of the property underlain by ejido lands. Arian has completed the process of negotiating a formal agreement with the principal surface owner. Geologix has subsequently renegotiated the same terms for a longer period of time with the main private owner.

Mining taxes, or holding fees for mining concessions, in Mexico are based on the amount of time elapsed from the date the title was issued and the number of hectares covered by the concessions (Table 3.1). These taxes are paid twice per year and the resulting tax liabilities for the Tepal Property total Mx$783,458 or US$67,682 for 2011.

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

Clifton Associates is not aware of any environmental issues currently relating to the property."

Figure 3.1: **Location Map of the Tepal Property (taken from Priesmeyer, 2007)**

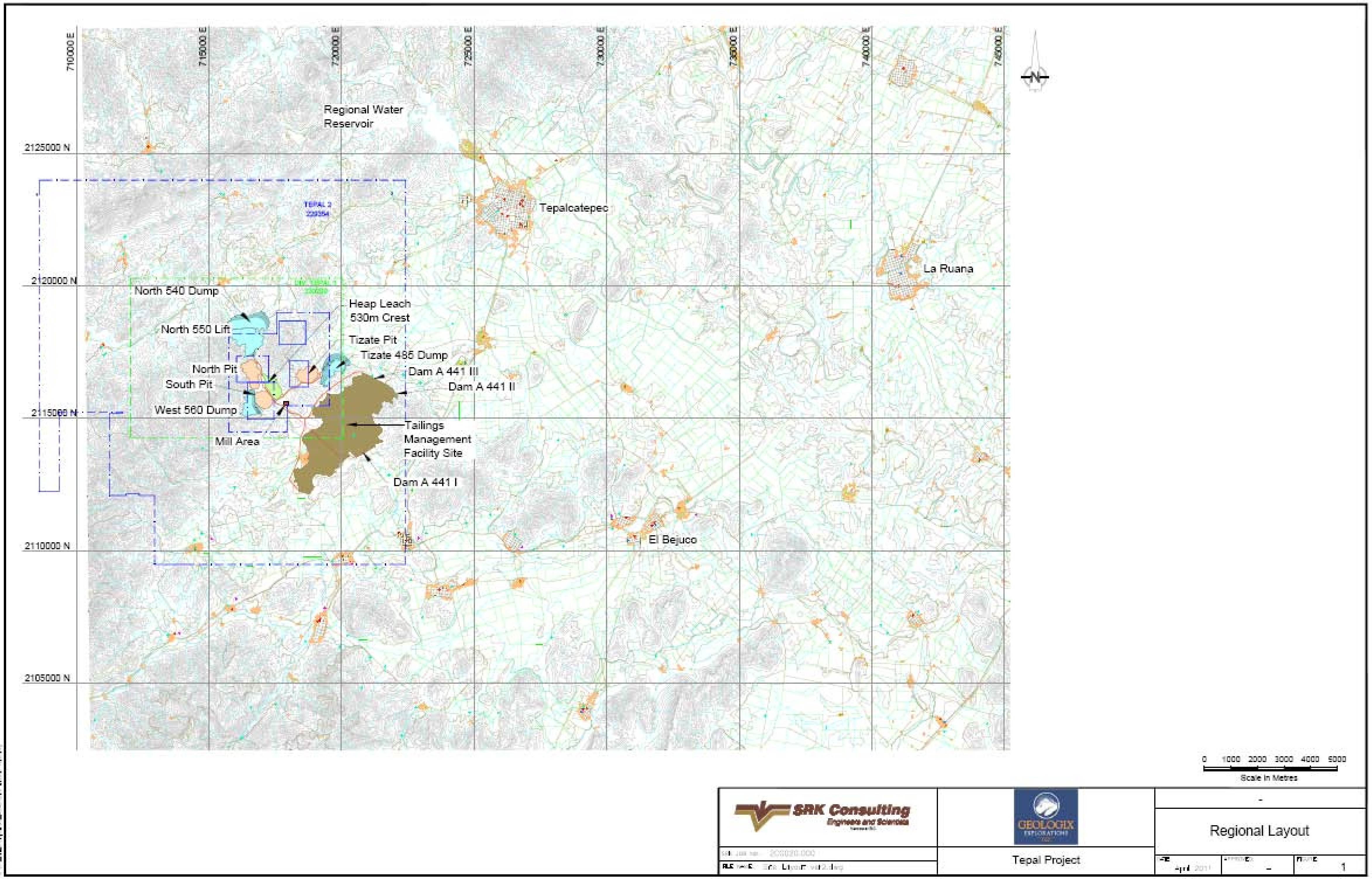

Figure 3.2: Tepal Regional View of Planned Facilities (SRK 2010)

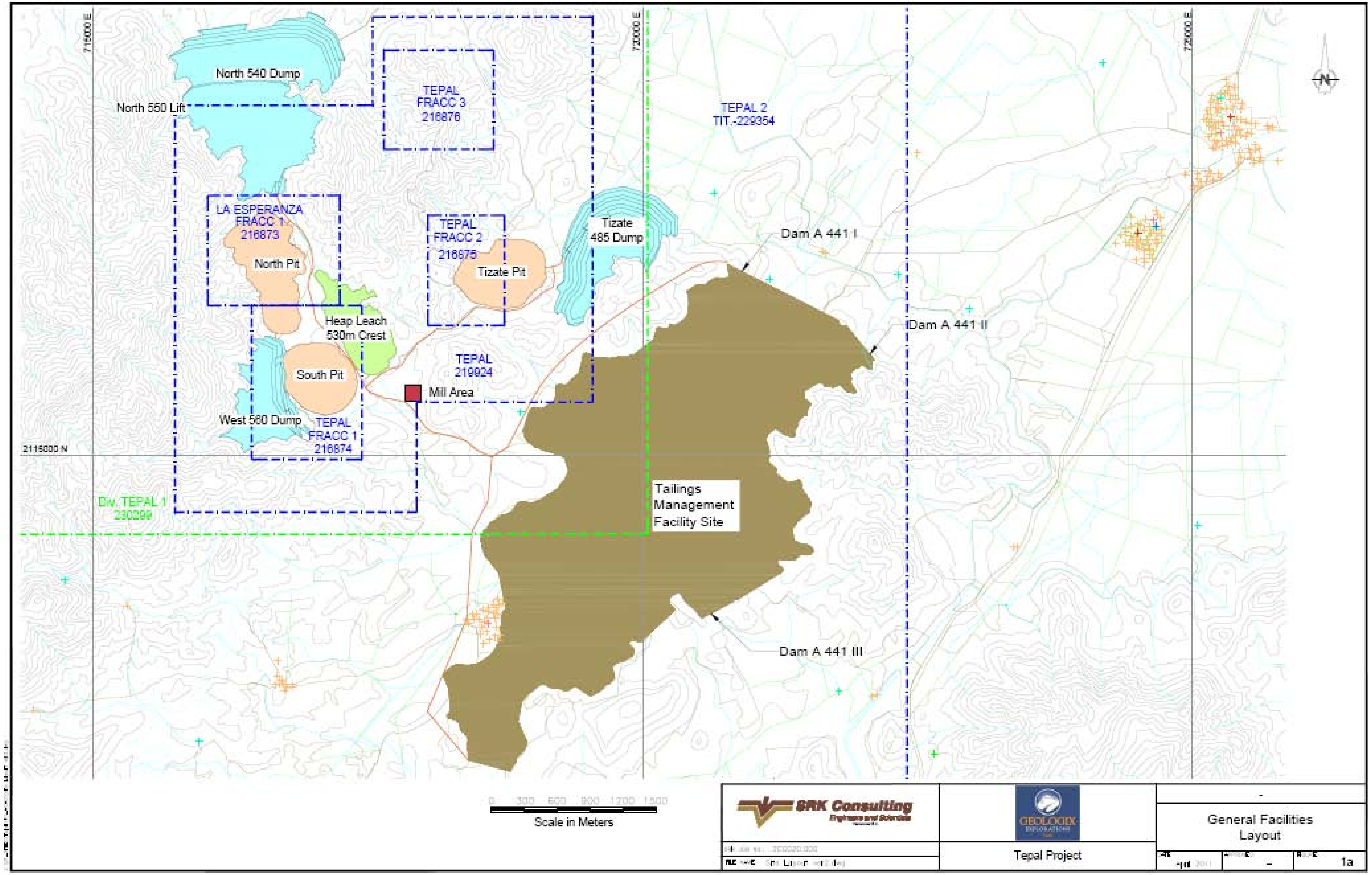

Figure 3.3: Planned Facilities Layout (SRK, 2010)

4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The following section is modified from Priesmeyer, 2007.

“Access to the property is good. The nearest town is Tepalcatepec, located 15.5 km to the northeast of the property. Tepalcatepec is reached by paved highway from Morelia, en-route from Mexico City. The final 7.5 km of access to the property are over unimproved dirt roads. Total driving time to Tepalcatepec from Morelia is about $3^{1/2}$ hours. Total driving time to the property from Tepalcatepec is about 30 minutes.

The climate of the region consists of a rainy season extending from June into October and a dry season extending from late November to May. Heavy rains during the rainy season can turn the dirt access roads to deep mud and produce wash outs making access difficult at times.

Average annual precipitation ranges from 500 mm to 700 mm. Daytime high temperatures range from 27°C in the December to February period to 38°C or 40°C in May and June. Average annual temperature is 28ºC to 30ºC.

The property lies in the steep hills on the northeast side of the Mexican Coastal Range at elevations between 500 m to 700 m. The elevation of the primary area of mineralization on the property ranges from around 550 m to around 650 m. Vegetation consists of thorny brush, small trees and occasional cactus.

The property is large enough but some topographically suitable locations for the development of facilities such as waste dumps and tailings disposal areas may be limited by the presence of mineralization, whose extent is presently unknown. Further study will be required to determine the suitability of the present land position for the development of all the mining-related facilities but at the present level of knowledge, the site appears to be adequate.

Tepalcatepec is the town nearest the property. It has a population of approximately 30,000. Services available in Tepalcatepec include lodging, a number of small restaurants, gasoline stations, a variety of small hardware, grocery, and retail stores, and an open air market, making it a suitable base for operations.

Apatzingán, located approximately 55 km east of Tepalcatepec, has a population of around 90,000. It is the closest town with scheduled air service and can be reached via daily commuter flights from Guadalajara.

Morelia is the capital of Michoacán State and has a population of around 550,000. There are daily air connections with Mexico City and the United States. Morelia is connected to the nation’s motorway, or highway system, with Guadalajara and Mexico City within half a day’s drive.

There is a three phase power line of unknown capacity located seven km east of the main mineralized area. There is also a power line of unknown capacity located 3 km north of the property. There is no power on the property.

There is, however, a major power substation located 2 km east of the town of Tepalcatepec and 14 km from the area of the mineral resources on the property. The Comisión Federal de Electricidad ("CFE"), the federal power authority in Mexico indicates that sufficient power is available to meet the needs of the project and a power line between the substation and the project could be constructed and power provided from the local electrical grid.

There are numerous reservoirs in the region. The water feeds a system of canals and is used primarily for irrigation purposes. Water may be available to the property from this reservoir system. If not, water appears to be shallow since it was encountered during both previous reverse-circulation programs (Personal Communication, Luis Gonzáles Barragán). There are a number of wells in the area of the project and the water table is generally located approximately 3 m below the surface.

The dominant land use on the property consists of cattle and goat grazing but sorghum and corn are raised in areas suitable for arable farming."

5 History

The following section is modified from Priesmeyer, 2007.

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") recognized the Tepal and the Tizate gossans (Tizate is located approximately 1,400 m east of the North Zone) and associated copper mineralization (Copper Cliff, 1974). INCO worked through its Mexican subsidiary DRACO although the sole surviving report from this time period was prepared by Copper Cliff. Limited data remains from the INCO period.

INCO explored the property during the period 1972-1974 by means of surface geochemistry, IP geophysics and drilling. INCO developed a small non NI 43-101 compliant resource of 27 Mt averaging 0.33% Cu and 0.65g/t Au but ultimately abandoned the property. INCO stressed that more drilling was required to further define the width of the mineralised zone.

Teck Resources Inc. ("Teck") acquired the property in late 1992. Work completed by Teck include geologic mapping, the collection of over 200 rock samples for multi-element analysis, the construction of more than 60 km of grid line, the collection of 1,268 soil samples and 50 rock chip samples from the grid, the construction of 15 km of access road and the completion of 50 reverse-circulation holes totalling 8,168 m in four phases of work. Teck also undertook some metallurgical testing, which is described in Section 12.2 of Priesmeyer (2007).

In 1994, Teck completed a non-NI 43-101 compliant resource estimate for the property. Results of the resource calculations are presented in Section 13.2 of the Priesmeyer report. The resource estimate is 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 is 78.82 Mt grading 0.4 g/t Au and 0.249% Cu with drill indicated resources totalling 55.84 million tonnes grading 0.514 g/t Au and 0.261% Cu. Of the 55.84 million tonnes drill indicated resource, 24.28 Mt averaging 0.545 g/t Au and 0.251% Cu are in the South Zone and 31.56 Mt averaging 0.489 g/t Au and 0.269% Cu are in the North Zone. It should be noted that the resource categories defined by Teck were drill indicated, drill inferred and projected do not directly correspond to the categories of mineral resources prescribed in NI43-101 but are broadly correlative with, measured, indicated and inferred resource categories as defined in CIM Definition Standards on Mineral Resources and Reserves (Canadian Institute of Mining, Metallurgy and Petroleum, 2005).

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

In late 1996, Minera Hecla S.A. de C.V. ("Hecla") visited the property and initiated a work program in the spring of 1997. 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 ship 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 reverse-circulation 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. Hecla's primary focus on the property was as a large tonnage, low-grade gold target. Although all samples were analyzed for copper and gold, Hecla did not include copper in its resource estimate.

The work completed by Hecla is the best documented of all previous work and is described in Section 8.3.1 of Priesmeyer (2007).

In 1997, Hecla completed a resource estimate for the property. The resource estimate is a polygonal block estimate based on manual definition of polygonal blocks on computer drafted drill sections using manual composited intercept intervals.

The results of the resource calculation for the North and South zones are detailed in Section 13.3 of the Priesmeyer report. The total resource for oxide and sulphide material is 9.063 Mt averaging 0.90 g/t Au and containing 262,359 ounces of gold. In addition to the resource for the North and South Zones, Hecla estimated a combined resource for the East and West Zones of 5.055 Mt averaging 0.36 g/t gold and containing 58,512 ounces of gold."

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

6 Geological Setting

The following section is excerpted from Priesmeyer (2007).

6.1 Regional Geology

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

The property lies just south of the Huacana Batholith (Figure 6.1), a Cretaceous to early-Tertiary batholith that ranges from quartz diorite to tonalite and granodiorite in composition.

The mineralized hyp-abyssal intrusions at the Tepal prospect are thought to be marginal phases of this batholith (Shonk, 1994).

6.2 Property Geology

Teck geologists identified three layered units and ten distinct intrusive rocks, some with multiple variations.

The layered units include a mixed unit of andesitic volcanics and interlayered volcanoclastic sediments, an andesitic to dacitic volcanic unit with minor interlayered volcanoclastic sediments (greywackes and siltstones) and a predominantly sedimentary unit of greywacke, shale, minor limestone and subordinate flows, tuffs and lahars.

Intrusive rocks on the property are only known north of a major east-northeast-trending fault on the southern part of the property. Nearly all fall in the tonalite/low-K dacite compositional range with the exception of post-mineralization and post-alteration andesite dikes. Intrusive rocks also display a wide variation in texture and phenocrysts abundance indicating diverse cooling histories and suggest multiple intrusive events and relatively high levels of emplacement. A detailed discussion of these lithologic units is presented in Shonk (1994).

Several inferred north-northwest-trending and east-northeast-trending faults cut the property dividing it into several parallelogram-like blocks. The southernmost east-northeast-trending fault separates two different domains of pre-intrusive rocks.

The rocks to the south form a homoclinal, south-dipping sequence which displays only weak thermal metamorphism, no alteration, and includes no intrusive rocks. North of the fault, the units are folded, faulted, more strongly thermally metamorphosed, and extensively intruded. The central north northwest-trending fault appears to juxtapose two different erosional levels and is parallel to a prominent structural grain seen in Landsat TM images of the property. The evidence for different erosional levels lies in the characteristics of the intrusive rocks. Intrusions east of the fault are typically large, equigranular, and medium-grained while porphyritic tonalite porphyry is virtually restricted to the western block south of the northern east-northeast-trending fault.

All of the defined resources are also located within this block. The deeper drilling in this area also shows a transition in the three small stocks in this area from tonalite porphyry and intrusion breccia near the surface to equigranular, medium grained tonalite at depth similar to those to the east of the fault. The presence of coarsely crystalline sericite north of the northern east-northeast-trending fault also supports the interpretation that deeper structural levels are exposed to the north and east.

Thermal metamorphism has converted andesitic volcanics to gray biotite hornfels and limestones to marbles and skarn peripheral to the intrusive rocks. Development of chlorite, clay, and carbonate in the volcanics and volcaniclastics may be due to weak regional metamorphism.”

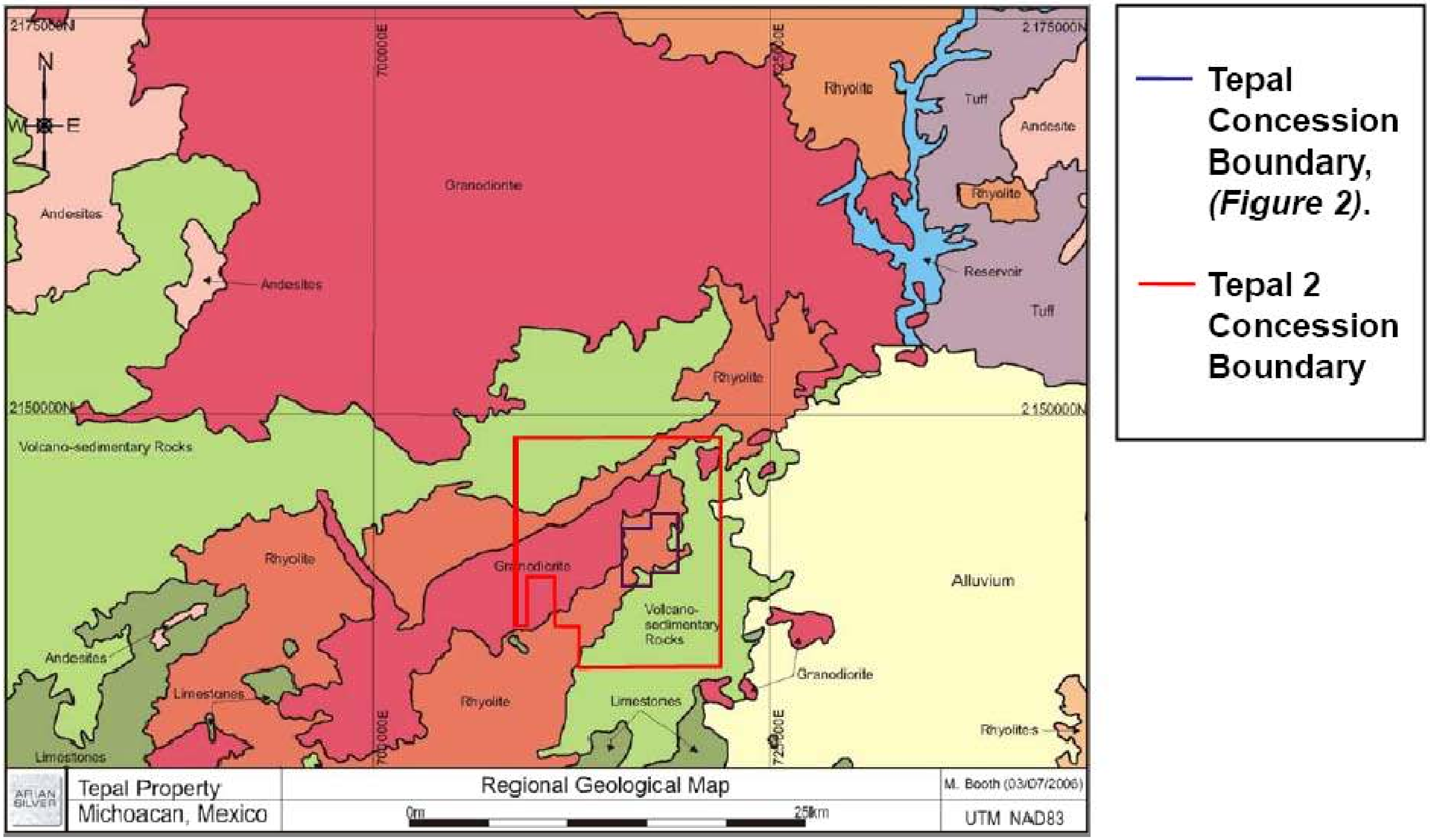

Figure 6.1: Geological Map of the Tepal Property Including Major Concession Boundaries (adapted from Priesmeyer, 2007)

7 Deposit Types

The following section is excerpted from Priesmeyer, 2007.

7.1 Deposit Type

“Mineralization on the property is characteristic of porphyry copper-gold mineralization. Porphyry-type depospits in Mexico occur in a nonrthwest 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, Sierraa Madre Occiendental and Sierra Madre de Sur covering the states of Sonora, Sinaloa, Chichuahua, Durango and Michoacan.

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 km2 in size. These altered zones are commonly coincident with shallow intrusives and/or dike swarms and hydrothermal or intrusion breccias. Deposit boundaries are determined by economic factors, which outline ore zones within larger areas of low-grade concentrically zoned mineralization.

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

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

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. The process results in a copper-poor leached cap lying above a relatively thin higher-grade zone of supergene enrichment that in turn overlies a thicker zone of lower grade primary hypogene mineralization at depth.

Porphyry systems may also exhibit hypogene enrichment. The process of hypogene enrichment may relate to the introduction of late hydrothermal copper-enriched fluids along structurally prepared pathways or the leaching and re-deposition of hypogene copper, or a combination of the two. Such enrichment processes result in elevated hypogene grades.

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 (< 100 ppm) 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, and;

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.

8 Mineralization

The following section is a modified excerpt from Priesmeyer (2007).

8.1 Mineralization

"Mineralization on the property consists of structurally controlled zones of stockwork and disseminated copper sulphide with elevated gold values. Mineralization occurs along a line of three small tonalite stocks just west of the north-northwest-trending fault that trends through the centre of the property. All three stocks are composed of multiple intrusive phases with tonalite porphyry and tonalite porphyry intrusion breccia phases hosting the highest grade mineralization. Most of the historic resource is hosted by these lithologies in the northern and southernmost stocks (North Zone and South Zone respectively). Both the North and South zone are crudely zoned from a gold-rich core with the highest gold and copper values and highest Au:Cu ratios to a copper dominant periphery with lower Au:Cu ratio to a barren pyritic halo (Shonk, 1994).

Mineralization within the Tizate deposit is similar to at Tepal but generally containing slightly lower gold and copper value, however, the Tizate deposit also contains molybdenum and silver mineralization in addition to gold and copper.

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).

The depth of oxidation ranges from 20 m 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. Thin supergene-enriched zones do exist locally at the base of the oxide zone and consist of chalcocite and covellite coatings on sulphide grains and local areas of poddy, massive chalcocite (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 later. Veins of all generations display a prominent 325o-350o orientation parallel to the central fault zone. Dips are generally vertical to steep either east or west. Other orientations are also present with a near east-west orientation and moderate south dip of secondary prominence.

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).

Mineralization on the property is consistently hosted by tonalite porphyry intrusions with at least the local presence of tonalite intrusion breccia showing chilled porphyritic to glassy porphyritic textures indicative of a near-surface environment. Intensity of mineralization is strongly related to the presence of late magmatic quartz and the density of late magmatic veining. The strong preferred orientation of these veins and evidence of shearing suggests development of a late magmatic age structure is required to focus mineralizing fluids.

Fracturing of the carapaces of the intrusive tonalite porphyritic units is likely related to continued movement on the north-northwest-trending structure controlling emplacement rather than volatile release (Shonk, 1994).

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. Inter-element relationships and zoning have not been systematically analyzed because all soil samples and most drill samples were only analyzed for Cu and Au. Cu and Au are strongly correlated with the highest Au:Cu ratios present in core of the North and South Zone resource areas. Au:Cu ratios appear to decline toward the periphery of these zones. Mo, Zn, and Ag are also elevated within the cores of the resource areas but the highest Zn and Ag values appear to occur on the periphery. The highest Pb and As values tend to occur in veins and mineralized structural zones outside of the resource areas. Sporadic high As values are most common in altered sediments (Shonk, 1994).

8.2 Alteration

Tonalities hosting the mineralized zones display alteration features typically associated with immature island arc-type porphyry systems. Potassic alteration is poorly developed and represented dominantly by secondary biotite when present. It is restricted to the core of the system in both the North and South Zones where it occurs as late magmatic biotite replacement of hornblende phenocrysts and in hydrothermal quartz-biotite-sulphide-magnetite veins. It is closely associated with copper-gold mineralization and the best grades.

Hydrothermal potassium feldspar is locally present but uncommon to rare. It occurs in quartz veins and after plagioclase. Hydrothermal amphibole has also been recognized. Both secondary biotite and amphibole are almost always strongly to completely chloritized.

The most visible and conspicuous alteration assemblage consists of sericite-pyrite-clay-silica/ quartz±tourmaline in veins and veinlets. This alteration assemblage is best developed in dacite volcanic rocks and domes adjacent to the mineralized zones and locally overprints mineralization.

Associated sericite-clay-pyrite alteration also affects post-mineralization dacite dikes which cut the North Zone, reflecting overprinting of this alteration on earlier alteration types.

Anomalous gold and copper values are often associated with this type of alteration but higher grade mineralization is absent. Associated quartz veins are generally uncommon but when present are typically pale gray and chalcedonic to cherty in appearance.

In the dacite unit, this alteration type is characterized by sparsely vegetated, red-brown to red colour outcrops of argillized rock as a consequence of supergene argillization due to oxidation of the 3-15% disseminated pyrite. Supergene minerals include kaolinite, illite, diaspore, pyrophyllite, and silica. Structurally controlled quartz-sericite-pyrite alteration is present locally elsewhere on the property.

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. Other alteration minerals sporadically associated with these assemblages include albite, calcite, epidote, clinozoisite, leucoxene, hematite, tourmaline, apatite, rutile and gypsum after anhydrite.

Whole rock analyses of altered and unaltered rocks available in the INCO data demonstrate significant addition of potassium associated with mineralization and alteration in spite of the scarcity of potassic alteration phases such as potassium feldspar or biotite. Potassium addition is probably reflected by the abundance of sericite.

Veinlets and replacements of quartz-chlorite-pyrite-epidote-calcite were noted in several drill holes peripheral to the South Zone and interpreted as peripheral to mineralization in location and post-mineralization in timing. This alteration type is associated with only very weakly anomalous gold and copper values. It often overprints assemblages more closely related to mineralization.

Chlorite-calcite-epidote with calcite and/or epidote veining or fracture coatings is the main alteration type in the post-mineralization andesite and diorite dikes. Propylitic alteration of this type is also pervasive in the andesitic volcanic rocks. It is probably related to regional, low grade metamorphism (Shonk, 1994)."

9 Exploration

The following section is a modified excerpt from Priesmeyer (2007).

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 North Zone) and associated copper mineralization (Copper Cliff, 1974).

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

9.2 Teck

Teck Resources Inc. (“Teck”) acquired the property in late 1992. Work completed by Teck includes 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 totalling 8,168 m in four phases. Total expenditure by Teck was approximately $875,000 (Shonk, 1994). Teck also completed metallurgical testing, which will be described in Section 12.2 of the Priesmeyer report.

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 Tech completed a soil sampling program. Grid lines were spaced 200 m apart and sample spacing was 100 m 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 Cu and Au and most rock chip samples were analyzed using multi-element Inductively-Coupled Plasma (“ICP”). According to Shonk (1994), values from both soil and rock samples showed a strong positive correlation.

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

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

9.3 Hecla

In late 1996 Minera Hecla S.A. de C.V. (”Hecla”) visited 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 ship 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 reverse-circulation 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 limited documentation for the resource estimate. Hand-written drill logs are also available. Most of the maps generated by Hecla remain, at least in electronic form. Sample splits and chip tray remain from the Hecla drilling. Four of the sample splits were resampled by Howe for grade verification purposes for the Report.

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 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 actuate copper anomaly with the concave portion of the anomaly open to the southwest. The anomaly is defined by copper values in excess of 301 ppm copper in rock.

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

The gold anomaly defined by Hecla is more restricted in aerial extent. The anomaly is defined by gold values in excess of 200 ppb or 0.2 g/t Au in rock and is open to the south and southeast. The anomaly trends 320^{o} and measures approximately 700 m by 215 m.

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

In order to confirm the analytical results from the Teck drilling, Hecla reanalyzed 298 pulps from Teck diamond drill holes T-9, T-13, T-23, T-24, T-25 and T-30. Results of the Hecla reanalysis indicated that the values obtained by Hecla were 7% higher than those obtained by Teck. Since Hecla’s primary focus was gold, Howe presumes that this difference is for gold values only.

9.4 Exploration by Arian

Exploration by Arian was initiated in April 2007. Exploration to date has consisted of the Tepal Phase 1 diamond drill program highlighted in the Section 10 Drilling.

9.5 Exploration by Geologix

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

By the end of the 1st quarter of 2010 the geophysical survey had been completed with a total of 78.4 line-kilometres 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 test for additional mineralization on the remainder of the property. Targets being evaluated in the latter areas are defined by geological, geochemical and geophysical anomalies as outlined in historic surveys as well as the geophysical survey completed by the Company in the 1st quarter of 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, 14 holes in the Tizate zone where no previous resources had been outlined, and two one other exploration targets on the property.

At the time of the report, drilling is continuing with two drill rigs but no drilling results have been reported since January 2011 and therefore the results are not considered further as part of the work for the PA. In addition in 2011 the Company has initiated detailed property geological mapping and silt sampling programs and is initiating an airborne geophysics survey that will include magnetics, radiometrics and EM.

10 Drilling

10.1 INCO Drilling

In early 1973 INCO drilled six diamond drill holes (57001 – 57006). Drilling continued through the winter of 1973-74 with seven widely spaced holes (57007 – 57013) on what was formerly known as the Tizate portion of the property and another seven holes (57014 – 57020 and 57026) were drilled on the Tepal gossan (Table 10.1). There is some discrepancy as to the number of holes drilled by INCO as collar details and assays are available for only 21 holes but according to Shonk (1994) it is possible that 26 diamond drill holes were actually completed. Howe has found nothing to support Shonk's contention that 26 holes were drilled by INCO.

Diamond drilling was conducted by Boyles Brothers drilling using a Longyear 38 core rig. Core was NX-sized (diameter = 54.7 mm) to 50 m and BX-sized (diameter = 42.0 mm) below 50 m. Sample interval for the INCO diamond drilling program ranged from 0.2 to 3.0 m but averaged about 2.0 m. This sampling length is acceptable when exploring for disseminated mineralization which, in this case, can reach thicknesses of over 50 m. The orientation of the mineralization is unknown as core was un-orientated.

INCO's drilling was confined to the North Zone and the Tizate area (Figure 10.1). The South Zone was unknown at the time. A summary of INCO drill hole results is presented below.

Table 10.1: Summary of INCO Diamond Drilling Results

DD Hole Number	Area	From (m)	To (m)	Thickness (m)	Au (g/t)	Cu (%)
57001	Tepal	0.0	11.4	11.4	0.19	0.51
		55.5	60.2	4.7	0.13	0.41
57002	Tepal	0.0	180.0	180.0	0.80	0.34
57003	Tepal	10.2	17.0	6.8	1.23	0.34
57004	Tepal	None				
57005	Tepal	20.0	40.4	20.4	0.47	0.41
57006	Tepal	None				
57007	Tepal	0.0	6.0	6.0	0.42	0.37
		24.0	36.0	12.0	0.45	0.14
		146.0	160.0	14.0	0.57	0.05
57008	Tizate	0.0	30.0	30.0	0.23	0.18
57009	Tizate	30.0	40.0	10.0	0.11	0.24
57010	Tizate	36.0	74.6	38.6	0.11	0.17
57011	Tizate	43.0	49.0	6.0	0.09	0.26
57012	Tizate	100.0	128.0	28.0	0.23	0.11
57013	Tizate	0.0	11.0	11.0	0.06	0.38
		20.2	32.0	11.8	0.43	2.30
57014	Tepal	0.0	12.0	12.0	0.23	0.24
57015	Tepal	0.0	112.0	112.0	0.68	0.38
		122.0	142.0	20.0	0.27	0.12
57016	Tepal	0.0	17.7	17.7	0.48	0.16
57017	Tepal	0.0	50.0	50.0	0.68	0.24
		96.0	108.0	12.0	0.25	0.18
57018	Tepal	None				
57019	Tepal	0.0	68.2	68.2	0.17	0.27
57020	Tepal	21.0	150.0	129.0	0.55	0.30
57026	Tepal	194.0	200.1	6.1	0.47	0.40

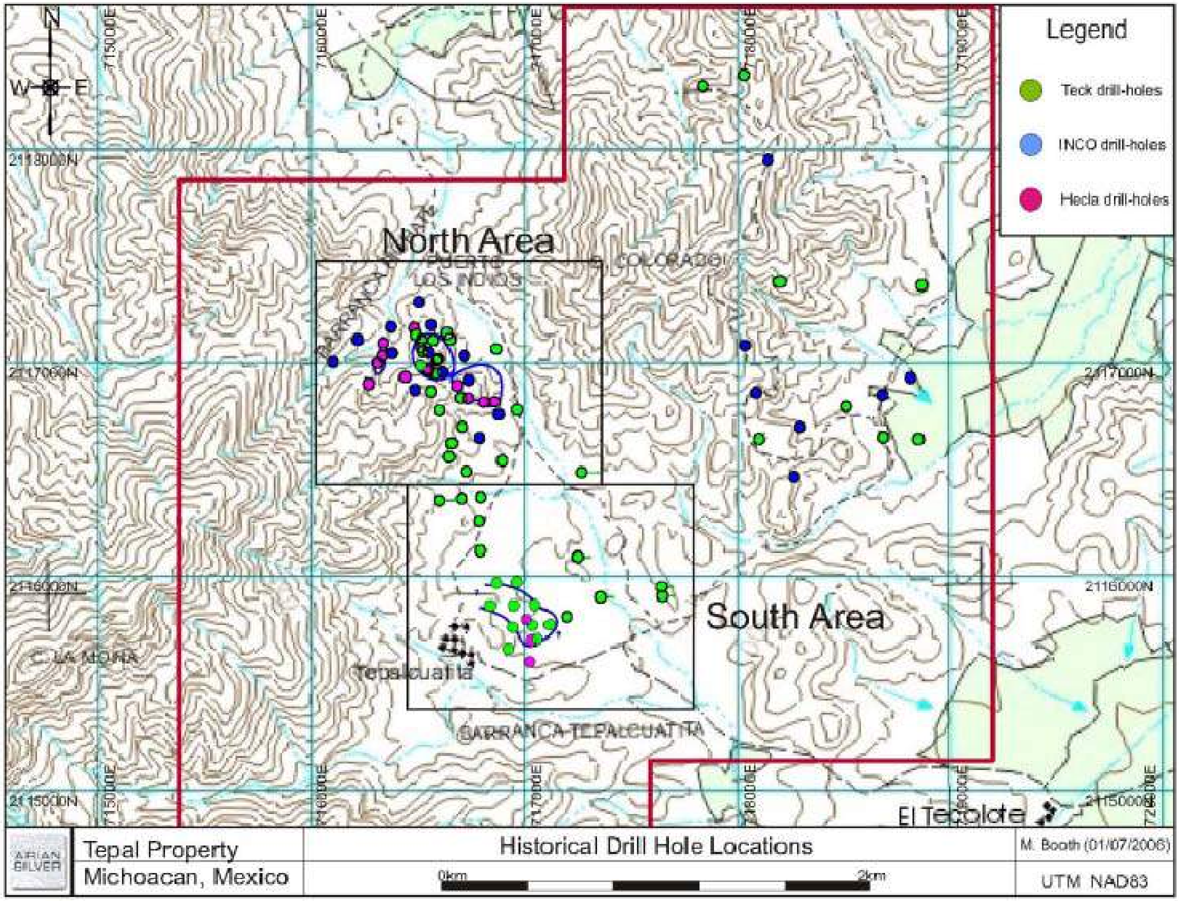

Figure 10.1: Tepal Historical Drill Plan

10.2 Teck Drilling

In 1994 Teck drilled 50 reverse-circulation (“RC”) drill holes totalling 8,168.8 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 (Figure 10.1). A differential GPS survey was conducted in late January, 1994 to locate the INCO holes and the first 24 Teck holes as well as roads, key grid points, concession monuments and planned drill holes. Compass and tape surveys were used to establish coordinates of later drill holes and map access roads constructed after the survey.

Samples were collected every 2.03 metres (3 per 20-foot drill rod) for the first 24 holes and every 1.52 metres (5 ft intervals) for holes T-25 through T-50. This is acceptable when exploring for disseminated mineralization which, in this case, can reach thicknesses of over 50 m. The orientation of the mineralization is unknown due to the nature of the drilling.

A duplicate analytical sample was collected every tenth sample interval. All drill samples were analyzed for Cu and Au at Chemex (now ALS Chemex). An additional 123 samples from selected intervals were analyzed for Ag, Co, Cu, Fe, Mn, Mo, Ni, Pb, and Zn using a multi-element ICP procedure. Given the fact that mineralization is disseminated or stockwork-controlled, this sample interval is adequate. Results are summarized in Table 10.2.

Drilling at Tepal generally indicates that the best values are present within 150 m of the surface. Significant intercepts at greater depths are 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 hole IN57002.

Table 10.2: Summary of Teck Reverse Circulation Drilling Results

RC Hole Number	Area	From (m)	To (m)	Thickness (m)	Au (g/t)	Cu (%)
T-1	North	20	80	60	0.2	0.15
		184	190	6	0.19	0.27
T-2	West	6	68	62	0.17	0.46
		88	106	18	0.33	0.23
T-3	North	0	156	156	0.83	0.33
		188	194	6	1.46	0.17
T-4	North	0	116	116	0.56	0.28
incl.		42	98	56	0.95	0.37
T-5	East	6	26	20	0.18	0.47
T-6	North	0	36	36	0.36	0.22
		80	112	32	0.57	0.22
T-7	Between north and south	117	198	86	0.32	0.14
T-8		0	26	26	0.44	0.15
		54	70	16	0.46	0.14
T-9		44	154	110	0.4	0.16
T-10	None	6	26	20	0.46	0.22
		82	130	46	0.65	0.25
T-11	Between north and south	16	42	26	0.41	0.25
T-12		42	96	54	0.47	0.2
T-13		24	78	54	0.47	0.18
T-14		NIL				
T-15	South	0	28	28	0.4	0.26
T-16	South	44	166	120	0.44	0.2
T-17	South	0	116	116	0.69	0.3
T-18	South	0	164	164	0.76	0.27
T-19	East	NIL				
T-20	East	NIL				
T-21	North	NIL				
T-22		NIL				
T-23	North	0	44	44	0.67	0.53
		56	122	66	0.28	0.22
T-24	North	0	188	188	1.04	0.4
T-25	South	4.6	199.6	195	0.82	0.3
T-26	South	7.6	86.9	79.3	0.34	0.15
		100.6	161.5	60.9	0.42	0.2
		172.2	201.2	29	0.66	0.32
T-27	South	0	32	32	0.24	0.18
T-28	South	0	36.6	36.6	0.67	0.21
		61	70.1	9.1	0.28	0.19
T-29	None	1.5	9.1	7.6	0.35	0.03
T-30	North	0	182.8	182.8	0.79	0.25
incl.		25.9	65.5	35.6	1.35	0.31
T-31	North	30.5	39.6	9.1	0.22	0.44
		96	112.8	16.8	0.25	0.24
		143.3	153.9	10.6	0.26	0.48
T-32	North	59.4	83.8	24.4	0.2	0.24
		108.2	112.8	4.6	0.23	0.45
		155.5	170.7	15.2	0.23	0.2
T-33	Between north	NIL				
T-34		54.9	112.8	57.9	0.29	0.44

10.3 Hecla Drilling

In late 1997, Hecla conducted a 17-hole reverse-circulation (“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. Results are presented in Table 10.3, which was taken from Gómez-Tagle (1998).

Sample interval for the Hecla reverse-circulation drilling program was 1.0 m. This is acceptable when exploring for disseminated mineralization which, in this case, can reach thicknesses of over 50 m. The orientation of the mineralization is unknown.

Table 10.3: Summary of Hecla Reverse Circulation Drilling Results

RC Hole Number	Interval (m)		Type of Mineral	Grade	Grade	Subinterval (m)		Thickness (m)	Grade	Grade
	From	To		Au (g/t)	Cu (%)	From	To		Au (g/t)	Cu (%)
MHT-5	67	182	S	0.88	0.32	67	71	4	1.4	0.39
						75	97	22	1.39	0.39
	128	150	S	0.09	0.04					
	1	27	O	1.24	0.47	17	25	8	2.05	0.56
	27	30	M	1.1	1.02					
	30	108	S	0.78	0.44	30	44	14	1.04	0.52
						53	61	8	1.56	0.96
						76	81	5	1.04	0.51
						98	108	10	0.88	0.39
	108	150	S	0.17	0.12					
MHT-6	1	42	O	0.67	0.2	15	23	8	0.96	0.23
	42	59	M	0.26	0.37	46	53	7	0.51	0.58
	59	150	S	0.23	0.14	80	114	34	0.44	0.16
MHT-7	1	14	O	0.19	0.48	1	4	3	0.44	0.18
	14	16	O	0.18	0.48	1	4	3	0.44	0.18
	16	38	S	0.27	0.15					
	38	51	S	0.18	0.12					
MHT-8	0	13	O	0.41	0.09					
	13	16	M	0.37	0.82					
	16	51	S	0.24	0.23	16	23	7	0.33	0.44
MHT-9	0	14	O	0.45	0.07					
	14	15	M	0.3	0.64					
	15	50	S	0.21	0.22	15	27	12	0.33	0.37
MHT-10	0	10	M	0.03	0.03					
	10	51	S	0.03	0.02					
MHT-11	0	12	O	0.05	0.01					
	12	31	M	0.04	0.01					
	31	51	S	0.03	0.03					
	51	81	S	0.4	0.2	77	81	4	0.67	0.28
MHT-12	0	30	O	0.13	0.17					
	30	32	M	2	0.19					
	32	80	S	0.21	0.23	41	54	13	0.41	0.25
MHT-13	0	29	O	0.35	0.12	14	29	15	0.48	0.12
	29	35	M	0.56	0.31					
	35	50	S	0.45	0.51	38	50	12	0.49	0.38
MHT-14	0	24	O	0.18	0.2					
	24	26	M	0.1	0.34					
	26	50	S	0.13	0.08					
MHT-15	0	33	O	0.31	0.93	6	11	5	0.44	0.39
						13	18	5	0.52	0.59
						28	32	4	0.29	2.75
	33	41	M	0.11	1.05					
	41	51	S	0.07	0.21					
MHT-16	0	19	O	0.45	0.1	0	4	4	0.54	0.06
						6	17	11	0.49	0.11
	19	20	M	0.54	0.43					
	20	50	S	0.43	0.23	26	36	10	0.64	0.32
						45	50	5	0.43	0.24
MHT-17	8	19	O	0.02	0.11					
	19	21	M	0.01	0.7					
	21	50	S	0	0.05					

O = oxide, M = mixed oxide/sulphide, S = sulphide

10.4 Arian Drilling

The Phase 1 diamond drilling (“DD”) campaign was completed in June 2008, consisting of 42 holes totalling 7,180 m. See Figures 10.2-10.4.

Drilling has been carried out using two Boart Longyear 38 drill rigs owned and operated by GICSA (Geotechnica, Igenieria y Construction, S.A. de C.V.), of Paseos de Taxquena, Mexico, D.F.

The majority of the initial diamond drilling was carried out using HQ sized drill rods (core diameter = 63.5 mm) except where, due to technical problems, the rod size was reduced to NQ (core diameter = 47.6 mm). Drill core was not oriented for the Phase 1 program.

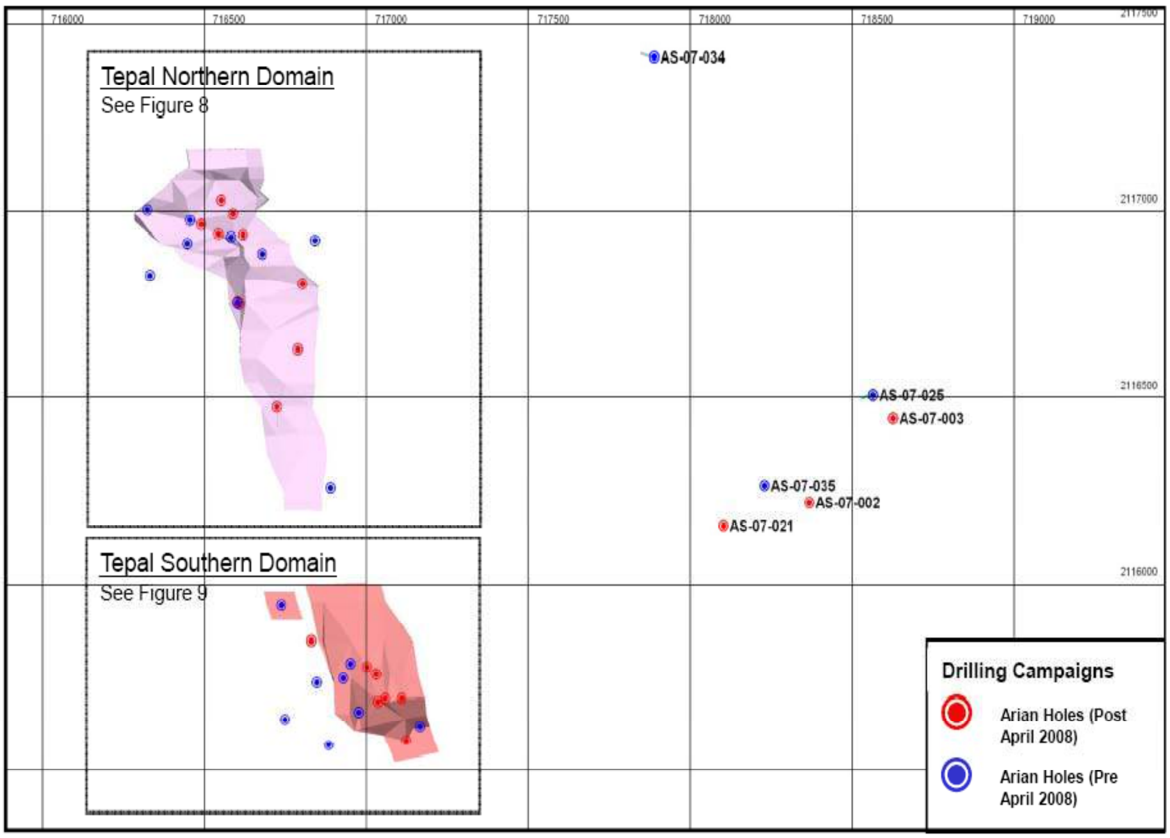

Figure 10.2: Location Plan – All Arian Phase 1 Drill Holes and Mineralized Domains

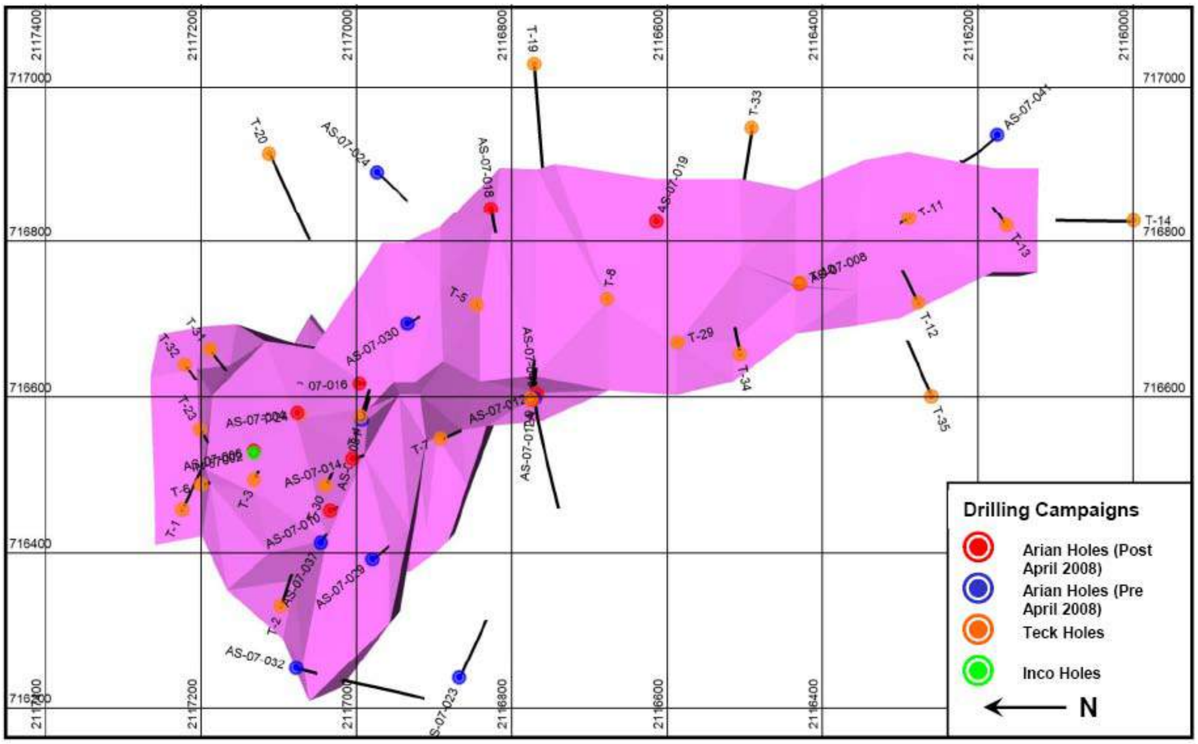

Figure 10.3: Location Plan – All Northern Domain Drill Holes

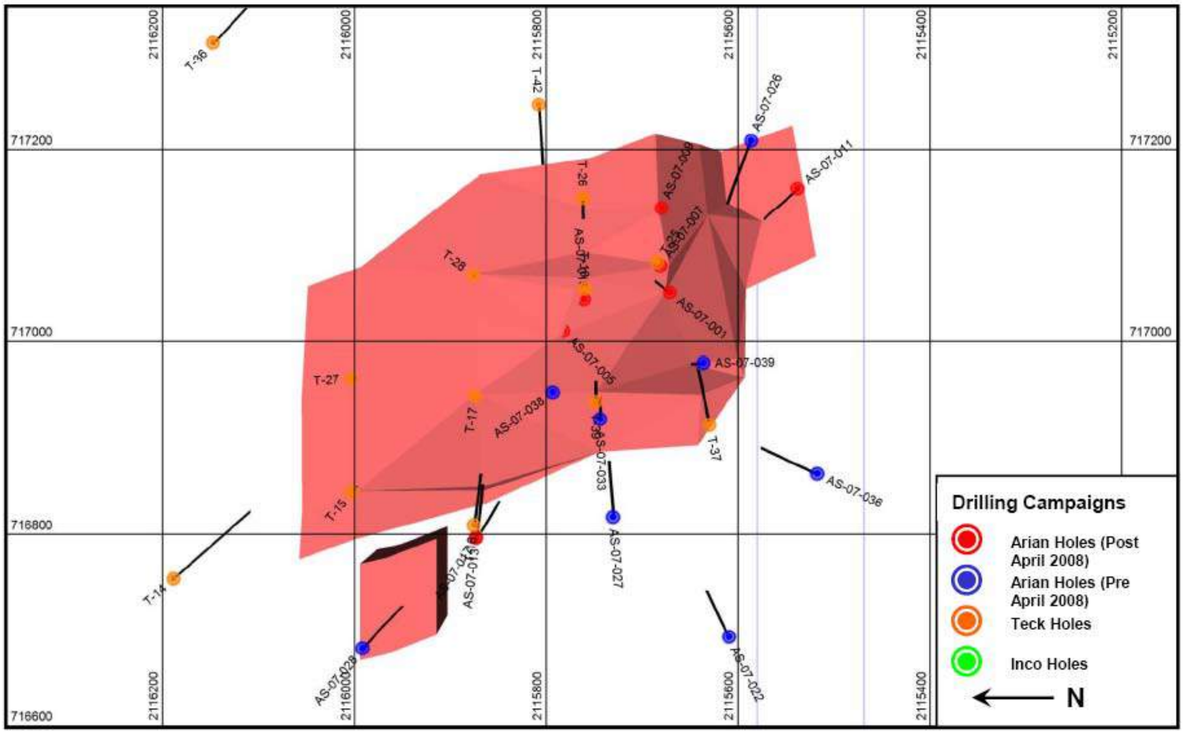

Figure 10.4: Hole Location Plan – All Southern Domain Drill Holes

10.5 Drill Hole Summary

Summary details of Arian drill hole data for the Tepal project are contained in Table 10.4.

Table 10.4: Arian Tepal Drill Hole Summary

Database Name	Micromine Tepal Drill Hole DH Database		
Date Created	February 2008		
Number of Holes	42		
Average Hole Spacing	150-170m x 50-100m within mineralized zones		
DD Hole ID	Depth (m)	Hole Azimuth	Hole Dip (Collar)
AS-07-001	200.1	045	-45
AS-07-002	151.45	000	-90
AS-07-003	101.65	000	-90
AS-07-004	200.4	000	-90
AS-07-005	150.9	045	-45
AS-07-006	200.85	000	-90
AS-07-007	250.05	000	-90
AS-07-008	152.75	000	-90
AS-07-009	150.7	000	-90
AS-07-010	100.3	000	-90
AS-07-011	151.3	000	-90
AS-07-012	60.1	000	-90
AS-07-012a	165.85	000	-50
AS-07-013	185.8	000	-50
AS-07-014	201.65	000	-90
AS-07-015	180.65	270	-80
AS-07-016	151.4	000	-90
AS-07-017	201.4	000	-90
AS-07-018	75.9	270	-45
AS-07-019	75.4	000	-90
AS-07-020	75.35	000	-90
AS-07-021	101	000	-90
AS-07-022	150.25	000	-90
AS-07-023	200.6	000	-90
AS-07-024	150.35	000	-90
AS-07-025	161	000	-90
AS-07-026	250.1	270	-80
AS-07-027	172.95	090	-80
AS-07-028	201.1	000	-90
AS-07-029	201	000	-90
AS-07-030	151.3	140	-45
AS-07-031	200.55	090	-50
AS-07-032	200.1	220	-45

10.6 Geologix Drilling

As of January 1, 2011, Geologix had carried out a diamond drilling program totalling 10,656 m on the Tepal property. The drill program was started on June 16, 2010 and utilized two diamond drilling machines. The purpose of the drill program was to evaluate the “near resource” potential for additional mineralization located near the Arian Silver/ACA Howe resource outlines and test for additional mineralization on the remainder of the property. No drilling was completed within the resource limits.

In total, Geologix has drilled 42 core holes on the property, including 26 holes targeted around the Tepal zone, 15 holes were targeted at the Tizate zone and two holes tested exploration targets in the area between Tepal and Tizate (Figure 10.5).

At the time of the report, drilling is continuing with two drill rigs but no drilling results had been reported since January 2011 and therefore the results are not considered further as part of the work required in the PA and is only mentioned for informational purposes only.

Table 10.5 Significant drill hole intersections for the Tepal deposit

Hole ID	Zone	From	To	INT (m)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (%)
TEP-10-004	North	0.00	149.00	149.00	0.18	0.12	2.5	0.004
inc.	North	0.00	86.50	86.50	0.25	0.13	3.4	0.004
inc.	North	0.00	31.00	31.00	0.36	0.15	7.8	0.004
inc.	North	49.00	86.50	37.50	0.27	0.17	1.2	0.005
inc.	North	141.70	149.00	7.30	0.29	0.29	4.1	0.005
TEP-10-006	North	0.00	35.00	35.00	0.16	0.12	0.8	0.003
TEP-10-006	North	47.00	75.00	28.00	0.21	0.18	0.9	0.002
TEP-10-006	North	107.00	169.00	62.00	0.51	0.25	0.5	0.001
TEP-10-007	North	0.00	28.00	28.00	0.02	0.34	0.4	NSV
TEP-10-012	North	11.00	22.00	11.00	0.12	0.21	1.1	0.015
TEP-10-015	North	152.00	176.30	24.30	0.16	0	17.0	0.002
TEP-10-017	North	108.35	146.45	38.10	0.18	0.18	1.9	0.002
TEP-10-019	North	63.00	175.20	112.20	0.35	0.21	1.8	0.001
TEP-10-019	North	67.00	95.00	28.00	0.67	0.23	2.9	0.001
TEP-10-019	North	128.00	157.05	29.05	0.41	0.30	1.1	0.002
TEP-10-020	South	159.00	177.00	18.00	0.33	0.20	5.4	0.005
TEP-10-020	South	191.00	286.00	95.00	1.18	0.39	1.3	0.002
inc.	South	217.00	286.00	69.00	1.43	0.45	1.4	0.002
TEP-10-022	South	64.00	138.00	74.00	0.25	0.16	0.6	0.002
TEP-10-022	South	160.00	172.70	12.70	0.26	0.19	0.4	0.002
TEP-10-024	North	57.85	84.00	26.15	0.22	0.17	1.1	0.002
TEP-10-024	North	101.00	143.00	42.00	0.29	0.24	3.3	0.005
TEP-10-026	North	192.00	200.00	8.00	0.11	0.12	0.9	0.008
TEP-10-026	North	211.35	229.00	17.65	0.15	0.10	1.0	0.001
TEP-10-027	North	0.00	86.00	86.00	0.37	0.26	3.9	0.002
TEP-10-027	North	104.95	137.00	32.05	0.39	0.25	1.0	0.001
TEP-10-027	North	166.00	205.00	39.00	0.40	0.39	2.6	0.004
TEP-10-028	North	2.00	28.00	26.00	0.23	0.20	0.5	0.003
TEP-10-030	North	164.00	168.00	4.00	0.14	1775.00	0.8	0.003
TEP-10-030	North	230.00	259.00	29.00	0.21	1184.20	0.7	0.001

Table 10.6 Significant drill hole intersections for the Tizate deposit

Hole ID	Zone	From	To	INT (m)	Au g/t	Cu %	Ag g/t	Mo %
TEP-10-005	Tizate	59.00	123.00	64.00	0.21	0.18	1.5	0.009
TEP-10-005	Tizate	160.40	199.00	38.60	0.22	0.23	2.2	0.010
TEP-10-008	Tizate	20.2	32.5	12.2	0.20	0.2	1.5	0.006
TEP-10-008	Tizate	56.00	136.00	80.00	0.18	0.24	3.6	0.008
inc.	Tizate	56.00	118.00	62.00	0.22	0.28	3.8	0.009
and	Tizate	84.00	118.00	34.00	0.26	0.33	5.0	0.01
TEP-10-009	Tizate	37.00	202.70	165.70	0.36	0.16	1.0	0.003
inc.	Tizate	37.00	159.95	122.95	0.37	0.16	1.1	0.003
inc.	Tizate	43.00	63.00	20.00	0.60	0.23	2.0	0.003
and	Tizate	173.00	201.00	28.00	0.53	0.20	1.1	0.003
TEP-10-029	Tizate	0.00	115.00	115.00	0.19	0.20	1.6	85
TEP-10-029	Tizate	232.00	236.00	4.00	0.60	0.16	5.9	9
TEP-10-031	Tizate	0.00	33.00	33.00	0.41	0.24	3.1	64
TEP-10-032	Tizate	23.40	66.00	42.6	0.07	0.23	5.1	179
TEP-10-033	Tizate	10.00	126.10	116.1	0.18	0.16	1.5	61
TEP-10-033	Tizate	145.00	389.25	244.25	0.11	0.19	4.2	69
TEP-10-034	Tizate	0.00	33.00	33.00	0.24	0.23	2.4	58
TEP-10-034	Tizate	89.00	113.00	24.00	0.06	0.19	5.9	12
TEP-10-034	Tizate	191.00	200.65	9.65	0.56	0.06	5.4	2
TEP-10-034	Tizate	313.80	324.70	10.90	0.44	0.05	8.1	1
TEP-10-034	Tizate	339.00	345.00	6.00	0.47	0.05	5.1	2
TEP-10-034	Tizate	369.00	379.45	10.45	2.50	0.14	7.3	5
TEP-10-036	Tizate	2.00	114.20	112.20	0.36	0.22	2.4	35
TEP-10-036	Tizate	140.20	148.00	7.80	0.26	0.18	1.2	34
TEP-10-036	Tizate	176.00	186.00	10.00	0.23	0.12	1.3	28
TEP-10-036	Tizate	219.00	269.15	50.15	0.24	0.18	1.6	107
TEP-10-037A	Tizate	75.00	108.00	33.00	0.12	0.12	1.5	20
TEP-10-037A	Tizate	129.20	241.95	121.75	0.17	0.15	2.7	47
TEP-10-037A	Tizate	342.00	350.00	8.00	0.12	0.15	1.3	68
TEP-10-038	Tizate	2.00	9.00	7.00	0.11	0.22	3.9	24
TEP-10-038	Tizate	29.00	132.00	103.00	0.17	0.19	1.1	71
TEP-10-039	Tizate	0.00	98.60	98.60	0.28	0.23	2.9	90
TEP-10-040	Tizate	2.20	71.50	68.95	0.20	0.25	0.53	102.00

The second phase of drilling encompassed about 5,000 m and was targeted at expanding the known resource at Tepal and Tizate. In total, between he two phases of drilling, Geologix has drilled 42 core holes on the property, 14 holes were targeted at the Tizate zone, 26 holes were targeted at the Tepal zone and three holes were exploration holes targeting the area between Tepal and Tizate (Figure 10.5).

At the time of the report, drilling is continuing with two drill rigs but no drilling results had been reported since January 2011 and therefore the results are not considered further as part of the work required in the PA and is only mentioned for informational purposes only.

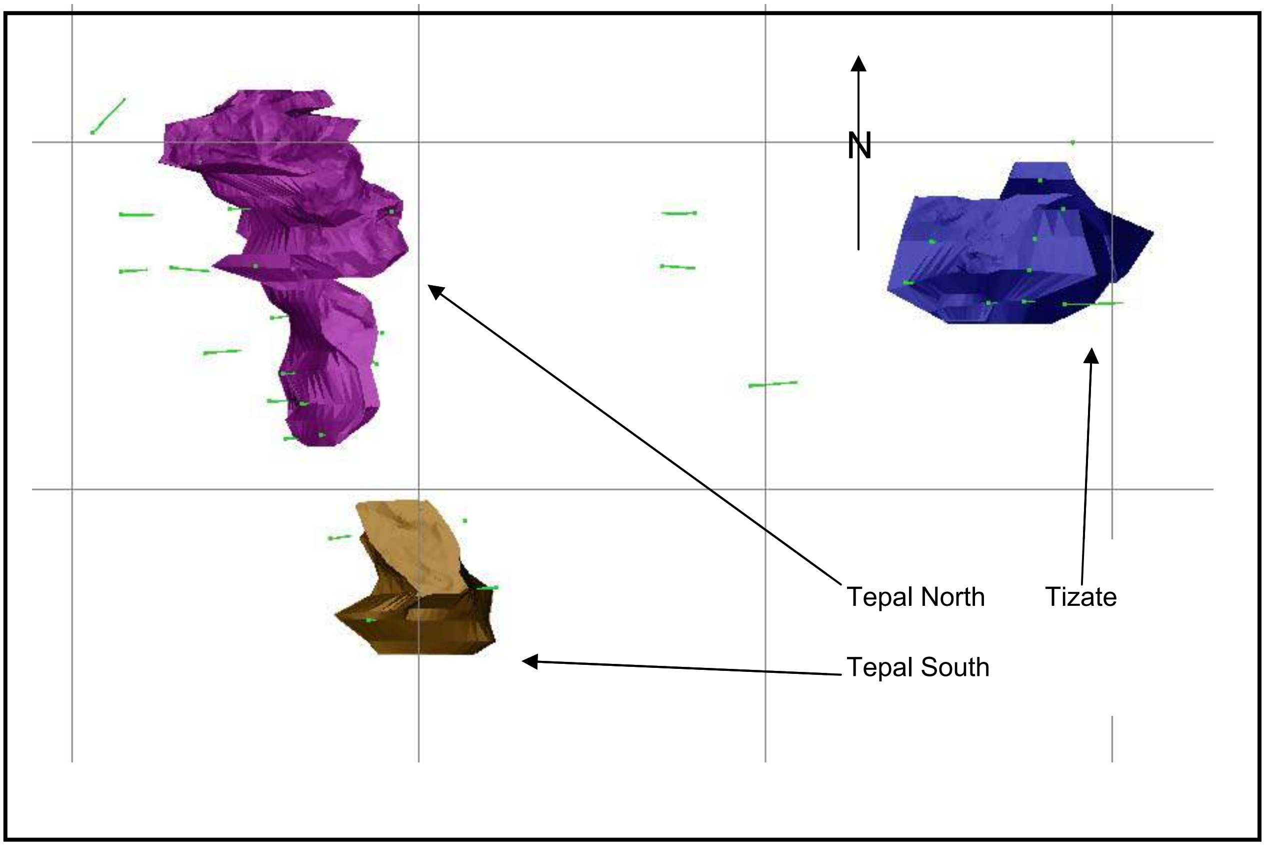

Figure 10.5 Location of Geologix 2010 Drilling

Note: grid is 1,000 by 1,000 m

11 Sampling Method and Approach

The following section is a modified excerpt from Priesmeyer (2007).

11.1 INCO Program

Little is known of the sampling method and approach employed by INCO for their soil and rock sampling programs. Soil samples were collected on a grid. Sampling methodologies are not discussed in the Copper Cliff report (Copper Cliff, 1973).

Sample interval for the INCO diamond drilling program ranged from 0.2 m to 3.0 m but averaged about 2.0 m. Diamond drill core was NX size (diameter = 54.7 mm) to 50 m and BX size (diameter = 42.0 mm) below 50 m. It is not known whether drill core was split, and if so how it was split, or whether whole core was analyzed. Core recoveries ranged from over 90% in un-weathered rock to between 40 to 90% in fractured rock. Without a detailed study it is difficult to determine the impact of low recovery on the validity of assay results although, in theory, the results could be affected. No core, duplicate samples, coarse rejects or sample pulps from the INCO drilling remain.

11.2 Teck Program

Little is known of the sampling method and approach employed by Teck for their soil and rock sampling programs. Rock samples were collected as part of Teck's property-wide mapping program. Presumably these samples were rock chip samples, rather than channel samples, collected from outcrops of interest around the property.

Soil samples were collected on a grid as discussed in Section 8.2 of the Priesmeyer Report. The grid covered most of the property. Sampling methodology is not discussed in the Teck report (Shonk, 1994).

Samples from the reverse-circulation program were collected every 2.03 metres (3 per 20" drill rod) for the first 24 holes and every 1.52 m (5-foot intervals) for holes T-25 through T-50. A duplicate analytical sample was collected every tenth sample. Recovery was not recorded on Teck drill logs. Property owner Luis Gonzáles Barragán (personal communication, 2006) indicated that Teck encountered problems when trying to drill below the water table with reverse-circulation drilling. This may have affected the recovery of drill cuttings and the results. Sample pulps from Teck's reverse-circulation drilling program have been preserved and are in Tepalcatepec.

11.3 Hecla Program

Little is known of the sampling method and approach employed by Hecla. A rock chip sampling program was completed by Hecla but Hecla did not collect soil samples. A total of 885 rock chip samples were collected from road cuts, trenches and the aforementioned grid.

In order to collect representative samples from the grid, samples were collected from outcrops within an area of five or ten metres surrounding each samples point (Figures 11.1 and 11.2).

Samples from reverse-circulation drilling were collected every meter down the hole. A duplicate analytical sample, or a split of the main sample, was collected from every sample interval. These duplicate samples have been preserved and are in Tepalcatepec. Recoveries were not recorded. Property owner Luis Gonzáles Barragán (personal communication, 2006) indicated the Hecla encountered problems when trying to drill below the water table with reverse-circulation drilling. This may have affected the recovery of drill cuttings and the results. Chip trays containing representative lithological samples for logging purposes are have also been preserved and are in Tepalcatepec.

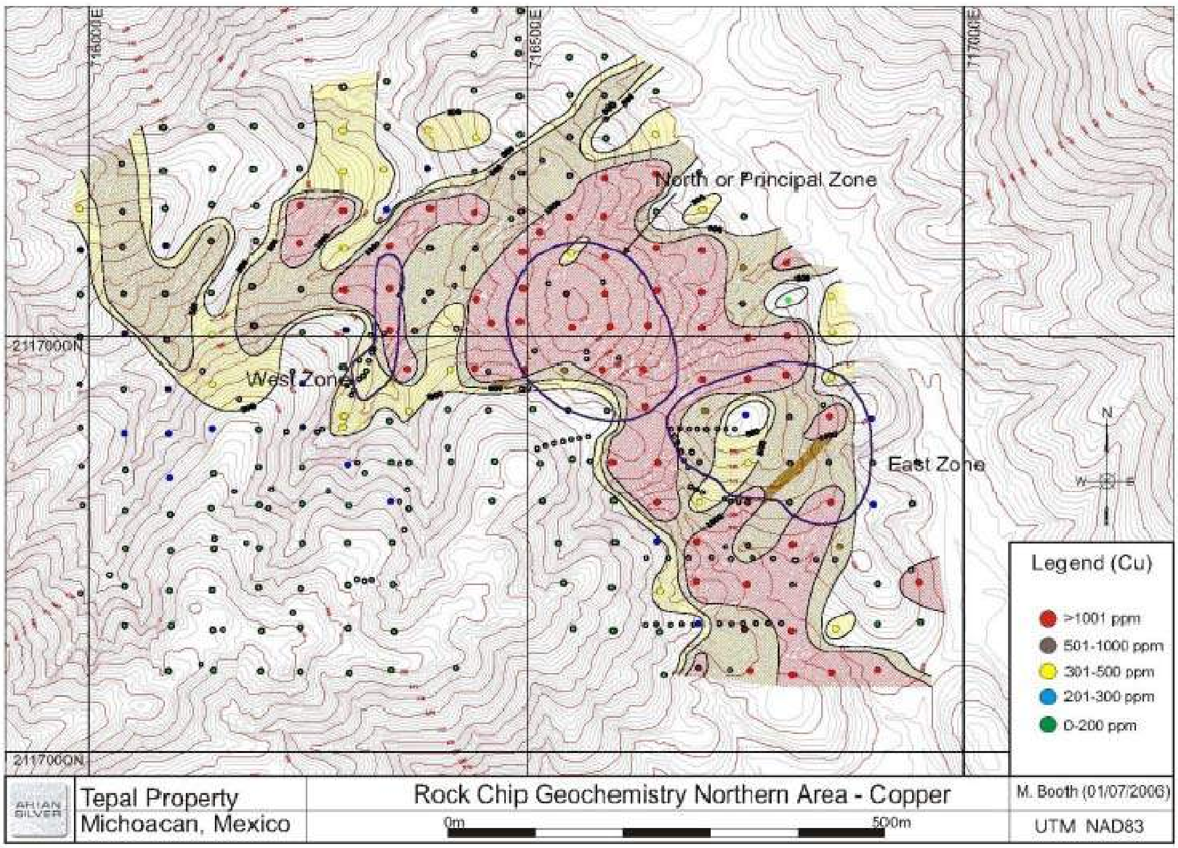

Figure 11.1: Hecla Rock Chip Cu Geochemistry Map for Tepal North Zone

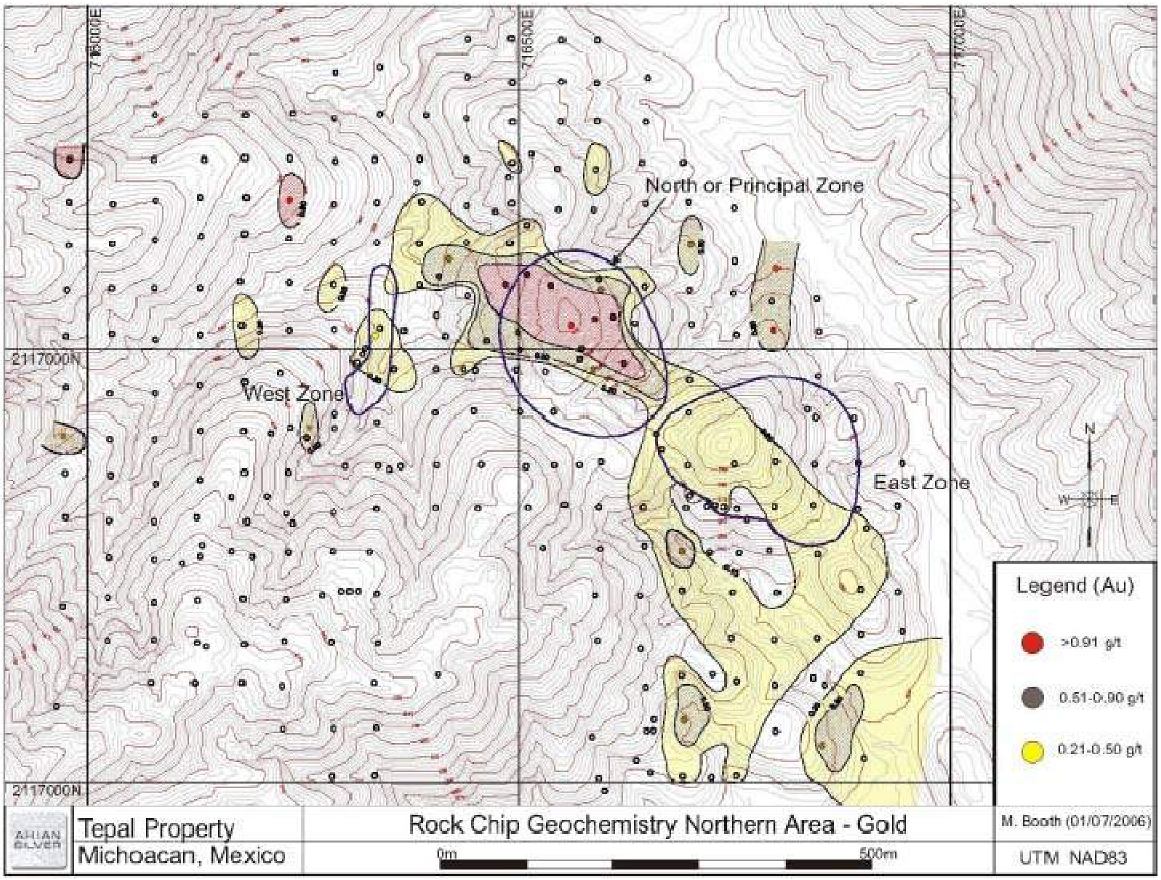

Figure 11.2: Hecla Rock Chip Au Geochemistry Map for Tepal North Zone

11.4 Arian Program

Procedures for the Tepal drill hole sampling method and approach are similar to those employed at Arian's San Jose property near Zacatecas, and taken from discussions with Arian staff geologists Mr. M. Booth and Mr. H. Parker and from internal documents 'San Jose –Sampling Methodology and QA/QC.doc' and 'San Jose Exploration by Arian.doc' provided to Howe for review. Arian's QA/QC and sampling methodology and procedures were developed following Howe's recommendations in the previous technical study for the project reported in Priesmeyer (2007).

HQ drill core is retrieved in approximate 2.4 m runs where possible and 3.05 m runs for NQ core. Run length is less where broken ground is encountered.

All drill-core was stored in plastic core boxes (with lids) that were able to hold 3m of core. The plastic core boxes were transported (by Arian personnel) with a large elastic band wrapped around them so to prevent the lids from blowing away when they were being transported (Booth, 2007a).

Drill-core was collected from the drill-rig(s) at the end of each day. The core was transported by Arian to the logging shed for storage, where it was cleaned and marked up (highlighting lithological and structural features), and then it was photographed. The photographs were saved, every day onto a computer at the property.

Once the core was photographed, it was logged, with geology, recovery, and RQD information noted on the logs and entered into an Access database on a daily basis (Booth, 2007b).

Where applicable, samples were marked on the core box, with a red mark, and the sample number recorded on the logs and inside the core boxes next to the relevant sample point. An aluminum ticket, on which the sample number was written, was also placed into the core box at the relevant position. The sample information was also entered in the access database.

Once a week, the Access databases are saved on the company's network in the Zacatecas office. The network is backed-up monthly on DVD which is stored in a safe location (Booth, 2007b).

Once the core was photographed, it was logged, with geology, recovery, and RQD information noted on the logs and entered into an Access database on a daily basis (Booth, 2007b).

Where applicable, samples were marked on the core box, with a red mark, and the sample number recorded on the logs and inside the core boxes next to the relevant sample point. An aluminum ticket, on which the sample number was written, was also placed into the core box at the relevant position. The sample information was also entered in the access database.

Once a week, the Access databases are saved on the company's network in the Zacatecas office. The network is backed-up monthly on DVD which is stored in a safe location (Booth, 2007b).

11.5 Geological Core Logging

Discussion with site personnel and a review of geological logging procedures and log sheets indicates that detailed geological logging was routinely undertaken during drilling.

Observations are recorded on hardcopy graphical logging sheets and capture pertinent geological information for each deposit including lithology, weathering, facies, texture, structure, mineralogy, colour, and grain size as well as presenting a graphic log. Site specific information such as relevant ore types and alteration assemblage characteristics are being recorded. Based upon review of the logs SRK is satisfied the logging is consistent and conducted to a satisfactory standard.

Geological information recorded as hand written sheets is then transferred to Access database on a daily basis, cross checked with the original sheets and validated by the Project Geologist.

Basic geotechnical core recovery and RQD information was captured for all drill holes, including weathering state and oxidation boundaries. These are entered on to the hand written sheets and then entered into an Access database.

The geological logs do capture basic geotechnical and structural information but discussion indicates that the core is not orientated and as such the orientations of potentially important fault and fracture sets remain unknown. No core orientation line referenced structural measurements have been taken.

11.6 Survey

Topographical survey data was acquired in February 2007 from PhotoSat of Vancouver, Canada, taken from IKONOS satellite images dated February 15 2007, and is accurate to 2 m.

Digital scaled contour topographic maps were produced from this data for the Tepal property. These were subsequently used to generate topographical DTMs in Gemcom for use in resource modelling.

Diamond drill holes were positioned using hand held GPS (UTM NAD83), providing +/- 5m accuracy. Once a drill-hole was completed, it was surveyed again with a hand-held GPS (UTM NAD83). The collar was capped and marked with a concrete monument that displayed the drill-hole name, azimuth, angle of dip and length. All drill holes since have been surveyed by total station and tied-in to a known Mexican government survey point situated on the property.

Drill hole surveys were routinely taken every 50 m down the hole using a Reflex instrument. Downhole survey results are provided by the drilling company in digital format. Drill hole survey measurements taken by this method can be considered reliable.

11.7 Core Recovery

At Tepal, 4,375 recovery measurements have been taken on the Arian drill core.

The average recovery value for all drill hole intervals is 96% and interval recovery values range from 0% to 200% recovery (See Figure 11.3).

32 spurious recovery readings of greater than 100% (including one reading of 200% recovery) occur within the database and require follow up. These discrepancies were found to be input errors: these were corrected and the core recovery database file was reviewed and validated prior to the resource estimation update.

975 core recovery measurements occur within the Tepal North mineralized domain. The mean core recovery within the mineralized zones is 93% with a range of 24% to 171%.

With spurious values excluded to remove bias from these error values, recovery remains at 93% which Howe considers satisfactory.

620 core recovery measurements occur within the Tepal South mineralized domain. The mean core recovery within the mineralized zones is 96% with a range of 24% to 200%.

Again spurious results require follow up. With spurious values excluded to remove bias from these error values, recovery remains at 95% which SRK considers satisfactory.

The core recovery through the mineralized zones is considered acceptable so as to be confident that core samples, and the assay values derived from them are representative of the material drilled and suitable for inclusion in resource estimation studies.

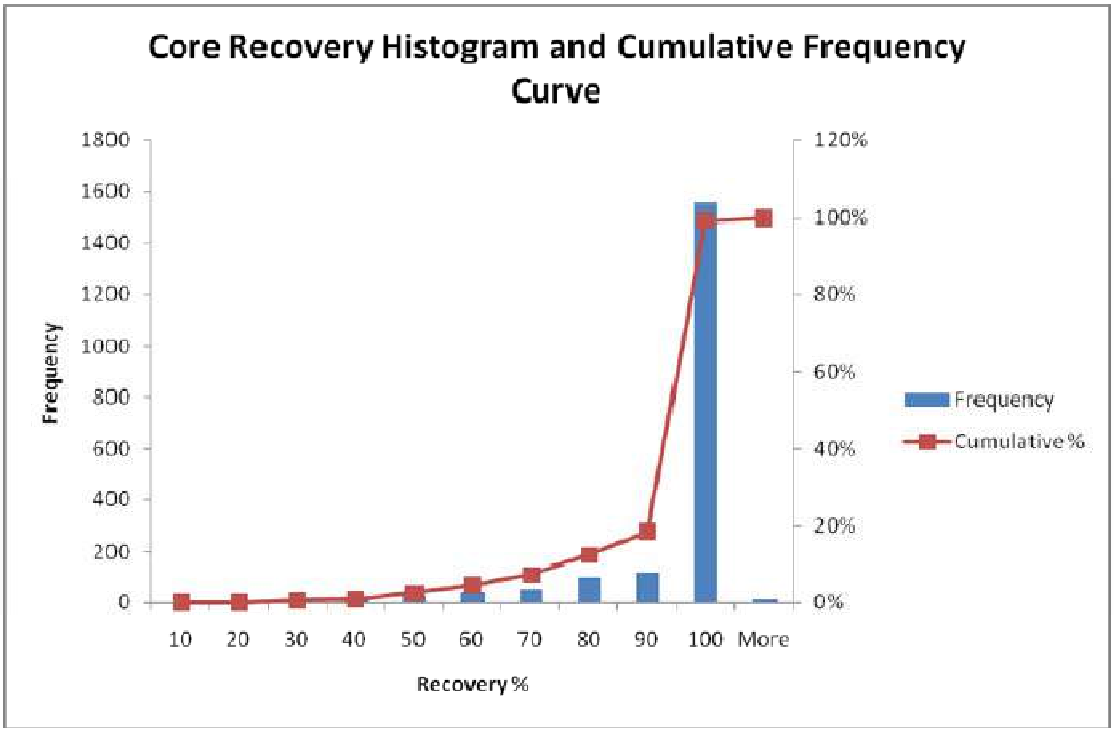

Figure 11.3: Arian Phase 1 Core Recovery Data

Core recovery should continue to be monitored as part of the proposed Phase 2 drilling campaign to ensure acceptable levels of core recovery are maintained, particularly through the mineralized zones.

11.7.1 Specific Gravity

During 2007, a total of 19 samples of core were collected from 13 DD drill holes at the Tepal property to facilitate specific gravity (SG) determination for use in the resource estimate and future mine planning. Geologix collected an additional 21 samples from the Tizate deposit for SG determination in 2010. A review of samples taken, indicate a reasonable spatial distribution, variety of mineralization and litho types and oxidation zones from the North and South Tepal mineralized zones.

Specific gravity determination for each sample was performed by ALS, Vancouver, BC. Specific gravity readings were calculated by gravimetric methods whereby two techniques are employed depending upon the material type.

For a bulk sample the rock or core section (up to 6 kg) is weighed dry or is covered in a paraffin wax coat and weighed. The sample is then weighed while it is suspended in water and SG determined by measuring the volumetric displacement of the rock in water and dividing the weight of rock by the volume.

For a pulverized sample, (3.0 g) is weighed into an empty pyncometer. The pyncometer is filled with a solvent (either methanol or acetone) and then weighed. From the weight of the sample and the weight of the solvent displaced by the sample, the specific gravity is calculated by the weight of sample divided by the weight of solvent displaced multiplied by the SG of solvent.

Specific gravity data is tabulated for Tepal core in Table 11.1 and for Tizate in Table 11.2. Weighted average bulk density values were calculated for fresh (sulphide) and weathered (oxide) material types for use in the resource tonnage estimations.

Table 11.1: Tepal Bulk Density Data

Rock Type	Oxidation	No of Samples	DD Drill Holes	Average Specific Gravity
Andesite	Oxide	2	AS-07-011	2.745
Andesite dyke	Oxide	2	AS-07-011	2.695
Rhyolite tuff	Fresh	2	AS-07-011	2.805
Quartz vein	Oxide	1	AS-07-011	2.800
Tonalite (North Zone)	Oxide	3	AS-07-008, 010, 012	2.783
	Fresh	3	AS-07-006, 012A, 019	2.827
Tonalite (North Zone)	Oxide	3	AS-007-007, 009, 022	2.807
	Fresh	3	AS-07-001, 005, 017	2.727

Table 11.2 Tizate Bulk Density Data

Rock Type	Oxidation	No. of Samples	DD Drill Holes	Average Specific Gravity
Tonalite	Fresh	2	AS-07-025 TEP-10-005 TEP-10-008 TEP-10-009 TEP-10-029 TEP-10-031 TEP-10-032 TEP-10-033 TEP-10-034	2.73
Tonalite	Fresh	2		2.77
Tonalite	Fresh	2		2.73
Tonalite	Fresh	3		2.77
Tonalite	Fresh	1		2.79
Tonalite	Fresh	3		2.75
Tonalite	Fresh	2		2.75
Tonalite	Fresh	4		2.75
Tonalite	Fresh	2		2.76
Average SG	Fresh	21		2.75

11.8 Geologix

Procedures for the Tepal drill hole sampling method and approach are similar to those employed at all of Geologix projects.

Drill core is retrieved in approximate 1.52 m for HQ runs where possible and 3.05 m runs for NQ core. Run length is less where broken ground is encountered or the core tube blocks.

All drill-core is stored in plastic core boxes (with lids) that are able to hold 3 m of core. The plastic core boxes are transported (by Geologix personnel) with a large elastic band wrapped around them so to prevent core loss when being transported.

Drill-core was collected from each of the drill-rigs at the end of each shift. The core was transported by Geologix personnel to the core logging area, where it was cleaned and marked up (highlighting lithological, structural, alteration, etc. features), and then photographed. The photographs are saved, every day onto a computer at the property and a periodic back-up is made. Access to the core logging and storage facility is restricted to Geologix personnel and locked during periods of non-use. The core logging and storage facility is located within an office/building complex which is also secured and locked during non-peak hours.

12 Sample Preparation, Analyses and Security

The following section is modified from Howe (2009).

12.1 INCO Program

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

12.2 Teck Program

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

All samples collected by Teck were analyzed by ALS Chemex (“ALS”) in Vancouver. The analytical methods utilized by Teck for gold consisted of a standard fire assay followed by an atomic absorption finish. The method requires that a sample weighing about 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 aqua regia digestion followed by ICP analysis. Samples collected from the oxide were analyzed for non-sulphide copper minerals by digestion in dilute sulfuric acid and AA finish.

SRK is not aware of the certification ALS had in the mid-1990’s but current ALS laboratories in North America are registered to 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”. CAN-P-1579 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 agreed well with the original assays and documented in Section 13 of this report.

12.3 Hecla Program

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

All samples were analyzed by ALS. 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.

12.4 Arian Program

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

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

After the core has been logged and photographed, all information was entered into an Access Database (Booth, 2007b).

The samples (in groups of 10 samples) are placed inside nylon rice-bags and sealed with a cable-tie to prevent access (Booth, 2007b).

Details of sample type for the Tepal drilling are contained in Table 12.1.

Table 12.1: Tepal Sample Types

Prospect	Sample Type	Number of Samples	Sample Length
Tepal	HQ (NQ) half core	3,532	Non-uniform (commonly 2m)*

**sample lengths vary between 0.25m and 5.95m, contained to mineralized and/or geological and geotechnical boundaries*

12.4.1 Previous Analytical Techniques

Following QA/QC issues identified in the April 2008 ACA Tepal Resource Estimation Study, the initial sample Assay methodology was changed as copper CRMs assayed at Inspectorate using the 3 acid digestion and ICP finish method returned results that were generally erratic and higher than expected.

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

Once re-analysis was complete the CRM and duplicate results were greatly improved for gold and are 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 methods and the final methods of analysis employed by each lab are presented in the following sections.

12.4.2 Sample Preparation

Inspectorate Labs

Initially samples sent to Inspectorate Labs for analysis, were collected from Arian’s warehouse on a fortnightly basis by Inspectorate, who transported the samples to their preparation facility in Durango, Durango, Mexico.

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

ALS Chemex

Samples analyzed by ALS were collected from Arian’s warehouse and transported the samples to the sent to ALS’s sample preparation facility in Guadalajara, Mexico.

Once the sample is received by ALS the entire half-core is crushed and pulverized to 85% passing 75 micron mesh. The pulp samples are then air freighted to the ALS analytical laboratories in Vancouver, Canada, for analysis.

At no time after the sample bags are sealed, are the samples handled by Arian personnel or contractors working for Arian.

12.4.3 Sample Analysis

A summary of samples analyzed and methodologies used is contained in Table 12.2.

Table 12.2: Tepal Sample Analysis Methodology

Analyte	Sample Range	Lab	# of Samples	Assaying Methodology	Limits of Detection*
Au	142001-143419, 145501-146000	Inspectorate	1,700	<3ppm: Aqua Regia digest with AAS finish;	LLD:<0.005ppm ULD:>10ppm
				>3ppm: Fire Assay with Gravimetric finish	LLD:<0.005ppm ULD:>100ppm
	143420-145500, 212251-217350	Chemex	1,829	<3ppm: Aqua Regia digest with AAS finish;	LLD:<0.005ppm ULD:>10ppm
				>3ppm: Fire Assay with Gravimetric finish	LLD:<0.005ppm ULD:>100ppm
Cu	142441-142445, 142465-142473, 142480-142485, 143032-143050, 143306-143335, 143344-143419	Inspectorate	142	Aqua Regia digest with AAS finish;	LLD:<0.2ppm ULD:>10,000PPM
	142001-142440, 142447-142464, 142474-142479, 142487-143031, 143051-143304, 143336-143342, 143420-144350, 144401-146000, 212251-217350	Chemex	3,342	<10,000: 3 Acid digestion with ICP	LLD:<0.2ppm ULD:>10,000PPM
				>10,000 Aqua regia Digest with AAS	LLD:<0.01% ULD:>3%

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.

Inspectorate Labs

Samples were assayed for gold by Aqua Regia digest with AAS finish in a 30 g sample. High grade gold (>3 ppm) samples were re-analyzed using fire assay with a gravimetric finish.

Copper was analyzed using an Aqua Regia digestion and an AAS finish.

ALS

Samples were assayed for gold by Aqua Regia digest with AAS finish in a 30 g sample. High grade gold (>3 ppm) samples were re-analyzed using fire assay with a gravimetric finish. The majority of copper assays were undertaken at ALS using a 3 Acid digestion with ICP finish. High grade (>10,000 ppm) copper samples were re analyzed using an Aqua Regia Digest with AAS finish.

12.5 Geologix Program

Samples analyzed by ALS Chemex were collected from Geologix's warehouse and transported to ALS Chemex's sample preparation facility in Guadalajara, Jalisco with the analytical work being completed at their laboratory facilities in North Vancouver, B.C. A QA/QC program has been implemented to ensure all core and sample handling procedures are in accordance with the best possible practices. The assay protocol includes the insertion of standards, blanks and duplicates into the sample stream on an average basis of one standard, one blank, and one duplicate sample within every 30 samples. At no time after the sample bags are sealed and placed inside nylon rice-bags and sealed with a cable-tie to prevent access, are the samples handled by Geologix personnel or contractors working for Geologix.

Once the sample is received by ALS the entire half-core is crushed and pulverized to 85% passing 75 micron mesh and the pulp samples being then air freighted to the analytical laboratories for analysis.

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

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

Core was transported from site to the processing facility, housed in the grounds of the house that the company currently occupies in Tepalcatapec, 15 kms 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), and the same sample number was written the same number. The bag was then sealed on site.

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

The samples (in groups of ten samples) are placed inside nylon rice-bags and sealed with a cable-tie to prevent access.

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 ore grade 4 acid digestion with ICP-AES finish.

Results are received from the lab via email and hardcopy certificate. For the 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.

13 Historical Data Verification

During the Howe site visit, Galen White selected 25 representative pulp samples from the Phase 1 drilling which were to be submitted to ALS Laboratories for check assay.

All pulp re-assays correlated well with the original assay data and no issues were identified.

13.1 Arian QA/QC

A quality assurance and quality control 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. 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.

An assessment of QA/QC samples submitted to Inspectorate laboratories was completed in the report 2008 ACA Resource Estimation Study on the Tepal, Gold-Copper Prospect, Michoacán, Mexico. Inspectorate gold results were sufficient to be, on the whole, confident in assay precision and accuracy.

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

The QA/QC program consisted of:

The inclusion of Certified Reference Material standards (CRM's) in sample batches sent to both Inspectorate and ALS laboratories, to assess analytical accuracy (4 per 100 samples).

The inclusion of field blanks and pulp blanks to assess laboratory sample preparation and analytical accuracy (3 per 100 samples).

The inclusion of field duplicates and externally assayed pulp duplicates to asses sample preparation and precision (3 per 100 samples).

Details of the QA/QC program are contained in the Table 13.1:

Table 13.1: Arian Assay QA/QC Details

QA/QC Sample/Assay Type	# of Samples	% of Total Samples*	Ratio
Standard Samples	60	2%	1:60
Field Blank Samples	33	1%	1:107
Pulp Blank Samples	33	1%	1;107
Coarse Reject Duplicates	35	1%	1:104
Pulp Duplicates	34	1%	1:101

**total number of samples submitted = 3532*

Approximately 6% of all samples submitted to the laboratory were quality control samples.

13.1.1 Blanks

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.

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

Gold grades in Field Blanks submitted to ALS showed that only 3 results returned values marginally greater than the lower limit of detection 0.5 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 1 standard deviation of 0.002% Cu based on all samples. There are two outliers of 0.007% and 0.008% however these are considered insignificant and within tolerance limits.

As part of the Phase 1 quality control sample resubmission 33 pulp blanks, prepared by Inspectorate, were submitted for reanalysis. Gold grades for Pulp Blanks show that 67% of returned grades are below the limit of detection. Of the remaining samples 8 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 1 standard deviation of 0.007% Cu based on all samples, with the majority of samples returning grades of 0.009% Cu. There was one outlier, again sample 145521, which returned a grade of 0.04% which is considered beyond acceptable limits.

On the whole the results of Blank Sample Analysis are acceptable; however there are some anomalous assays for both field and pulp Blanks. Field Blanks are acceptable indicating that is 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.

13.1.2 Standard Samples

Certified Reference Material samples were prepared from mineral matrices that contain gold and copper values similar to the grade of the Tepal deposit, which are uniformly distributed throughout the pulverized rock. Standard statistical techniques are used to assign a recommended assay value with associated 95% confidence interval (see Table 13.2). CRM's were prepared by WCM Minerals, Burnaby, British Columbia and Rock Labs, New Zealand.

CRM samples were routinely submitted for assaying with core at a ratio of up to 1:60, totalling 2% of all samples. Three CRM samples were used CU139, to assess lower grades, CU150 and OX14 for higher grades. A total of 60 CRM check samples were undertaken to check lab accuracy. Error plots for each CRM for gold and copper are presented in the following pages (Figure 13.1 to Figure 13.5).

Table 13.2: Arian CRM Assessment List

CRM ID	Recommended Values		Standard Deviation		No of CRM's submitted
	Au ppm	Cu (%)	Au ppm	Cu (%)	
CU139	0.55	0.43	0.031	0.007	34
CU150	0.79	0.59	0.033	0.012	11
Ox14	1.22	NA	0.057	NA	15

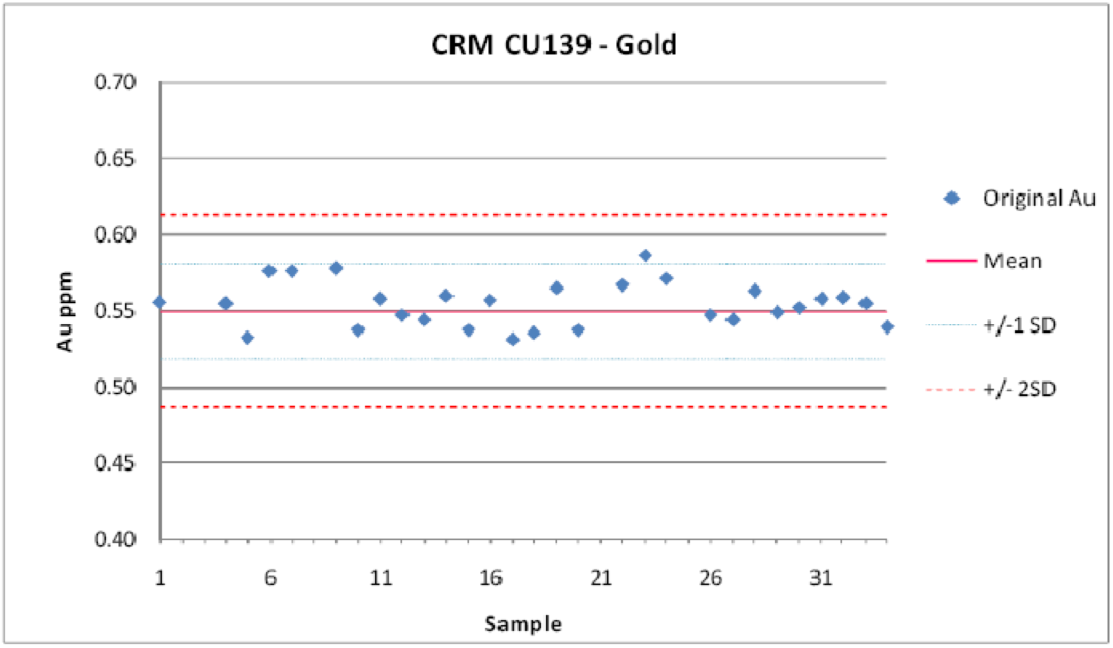

Figure 13.1: Control Plot for CRM CU 139-Gold

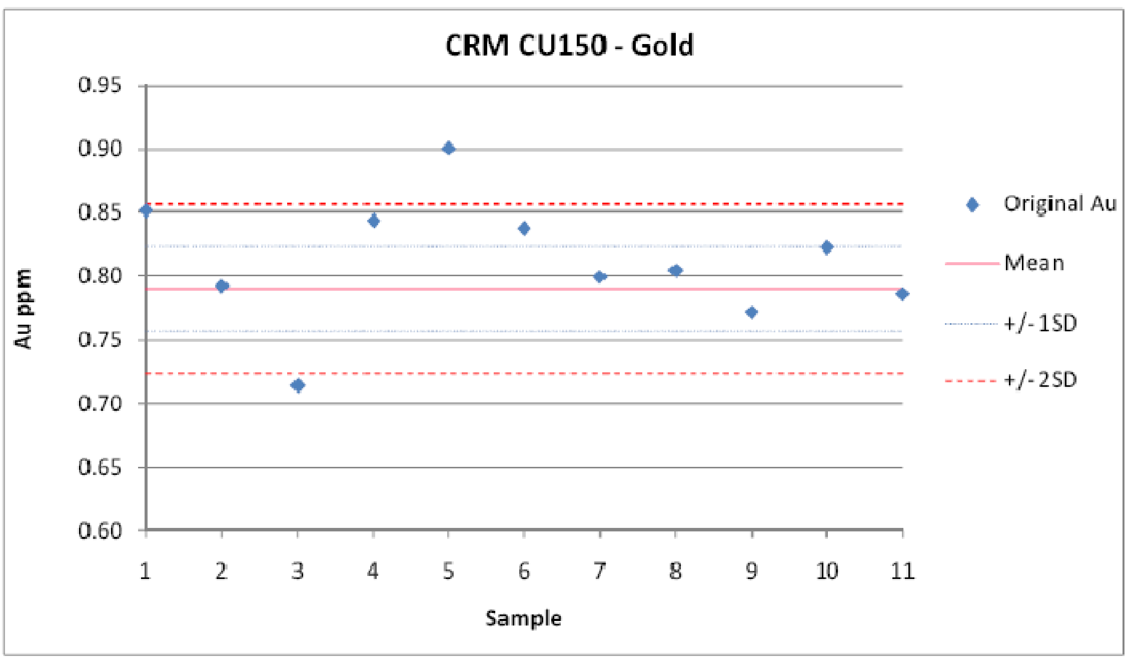

Figure 13.2: Control Plot for CRM CU150-Gold

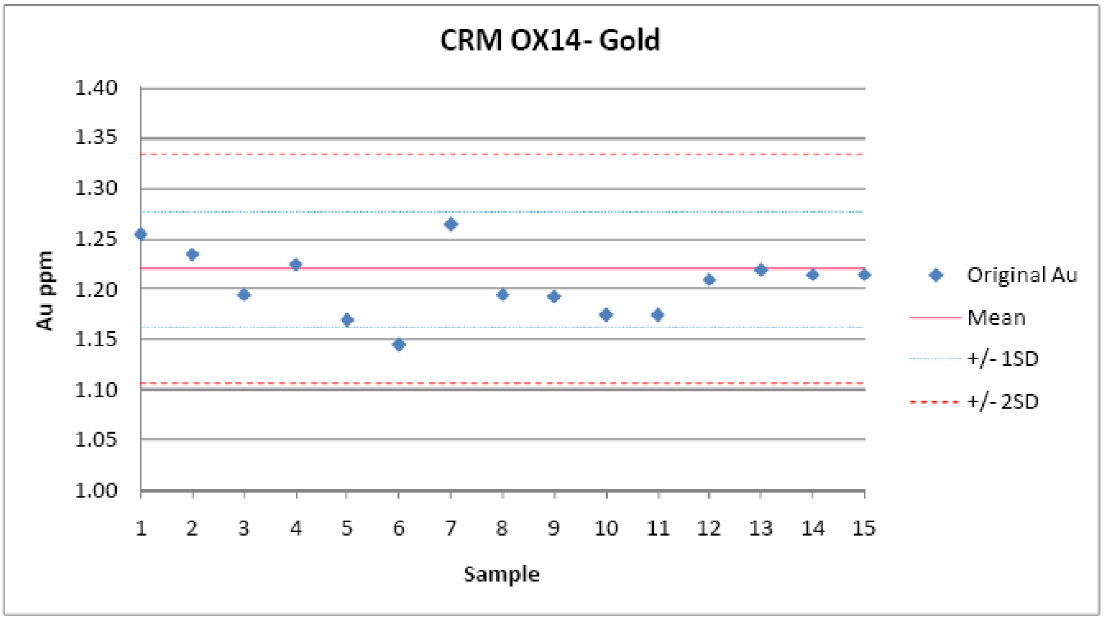

Figure 13.3: Control Plot for CRM OX14-Gold

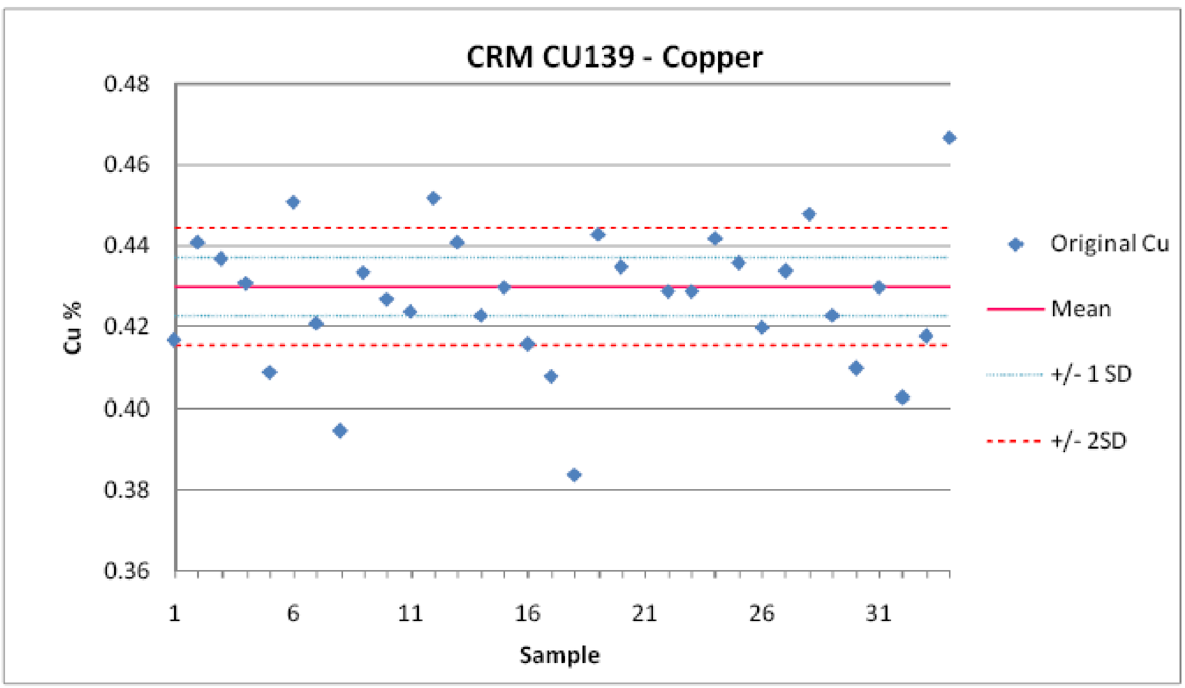

Figure 13.4: Control Plot for CRM CU139-Copper

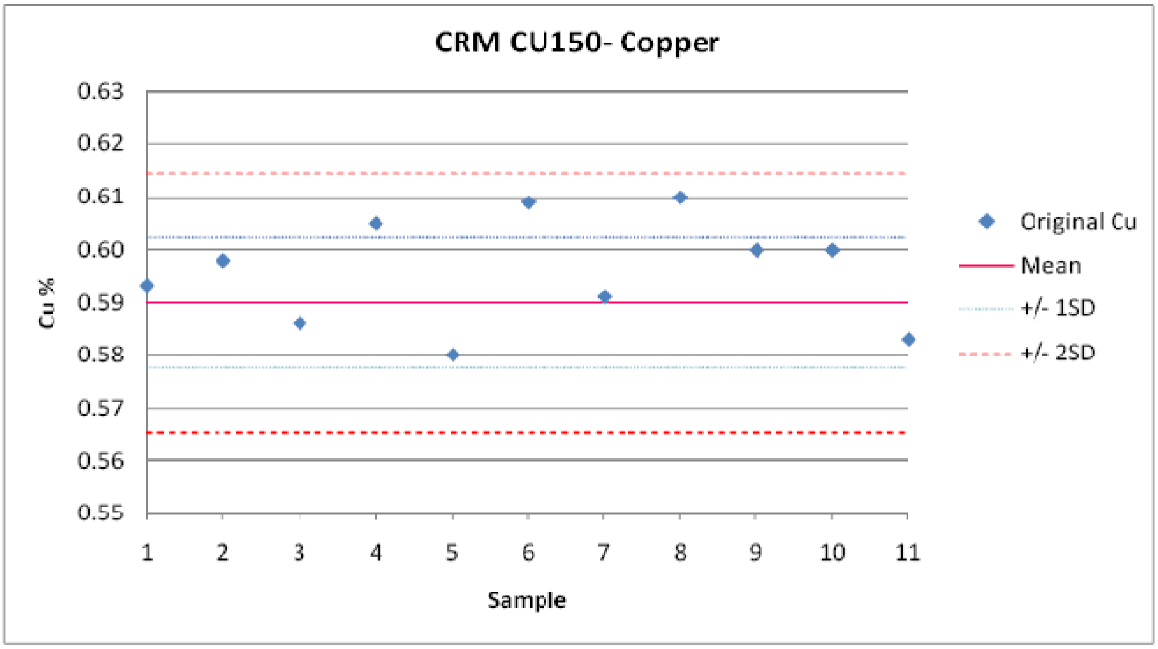

Figure 13.5: Control Plot for CRM CU150-Copper

The error plots for gold CRM assays show that 96.4% are within 2SD of the expected value. All samples fall within 10% of the expected grade aside from CRM CU150 sample 144892 assayed at 0.900 ppm, 13.924% higher than the expected CRM value of 0.790 g/t Au.

For copper 77.3% of samples were within +/- 2SD of the expected CRM grade. All samples were within 10% excluding CRM CU139 sample 142897 which returned an assay of 0.384% Cu, 10.7% lower than the CRM expected value of 0.430%.

In general, submitted standard samples showed good repeatability for both copper and gold at both low and high grades. There are only few significant outliers, however those identified should be investigated. Gold results for CRM CU139 are over reported by a mean value of 7.5% however on the whole there appears to be no evidence of a strong systematic bias to either over or under reporting for either copper or gold, with results being generally well distributed around the expected grade.

It should be noted that the sample number on the (x) axis of the control plots also represent a time axis and analysis of the control plots suggests some analytical drift, resulting in cyclic peaks and troughs. This is acceptable given that the majority of assays fall within acceptable limits, but erroneous outliers may be caused by re-calibration of analytical equipment.

The use of only one medium and one higher grade CRM type limits this assessment to one specific grade range for each analyte. It is highly recommended that a broader range of CRM's are used for any further work to identify bias in analysis, particularly for lower grade ranges for gold. It is also considered that an insufficient number of CRM samples have been taken to ensure a reliable determination of analytical bias. It is recommended that a minimum of 2% CRM samples are inserted for any further work.

13.1.3 Duplicates

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

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

There is 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 (Figure 13.6 and 13.7).

A lesser level of precision between original and duplicate assays is shown for 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.

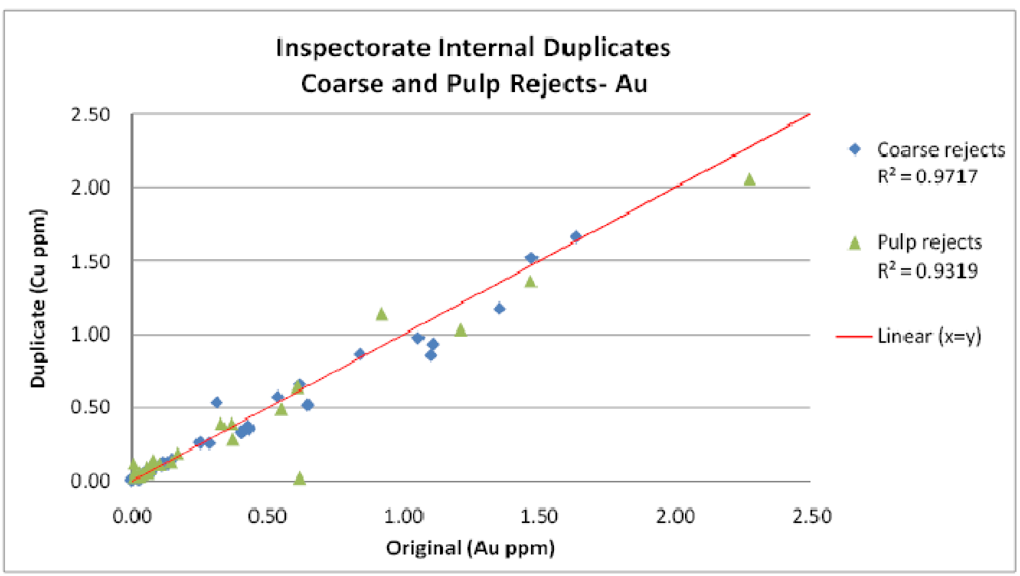

Figure 13.6: Inspectorate Coarse and Pulp Duplicates - Gold

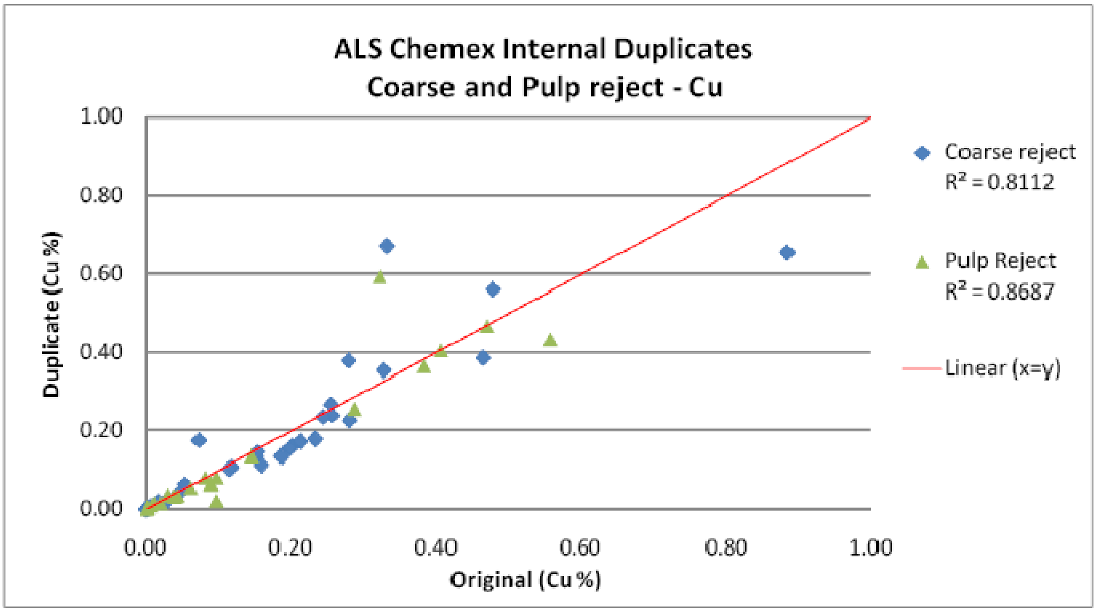

Figure 13.7: ALS Chemex Coarse and Pulp Duplicates - Copper

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

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

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

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

Duplicate analysis shows a good level of precision for both gold and copper. However it is noted that there have been no field duplicates submitted for reanalysis during the analysis of holes beyond borehole AS-07-23. For future drilling operations it is essential that duplicates are continuously submitted throughout the drilling campaign. It is recommended that a minimum of 2% of samples should be duplicates.

13.1.4 Arian Twin Drilling Program

A verification study of twin drill hole data conducted by Arian geologists indicated poor correlation between Arian diamond drill hole results and historical Hecla (MHT prefix) RC drill grades (Table 13.3).

The 'average' difference for Au was 19% and 16% for copper (with maximums of 72% and 142% respectively). For this reason, Howe decided that the historic assay results provided by Hecla were inaccurate and therefore removed from the Tepal database.

Table 13.3: Summary of Arian Twin Drill Holes

Arian Drill Hole	Original Drill Hole	Comment
AS-07-001	MHT-2	Hecla drill hole
AS-07-004	T-24	Teck Drill hole
AS-07-005	MHT-3	Hecla drill hole
AS-07-006	IN-57002	INCO drill hole - retained
AS-07-007	T-25	Teck Drill hole
AS-07-008	T-10	Teck Drill hole
AS-07-012	T-9	Teck Drill hole
AS-07-013	T-16	Teck Drill hole
AS-07-014	IN-57020	INCO drill hole - removed
AS-07-015	T-18	Teck Drill hole
AS-07-016	IN-57015	INCO drill hole - removed
AS-07-018	MHT-15	Hecla drill hole
AS-07-019	IN-57017	INCO drill hole - removed
AS-07-020	IN-57013	INCO drill hole - removed

13.2 Geologix QA/QC

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

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

The QA/QC program consisted of:

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

Details of the QA/QC program are contained in Table 13.4 below:

Table 13.4: Geologix Assay QA/QC Details

QA/QC Sample/Assay Type	# of Samples	% of Total Samples*	Ratio
Standard Samples	289	4.0%	1:30
Field Blank Samples	287	4.0%	1:30
Duplicate samples	274	3.7%	1:30
Check assays	292	4.0%	1:30

**total number of samples submitted = 5839*

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

13.2.1 Blanks

Field blanks were prepared from samples of unmineralized porphyritic andesite collected from an area on the access road to the property and submitted along with the core samples.

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 1:30 and totalled 4% of all samples. A total of 287 blanks were submitted.

Gold grades in field blanks submitted to ALS showed that only 5 results returned values marginally greater than the lower limit of detection 0.015 ppm Au and were well within tolerance limits, returning values of up to 0.027 ppm Au (Figure 13.8).

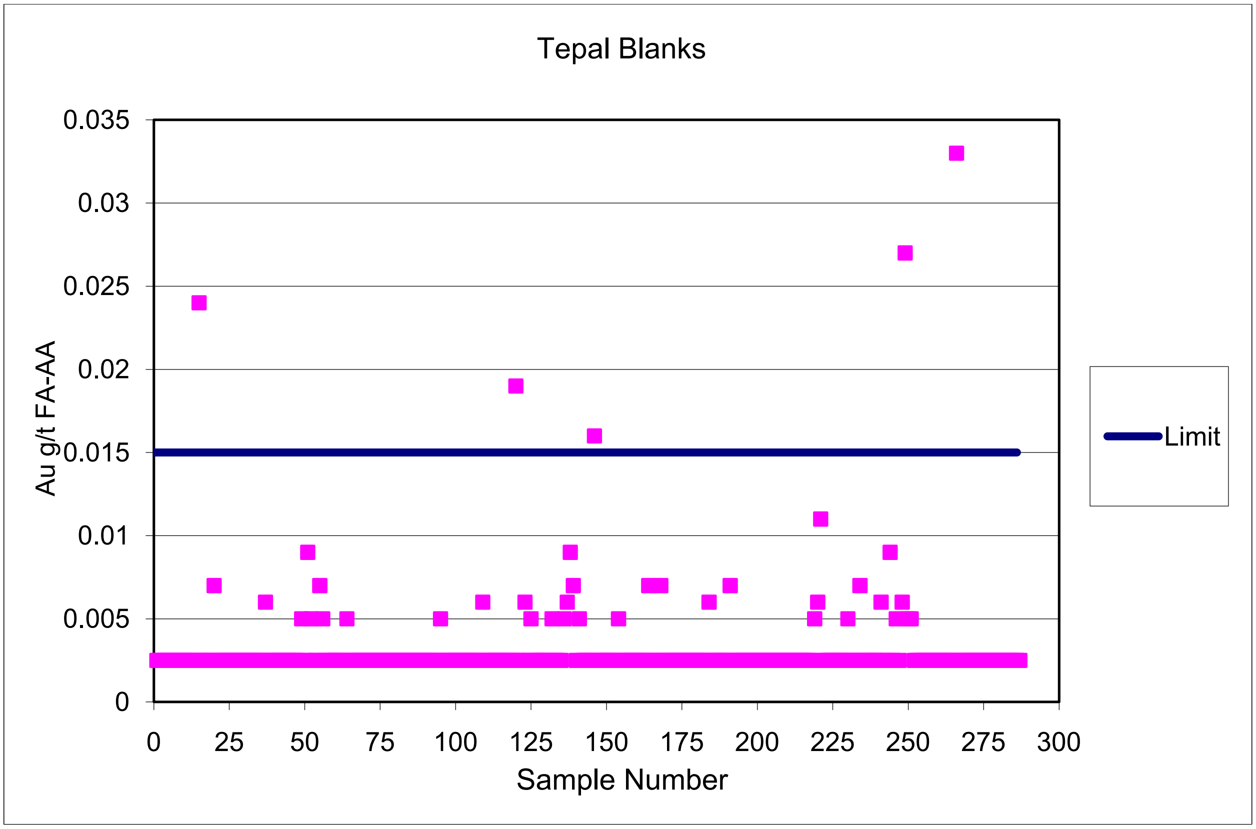

Figure 13.8 Results of Field Blank Reference Material

On the whole the results of Blank Sample Analysis are acceptable indicating that is no significant contamination issues in field sample preparation.

13.2.2 Standard Samples

Certified Reference Material samples were prepared from mineral matrices that contain gold and copper values similar to the grade of the Tepal deposit, which are uniformly distributed throughout the pulverized rock. Standard statistical techniques are used to assign a recommended assay value with associated 95% confidence interval (Table 13.4). CRM's were prepared by CND Laboratories Langley, British Columbia and Ore Research and Exploration Pty Ltd. of Australia.

CRM samples were routinely submitted for assaying with core at a ratio of up to 1:30, totalling 4% of all samples. Four principal CRM samples were used CDNCGS-21, CDNCGS-23, 50pb and 52pb. In addition, Geologix submitted 292 samples representing 5% of all the samples collected to ACME Analytical as additional quality assurance check. Error plots for each CRM for gold and copper are and the results of the check assay program are presented in the following pages (Figures 13.9 to 13.18).

Table 13.5: Tepal Geologix CRM Assessment List

CRM ID	Recommended Values		2 Standard Deviation		No of CRM's submitted
	Au ppm	Cu (%)	Au ppm	Cu (%)	
CDNCGS-21	0.99	1.30	0.09	0.084	65
CDNCGS-23	0.218	0.182	0.036	0.01	57
50pb	0.841	0.744	0.063	0.042	37
52pb	0.307	0.035	0.333	0.014	48
53pb	0.623	0.546	0.10	0.027	75

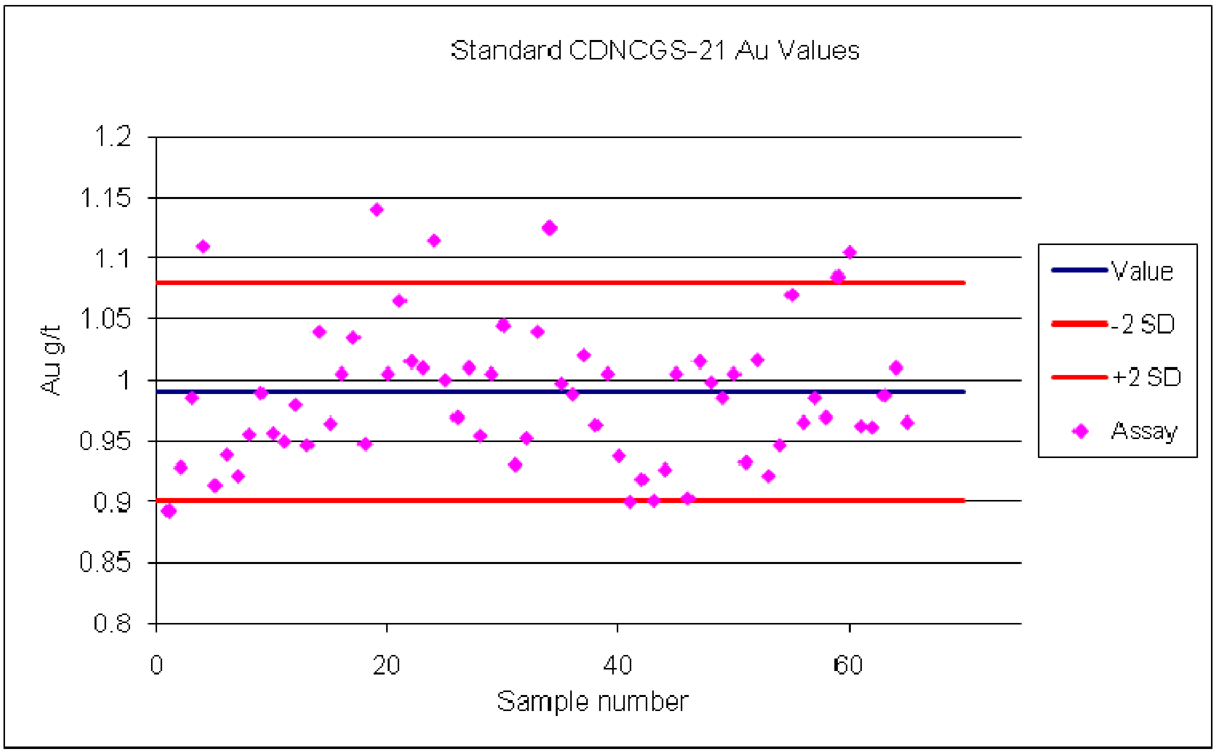

Figure 13.9: Control Plot for CRM CDNCGS-21-Gold

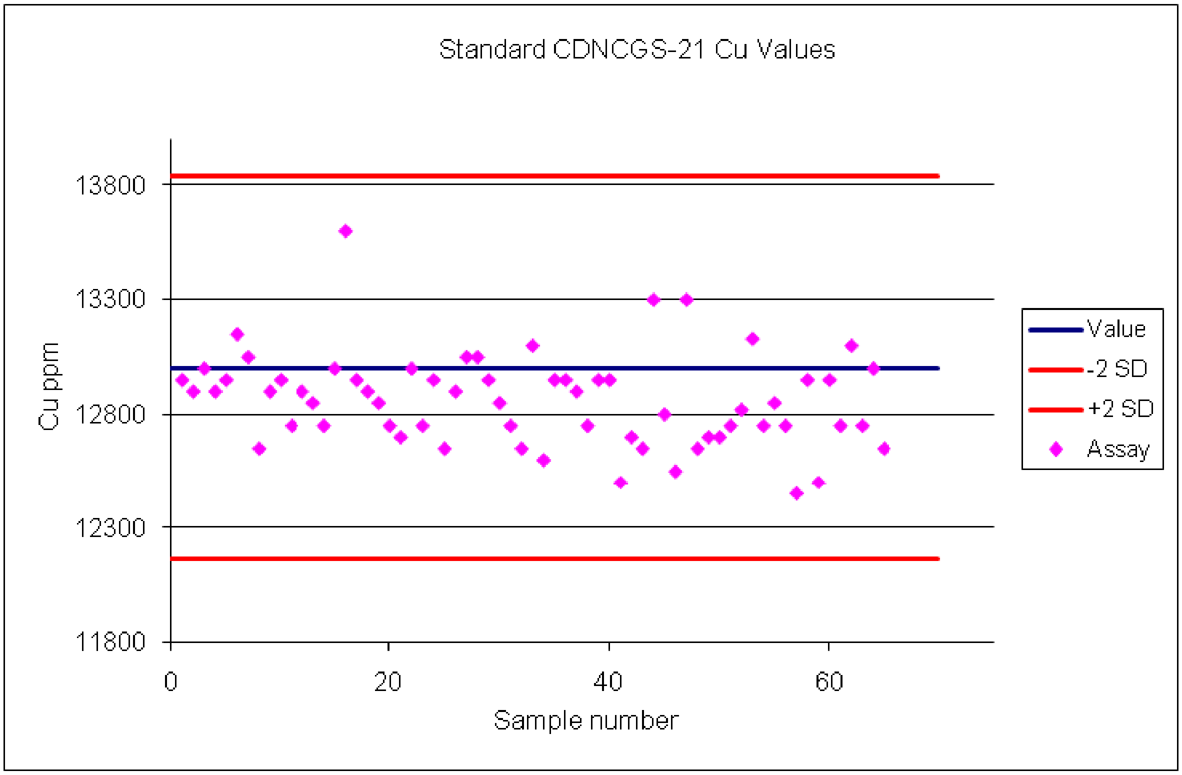

Figure 13.10: Control Plot for CRM CDNCGS-21-Copper

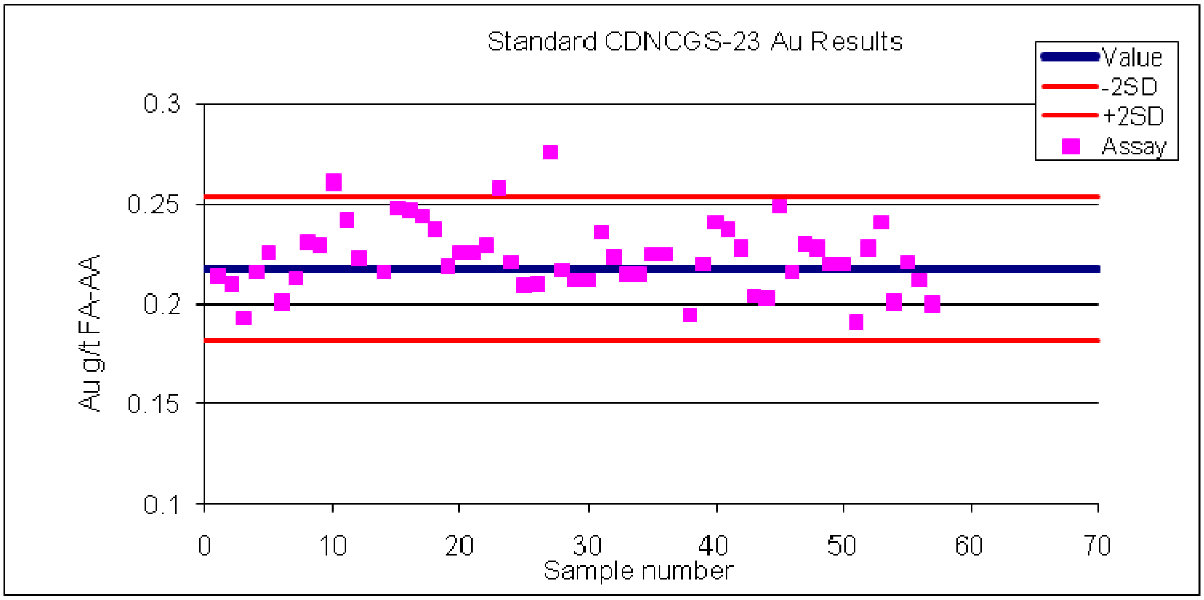

Figure 13.11: Control Plot for CRM CDNCGS-23-Gold

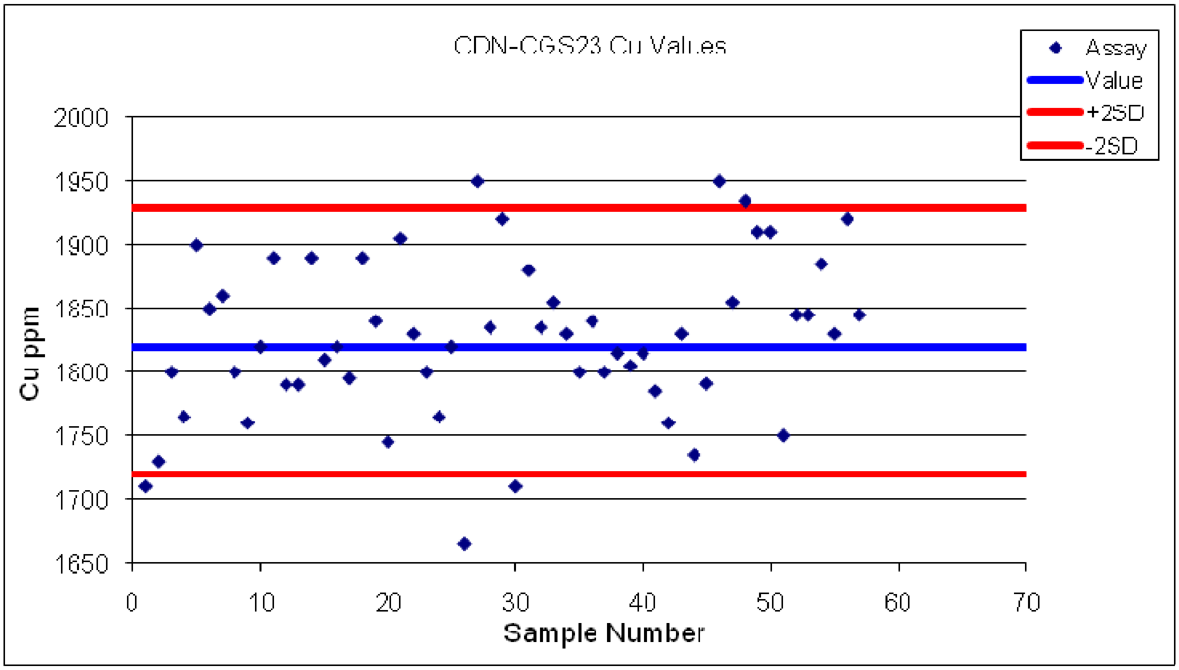

Figure 13.12: Control Plot for CRM CDNCGS-23-Copper

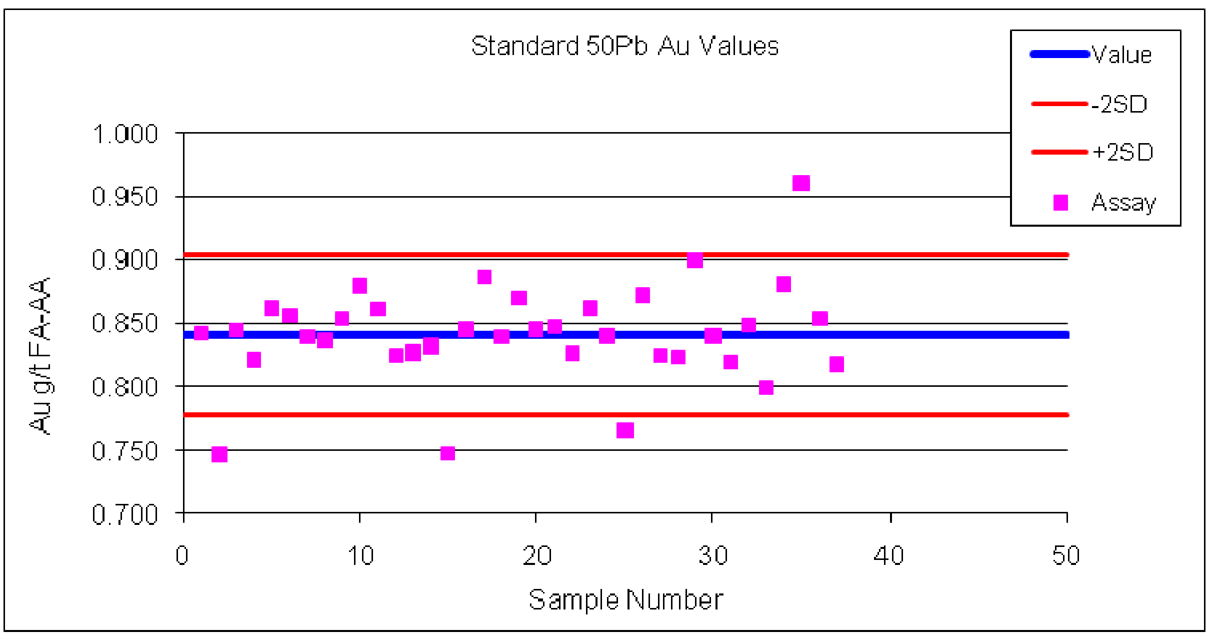

Figure 13.13: Control Plot for CRM 50pb-Gold

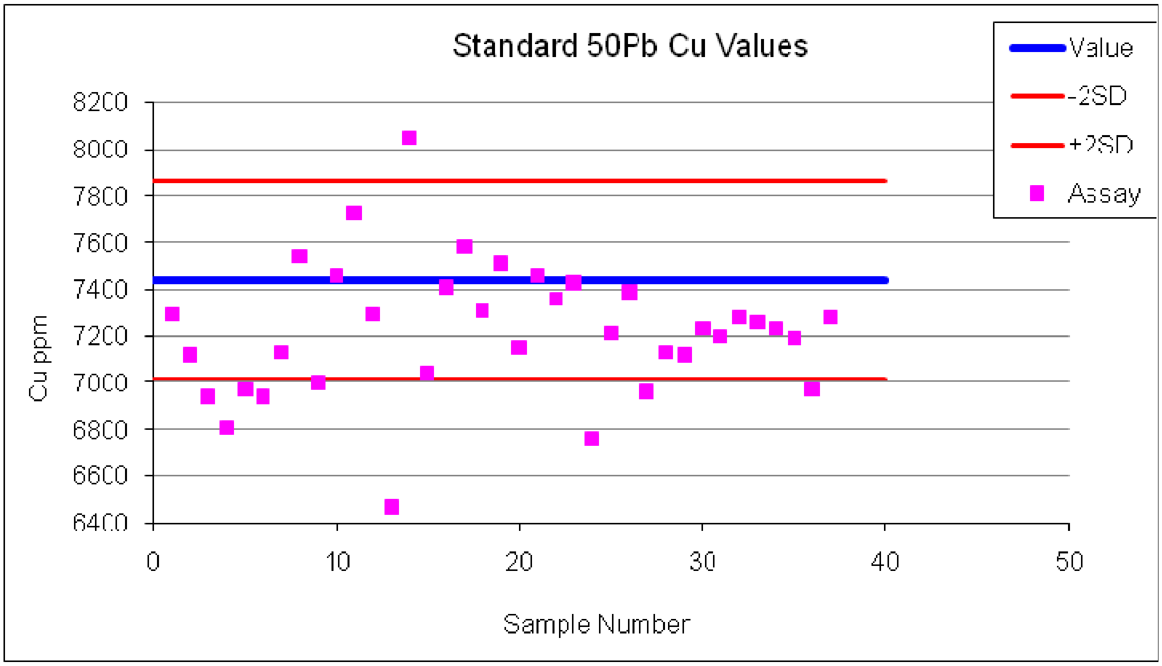

Figure 13.14: Control Plot for CRM 50pb–Copper

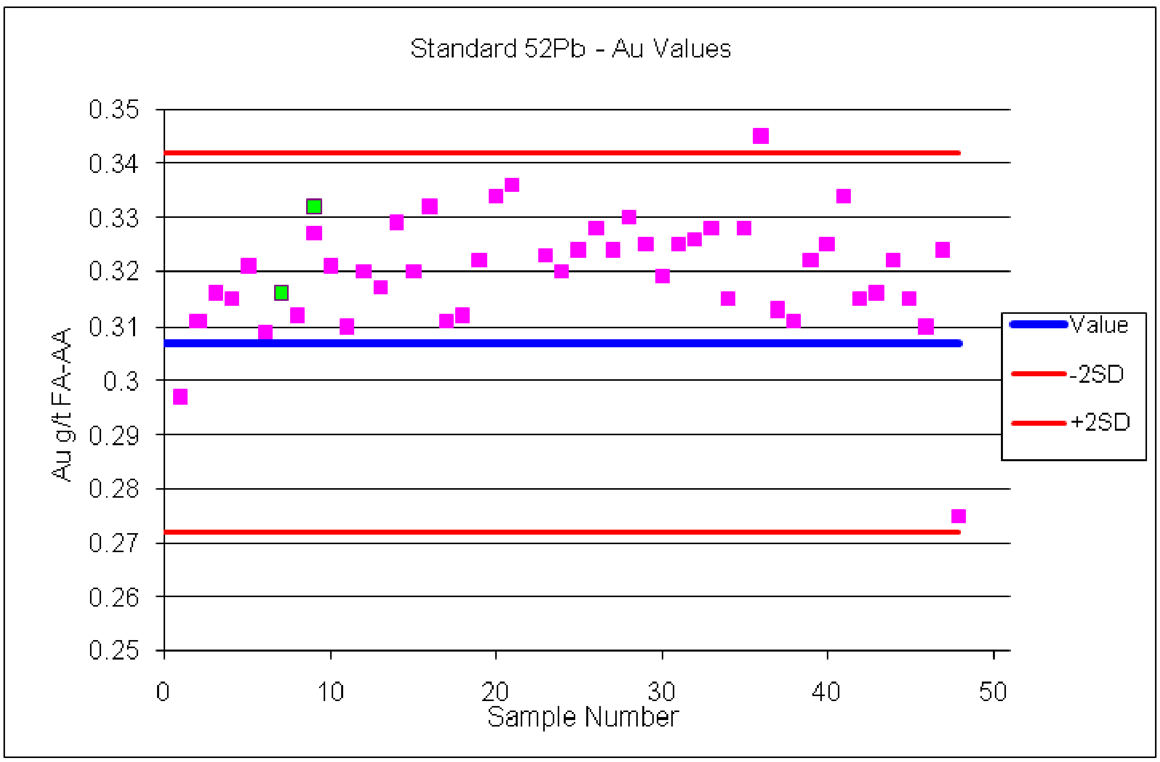

Figure 13.15: Control Plot for CRM 52pb-Gold

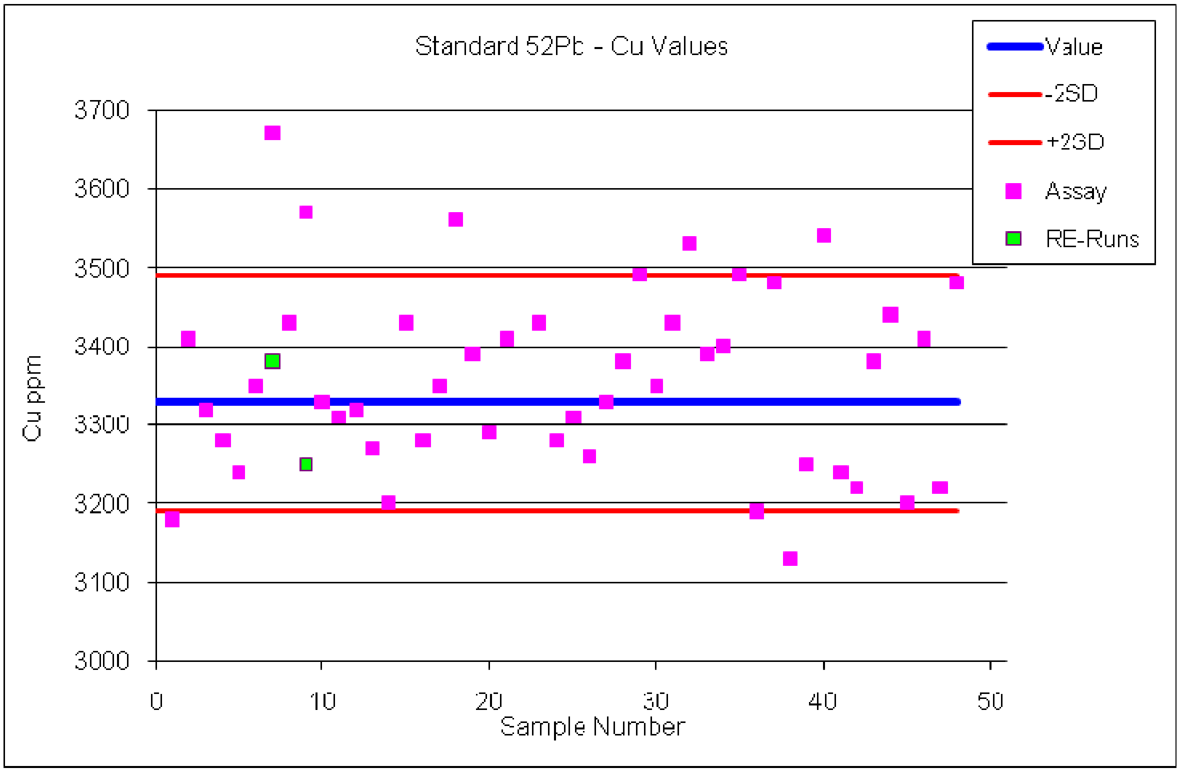

Figure 13.16: Control Plot for CRM 52pb-Copper

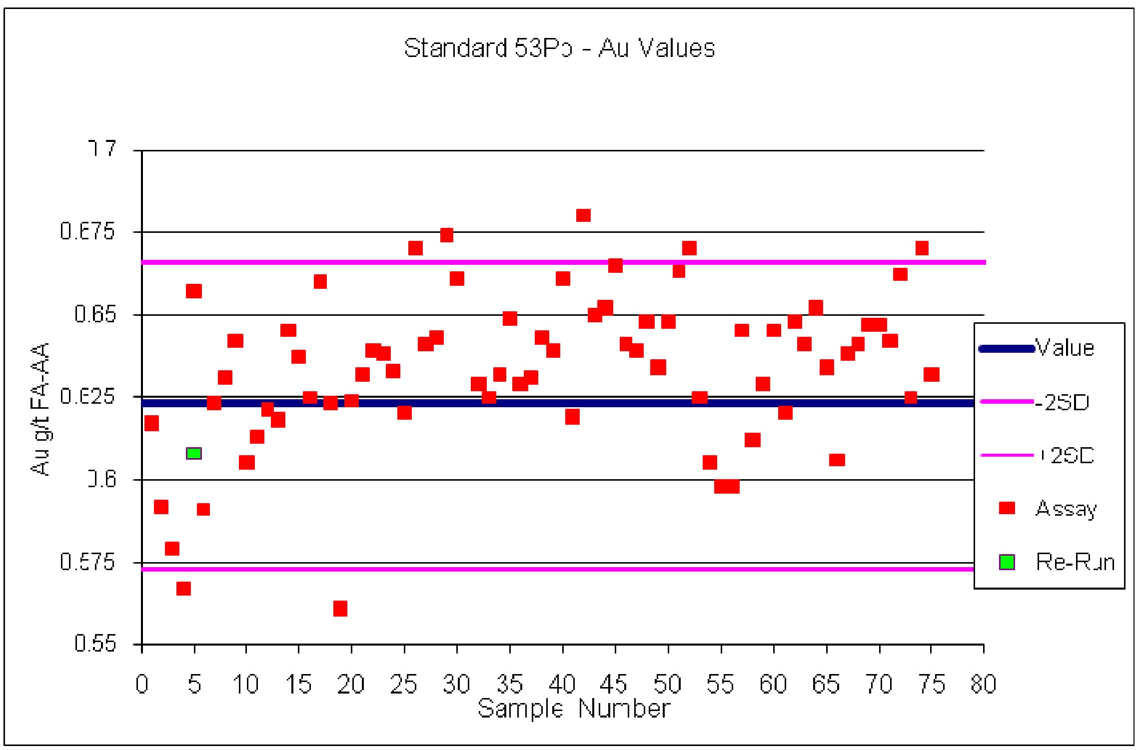

Figure 13.17: Control Plot for CRM 53pb-Gold

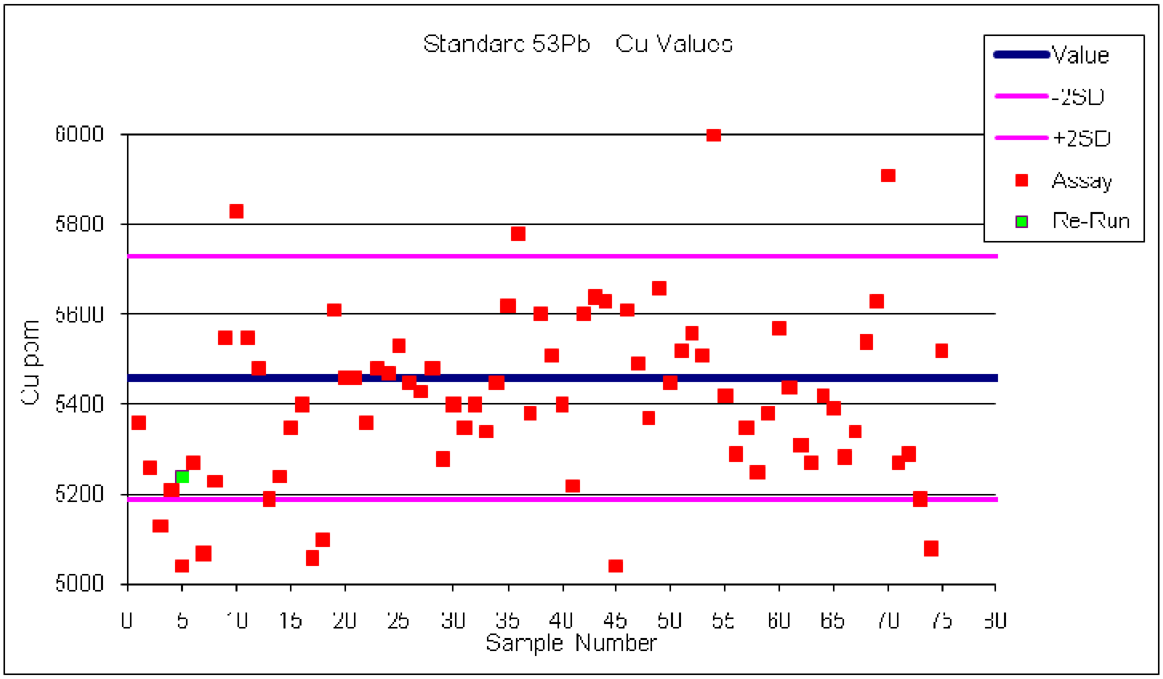

Figure 13.18: Control Plot for CRM 53pb-Copper

Most of the CRM for both gold and copper fall well within the ± 2 SD of the expected value. Eight of the sample batches failed the ± 3SD test and were re-assayed with the re-assayed bath retuning acceptable values for the CRM. 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. Reference material 53pb and 52pb seem to consistently report above the expected value for gold but well within the accepted value for the standard. Similarly, CRM CDNCGS-21 reports low for copper but also well within the acceptable limits for the standard.

It should be noted that the sample number on the (x) axis of the control plots also represent a time axis and analysis of the control plots suggests some analytical drift, resulting in cyclic peaks and troughs. This is acceptable given that the majority of assays fall within acceptable limits, but erroneous outliers may be caused by re-calibration of analytical equipment.

13.2.3 Coarse Duplicates

274 duplicate samples were submitted for assays and 292 samples were sent to ACME laboratories for re-analyzed, accounting for 8% of all samples.

Duplicates samples were prepared by sawing the core in half and sending both halves of the core for assay. There is a very good correlation for both gold and copper for the duplicate assays from coarse reject (Figure 13.19 and Figure 13.20). There is good level of precision between original assays and duplicate assays with most data plotting within +/-10% of the original assay value.

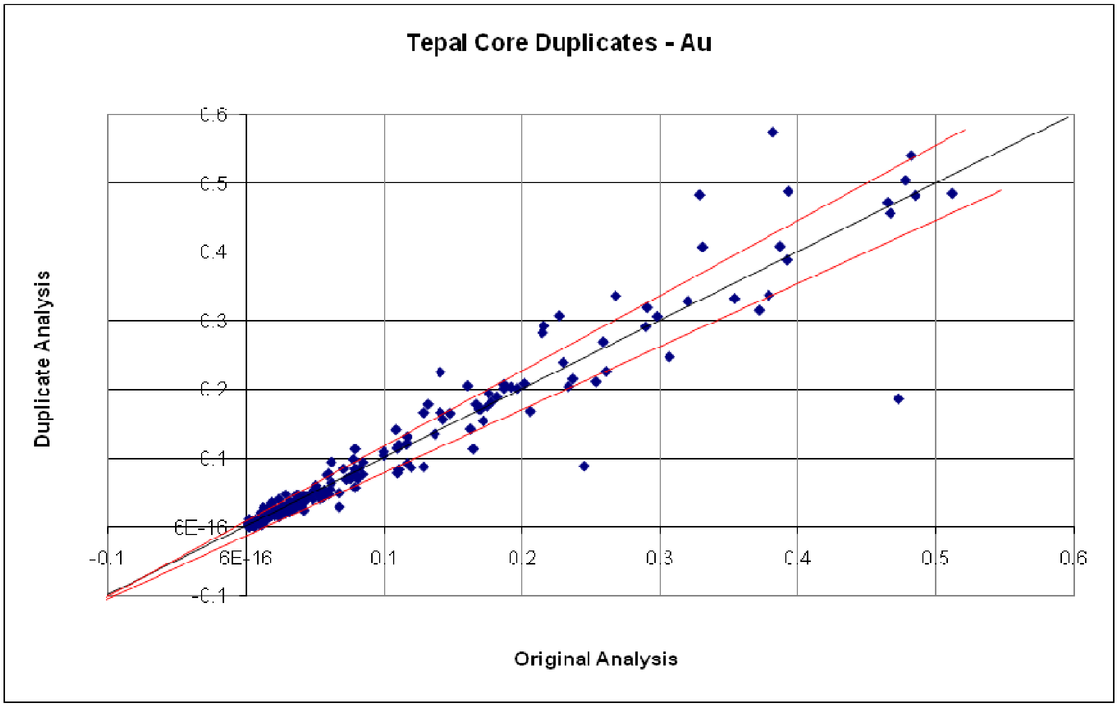

Figure 13.19: ALS Core Duplicates - Gold

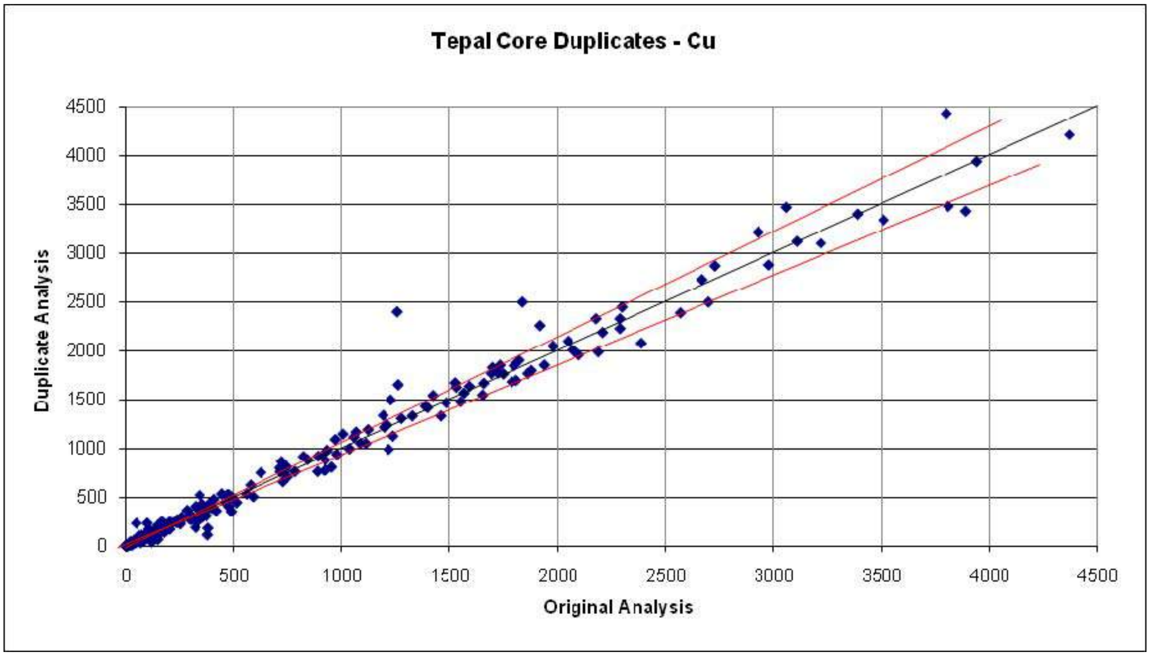

Figure 13.20: ALS Core Duplicates – Copper

13.2.4 Pulp duplicates

Geologix selected 292samples for re-assay to an Empire laboratory; the samples were selected from pulp rejects from ALS and forwarded to ACME for re-assay. ACME is a well recognised laboratory based in Vancouver. The laboratory maintains ISO 9001:2000 certification. The lab is a participant on the CALA proficiency testing program and is registered by the BC Ministry of Water Land and Air Protection under the Environmental Data Quality Assurance Regulation.

The results from the pulp re-assay program seem to indicate that ALS is reporting slightly lower than ACME for both gold and copper but the results are not significantly different (Figure13.21 and Figure 13.22). Values for silver and molybdenum appear to correlate very well between the original lab and the Empire lab. (Figure 13.23 and Figure 13.24).

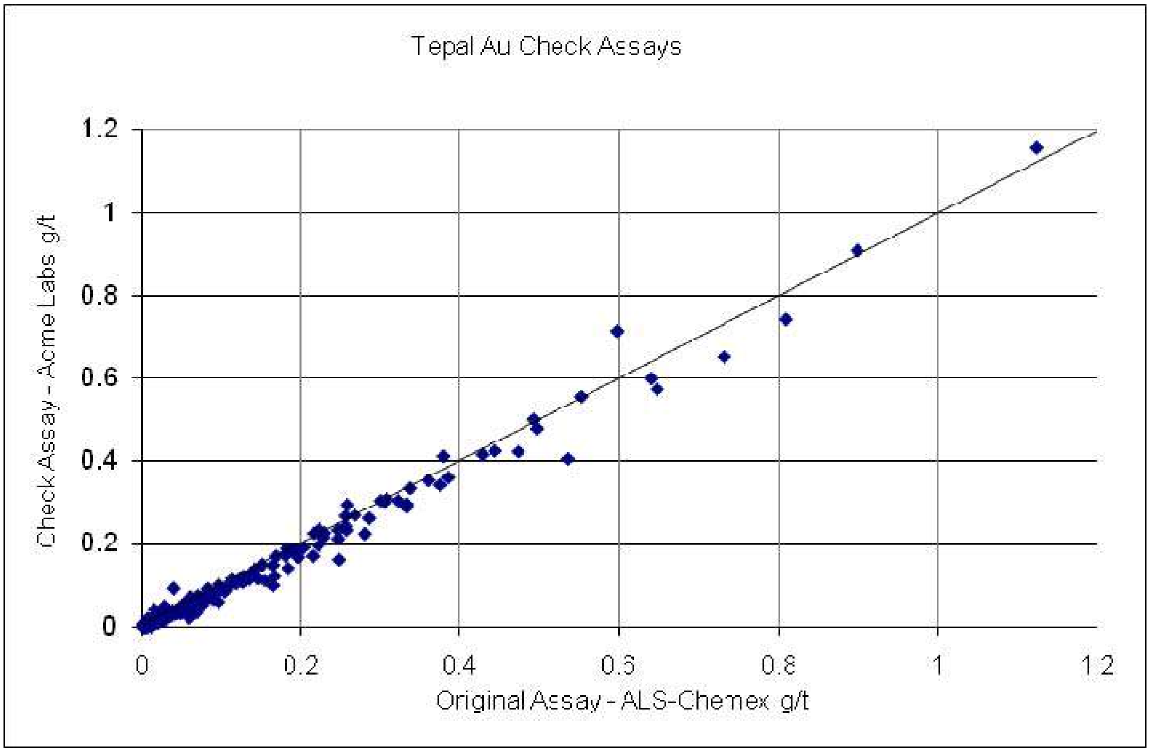

Figure 13.21: Comparison between Original ALS and ACME Pulp Re-assay for Gold

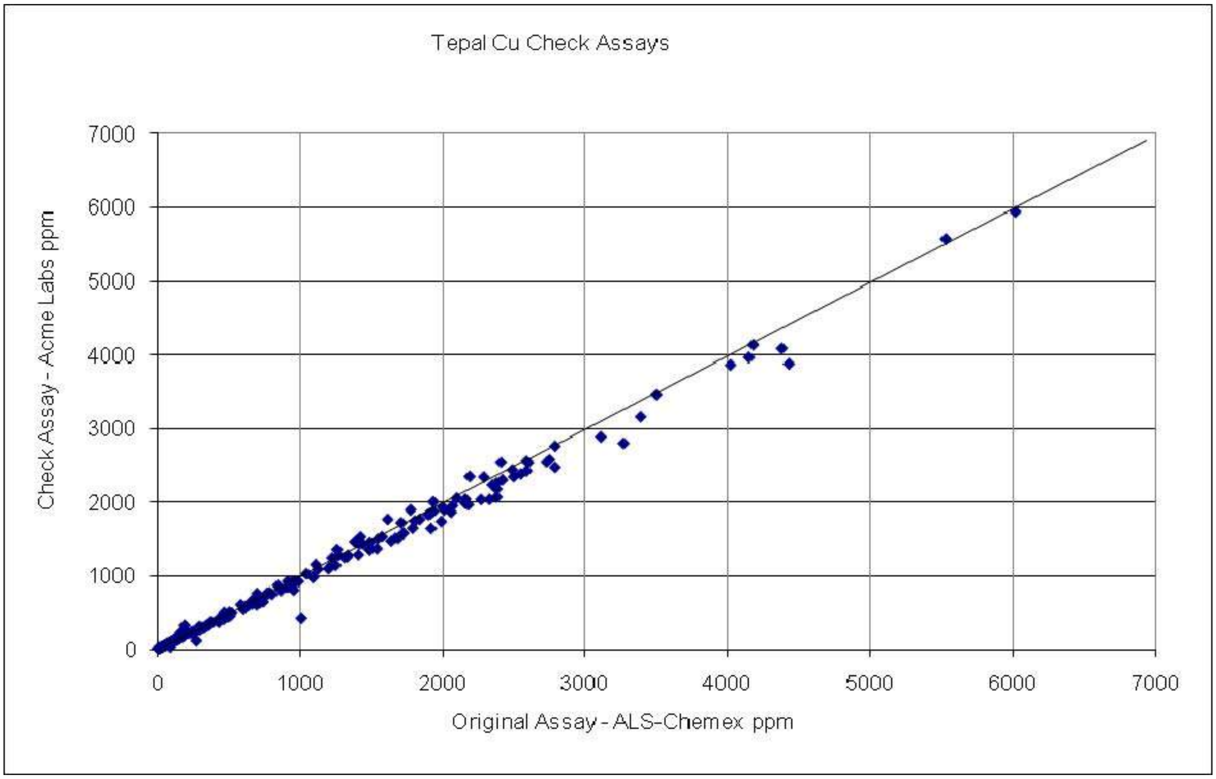

Figure 13.22: Comparison between Original ALS and ACME Pulp Re-assay for Copper

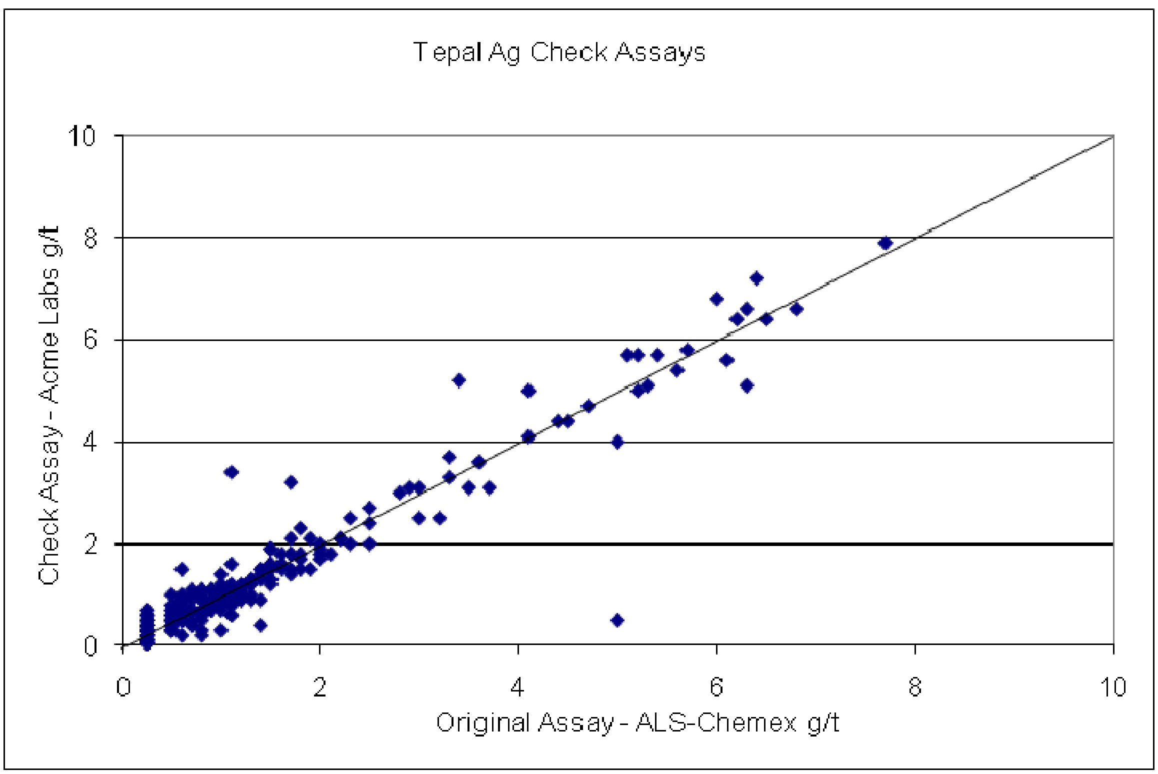

Figure 13.23: Comparison between Original ALS and ACME Pulp Re-assay for Silver

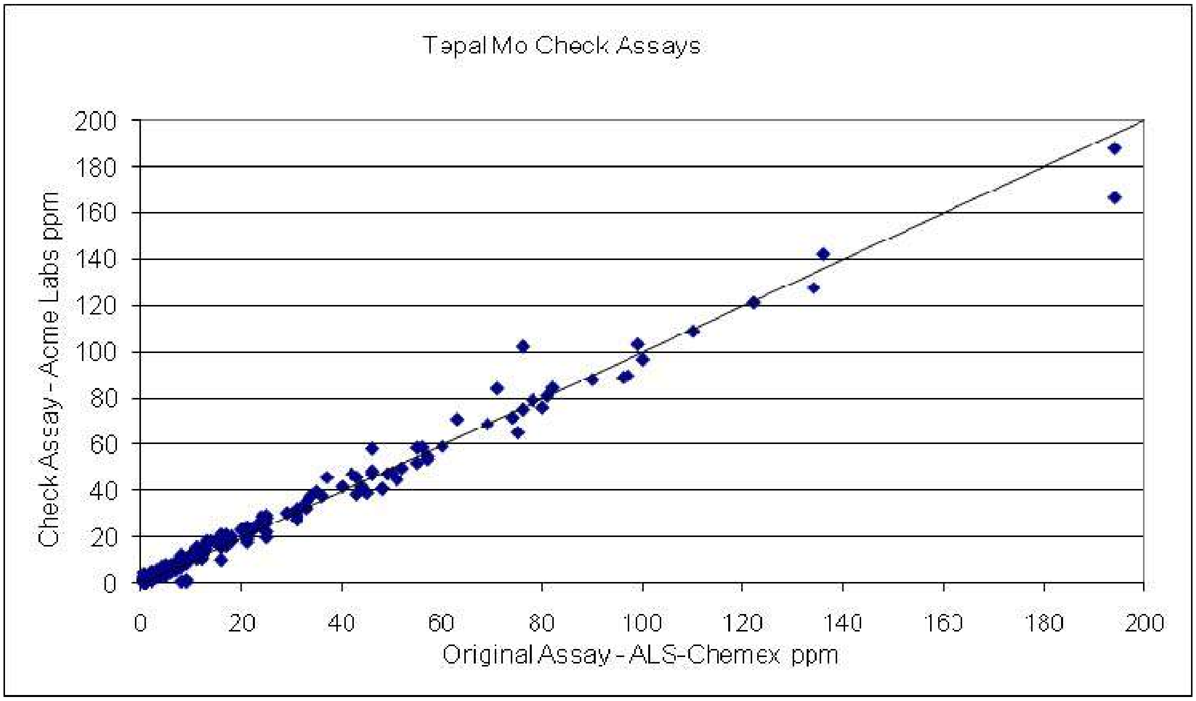

Figure 13.24: Comparison between Original ALS and ACME Pulp Re-assay for Molybdenum

Geologix also 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 and 234 Teck pulps were selected and sent for re-assay. The Hecla pulp re-assays were carried by ACME laboratory while the Teck re-assays were carried out by ALS. Results of the re-assay program returned very similar results to the original data entered in the database for the historical drill holes. Figure 13.25 and Figure 13.26 display the comparison between the original Hecla assay and the Geologix pulp re-assay program. The Geologix re-assays display very good correlation with the original Hecla assays.

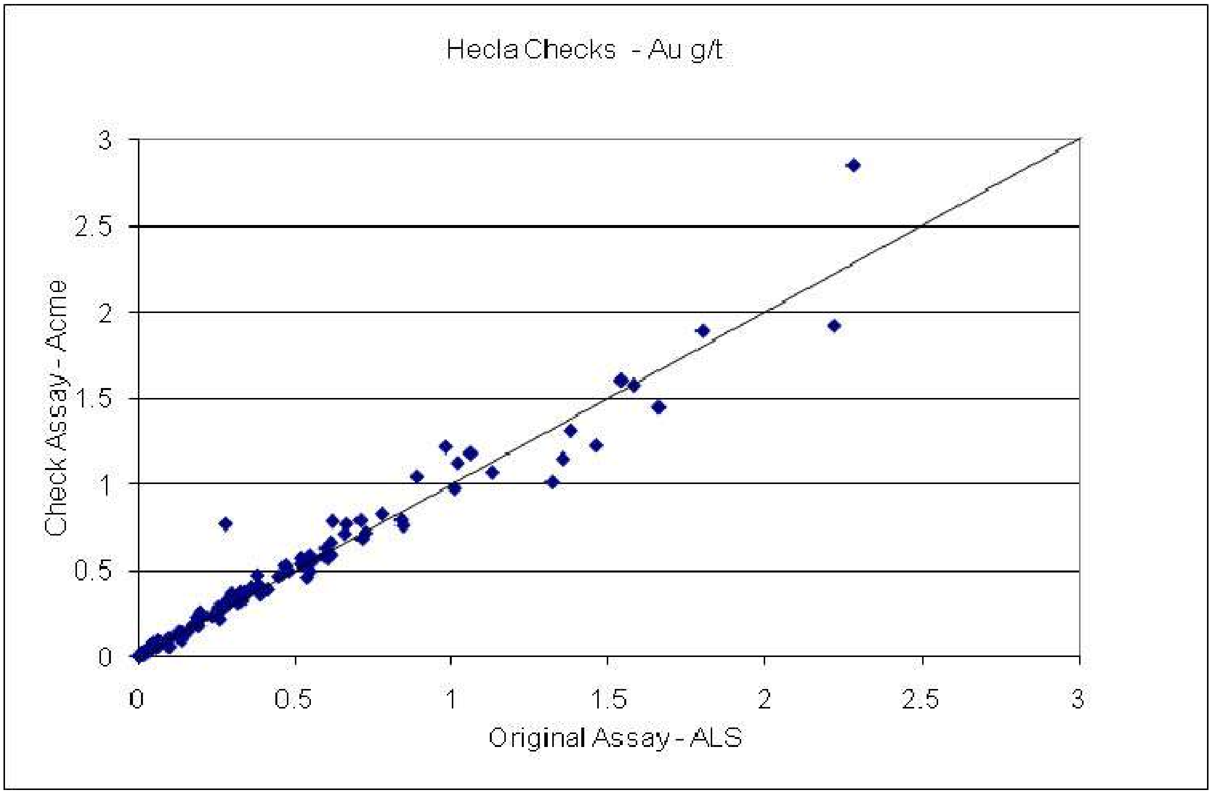

Figure 13.25: Comparison of Hecla assay and Geologix pulp Re-assay for gold

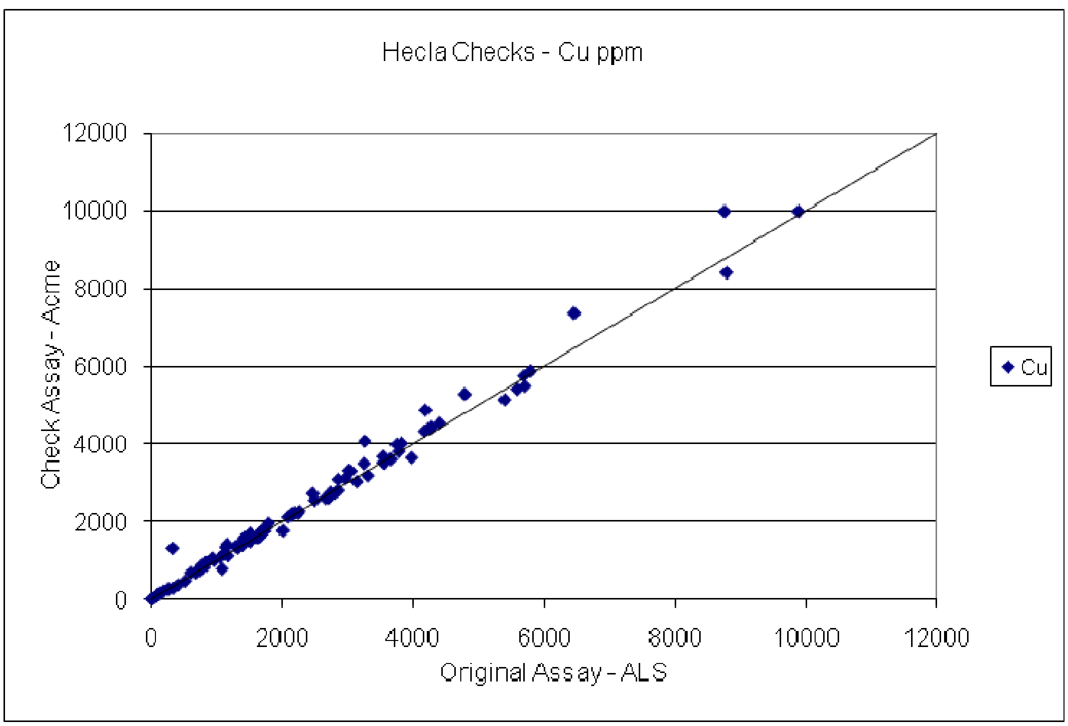

Figure 13.26: Comparison of Hecla Copper Values and Geologix Pulp Re-Assay for Copper

Figure 13.27 and Figure 13.28 show the result of the Teck pulp re-assay program. The re-assayed pulps agree well with the original data. Gold displays a wider scatter than copper and the re-assays seem to return slightly higher gold values than the original Teck assays.

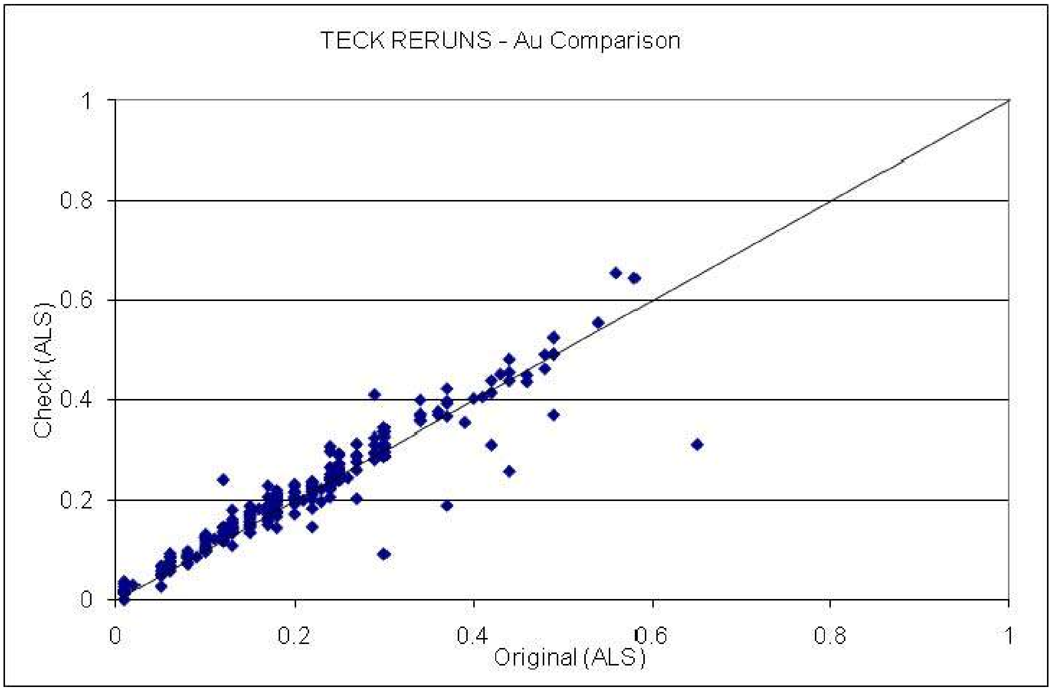

Figure 13.27: Comparison of Teck gold values and Geologix pulp re-assay program

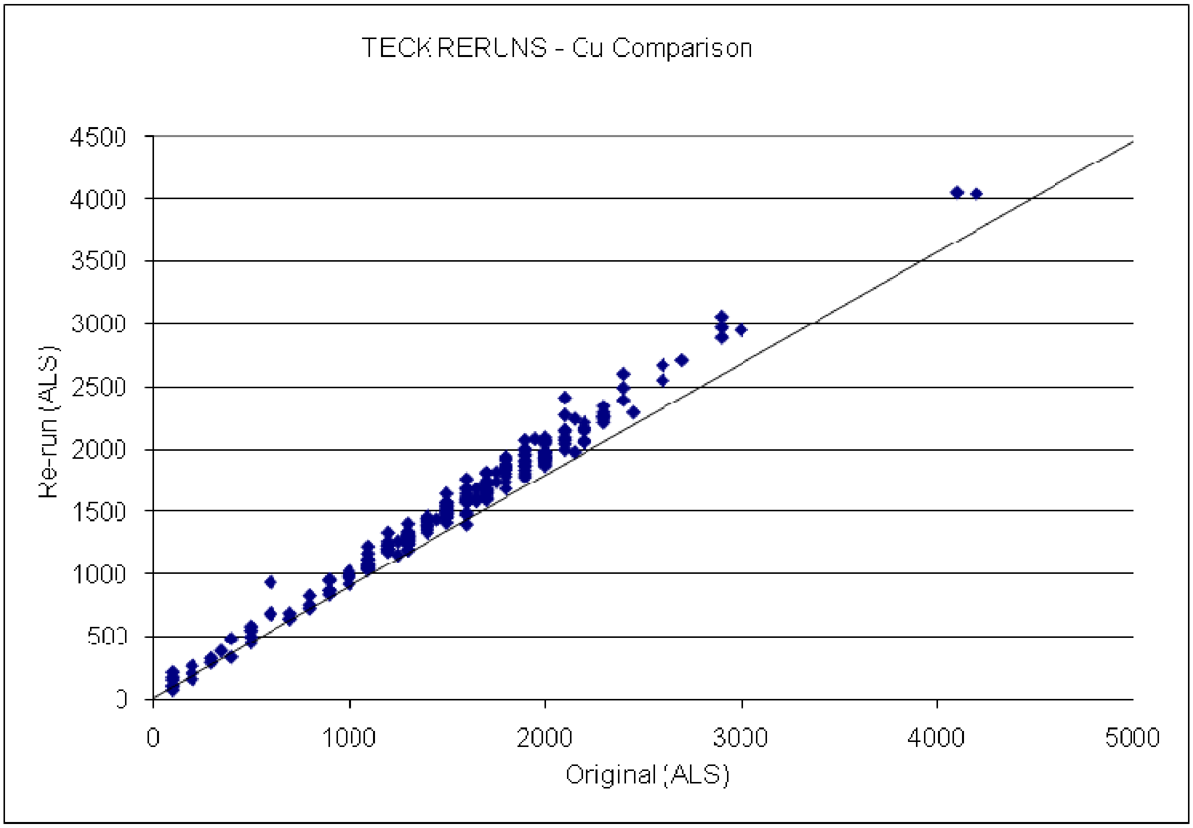

Figure 13.28 Comparison of Teck Copper Values and Geologix Re-assay Program

Pulp duplicate assessment shows repeatability of historical Au assay data is reasonable with correlation coefficients of 0.65 for Teck gold and 0.96 for the Hecla samples respectively. Pulp duplicate assessment of Cu values returned equally satisfactory correlation coefficient values of 0.98 for both the Teck and Hecla re-assays.

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

Duplicate analysis shows a good level of precision for both gold and copper.

Because the geology in the Hecla drill-holes indicate a good correlation with Arian's drill-holes, and because of the excellent correlation between the original Hecla assays and the Geologix re-assay program, SRK decided to include the Hecla drill holes in the Tepal database.

13.2.5 QA/QC Conclusions

On the whole, it is considered that QA/QC results do not demonstrate a systematic sample bias. Results of this work indicate that the analytical techniques employed by Inspectorate and ALS are generally reliable in producing assay data that demonstrates a good level of accuracy and precision with ALS performing slightly better than Inspectorate. However the occurrence of significant errors in a limited number blank samples show that there has been a potential miss-numbering during the Arian part of the program. These issues have since been corrected by the Geologix staff. CRM and duplicate analysis indicate that there is no significant bias to over or underreporting of assay results.

SRK is of the opinion that of the number of CRM samples and blanks used by Geologix is in keeping with best industry practices and sufficient for the estimation of mineral resources.

Assay results from drilling and sampling programs implemented during 2006-2007 may be regarded as representative of the samples collected.

13.2.6 Analytical Laboratories

Inspectorate Laboratories are accredited to relevant national and international standards and ISO 9001:2000 registration ISO 17025 quality assurance accreditation.

ALS laboratories in North America are registered to ISO 9001:2000 for the “provision of assay and geochemical analytical services” by QMI Quality Registrars. In addition to ISO 9001:2000 registration, ALS’s North 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”. CAN-P-1579 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).

13.3 Previous Howe Verification Sampling

During previous studies on the Tepal project by Priesmeyer in 2007, Howe collected a total of eleven samples from the property (Table 13.6). All samples were collected under Howe's direct supervision and were placed in appropriately numbered sample bags and sealed at the project site. These samples were sealed in sacks and transported by Howe to the ALS sample preparation facility in Guadalajara, Mexico.

The Howe samples were crushed to 75% passing 2 mm followed by the pulverization of a 250 g split in chromium steel to 85% passing 75 microns. The gold content of these samples was determined by means of atomic adsorption on a 50 g sub-sample. Each sample was also analyzed for 32 other elements by inductively coupled plasma preceded by an aqua regia digestion.

Seven rock chip samples were collected from the property for the purpose of data verification. Due to the fact that samples collected by previous operators were all collected nearly ten years ago or more, it was difficult to identify sample locations from previous operators.

Howe collected five samples from areas in which the metal content was unknown and two from locations that had been previously samples by Arian. For the two locations sampled by both Arian and Howe, Howe's copper values were slightly higher. For one of the samples Howe obtained a significantly higher gold grade and for the other a significantly lower gold grade.

The inconsistency probably results from discontinuous chip samples being collected from slightly different areas than the originals. In addition, in Howe's experience it is common to have a high degree of variability in the reproducibility of gold assays. Howe is satisfied that its check samples have confirmed the presence of copper and gold in the selected samples.

There are no known coarse rejects or pulps that remain to be sampled for the purpose of verifying the data from the Hecla drilling, however core duplicates and sample splits from the Hecla drilling program have been preserved by the property owner in Tepalcatepec. The samples are stored in the original sample bags and for the most part are clearly marked. In some cases, the sample bags are stacked by drill hole and in others they are grouped by hole number and sample number in large sacks. Chip trays are also present and available for review.

Howe selected a further four samples from three drill holes to verify the original drill assays based on electronic files of analytical results from the Hecla drilling. Results from all four samples are very close to the original results, with two copper assays from the Howe sampling being higher and two being lower. Three of Howe's samples returned higher gold values that the Hecla results.

On the basis of Howe's data verification sampling, Howe was satisfied that its check samples have confirmed the presence of gold in the selected samples (Priesmeyer, 2007). However, the study highlights significant discrepancy in assay grades between original analyses and verification analyses.

Table 13.6: Howe's Previous Data Verification Sampling

Sample Number	Arian Sample Number or Drill Hole	Sample Width/Length (m)	UTM coordinates or From - To (m)		Original Copper Value (%)	Howe Copper Value (%)	Original Gold Value (g/t)	Howe Gold Value (g/t)
70258	37902	4.2	2116945	716547	0.25	0.52	1.24	3.33
70259	NA	4.3	2116992	716644	NA	0.24	NA	0.97
70260	NA	4.0	2117040	716624	NA	0.47	NA	1.32
70261	NA	3.0	2117002	716326	NA	0.11	NA	0.5
70262	NA	3.0	2116994	716594	NA	0.44	NA	1.17
70263	NA	3.8	2116847	716695	NA	0.11	NA	0.32
70264	37904	3.0	2115643	716760	0.04	0.06	0.41	0.13
70265	MHT-12	1.0	33	34	0.99	0.94	0.14	0.17
70266	MHT-3	1.0	39	40	0.85	0.91	3	3.37
70267	MHT-12	1.0	6	7	0.34	0.32	0.33	0.4
70268	MHT-6	1.0	109	110	0.18	0.19	0.67	0.66

14 Data Verification

14.1 Metallurgical Data Verification

Geologix has prepared metallurgical composites from both the oxide and sulphide portion of the Arian drill core material and the head assays reported from the North and South zone composites. Geologix drill core was used for Tizate zone composites. The analyses completed by G & T Metallurgical Services Ltd. and McClelland Laboratories Inc., described in more detail in Section 16 of this report, showed a strong agreement between the assay grades reported by Arian and Geologix and those reported from the metallurgical testwork. Supervision of the metallurgical testwork was completed by William Joseph Schlitt, QP to the metallurgical program.

14.2 SRK Verification

As part of the PEA, SRK carried out a site visit to the Tepal project. The purpose of the site visit was to verify access and infrastructure, review the geology of the property, review drill core and logging procedures and collect some surface samples of the mineralization. During the site visit, SRK collected four chip samples from surface outcrops of the North Tepal zone. The samples were collected from road cuts along drill roads. SRK is unaware if these sample sites have been previously sampled by Geologix. The samples were collected as a mean to independently verify the presence of copper and gold mineralization on the property at levels documented by the company. The results of the SRK sampling are presented in Table 14.1.

Table 14.1 assay results of SRK check samples

Sample No	Description	Location	Au (g/t)	Cu (%)
C048173	Grab of oxidized tonalite	718,519E; 2,116,594N	0.036	0.030
C048174	1m chip sample of weathered and oxidized tonalite	716,589E; 2,116,991N	1.000	0.530
C048175	2m chip sample of weathered and oxidized tonalite	716,585E; 2,116,545N	0.036	0.002
C048176	Same as C048175	716,453E; 2,116,809N	0.024	0.004

The SRK samples were delivered to ALS Chemex in Vancouver by SRK. At ALS Chemex, the samples were crushed to 75% passing 2 mm followed by the pulverization of a 250 g split in chromium steel to 85% passing 75 microns. The gold content of these samples was determined by means of atomic adsorption on a 30 g sub-sample. Each sample was also analyzed for 32 other elements by inductively coupled plasma preceded by an aqua regia digestion. The samples returned values that were to be expected given the random site selection. The random sampling does confirm that copper and gold mineralization occurs in surface outcrop in grade similar to what has been previously reported.

In addition to the site visit in March of 2011, SRK carried out a site visit in July 2010, to verify the geological characteristics of the deposit and to evaluate to possible slope angles and stability for possible pit designs. QP's Dino Pilotto and Bruce Murphy visited the site. During their site visit they inspected the area of the potential pit, waste dump, tailings facility and mill areas and verified that the sites were appropriate to support the designed infrastructures. They also viewed drill core to verify general geotechnical characteristics and rock type. The tour included a visit to local towns to view existing roads and electrical power infrastructure. General site conditions and geotechnical characteristics were verified. No restrictions were placed on the SRK QP's during the site visit.

15 Adjacent Properties

This report does not rely upon, nor is affected by, information from adjacent properties.

16 Mineral Processing and Metallurgical Testing

16.1 Introduction

The Tepal deposit is dominantly a copper-gold (Cu-Au) resource. The bulk of the resource (85 to 90%) is sulphidic, but is overlain by a distinct oxide zone. The sulphide responds well to milling, with production of a Cu-Au flotation concentrate. However, based on the current mine schedule, most of the oxide would be mined first. This material is a candidate for cyanide leaching, either in crushed ore heaps or coarse ore dumps. This would produce gold and some cyanide soluble copper. The latter would be removed from the gold circuit as a sulphide and combined with the concentrate using SART (sulphidation-acidification-recycling-thickening) technology.

To ensure that all process options were considered, milling and flotation of the oxide was also briefly investigated. This did produce a Cu-Au concentrate that could be leached. However, this option did not appear to offer any advantages over the more conventional heap leach approach in terms of recovery or cost.

Very little oxide-to-sulphide transition material has been encountered. Where it exists, most of the copper is still sulphidic and it responds well to flotation. Thus, any transition material will be mined and processed through the mill, along with the primary sulphide ore.

The balance of this section addresses the metallurgical testing that has been done on samples from the North, South and Tizate zones. It starts with a brief review of the limited testwork programs conducted by previous owners. Then the focus shifts to the current program being conducted by Geologix. This portion contains material on sample selection, the three phases of the milling and flotation program on the sulphide ores, and the bottle roll and column leach testing done on the oxide ore. G&T Metallurgical Services, Limited ("G&T") of Kamloops, British Columbia conducted the milling and flotation studies. McClelland Laboratories, Inc. ("MLI") of Sparks, Nevada conducted the majority of the leaching testwork. The quality assurance/quality control ("QA/QC") practices at both laboratories are discussed in a separate subsection. The final portion covers the conclusions.

Metric units are used throughout this section. Where English units are widely used, they are given in parentheses.

16.1.1 Historical Background

Apparently, neither Arian nor Hecla pursued a metallurgical testwork program on the property. Work done by two other previous owners is summarized below.

The International Nickel Company of Canada, Limited (INCO)

The earliest testwork done on the property was conducted by INCO at their J. Roy Gordon Research Laboratory in mid-1973. INCO viewed the property as a Cu-Au porphyry and focused on production of a co-product concentrate. The composite tested was from the first 88 metres of drill hole 57002. The head grade assay for this composite was 0.43% Cu, 1.3 ppm Au and 1.25 ppm Ag.

Following some preliminary grinding and flotation trials, two locked cycle tests were performed. The primary grind size was a P86 of 325 mesh (44 µm). The ore charge was conditioned for 10 min at 20% solids and a pH of 11 using lime, xanthate (0.1 g/kg) and a frother. Then rougher flotation was run for 10 min. This was followed by three stages of cleaning, apparently without regrinding, using the same pH and xanthate concentration. Flotation times were too long in the first locked cycle test and were shortened to 5, 4 and 3 min., for the three cleaner stages respectively. Results for the second test are summarized in Table 16.1.

Table 16.1: INCO Flotation Recoveries and Grade

Constituent	Final Cleaner Assay, % Or Ppm	Distribution In Concentrate, %
Cu	12.7	74.2
Au	41	~76
Ag	39	~75
Mo	260	~62

As can be seen, the INCO recoveries are reasonable, especially for the precious metals. However, the grade would be unacceptable and probably reflects the lack of a regrind step on the rougher concentrate. The tailings assayed 0.11% Cu, mostly as non-floating oxides. The gold content of the tailings was 0.25 ppm. The mode of occurrence of the gold in the tailings was not indicated.

Teck Corporation (Teck)

Unlike INCO, Teck viewed Tepal as a gold project and focused on cyanide leaching. The metallurgical work was done under contract at Lakefield Research, Peterborough, Ontario in mid-1993. Lakefield received six samples identified as T-101, 102, 103, 104, 110 and 114 and weighing about 5.5 kg each. Since the sample numbers do not match the Teck drill hole numbers, the origin of the samples is uncertain. Only samples T-103, 104, 110 and 114 were used to prepared composites to be tested. These had the highest gold grades, ranging from 1.07 to 1.36 g/t. Each of the four samples was blended and split in half. The halves were then blended to produce two composites. Composite 1 was crushed to minus 10 mesh (-2 mm). Composite 2 was retained in as-received condition with a ½-in. (12.5 mm) top size. The expected composite grade was 1.21 g/t Au and 4,775 g/t copper, of which 3,775 g/t (79%) was acid soluble. This composite appears to be similar in nature to the oxide ores currently being tested.

Composite 1 was further ground to a P100 of 65 mesh (~225 µm) and then subjected to cyanide bottle roll leach tests. The tests were run for 48 h on 500 g charges at 40% solids and pH 11. Three cyanide levels were tested: 5, 10 and 20 kg/t NaCN. The latter represented 100% stoichiometry for complete gold extraction. The best results were obtained at 5 kg/t, with 90% gold extraction in 24 h; increasing to 95% after 48 h. Corresponding levels of copper extraction were 4.5% and 5.3%. Cyanide consumption was 0.91 kg/t, similar to that in the current tests.

Composite 2 was split into three size fractions and leached for seven days at pH 11 and 1.5 kg/t NaCN, with cyanide added as needed to maintain 0.5 g/L NaCN. After just three days, the gold extraction was essentially compete and was the same for all three splits. This extraction level averaged 84%, with 0.75 kg/t cyanide consumption. The copper extraction was slower (5.5% after three days), so stopping the leach after just three days minimized cyanide consumption.

Because the bulk of the copper was present in oxide form, an acid leach test was also performed on the coarse ore sample. This was run at 40% solids for seven days using a sulphuric acid solution at pH 1.5. Copper extraction was fast, with 60% recovery in two days. At this point acid consumption was 20 kg/t. Extending the leach to seven days only increased extraction to 63%, but caused a 50% increase in acid consumption.

16.2 North and South Zone Metallurgical Programs

16.2.1 Sample Selection

None of the material that has been tested came from core or reverse circulation (RC) cuttings drilled by Geologix. This is because the metallurgical work began before Geologix undertook its first drilling campaign. Therefore, all samples were taken from core drilled by Arian. Details are shown in the following tables. The samples include material from the North Sulphide Zone (NSX), the North Oxide Zone (NOX) and the South Sulphide Zone (SSX). For some tests, the North Zone was divided into a northern section and a southern section. Later, samples from the South Oxide Zone (SOX) were included in the leach program at MLI.

All source-of-sample tables follow the same format. Each gives the composite or laboratory sample number, the drill hole number, the beginning and ending depth for the interval, and the Arian gold and copper assays for the interval.

Table 16.2 identifies the source of the samples used in the initial testwork at G&T. A 2-m interval from each drill hole was selected for preparation of the composites for the testwork. These composites were identified as NSX-1, NOX-1, and SSX-1. These samples were also used in the second program conducted at G&T. An additional sulphide composite from the North Zone, NSX-2, was included in the second G&T program. This was prepared the same way as the others, with source details given in Table 16.3.

The third phase of the testwork at G&T utilized two new sulphide composites, one from each zone. These were identified as NSX-3 and SSX-2. Preparation of these composites followed the same procedures as the earlier ones. The source details are given in Table 16.4.

All testwork conducted by MLI was performed on material from the oxide, rather than the sulphide zones. The oxide composites were drawn from both the South and North zones, with the latter further divided into north and south areas. Bottle roll leach tests were run on 11 samples taken from all areas of the resource, thus representing a variability study. Source information on these samples is presented in Table 16.5. As discussed later, bottle roll tests were also performed on pulverized splits from the oxide column composites. The sources for these composites are shown in Table 16.6.

The column composites are NOXCL01 (north end of North Oxide Zone), NOXCL2 (south end of North Oxide Zone), and SOXCL1 (South Oxide Zone).

Table 16.2: Drill Core Identification for Initial Set of G&T Samples

Composite	Drill Hole	Drill Hole Interval, m		Au ppm	Cu ppm
		From	To		
SSX-1	AS-07-013	96	130	0.302	1,440
SSX-1	AS-07-038	48	100	0.227	1,560
SSX-1	AS-07-015	40	90	1.196	6,500
SSX-1	AS-07-007	174	216	0.632	2,620
SSX-1	AS-07-009	16	56	0.127	670
Average of five selected sample				0.497	2,558
NSX-1	AS-07-004	62	106	1.273	6,600
NSX-1	AS-07-037	50	96	0.143	1,640
NSX-1	AS-07-014	134	180	0.528	2,650
NSX-1	AS-07-012A	122	152	0.358	1,970
NSX-1	AS-07-008	90	132	0.365	2,200
Average of five selected samples				0.533	3,012
NOX-1	AS-07-006	6	50	1.439	4,900
NOX-1	AS-07-014	10	40	0.112	2,580
NOX-1	AS-07-010	0	24	0.357	2,160
NOX-1	AS-07-030	16	46	0.463	3,010
NOX-1	AS-07-012A	36	64	0.387	1,970
Average of five selected samples				0.552	2,924

Note: From and To give the interval from which a 2-m section was selected.

Five 2-m intervals comprise each composite.

Table 16.3: Drill Core Identification for Composite NSX-2

Composite	Drill Hole	Drill Hole Interval (m)		Au ppm	Cu ppm
		From	To		
NSX-2	AS-07-004	62	106	1.328	5,500
NSX-2	AS-07-012A	122	152	0.358	2,150
NSX-2	AS-07-008	90	132	0.32	2,190
NSX-2	AS-07-037	50	96	0.215	2,550
NSX-2	AS-07-014	134	180	0.546	2,770
Average of five selected samples				0.553	3,032

Note: From and To give the interval from which a 2-m section was selected.
Five 2-m intervals comprise each composite.

Table 16.4: Drill Core Identification for Composites NSX-3 and SSX-2

Composite	Drill Hole	Drill Hole Interval, m		Au ppm	Cu ppm
		From	To		
NSX-3	AS-07-012A	144	146	0.447	2,040
NSX-3	AS-07-006	164	166	0.519	2,930
NSX-3	AS-07-006	76	78	1.066	6,100
NSX-3	AS-07-030	58	60	0.427	2,380
NSX-3	AS-07-008	120	122	0.385	1,830
NSX-3	AS-07-004	110	112	0.505	2,180
NSX-3	AS-07-014	164	166	1.41	6,800
NSX-3	AS-07-014	52	54	0.115	2,270
NSX-3	AS-07-016	46	48	1.005	6,500
NSX-3	AS-07-037	70	72	0.204	2,340
NSX-3	AS-07-037	162	164	0.12	2,030
NSX-3	AS-07-038	120	122	0.234	2,170
NSX-3	AS-07-006	182	184	0.231	2,060
NSX-3	AS-07-010	78	80	0.305	2,630
Average of five selected samples				0.498	3,161
SSX-2	AS-07-039	94	96	0.409	1,240
SSX-2	AS-07-009	92	94	0.473	2,300
SSX-2	AS-07-007	48	50	0.339	2,890
SSX-2	AS-07-007	196	198	0.641	2,280
SSX-2	AS-07-038	104	106	0.476	2,570
SSX-2	AS-07-001	160	162	0.714	3,860
SSX-2	AS-07-001	174	176	0.87	5,300
SSX-2	AS-07-015	34	36	0.37	1,190
SSX-2	AS-07-033	34	36	0.119	1,180
SSX-2	AS-07-033	58	60	0.316	2,460
SSX-2	AS-07-013	76	78	0.418	1,940
SSX-2	AS-07-005	50	52	0.57	1,660
SSX-2	AS-07-005	68	70	0.64	1,810
SSX-2	AS-07-005	98	100	1.017	6,800
Average of five selected samples				0.527	2,677

Table 16.5: Drill Core Identification for MLI Bottle Roll Tests

MLI Sample No.	Drill Hole	Drill Hole Interval (m)		Au ppm	Cu ppm
		From	To		
CY-1	AS-07-006	20	22	0.522	3,760
CY-2	AS-07-004	20.1	21.9	1.68	6,500
CY-3	AS-07-037	10	12	0.659	610
CY-4	AS-07-014	24	26	0.265	3,130
CY-5	AS-07-016	6	8	0.288	2,830
Average of North-North Zone				0.683	3,366
CY-6	AS-07-030	18	20	0.369	680
CY-7	AS-07-008	18	20	0.781	1,120
Average of South-North Zone				0.575	900
CY-8	AS-07-038	10	12.65	0.385	1,480
CY-9	AS-07-005	8	10	0.445	4,040
CY-10	AS-07-015	4	6	0.67	1,460
CY-11	AS-07-001	5.3	8.9	0.252	1,980
Average of South Zone				0.438	2,240

Table 16.6: Drill Core Identification for MLI Oxide Column Test Composites

Composite	Drill Hole	Drill Hole Interval		Au ppm	Cu ppm
		From	To		
NOXCL01	AS-07-006	6	8	1.533	3,850
		22	24	0.414	3,940
NOXCL01	AS-07-004	11.1	12.4	0.733	5,100
		16.5	18.55	1.022	8,300
NOXCL01	AS-07-037	14	16	0.459	610
		20.25	22.05	0.241	8,080
NOXCL01	AS-07-010	2	4	0.565	2,860
		12	13.66	0.22	1,530
NOXCL01	AS-07-014	20	22	0.102	1,690
		30	32	0.139	1,850
NOXCL01	AS-07-016	2.2	4	0.321	3,270
		14	16.35	0.345	3,100
Average of selected samples				0.523	3,735
NOXCL02	AS-07-030	8	10	0.268	1,400
		22	24	0.451	1,940
		30	32	0.714	1,800
NOXCL02	AS-07-018	5.95	8.1	0.28	2,480
		12.1	14.2	0.271	1,210
		16	18	0.198	2,080
NOXCL02	AS-07-019	4	6	0.526	1,430
		10	12	0.225	700
		12	14	0.339	1,160
NOXCL02	AS-07-008	6.01	8	0.361	2,050
		10	12	0.398	1,430
		14	16	0.427	1,390
Average of selected samples				0.372	1,589
SOXCL01	AS-07-038	4	6	0.228	1,800
		15.5	17.5	0.406	2,570
SOXCL01	AS-07-005	6	8	0.803	4,020
		10	12	0.93	3,600
SOXCL01	AS-07-015	2	4	0.668	1,340
		6	8	0.644	5,600
SOXCL01	AS-07-001	8.9	10.5	0.231	2,400
		10.5	12	0.258	2,520
SOXCL01	AS-07-009	6	8	0.514	5,300
		4	6	0.549	4,500
SOXCL01	AS-07-007	0	2	0.5	2,630
		4	6	0.445	2,530
Average of selected samples				0.52	3,239

The single most important factor in a metallurgical testwork program is how well the samples being tested represent the ore type or portion of the resource being studied. The samples for the program were selected by the Geologix geologist in an effort to provide representative material. Best efforts were made in selecting samples that met the following criteria:

Collect samples that were spatially representative of each zone.

Collect samples that were representative of all grade ranges within each zone.

Ensure that the weighted average grade for each zone was a close as possible to average deposit grade.

The spatial representation of the samples can be seen in Figure 11.1, which shows the location of the Arian drill holes. Material available for selection of the oxide composites was more limited than the sulphides. A as a result, preparing a representative composite was more difficult and the variation from the average grade of the deposit was greater than it was for the sulphides.

Table 16.7 shows a comparison between the composite grades and the grades given in the resource report. The overall average gold and copper composite grades are slightly higher than resource grades. However, most gold grades are less than 0.1 g/t higher and most copper grades differ by 0.1% Cu, or less. The only significant difference is in the low values for NOXCL02. However, this reflects reality, as the southern portion of the north zone has lower gold and copper grades than the northern portion.

Table 16.7: Comparison of Composite Sample Grades and Resource Grades

Composite	Weighted Sample Grades		Resource grades	
	Au, g/t	Cu, %	Au, g/t	Cu, %
NSX-1	0.533	0.3	0.45	0.25
NSX-2	0.553	0.32	0.45	0.25
NSX-3	0.498	0.32	0.45	0.25
SSX-1	0.497	0.26	0.44	0.21
SSX-2	0.527	0.27	0.44	0.21
NOX-1	0.552	0.29	0.5	0.27
NOXCL01	0.523	0.37	0.5	0.27
NOXCL02	0.372	0.16	0.5	0.27
SOXCL01	0.52	0.32	0.44	0.22

16.3 Metallurgical Testing at G&T Metallurgical Services, Ltd. (G&T)

Three separate metallurgical testwork programs have been conducted at G&T. All have focused primarily on standard milling and flotation of the sulphidic portion of the Tepal deposit. The first was a broad scoping study undertaken in November 2009. Testing was completed in December 2009 and the final report on that work was issued in January 2010. The second program addressed gold recovery from the North and South Zones. This work began in January 2010 and was completed in February 2010, with the final report released in March 2010.

The third program once again focused on recovery of copper and gold using milling and flotation. The work began in late May 2010 and was concluded in July 2010. The final report was issued in August 2010. Each G&T program is discussed in more detail in the following sections.

16.3.1 G&T Scoping Study

This G&T program was intended to be a broad initial study on the metallurgy of the Tepal deposit. The composites tested included NSX-1, NOX-1 and SSX-1. There were four objectives in this part of the program:

- Characterize the chemical and mineralogical makeup of the ore based on the composites from the North Zone sulphide and oxide material and from the South Zone sulphide.
- Determine the grinding power requirements for the North Zone sulphide, which is the most important part of the resource.
- Begin development of a conventional milling and flotation process to recover the copper and gold in a salable concentrate.
- Asses the potential for cyanide leaching of the North Zone oxide to recover the gold.

The chemical analyses and mineralogical nature of the three composites are summarized in Table 16.8. The chemical and mineral contents were determined by using standard analytical techniques and QEMSCAN particle mineral analysis. As can be seen, the copper and gold head grades of all three composites are nearly the same. The sulphide zones are dominated by hypogene mineralization, while the oxide zone is nearly devoid of any copper sulphides. In the oxide, the copper is largely embedded in the chlorite or limonite.

Table 16.8: Chemical and Mineralogical Makeup of the Tepal Composites

Name	Symbol	Units	NSX-1	SSX-1	NOX-1
Elements					
Copper	Cu	%	0.25	0.21	0.26
Iron	Fe	%	4.3	4.7	6.3
Sulphur	S	%	2.11	2.16	0.08
Gold	Au	%	0.46	0.46	0.48
Silver	Ag	%	2	1	1
Acid Sol. Cu	CuOx	%	0.01	0.01	0.08
Cyanide Sol. Cu	CuCN	%	0.01	0.01	0.05
Minerals					
Chalcopyrite & Bornite	Cp & Bn	%	0.72	0.61	0.04
Covellite	Cv	%	0.003	0.02	0.02
Cuprite	Cup	%	0.01	- - -	0.01
Cu-Chlorite	Chl	%	- - -	- - -	0.18
Limonite	Lim	%	- - -	- - -	1.45
Pyrite	Py	%	2.59	4.08	0.09
Gangue	Gn	%	96.7	95.3	98.1

The Bond grinding work index ("Wi") was determined at a sieve size of 106 µm. The P80 of the feed was 1949 µm and the P80 of the final product was 78 µm. The resulting value of Wi for NSX-1 was 19.8 kWh/metric tonne ("mt"). Such an ore would be classified as "hard". No other comminution parameters were determined for any of the sulphide composites.

Most of the flotation testing was done on the North Zone composite. In the first test, the copper rougher float was followed by a pyrite float in an effort to maximize gold recovery. The intent was to determine whether or not a pyrite concentrate could be produced that was suitable for further processing to improve overall gold recovery.

With an initial grind of 150 µm, the first rougher concentrate contained 12% Cu and 19.4 g/t Au, for recoveries of 60% and 46.5%, respectively. The first pyrite concentrate contained 0.54% Cu and 2.20 g/t Au, giving recoveries of only 4% and 8%, respectively. Due to the high gold recovery in the copper concentrate, the pyrite option was not pursued further during this part of the program.

The test was repeated without the pyrite circuit. The first rougher concentrate graded 16.0% Cu and 21.9 g/t Au, with recoveries of 74.5% and 46.4% respectively. Overall rougher recovery was 87% and gold recovery was 63% at a mass pull of 3%. When the rougher concentrate was reground to 47 µm and floated in the cleaner circuit, the concentrate assayed 25.7% Cu and 25.8 g/t Au, with recoveries of 72.8% and 36.1% respectively. A second cleaner test at a finer regrind (31 µm) gave better results. The final concentrate graded 26.9% Cu and 33.9 g/t Au, giving recoveries of 82.1% and 46.9%, respectively. The silver content was 53 g/t, for a recovery of about 25%.

The concentrate quality was quite satisfactory. No minor elements were present at levels that would incur penalties. On the other hand, the gold and silver contents were both high enough to warrant payment for by-product credits.

An effort was then made to float the North Zone oxide composite without using any type of sulfidizer to improve oxide flotation. Only a rougher test was run. The combined concentrate assayed 0.92% Cu and 9.4 g/t Au, with a mass pull of 2.8%. Copper recovery was only 10.2%, but gold recovery was 56.2%.

A single flotation test was then run on the South Zone sulphide composite. Metallurgical performance was not as good as it had been with the North Zone material. At a nominal P80 of 150 µm, the first south sulphide rougher concentrate graded 7.2% Cu and 10.0 g/t Au, giving recoveries of 59.4% and 39.4%, respectively. Overall recovery at the rougher stage was 78.8% for copper and 59.0% for gold, but at a 12% mass pull. When reground to 27 µm and cleaned, the final concentrate assayed 26.8% Cu and 28.8 g/t Au, with respective recoveries of 62.3% and 27.9%. The latter is significantly lower that the gold recovery from the oxide composite.

Following the flotation tests, a standard 48-h bottle roll test was run on a 0.5-kg sample of the north oxide composite ground to a P80 of 162 µm. Cyanide additions totalled 2 g/kg at a pH of 11. Results were encouraging, with maximum gold recovery (79.8%) achieved in 24 h. Silver recovery was about 25%, the same as in flotation. Cyanide consumption was 1.4 kg/t and lime consumption was 2.9 kg/t. Extraction of copper by cyanide was not reported, but based on the cyanide consumption, it was likely significant.

The final part of the scoping study was to assess the potential for recovering gold from the rougher tailings using gravity techniques. This was a two-step process. A Knelson concentrator was first used to recover the gold. Then the gold was further concentrated by panning. The Knelson concentrator recovered about 60% of the gold in the tailings. The pan concentrate assayed just under 8 g/t, representing 39% of the gold in the tailing. This grade is relatively low, indicating that more work would be needed to optimize any tailings recovery process. Much of this gold is associated with pyrite, which may warrant further processing.

16.3.2 G&T Gold Recovery Study

The second study conducted by G&T focused primarily on gold recovery from the two sulphide zones. The South Zone composite was SSX-1, as used in the first study. The North Zone composite was a new one designated NSX-2. The objectives of the study were as follows:

- Compare the characteristics of NSX-2 with those of the sulphide composites used in the first study.
- Perform four bench scale cleaner tests to assess the metallurgical response of the samples at a targeted P80 87µm grind size for SSX-1 and a P80 158 µm grind size for NSX-2, using a conventional copper-gold flowsheet with sequential pyrite flotation.
- Assess the quality of the pyrite concentrates for gold recovery through cyanide leaching using standard bottle roll tests.

- Assess the quality of the final pyrite rougher tailings for gold recovery using the Knelson gravity concentration unit, followed by hand panning of the Knelson concentrate.
- Evaluate the gold occurrence in the pan concentrates, using an Automated Digital Imaging System (ADIS).

The South Zone composite was tested first. Flotation parameters were those established in the first program. At the P80 87 µm primary grind, flotation performance was much better than it had been with the P80 150 µm grind. With a 7% mass pull, the rougher concentrate contained 2.39% Cu and 3.63 g/t Au, giving recoveries of 82.8% and 57.6%, respectively. The cleaner concentrate graded 28.5% Cu and 37.6 g/t Au, with corresponding recoveries of 76.3% and 46.3%. The copper recovery was 20% better than it had been at the coarser grind and the gold recovery was nearly 50% better. Silver recovery was 23.6%. Even at the finer grind the pyrite rougher concentrate was low grade. It contained 0.19% Cu (6.0% recovery) and 1.38 g/t Au (19.7% recovery) while the rougher tails carried 22% of the gold.

The pyrite concentrates from the South Zone tests were combined and subjected to a standard 48-h bottle roll cyanide leach test. This extracted 8% of the feed gold. The bottle roll leach residue was the reground to 14 µm and given another 48-h bottle roll leach test. This extracted an additional 5.5% of the feed gold. Cyanide consumption was 1.2 kg/t and lime consumption was also 1.2 kg/t in the first test. In the test at the finer size, cyanide consumption rose to 2.4 kg/t, with lime consumption at1.9 kg/t. Again, copper extraction was not reported.

An effort was made to produce a gravity concentrate from the pyrite rougher tailings using the Knelson concentrator. The Knelson concentrate was then upgraded by hand panning. The final pan concentrate contained only 2.8% of the feed gold at a grade of 10.8 g/t.

The new North Zone sulphide composite was similar to NSX-1, having the same copper grade but a higher gold content (0.63 g/t vs. 0.46 g/t). Flotation performance was also similar. With a 5% mass pull the rougher concentrate assayed 4.92% Cu and 7.73 g/t Au, with recoveries of 87.3% and 64.5%, respectively. Silver recovery was 26%. The cleaner concentrate graded 26.0% Cu and 35.6 g/t Au, corresponding to respective recoveries of 78.3% and 50.5%. The pyrite concentrate was very similar to the South Zone product. The North concentrate ran 0.22% Cu (6.3% recovery) and 1.44 g/t Au (18.9% recovery). Leaching of this product extracted 51.4% of the gold in the concentrate, or about 10% of the gold in the feed. Cyanide consumption was 0.6 kg/t and lime consumption was 1.1 kg/t. The low gold grades of these tailings products, combined with the low recoveries, suggests that these approaches to increased gold recovery may not be economic.

16.3.3 G&T Flotation Optimization Study

The third program at G&T focused on optimizing the copper-gold flowsheet for flotation. Two new sulphide composites were used in this study, one from each zone (composites NSX-3 and SSX-2). The objectives of this program were as follows:

- Compare the characteristics of the new composites with those of the previous ones.
- Optimize the metallurgical response of each composite in rougher, cleaner and locked cycle testing.
- Analyze the concentrates from the best locked cycle tests to determine concentrate quality and the concentration of any impurities that might be above the threshold penalty levels.

Table 16.9 compares the chemical composition of all five sulphide composites used by G&T. As can be seen, all five had about the same copper, iron and silver contents. Gold grades ranged from 0.46 to 0.63 g/t. SSX-2 had a slightly lower sulphur level than the others.

Table 16.9: Comparison of Head Assays for the G&T Composites

Composite	Assays, % or g/t				
	Cu	Fe	S	Au	Ag
SSX-1	0.21	4.7	2.16	0.46	1
SSX-2	0.26	4.1	1.69	0.6	2
NSX-1	0.25	4.3	2.11	0.46	2
NSX-2	0.26	4.1	2.49	0.63	2
NSX-3	0.27	4	2.23	0.47	2

Table 16.10 compares the major mineral content of four of the five composites. SSX-1 had a higher pyrite content than the others, while NSX-1 had a lower quartz content. However, all are high in silica and have similar compositions and mineralogies. The main difference is in the calcite, which is lower in the North Zone than in the South Zone.

Table 16.10: Comparison of Composite Mineralogy

Mineral	Mineral Content, %			
	SSX-1	SSX-2	NSX-1	NSX-3
Cu Sulphides	0.73	0.85	0.85	0.73
Pyrite	4.7	2.25	2.87	2.73
Hematite	1.64	1.67	1.73	1.8
Quartz	32.4	38.5	26.9	32.7
Chlorites	10.9	10.2	14.8	10.2
Feldspars	19.8	17.9	33.5	27.6
Micas	19	18.1	9.6	17.9
Calcite	5.25	4.77	3.39	1.11
9 Others	5.6	5.8	6.4	5.2

Three types of laboratory tests were conducted to optimize copper-gold flotation. These started with rougher tests where the grind size was varied from 150 µm down to 100 µm. Five different collectors were also screened, including PAX to boost gold recovery. Various pH levels were tried, as well. These tests were followed by cleaner optimization tests. Variables included regrinding to a range of 49 µm down to 13 µm. Various collectors and dosages were also screened, along with different pH levels. The third type of tests involved locked cycle runs to simulate continuous operations. These were conducted utilizing the optimal conditions obtained in the rougher and cleaner tests.

Nine rougher tests were conducted at three nominal grind sizes (150, 125 and 100 µm) and three pH levels (9.5. 10.5 and 11.0), plus the reagent screening. There was little difference at the two finer sizes, but recovery did drop off at 150 µm. The pH level had only a minor effect on copper recovery. However, for both composites, gold recovery improved at pH 9.5. However, this was due to the increased mass pull, with more gold-bearing pyrite reporting to the rougher concentrate.

Five collectors were investigated: 208, 3418A, SEX, 5100 and PAX. The choice of collector had little effect on copper recovery from either composite. However, 3418A gave the best gold recovery with NSX-3 while SEX gave the best recovery with SSX-2. PAX also improved gold recovery for both composites, but this was due mainly to pulling more pyrite into the concentrates.

Six cleaner tests were run by varying the regrind size (nominal 15, 25, 35 and 50 µm), the pH (10.5 and 11.0) and the collectors (3418A and PAX). The regrind size did not have a major impact on either copper or gold recovery from either composite. The 25 µm regrind was selected as the best choice. The pH had no effect on the North Zone composite, but a pH of 11 gave the best copper and gold recovery from the South Zone composite. There was no difference in copper or gold recovery from SSX-2 with the two collectors. Copper recovery from NSX-3 was unaffected by the choice of collector, but PAX boosted gold recovery. However, this was simply due to the stronger collecting capabilities of the PAX reagent, which pulled considerable gold-bearing pyrite into the north concentrate. On this basis, 3418A was selected as the preferred collector.

Three locked cycle tests were run. The first test was conducted on the South Zone composite. Recoveries were good at 85% for copper and 58% for gold, with silver in the mid-20% range.

However, the copper grade was below 20%, which could make it difficult to market. Therefore a second test was run with a lower reagent dosage in an effort to reject more pyrite to tailings. The resulting concentrate grade was much improved, at 26.1% Cu and 32.7 g/t Au. Metal recoveries dropped only slightly, to 84% for copper and 52% for gold.

Only a single locked cycle test was run on NSX-3. As in previous work, the North Zone composite out performed the South Zone sample. The North Zone cleaner concentrate ran 27% Cu at 90% recovery and 33.8 g/t Au at 65% recovery.

Complete assays from the two final concentrates are shown in Table 16.11. It does not appear that any of the impurities exceed threshold levels for smelter penalties. The high gold values should make these concentrates highly desirable for toll smelters.

Table 16.11: Comparison of North and South Zone Concentrate Quality

Elements	Units	SSX-2 (Test 32)	NSX-3 (Test 34)
Aluminum	%	0.8	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
Fluorine	g/t	125	141
Gold	g/t	28.1	33.8
Iron	%	33.7	32.4
Lead	%	0	0
Magnesium	%	0.23	0.19
Manganese	%	0.01	0.01
Mercury	g/t	<1	<1
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

16.3.4 Leach Testwork at McClelland Laboratories, Inc. (MLI)

Two types of cyanide leach tests were conducted; standard bottle roll testing and column leach testing, with each type further discussed below. Samples are those described in Tables 16.5 and 16.6. Acid-base-accounting tests were also performed on the column leach residues. Ancillary comminution tests and oxide flotation tests were conducted by third party vendors. The latter included cyanide leaching of the reground concentrate and tailings in an effort to increase gold recovery.

Bottle Roll Testing

A total of 14 bottle roll tests were performed; 11 on samples from throughout the deposit (CY-1 through 11) and three on splits from the three column composites (NOXCL01, NOXCL02 and SOXCL03). Thus, the samples represent a small-scale variability study. The three tests on column composites were done primarily to begin establishing a correlation between the gold and copper extractions from the fine bottle roll charges (minus 10 mesh or -2 mm) and the coarser column charges (+12.5 mm or + ½ in. top size).

All tests were run using a standard set of conditions. The samples were split from the blended composites, and then pulverized to a P80 of 1.7 mm. A nominal 0.5-kg sample was split out from the composite for a triplicate head assay and a1-kg sample was split out for leaching and the exact dry weight was recorded. Head assays were limited to gold, copper and silver. The natural pH was determined and the bottle roll charge was then loaded and leached at a nominal 40% solids. The exact volume of leach solution was recorded and the bottle was agitated on rollers for a total of 96 h. Agitation was interrupted after 2, 6, 24, 48, 72 and 96 h. A 100-ml aliquot of solution was withdrawn at each of these times. These solution samples were checked for pH and cyanide concentration and were assayed for copper and gold. Silver assays were attempted, but the silver content was generally at or below the level of detection. Cyanide concentration was maintained at 1 g/L by additions of sodium cyanide (NaCN) and the pH was held between 10.5 and 11.0 with additions of lime (CaO).

Results from the bottle roll tests are summarized in Table 16.12. In general, the results are positive. Gold extraction averaged just over 80%, with a fairly narrow range of 70 to 90%. There was little difference between the variability and composite samples, with the former giving slightly better recoveries on average. However, the variability samples exhibited a much lower average copper extraction than the composites (6 vs. 17%), even though there was little difference in the average head grades (2,195 vs. 2,385 g/t). The wide variation in copper extraction suggests that the copper mineralogy may vary across the deposit, being more soluble in some places than others. As a result of the higher copper extraction, cyanide consumption was also higher for the column composites than the variability samples (1.41 vs. 0.57 kg/t NaCN). Average lime consumption was much closer, 4.2 vs. 3.9 kg/t. Here lime consumption is based on the CaO content of the hydrated lime used as reagent.

Table 16.12: Summary of Bottle Roll Results for All Tests

Composite (Drill Hole or Composite)	MLI No	Au Head Grade (g/t)		Au Recovery (%)	Cu Head Grade (g/t)		Cu Extraction (%)	Ag Recovery (%)	Reagent Requirements		Natural pH
		Calc'd.	Assay		Calc'd	Assay			NaCN (kg/t)	Lime (kg/t)1	
MEX5601 (AS-07-006)	CY-1	0.53	0.49	90.6	2797	2773	4.4	10.5	0.45	1.7	7.4
MEX5602 (AS-07-037)	CY-3	0.53	0.49	83	602	647	4.2	16.7	0.43	6.8	4.3
MEX5603 (AS-07-016)	CY-5	0.16	0.16	81.3	2248	2227	3.9	10.5	0.38	4.2	6.8
MEX5604 (AS-07-014)	CY-4	0.2	0.19	70	2335	2327	4.2	50	0.4	2.6	7.2
MEX5605 (AS-07-004)	CY-2	1.34	1.18	80.6	5052	5033	15.5	55	2.08	2.1	5.9
MEX5606 (AS-07-030)	CY-6	0.37	0.34	78.4	710	700	6.6	18.2	0.22	4.6	5.4
MEX5607 (AS-07-008)	CY-9	0.9	0.83	83.3	1080	1070	4.6	16.7	0.52	9.2	3.9
MEX5608 (AS-07-038)	CY-7	0.41	0.38	85.4	1467	1450	13.2	10	0.67	2.2	7.1
MEX5609 (AS-07-005)	CY-10	0.47	0.44	78.7	4181	4093	3.1	14.8	0.45	3.4	6.2
MEX5610 (AS-07-001)	CY-11	0.38	0.37	76.3	2368	2320	5.4	18.8	0.53	2.6	7.7
MEX5611 (AS-07-015)	CY-8	0.58	0.56	84.5	1307	1287	0.5	9.1	0.15	3.8	7.8
Variability Average				81.1			6	20.9	0.57	3.9	6.3
N-N end Oxide (NOXCL01)	CY-14	0.48	0.47	77.1	2966	2943	25.7	15.8	2.35	4.9	4.3
N-S end Oxide (NOXCL02)	CY-12	0.38	0.39	84.2	1469	1457	10.3	16.7	0.68	4.9	5.7
S Oxide (SOXCL01)	CY-13	0.53	0.56	73.6	2717	2833	14.9	68.8	1.2	2.9	7.2
Composite Average				78.3			17	33.8	1.41	4.2	5.7
Overall Average		0.52	0.49	80.5	2236	2226	8.3	23.7	0.75	4	6.2

Note 1. The lime requirement is based on the CaO content of the hydrated lime used as reagent.

In most samples, the extraction of gold was very rapid, with at least 60% of the gold solubilized in six hours, or less. In a few cases extraction exceeded 80% in six hours. A typical leach curve is shown in Figure 16.1. About a third of the samples were leached to exhaustion in 24 h, while another third were still leaching, albeit quite slowly, when the tests were terminated after 96 h. The remaining tests reached their extraction limit in 48 to 72 h. One sample, MEX5605 (CY-2) had an atypical leach curve (see Figure 16.2) that was pseudo-parabolic. This sample was the highest grade of any tested, with 1.18 g/t Au. The large amount of gold was more than twice the average head grade. Thus, the sample may have simply required more time to leach that much metal. Copper leaching was somewhat slower than gold, with most copper leached to exhaustion between 48 and 72 h. In three samples, copper extraction was continuing slowly when the test was terminated.

Other results showed that the natural pH of the samples varied from acidic to neutral (pH 3.9 to 7.8). The average was near neutral at 6.2. Although not a major consideration due to its low value, silver recovery varied widely, from 68.8 to 9.1 %. The average was about 25%. The average back calculated head grades agreed closely with the average head assays. This indicates that there were no significant assay accountability issues. For gold the average back calculated grade was 0.52 g/t vs. 0.49 g/t for the head assay. For copper the respective averages were 2,236 vs. 2,226 g/t.

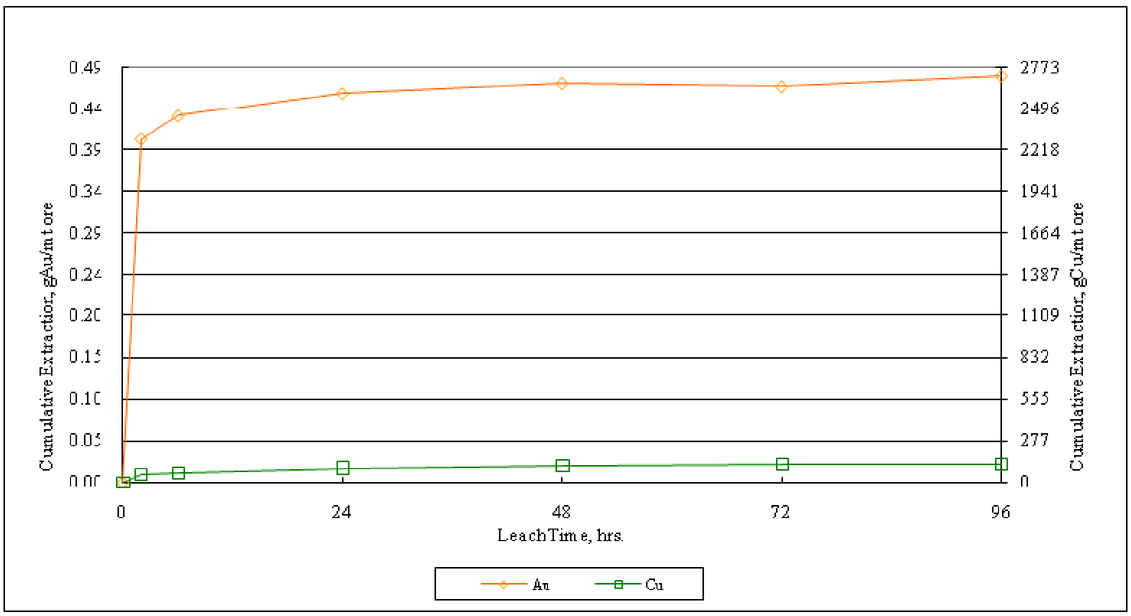

Figure 16.1: Typical Leach Curve for Fast Leaching Sample (MEX5601 or CY-1)

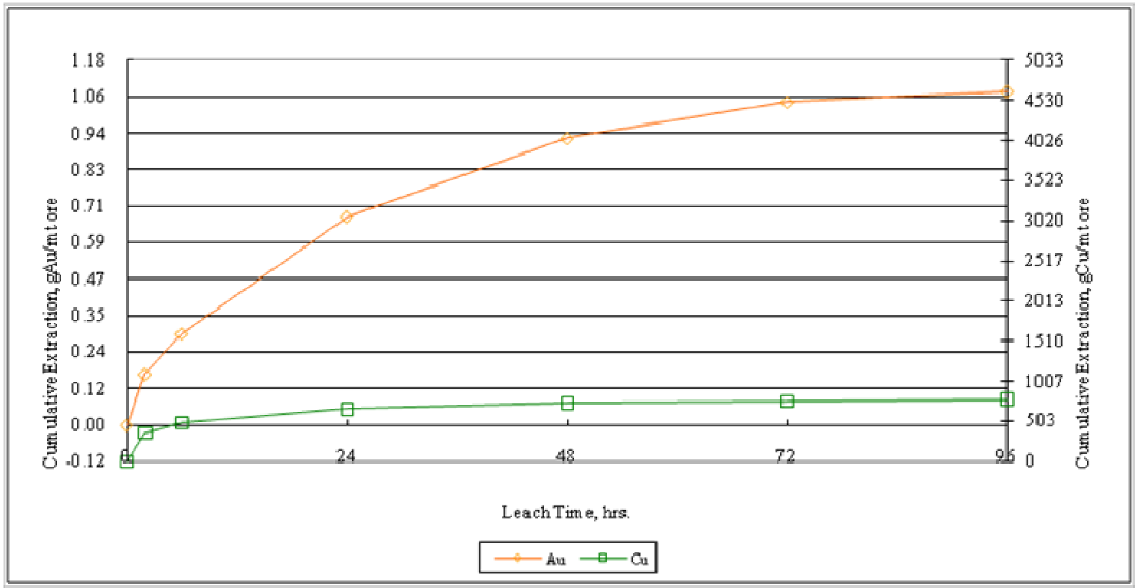

Figure 16.2: Atypical Leach Curve with Pseudo-Parabolic Form (MEX5605 or CY-2)

Column Leach Tests

Column Test Procedures.

Three column tests were run on Tepal oxide composites, one each from the north end of the North Zone (P1), the south end of the North Zone (P2) and the South Zone (P3). These followed the earlier bottle roll tests. Prior to testing, a split from each composite was screened into size fractions, which were then weighed and assayed for gold and copper. Another split was taken for triplicate heads assays covering gold, copper and silver.

The results from the composite bottle roll tests were used to estimate the lime additions for the columns. These were 3.9 kg/t for the North composites and 2.3 kg/t for the South based on 80% of the corresponding bottle roll requirements. The columns were 100 mm (4-in.) in diameter by 3 m tall. Each column was charged with about 30 kg of ore (exact weight recorded) with a nominal P80 crush size of 12.5 mm. This size meets the requirement that the column diameter be at least eight times the particle top size in order to avoid wall effects during leaching, i.e. the short circuiting of solution along the column walls. The material did not require agglomeration prior to charging into the columns.

After charging with the ore and lime mix, each column was leached with 2.40 L/day of solution containing 0.5 g/L NaCN. This leach rate was 0.2 L/min/m2 or 0.005 gpm/ft2. On a daily basis, the volume of pregnant leach solution (PLS) draining from each column was recorded and a 30 ml aliquot of solution was taken. This was checked for pH and analyzed for gold, copper and cyanide contents. Silver concentrations were found to be below the level of detection. The coarse composites proved to be more acidic than expected, especially the north-north column (P1). Thus, the initial pH levels were lower than desired, initially in the pH range of 6 to 7.

In order to get to the pH range of 10.5 to 11, caustic soda (NaOH) was added to the feed solution to each column, along with any cyanide additions needed to maintain the 0.5 g/L concentration. Total caustic additions to P1, P2 and P3 were 3.85, 3.32 and 2.84, kg/t, respectively.

All columns were run for 88 days, followed by a 5-day drain down period. The 88-day period included two 14-day rest cycles (only one for P1) and a rinse cycle to remove soluble gold and residual cyanide prior to drain down. The drainage volume was recorded to determine the solution holdup when under active leach. The results showed that the columns contained about 19% water under active leach and 8 to 10% when fully drained.

The rest cycles were run when the gold content of the PLS dropped down to values near the detection limit. During the rest cycle, the cyanide in the residual solution within the column was able to solubilize more metal values, maximizing extraction.

During leaching, the PLS was run through three carbon columns operating in series (four for P1). This was done to remove the gold and silver, producing a barren solution for recycling to the columns. At the end of the tests the carbon was also checked for copper loading.

Following each drain down, the leach residue was removed and a split was immediately removed for final moisture determination. The balance was then dried and reweighed. After drying, the residue was rescreened and each size fraction was assayed to determine the final gold and copper extractions as a function of particle size.

Column Test Results.

The predicted, back calculated and average triplicate head assays all agreed closely for gold. For copper the predicted values were slightly higher than the direct or calculated head assays. However, precision was still excellent. Results are summarized in Table 16.13 and 16.14. The small standard deviation and high level of precision indicate that there were no significant assay accountability issues that might affect the results.

Table 16.13: Comparison of Gold Head Assays for Column Composites

Determination	Composite Gold Head Grade (gAu/t)		
	North-North Oxide (NOXCL01)	**North-South Oxide (NOXCL02)**	**South Oxide (SOXCL03)**
Predicted Assay	0.51	0.37	0.53
Average Direct Assay1	0.47	0.39	0.56
Calc'd. Bottle Roll 1.7 mm	0.48	0.38	0.53
Calc'd. Head Screen 12.5 mm	0.49	0.37	0.51
Cacl'd. Column, 12.5 mm	0.45	0.36	0.51
Weighted Average	0.47	0.38	0.54
Standard Deviation	0.02	0.02	0.03
Precision, %	95.7	94.7	94.4

Note 1. Average of triplicate assays.

Table 16.14: Comparison of Copper Head Assays for Column Composites

Determination	Composite Copper Head Grade, ppm Cu		
	North-North Oxide (NOXCL01)	North-South Oxide (NOXCL02)	South Oxide (SOXCL03)
Predicted Assay	3640	1592	3266
Average Direct Assay1	2943	1457	2833
Calc'd. Bottle Roll 1.7 mm	2966	1457	2717
Calc'd. Head Screen 12.5 mm	2917	1463	2728
Calc'd. Column, 12.5 mm	3060	1470	2717
Weighted Average	2962	1460	2777
Standard Deviation	118	26	63
Precision, %	96	98.2	97.7

Note 1. Average of triplicate assays.

Recoveries and reagent consumptions for the three columns are shown in Table 16.15. Silver values were not reported by MLI for the columns due to the low silver head grades. The values shown are based the cumulative extraction in the columns, with head grades from the bottle roll composites. Key values for the column and bottle roll tests are compared in Table 16.16. Silver recovery is not shown, but averaged 23.6% in the columns and 17.0% in the bottle rolls. The much longer exposure period in the columns may explain the higher column extractions.

Care should be exercised when using the column test lime consumptions. These were based on the column composite bottle roll lime demand. The coarse ore proved to be more acidic than expected, so that the amount of lime added was insufficient to maintain the desired pH level. Thus, the reported lime additions are biased low. Additional alkalinity had to be provided during the test, in order to hold the pH level between 10.5 and 11. This was done by adding caustic (NaOH) to the barren solution being returned to the columns. Because the hydrated lime and the caustic solution may not react with the ore in the same manner, estimating the lime equivalent of the caustic is uncertain. One approach would be to base the estimate on the molecular weights of CaO and NaOH required. When this is done the equivalent lime requirements are 5.2 kg/t for P1, 4.8 kg/t for P2 and 3.5 kg/t for P3. However, further tests are required to confirm the lime demand. The lower lime consumption in the South Zone composite is likely due to the higher calcite content in this part of the deposit.

Table 16.15: Summary of Column Leach Results

Composite	Metal Recovery (%)			Reagent Consumption (kg/t)		
	Gold	Copper	Silver1	NaCN	Lime2	NaOH
NOXCL01	75.6	21.8	10	1.99	3.9	3.86
NOXCL02	86.1	11.2	31	1.38	3.9	3.32
SOXCL03	72.5	8.9	30	1.39	2.3	2.84

Note 1. Silver recovery was not reported by the columns by MLI. Approximate values are based on the reported cumulative silver extraction in the columns and the head grades reported by MLI for the bottle roll samples of each column composite.

Note 2. Lime consumption is based on the CaO content of the hydrated lime used as reagent.

Table 16.16: Comparison of Key Column and Bottle Roll Results

Composite	Metal Recovery, (%)				Cyanide Consumption (kg/t)	
	Gold		Copper			
	Column	BR	Column	BR	Column	BR
NOXCL01	75.6	77.1	21.8	25.7	1.99	2.35
NOXCL02	86.1	84.2	11.2	10.3	1.38	0.68
SOXCL03	72.5	73.6	8.9	14.9	1.39	1.2
Average	78.1	78.3	14	17	1.59	1.41

The close agreement between the column and bottle roll gold extractions is encouraging, albeit with significantly different cycle times. These results show that a significant portion of the gold is cyanide soluble and can be extracted from coarse as well as fine material. Although confirmatory testwork would be required, similar extraction levels may be achievable from even coarser ore, given enough time. At this point, no diagnostic work has been done on the column residues to determine the nature of the gold that was not extracted.

The copper extraction was actually lower in the columns than in the bottle rolls, in spite of the longer leach cycles. This may be related to the lower surface area per unit weight in the columns. The high cyanide consumptions undoubtedly reflect the high levels of copper extraction.

Although the total column cycle time was 88 days, this included both rest and rinse cycles that produced little additional gold but increased copper extraction. Most of the gold extraction occurred much faster. Table 16.17 shows how quickly each column achieved 80, 90 and 98% of the final gold extraction. As can be seen, nearly complete gold extraction was achieved in less than two months. Less than 2% of the gold was extracted during the third month of the leach cycle. All three columns exhibited leach curves with a pseudo-parabolic shape, similar to Figure 16.3. An example for NOXCL02 (P2) is shown in Figure 16.3. This figure also shows the “bump” in copper extraction due to the rest and rinse cycle.

The initial leach rate for the north-north composite (Figure 16.4) may be biased to the low side. This column operated for some time at an excessively low pH, which would have been detrimental to gold extraction. As shown in Table 16.17, it reached 80 and 90% extraction more slowly than the others. However, once the pH finally reached the range of 10 to 11, the extraction rate increased and it reached 98% extraction more quickly than the south zone column (P3). The leach rate in Column P1 may also have been affected by the limited availability of the cyanide reagent due to the high soluble copper content. Unfortunately, there was not enough material remaining to repeat the test,

Table 16.17: Leach Cycle Time to Reach Various Extraction Levels

% of Total Gold Extraction	Leach Time, days		
	NOXCL01 (P1)	NOXCL02 (P2)	SOXCL03 (P3)
80	28	10	15
90	38	16	23
98	53	34	59

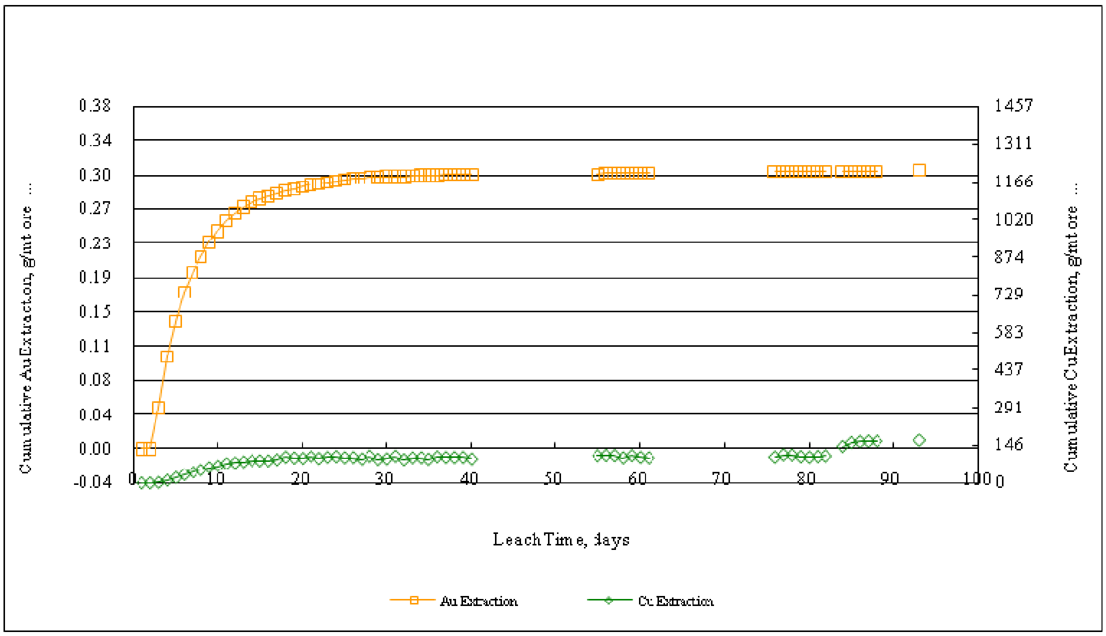

Figure 16.3: Column Leach Curve for Gold and Copper in Composite NOXCL02 (P2)

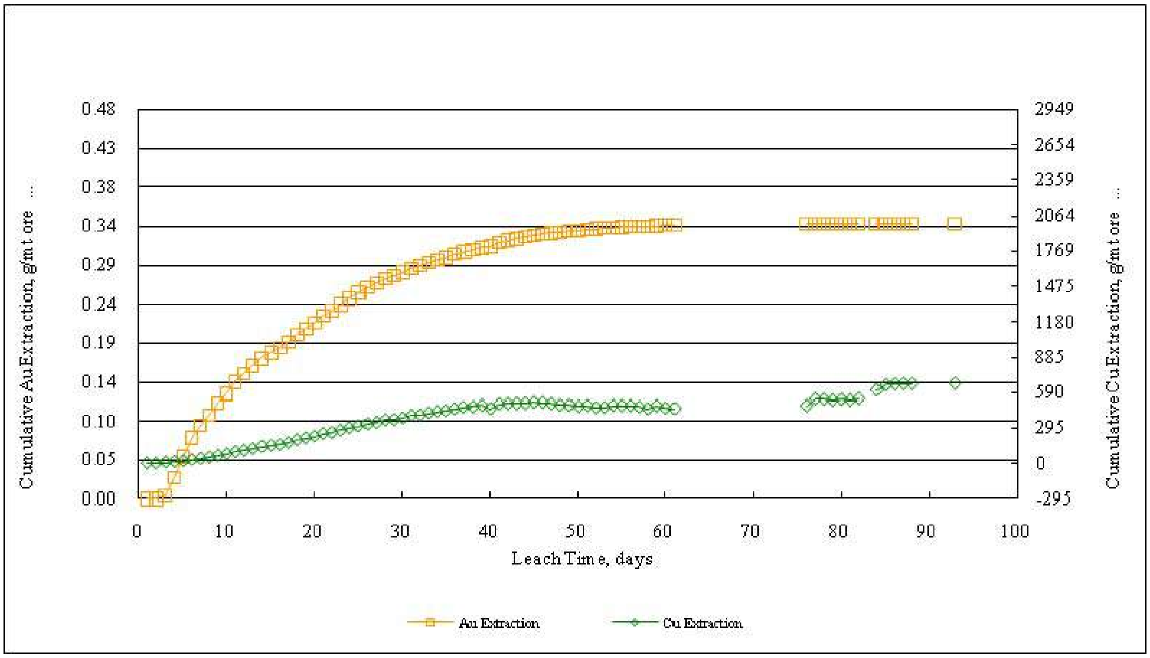

Figure 16.4: Leach Curves for NOXCL01 Showing the Slow Initial Leach Rate

In addition to whole ore assays and recoveries, screen size distributions were run on the heads and leach residues from all three composites. Each screen fraction was then weighed and assayed for gold and copper. The results provide information on recovery as a function of particle size, plus data on enrichment and the possible degradation of the ore during leaching.

Tables 16.18 to 16.20 give the size distributions, head and residue gold assays and the gold extraction by size fraction for each composite. Table 16.18 compares the copper assays in the heads and residues. As can be seen from the first three tables, there was no tendency for the particle size to decrease during leaching. In some cases the percentages of the coarsest sizes actually increased while the finest decreased. Apparently there was some chemical precipitation and particle adhesion taking place during the leach cycle. However, there was no evidence that this adversely impacted gold recovery or solution percolation. Table 16.21 shows that copper extraction increased as the particle size decreased.

All three composites had similar size distributions, with about 80% of the material in the plus 1.7 mm (plus 10 mesh) sizes and about 8% below 150 µm. There was little upgrading of gold with decreasing particle size, except in the finest size range. The fines represented 7% to 10% of the material, but carried 15 to 21% of the gold. Gold recovery from the fines averaged over 90%, while recovery from the coarser material was much lower and largely independent of the size. Gold recoveries from all but the finest fraction were below the average recovery. This demonstrates the importance of the fines to leach recovery. If coarser ore is heap leached, it is uncertain how the recovery will respond if the fines content is significantly decreased. This will have to be tested in future metallurgical programs.

Table 16.18: Size Distribution and Gold Recovery for NOXCL01 (P1)

Size Fraction	Weight Percent		Gold Content (g/t)		Gold Recovery, (%)
	Head	Residue	Head	Residue	
+12.5 mm	23.0	25.8	0.36	0.1	72.2
12.5 x 6.3 mm	33.7	32.8	0.46	0.13	71.7
6.3 x 1.7 mm	22.2	21.3	0.46	0.11	76.1
1.7 mm x 850 µm	5.3	4.9	0.39	0.1	74.4
850 x 420 µm	2.7	3	0.39	0.11	71.8
420 x 212 µm	1.7	1.8	0.52	0.18	65.4
212 x 150 µm	1	0.6	0.65	0.18	72.3
-150 µm	10.4	9.8	0.98	0.1	89.8
Composite	100	10	0.49	0.11	77.6

Table 16.19: Size Distribution and Gold Recovery for NOXCL02 (P2)

Size Fraction	Weight Percent		Gold Content (g/t)		Gold Recovery, (%)
	Head	Residue	Head	Residue	
+12.5 mm	21.3	21.4	0.36	0.07	80.6
12.5 x 6.3 mm	37.2	35.8	0.33	0.05	84.8
6.3 x 1.7 mm	23.1	22.9	0.33	0.05	84.8
1.7 mm x 850 µm	5.4	6.2	0.27	0.04	85.2
850 x 420 µm	3.2	3.6	0.26	0.04	84.6
420 x 212 µm	2.1	2.3	0.33	0.07	78.8
212 x 150 µm	0.9	0.9	0.33	0.08	75.8
-150 µm	6.8	6.9	0.88	0.05	94.3
Composite	100	100	0.37	0.05	86.5

Table 16.20: Size Distribution and Gold Recovery for SOXCL01 (P3)

Size Fraction	Weight Percent		Gold Content, g/t		Gold Recovery, %
	Head	Residue	Head	Residue	
+12.5 mm	19.4	21.8	0.51	0.16	68.6
12.5 x 6.3 mm	40.5	41.2	0.46	0.13	71.7
6.3 x 1.7 mm	21	19.5	0.46	0.16	65.2
1.7 mm x 850 µm	5.3	4.7	0.46	0.13	71.7
850 x 420 µm	2.6	2.8	0.47	0.14	70.2
420 x 212 µm	1.9	2.5	0.5	0.14	72
212 x 150 µm	0.9	0.7	0.64	0.14	78.1
-150 µm	8.4	6.8	0.9	0.09	90
Composite	100	100	0.51	0.14	72.5

Table 16.21: Comparison of Copper Assays in the Column Heads and Residues

Size Fraction	Copper Assays, g/t					
	NOXCL01 (P1)		NOXCL02 (P2)		SOXCL01 (P3)	
	Head	Residue	Head	Residue	Head	Residue
+12.5 mm	1,925	1.9	1,250	1,230	2,390	2,200
12.5 x 6.3 mm	2,750	2,490	1,470	1,240	2,670	2,470
6.3 x 1.7 mm	2,830	2,380	1,370	1,250	2,610	2,490
1.7 mm x 850 µm	3,170	2,420	1,465	1,320	2,750	2,500
850 x 420 µm	3,310	2,570	1,600	1,470	2,800	2,600
420 x 212 µm	3,700	2,850	1,875	1,660	3,190	2,760
212 x 150 µm	4,060	2,940	1,905	1,660	3,150	2,730
-150 µm	5,370	3,210	2,160	1,790	3,900	3,090
Composite	2,917	2,393	1,463	1,305	2,728	2,476

One area that requires further investigation is how to deal with the high concentration of cyanide soluble copper that builds up in the leach circuit. Table 16.22 summarizes the parameters related to the copper buildup in the leach circuit. As shown, on a mass basis there is far more copper being extracted than gold. As a result, the copper built up in solution, reaching levels as high as 2.1 g/L after a single leach cycle. The copper also tied up cyanide and loaded on the carbon, where it would end up contaminating the gold doré. Therefore, future work will be undertaken to study the removal of copper from the circuit, along with the recovery of the cyanide. One possibility is the SART (sulfidation-acidification-recycling-thickening) technology. This would remove the copper as a sulphide, which could be combined with the copper concentrate.

The columns displayed excellent stability during entire leach cycle. The column heights were unchanged, indicating that there was no “slumping” of the charge during leaching. In addition, there was no change in bulk density reported for the columns, showing that there was no decrease in the void space for solution flow. Finally, no standing solution was seen on top of the columns and there was no dry material in the residue when the columns were unloaded. This provides solid evidence that percolation through the ore was reasonably uniform and that no permeability problems developed during leaching.

Table 16.22: Parameters Related to Copper Build-up in the Leach Circuit

Parameter	Units	Composite (Column)		
		NOXCL01 (P1)	NOXCL02 (P2)	SOXCL03 (P3)
Head Grade	ppm Cu	2,962	1,460	2,777
Cu Extraction	g/t	664	165	240
Au Extraction	g/t	0.34	0.31	0.37
Extraction Ratio	Cu/Au	1,953	532	649
Max. Cu in PLS	ppm Cu	2,145	712	1,224
Carbon Loading	mg Cu/kg	283	12	22

While no solution problems were observed, the limited amount of material available restricted the column height to less than half the planned heap height (7 m). Because the oxide material proved to be soft (see below), it is possible that percolation problems could develop in a full-height ore lift or at the bottom of a multi-lift heap. Therefore, future tests will need to be run in full-height columns. Geotechnical testing of the fresh ore and leach residues should also be conducted to ascertain load bearing capacity of the ore and other geotechnical parameters that can influence solution flow and heap stability.

Following completion of the post mortem evaluation of the column leach residues, samples from each column residue were subjected to standard static acid/base accounting (ABA) tests. The objective was to determine if the residues would be considered non-acid generating wastes when exposed to the elements after heap leaching was terminated and closure was complete. The paste pH and complete sulphur speciation were determined for each residue. The results were used to calculate the acid generating potential (AGP), the acid neutralization potential (ANP) and the net neutralization potential (NNP). The latter was calculated as ANP – AGP = NNP. The ratio of ANP to AGP was also determined.

The results are summarized in Table 16.23. As can be seen from the table, with positive values of NNP and ratios > 1.0, all three residues would be classed as non-acid generating. In order of descending NNP and ratio values, the South Zone composite has the greatest neutralization potential, followed by the south area of the North Zone and then the north area of the North Zone. The appropriate regulatory agency will have to review the results and determine if further acid rock drainage (ADR) testing is required.

Following preparation of the column charges, there were about 8 kg of surplus North-South Zone composite (NOXCL02) remaining. This material was shipped to Phillips Enterprises L.L.C. in Golden, Colorado for comminution testing. Due to the limited amount of material and the relatively fine top size (~12.5 mm) only a Ball Mill Grindability Index (“Wi”) and an Abrasion Index (“Ai”) could be determined. The testing gave a Wi of 9 kW-h/mt, less than half the value exhibited by the North Zone sulphide ore. A value of 9 kW-h/t would be classed as moderately soft. Since the crushing work index is typically 1 to 2 kW-h/t lower than the grinding work index, the Tepal oxide should use much less power for crushing than the sulphide.

The comminution testing also gave an Ai of just 0.0245. Materials with values of Ai below 0.1 are considered to be only mildly abrasive. Thus, the value of 0.0245 indicates that the Tepal oxide is virtually non-abrasive. More comprehensive comminution testing to determine the crushing work index and other parameters requires whole core, preferably PQ. Such tests should be included in the next phase of the metallurgical program.

Table 16.23: Summary of results from static acid/base accounting tests

Sample ID	Paste pH	Sulphur Content, wt% as S		AGP	ANP	NNP	Ratio
		Sulphate	Pyritic S			(ANP – AGP)	(ANP:AGP)
P1	9.38	0.14	0.11	3.4	8.6	5.2	2.53
P2	9.77	0.09	0.09	2.8	11.6	8.8	4.14
P3	9.81	0.03	0.07	2.2	21.7	19.5	9.86

Material used for comminution testing retains its integrity. Therefore, following completion of the comminution program, the remaining north-south oxide composite was shipped to G&T for some additional flotation testing. This was prompted by results obtained during the first G&T program where rougher flotation of the oxide gave poor copper recovery but recovered almost 60% of the contained gold, more than was recovered from the South Zone sulphide.

At G&T, the NOXCL02 composite (6.3 kg) was blended with the remaining 20.9 kg of material from G&T composite NOX1. The blend was designated NOX3 and assayed 0.23% Cu, 5.3% Fe and 0.20% S, with 0.43 g/t Au. The copper included 0.08% in acid soluble form and 0.04% in cyanide soluble form. This composite was similar to NOX1, except for a higher sulphide content (0.20% vs. 0.08%).

After several rougher tests were run to optimize conditions, six tests were run to produce enough rougher concentrate for subsequent cyanidation tests. Flotation conditions included a 146 µm primary grind size at pH 9.0 and 50 to 60 g/t PAX as collector. With an average mass pull of 5.3%, the rougher concentrate graded 4.3g/t Au (52% recovery) and 0.61% Cu (14% recovery).

The rougher concentrate was then leached in cyanide, with and without regrinding to 13 µm. In both tests, a 48-h leach cycle was used with lime additions to pH 11 and a sodium cyanide concentration of 1,000 ppm. Results are summarized in Table 16.24. As shown, with a regrind, gold extraction approached 100%. However, even at 98% the recovery of gold from the ore drops from 52 to 50%. On the other hand, the cyanide consumptions appear high, but only apply to 5% of the ore mass.

As a result, the total quantity of cyanide consumed is only about one third of that consumed at 1.59 kg/t when applied to 100% of the ore.

Table 16.24: Summary of Concentrate Leach Results

Concentrate Regrind	Metal Recovery, %		Reagent Consumption, kg/t	
	Gold	Copper	NaCN	Lime
No	84	50	8	2.7
Yes	98	46	10.6	3.2

In addition to leaching the rougher concentrate, G&T also ran exploratory tests on leaching of the tails. Since 48% of the gold remained in the rougher tailings and the material was already finely ground, a tailings leach might be viable. Two tests were conducted under the same conditions as the concentrate leach tests. The head grade was 0.23 g/t Au and 0.21% Cu. In one test 78% of the gold was extracted, in the other, only 51%. Unfortunately, there were unresolved assay accountability issues and the results are questionable.

A third test was run after regrinding to 47 µm. In this trial, gold extraction was 89% and copper extraction was 13%. Cyanide consumption was 1.2 kg/t and lime consumption was 1.5 kg/t. This was better than the average bottle roll results on whole ore, but required fine grinding of the entire ore mass. However, the extraction was not as good as the average extraction from the -150 µm fraction in the column tests (91.4%).

From a process standpoint, one should note that the flotation recovery of gold from the oxide was significantly lower than the leach extractions in either the column or bottle roll leach tests. In addition, the entire oxide ore mass had to be crushed and coarse ground in order to achieve the particle size needed for flotation feed. Finally, there will be some further loss in recovery during concentrate leaching. Thus, in spite of the much smaller volume of material to leach, this route may be less attractive than heap leaching

16.4 Tizate Metallurgical Program

16.4.1 Milling and Flotation

Sample Selection, Preparation and Characterization

Geologix personnel selected 13 drill holes from their recent drilling campaign to represent the Tizate Zone sulfide zone. These included holes TEP-10-005, 008, 009, 029, 031, 032, 033, 034, 036, 037A, 038, 039 and 040. Three 2-m intervals were selected from each hole. In general these represent shallow, intermediate and deep depths. The intervals also covered a range of gold and copper values. The estimated average grade of the selected core was 0.22 g/t Au and 0.25% Cu. These are close to the average assays for all Tizate Zone holes, which were 0.20 g/t Au and 0.20% Cu. Overall best efforts were made in selecting samples that met the following criteria:

- Collect samples that are spatially representative of each zone.
- Collect samples that are representative of all grade ranges within each zone.
- Ensure that the weighted average grade for each zone is as close as possible to average deposit grade.

A total of 256 kg of split HQ core was supplied to the laboratory for metallurgical testing. All material was staged crushed to pass minus 3.36 mm (-6 mesh), then combined and homogenized. The blended master composite was rotary split into 2-kg charges for flotation. Each charge was sealed under nitrogen and stored at - 10°C to prevent any sulfide oxidation until used in flotation testing, The final back calculated head grade of the master composite ran 0.22% Cu, 0.18 g/t Au, 0.008% Mo and 3.5 g/t Ag. The gold and copper values were very close to the average grade of the Tizate Zone deposit, indicating the sample was quite representative. Pyrite was the dominate sulfide present and most of the copper was present as chalcopyrite.

Testwork Protocol

The intent of the flotation tests at G&T was to optimize the metallurgical response of the master composite in rougher, cleaner and locked cycle testing. A total of 15 flotation tests were conducted on the Tizate Zone composite. Seven were performed to establish the optimum rougher conditions, six were run to establish the optimum cleaner conditions, and two locked cycle tests were conducted to estimate the results of continuous operation.

The earlier flotation optimization testwork performed on the North and South Zone material included extensive reagent screening. This showed that a combination of 3418A and MIBC worked well. This combination also worked well on the Tizate Zone composite, so no reagent screening tests were run. However, the first test showed that addition of fuel oil during grinding was needed to improve molybdenum recovery. The fuel oil was therefore included in the reagent suite for all remaining tests. Note that no effort had been made to recover molybdenum in the earlier tests due to the low molybdenum head grades in the North and South Zones.

Rougher optimization focused on the primary grind sizes and pH. Three grind sizes were checked: 103, 147 and 191 µm at pH levels of 10, 10.5 and 11. The optimum values were found to be a primary grind of 147 µm with grinding and flotation at pH 11.

Cleaner optimization also focused on regrind times and pH. Regrind times were varied from 8 to 30 minutes, with K_{80} regrind sizes of 19, 20, 22 28 and 37 µm. The pH levels were checked at nominal values of10 and 11.

Two locked cycle tests were conducted at the conclusion of the cleaner tests using the optimum conditions established in the earlier tests. In the first test, the primary grind was done with a lime addition of 1.3 kg/t and a fuel oil addition of 10 g/t. Grind time was 18 minutes at pH 11.1 giving a nominal 147 µm K_{80} grind size. Reagent additions for rougher flotation included 15 g/t MIBC in the first rougher stage, with 5 g/t 3418A added in each of the four rougher stages. Conditioning and flotation times were 1 and 2 minutes, respectively. The regrind time was 15 minutes at pH 10, producing a regrind discharge of minus 16µm K_{80}. Reagents added during regrinding were 50 g/t lime and 10 g/t fuel oil. Three cleaner stages were run at pH 10. Conditioning times were 1 minute per stage with 2 minutes of flotation (3 minutes in stage 1). Both 3418A and MIBC were added to each cleaner stage. The 3418A dosages were 4, 3 and 1 g/t in stages 1, 2 and 3. MIBC additions were 15, 24 and 24 g/t for stages 1, 2 and 3.

The second locked cycle test was essentially the same as the first, except that the lime addition during regrinding was increased to 100 g/t so cleaning was done at pH 11. In addition, the flotation times were increased to 4, 3 and 2 minutes for the three cleaning stages. The MIBC dosage was also increased to 30 g/t per stage and the 3418A dosages were increased slightly to 5, 4 and 2 g/t for the three cleaner flotation stages.

Flotation Results

Overall the flotation test results were encouraging. The final concentrate from the first locked cycle test assayed 24.2% Cu, 0.68% Mo, 248 g/t Ag and 14.6 g/t Au. Recoveries were 85.3% for copper, 70.6% for molybdenum, 55.5% for silver and 66.2% for gold. The bulk rougher tailings assayed 0.03% Cu, 0.001% Mo, 0.04 g/t Au and 1.4 g/t Ag. Most of the pyrite was also in the tailings, which contained 82% of the iron and 71% of the sulphur in the flotation feed.

Increasing the pH in the second locked cycle test increased the copper and silver recoveries and grades, but gold and molybdenum losses increased. The final concentrate assayed 28.7% Cu, 0.59% Mo, 288 g/t Ag and 15.4 g/t Au. Recoveries were 86% for copper, 66.4% for molybdenum, 63.8% for silver and 49.7% for gold. The bulk rougher tailings assayed 0.03% Cu, 0.001% Mo, 0.95 g/t Ag and 0.10 g/t Au. The pyrite level in the tailings also increased, as 87% of the iron and 75% of the sulphur reported to the tailings. Since the pyrite is a known carrier for some of the gold, the higher pyrite rejection to tailings explains the decreased gold recovery.

At this point, not all the flotation work on Tizate Zone is complete. Although there was good molybdenum recovery to the cleaner concentrate, there was insufficient material to produce a separate molybdenite concentrate. An ICP-multi-element scan is also being run to determine the impurity content of the final locked cycle concentrate. However, results are still outstanding. Similar tests on the North and South Zone concentrates did not identify any constituents that exceeded the threshold for smelter penalty charges. This suggests there should not be a problem with Tizate Zone. Finally, ARD testing is planned for the flotation tailings to determine if they are likely to be net acid generating. These results are not yet available, either.

Overall the flotation results for Tizate Zone compare favourably with the earlier results from the North and South Zone tests. This is demonstrated in Table 16.1. As can be seen, copper concentrate grades for Tizate Zone are as good as those from the other areas, in spite of the lower head grades. The Tizate Zone copper recovery is better than the South Zone levels, but is slightly below the results for the North Zone. Gold recoveries are also comparable for all three zones, again in spite of the much lower head grade for Tizate Zone. However, the lower Tizate Zone head grade does impact the gold grade in the Tizate Zone concentrates.

At present the molybdenum and silver grades for the North and South Zones are not available. Pulps from these areas are now being assayed to see if either molybdenum or silver represent potentially viable by-products.

Table 16.25: Comparison of Key Locked Cycle Flotation Parameters for All Three Deposits

Parameter	Tizate Zone		North Zone	South Zone	
	1st Test	2nd Test		1st Test	2nd Test
Head Grade					
- Cu, %	0.22	0.22	0.3	0.3	0.3
- Mo, %	0.008	0.008	n/a	n/a	n/a
- Au	0.18	0.18	0.6	0.5	0.5
- Ag	3.5	3.5	n/a	n/a	n/a
Con. Grade					
- Cu, %	24.2	28.7	27	20	26
- Mo, %	0.68	0.59	n/a	n/a	n/a
- Au	14.6	15.4	34	28	33
- Ag	248	288	n/a	n/a	n/a
Con. Recovery, %					
- Cu	85	86	90	85	84
- Mo	71	66	n/a	n/a	n/a
- Au	66	49	65	58	52
- Ag	55	64	n/a	n/a	n/a

16.4.2 Cyanide Leaching of Gold

16.4.3 Sample Selection, Preparation and Characterization

Twenty (20) half-split HQ drill core intervals from ten (10) drill holes in the Tizate Zone oxide capping were selected from across the deposit by Geologix personnel. All intervals were relatively shallow, as the oxide zone at Tizate Zone is not as thick as it is for the North and South zones. Sample depths ranged from the surface down to a maximum of 18 m. As discussed below, the selected core not only varied spatially but also covered a range of gold values.

The core from each hole was shipped to McClelland Laboratories, Inc. in Sparks, Nevada Here the core intervals from each hole were combined in their entirety to create ten (10) composites. Each composite was crushed to 100% passing 19 mm and then blended and split to obtain a 2.5 kg sample for further crushing. Each 2.5 kg split was crushed to a P_{80} of minus 1.7 mm (P_{100} of minus 6.3 mm). The crushed composites were blended and split using a rotary splitter to obtain the 1 kg sample for bottle roll testing and three 0.5 kg samples for triplicate head assays for copper, gold and silver. Gold analysis was done by a conventional fire assaying fusion procedure with an atomic absorption (A.A.) finish. The copper and silver assays were done using a standard 4-acid digestion procedure with an A.A. finish. Table 16.2 shows the average gold assay head grade for each composite.

Table 16.26: Tizate Zone Oxide Composite Head Grade Comparison for Gold

Comp. #	DDH #	Interval From/To, (m)	Sub-Sample	Weight, (kg)	GX EST. Head Grade[1] Au, g/t	MLI Back Calc. Head Au, g/t	MLI Head Assay Au, g/t
TZOXBR01	TEP-10-009	0.00-2.00	MEX5620-1	2.9	0.06		
		4.00-6.10	MEX5620-2	4.945	0.04		
			Total	7.845	0.047	0.1	0.08
TZOXBR02	TEP-10-029	6.00-8.00	MEX5621-1	6.645	0.08		
		15.5-17.00	MEX5621-2	5.17	0.14		
			Total	11.835	0.106	0.1	0.11
TZOXBR03	TEP-10-031	0.00-2.00	MEX5622-1	1.305	1		
		3.15-5.00	MEX5622-2	5.985	0.57		
			Total	7.29	0.647	0.59	0.64
TZOXBR04	TEP-10-032	4.00-6.00	MEX5623-1	1.31	0.08		
		10.00-12.00	MEX5623-2	3.83	0.13		
			Total	5.14	0.117	0.1	0.1
TZOXBR05	TEP-10-033	6.00-8.00	MEX5624-1	0.55	0.06		
		10.00-12.00	MEX5624-2	1.58	0.2		
			Total	2.13	0.164	0.2	0.24
TZOXBR06	TEP-10-034	4.00-6.00	MEX5625-1	3.435	0.27		
		12.00-14.00	MEX5625-2	5.49	0.37		
			Total	8.925	0.332	0.3	0.32
TZOXBR07	TEP-10-036	6.25-8.25	MEX5626-1	4	0.5		
		10.25-12.20	MEX5626-2	6.39	0.33		
			Total	10.39	0.395	0.38	0.39
TZOXBR08	TEP-10-038	2.00-4.00	MEX5627-1	2.87	0.17		
		4.00-5.00	MEX5627-2	1.96	0.25		
			Total	4.83	0.202	0.18	0.17
TZOXBR09	TEP-10-039	7.55-9.00	MEX5628-1	2.455	0.22		
		12.00-14.00	MEX5628-2	5.82	0.17		
			Total	0.2	0.185	0.18	0.2
TZOXBR10	TEP-10-040	10.00-12.00	MEX5629-1		0.22		
		16.00-17.95	MEX5629-2		0.17		
			Total	0.14	0.185	0.13	0.14
AVERAGE[2]					**0.237**	**0.24**	**0.24**

Note 1. The assays shown in the Total rows are weighted on the basis of sample weights.

Note 2. The values shown for the averages are based on the ten (10) total values shown for each composite and do not take into account the different weights in each interval sample

Gold grades range from 0.64 down to 0.08 grams per metric ton (g/t). The standard deviations on the triplicate head assays average just over 0.01 g/t. The drill hole numbers and intervals used in each composite are also shown in the table, along with Geologix's estimate of the head grade and the back calculated head determined in the leach tests. The average for each of the three assay procedures was the same 0.24 g/t. The low standard deviation and close agreement on the various assays shows that there are no significant analytical issues associated with the gold analyses.

The average copper head grade was 1,522 ppm, with an average standard deviation of 33 ppm. Copper values ranged from 2,690 down to 300 ppm. This compares closely with the back calculated head grade of 1,486 ppm and the Geologix estimate of 1,556 ppm. As with gold, the low standard deviation and close assay agreement show that there are no significant analytical issues with the copper determinations.

The average silver head assay was 1.6 g/t, slightly lower than the back calculated or estimated head grades. The range was from 3.7 g/t to less than the detection limit. There is even a wider spread on some of the individual samples. In part, this is because the low grade assays are approaching the limit of detection, which adds some scatter. Some of the samples are currently being re-assayed for silver in an effort to reduce the analytical variance.

Overall, the head grades for Tizate Zone are somewhat lower than they were in the bottle roll samples for the North and South Zones. For the latter the average gold and copper grades were 0.48 g Au/mt and 0.22% Cu vs. 0.24 g Au/mt and 0.15% Cu for Tizate Zone. However, the silver grades for Tizate Zone are about the same as in the other zones.

In reviewing the Tizate Zone assays, it appeared that there might be a correlation between the copper and gold head assays. This correlation was checked, with the results shown in Figure 16-1. There clearly is a positive trend between the two variables. A linear relationship passing through the origin (0% Cu:0% Au) was first tried, but the correlation was poor, with a very low correlation coefficient (R^2 value). The best empirical fit proved to be a power function with an R^2 value of 0.63.

16.4.4 Testwork Protocol

Direct agitation cyanidation (bottle roll) tests were conducted on each of the 10 composites, at an 80%-1.7 mm feed size to determine the gold, silver and copper recoveries, recovery rates, and reagent requirements. Ore charges (~ 1 kg ea.) were mixed with water to achieve a slurry with 40% weight solids. Natural pulp pH levels were measured after allowing the slurries to equilibrate. Lime was then added to adjust the pH of the pulps to 11.0 before adding the cyanide. Sodium cyanide, equivalent to 1.0 g/L of solution, was added to the alkaline pulps.

Leaching was conducted by rolling the pulps in bottles on laboratory rolls for 96 hours. Rolling was continuous, but was suspended briefly after 2, 6, 24, 48, and 72 hours to allow the pulps to settle so aliquots of pregnant leach solution (PLS) could be taken for gold, silver, and copper analysis by A.A. methods. PLS volumes were measured and sampled. Cyanide concentration and pH were determined for each sample. Make-up water, equivalent to that withdrawn (100mL), was added to the pulps. Cyanide concentrations were restored to initial levels. Finally, lime was added when necessary, to maintain the leaching pH between 10.8 and 11.2. Rolling was then resumed.

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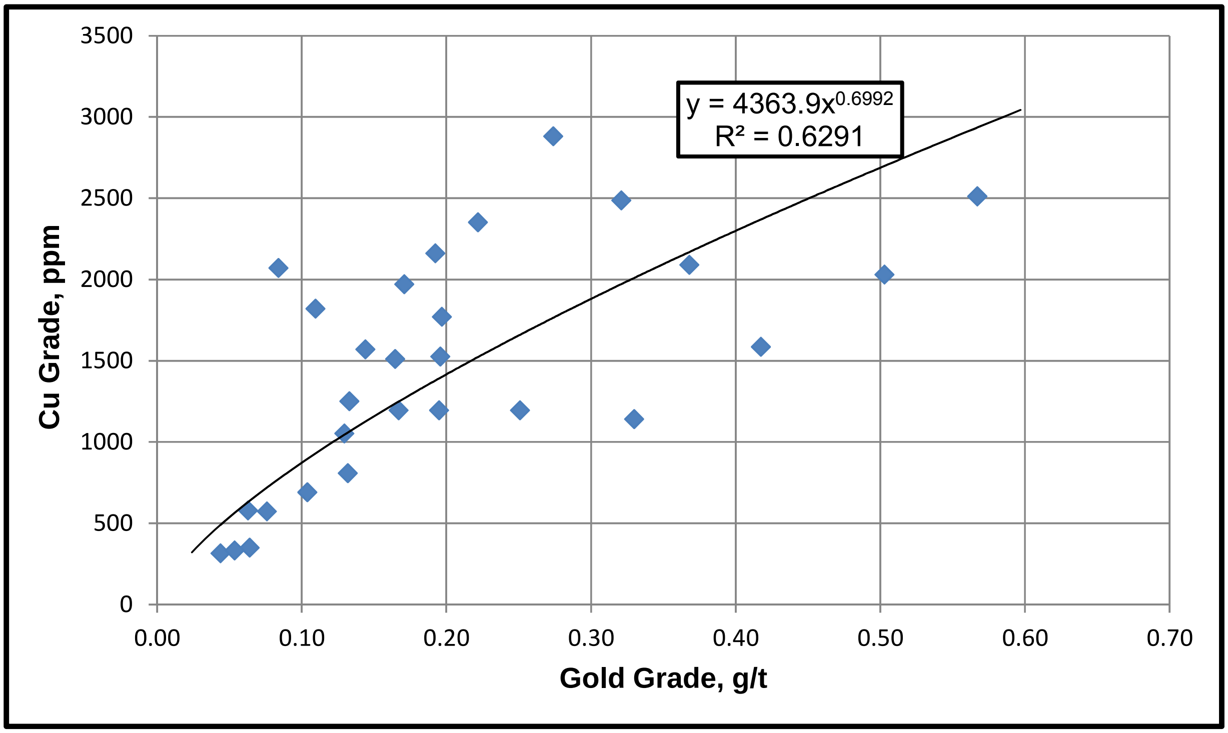

Figure 16.5: Correlation between Copper and Gold Head Grades in Tizate Zone Oxide Ore

After 96 hours, the pulps were filtered to separate liquids and solids. Final pregnant solution volumes were measured and sampled for gold, silver, and copper analysis. Final pH and cyanide concentrations were determined. Leached residues were washed, dried, weighed, and assayed in triplicate to determine the residual gold, silver, and copper contents.

Modified ABA tests are being conducted on a composite prepared by combining a split from each bottle roll residue. The tests are intended to determine if the leach residues are likely to be classed as potentially acid generating. Results of the tests are not yet available.

16.4.5 Leach Results

Summary metallurgical results for gold are shown in Tables 16.3 and 16.4. The leach curves (plots of cumulative extraction vs. leach time) are shown in Figures 16.2 and 16.3. The tables also include the final copper and silver extractions, the cyanide and lime consumptions, and the natural pH for each composite. Each table and its corresponding figure are shown together on the following pages.

Table 16.27: Summary Metallurgical Results for Tizate Zone Composites

TZOXBR01 through TZOXBR05

Metallurgical Parameter	COMPOSITE				
	TZOXBR01	TZOXBR02	TZOXBR03	TZOXBR04	TZOXBR05
Gold Extraction, % of total Au					
2 hours	45	60	53.4	45	30
6 hours	48	64	62	63	39.5
24 hours	50.9	83	71	67	49.5
48 hours	38.9	73	75.2	71	52.5
72 hours	40.9	77	76.3	75	55.5
96 hours	40	80	76.3	80	60
Calc'd. Head Grade Au	0.1	0.1	0.59	0.1	0.2
Final Ag Extraction, % of total	71.4	31.3	25	50	35.7
Final Cu Extraction, % of total	3.8	7.6	2.8	4	18.7
NaCN Consumed, k/t	<0.07	0.45	0.3	0.15	0.75
Lime Consumed, k/t	6.2	5.7	5.2	3.5	1.8
Natural pH (40% solids)	7.7	6.4	7.3	7.1	7.9

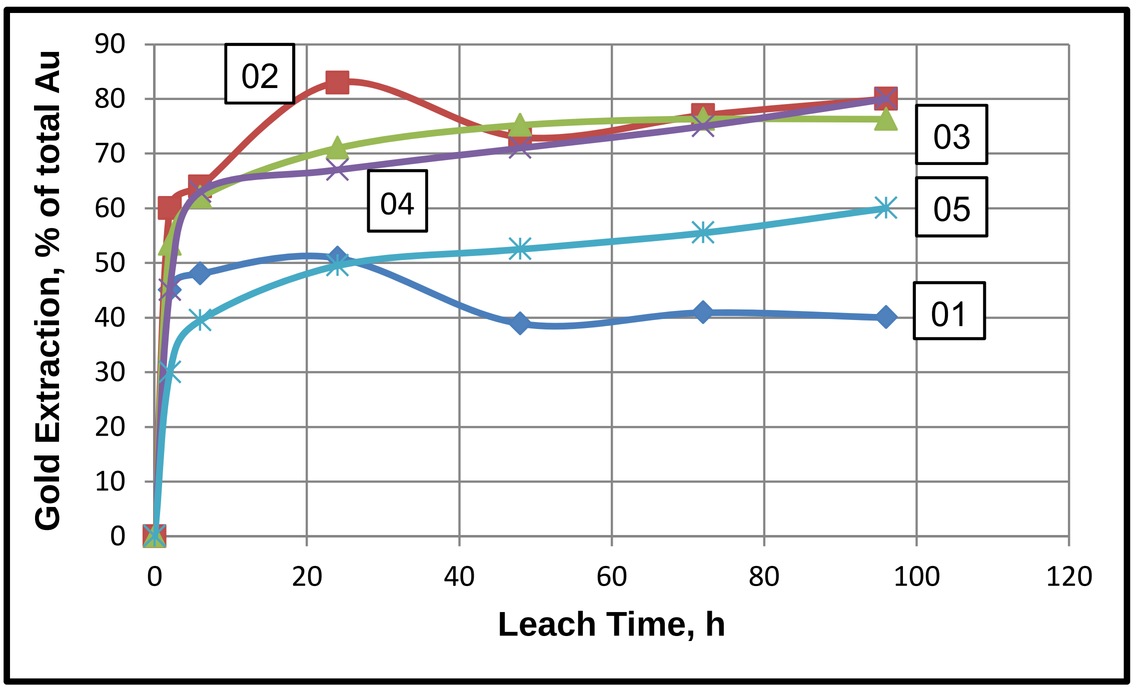

Figure 16.6: Gold Leach Curves for Composites TZOXBR01-05

Table 16.28: **Summary Metallurgical Results for Tizate Zone Composites TZOXBR06 through TZOXBR10.**

Metallurgical Parameter	Composite				
	TZOXBR 06	TZOXBR 07	TZOXBR 08	TZOXBR 09	TZOXBR 10
Gold Extraction, % of total Au					
2 hours	35	51.3	50	41.7	46.2
6 hours	47.3	62.6	61.7	52.8	49.2
24 hours	60.3	74.5	73.9	64.4	63.8
48 hours	64	75	78.3	68.3	56.1
72 hours	66.7	79.2	74.4	72.2	59.2
96 hours	66.7	78.9	77.8	66.7	61.5
Calc'd. Head Grade, Au	0.3	0.38	0.18	0.18	0.13
Final Ag Extraction, % of total	30.8	50	13.3	38.1	43.2
Final Cu Extraction, % of total	2.5	7.4	2.1	8.4	10.2
NaCN Consumed, k/t ore	0.15	0.3	0.15	0.45	0.52
Lime Consumed, k/t ore	3.7	4.8	7.5	2.9	2.9
Natural pH (40% solids)	7.1	6.1	7.4	7	7.1

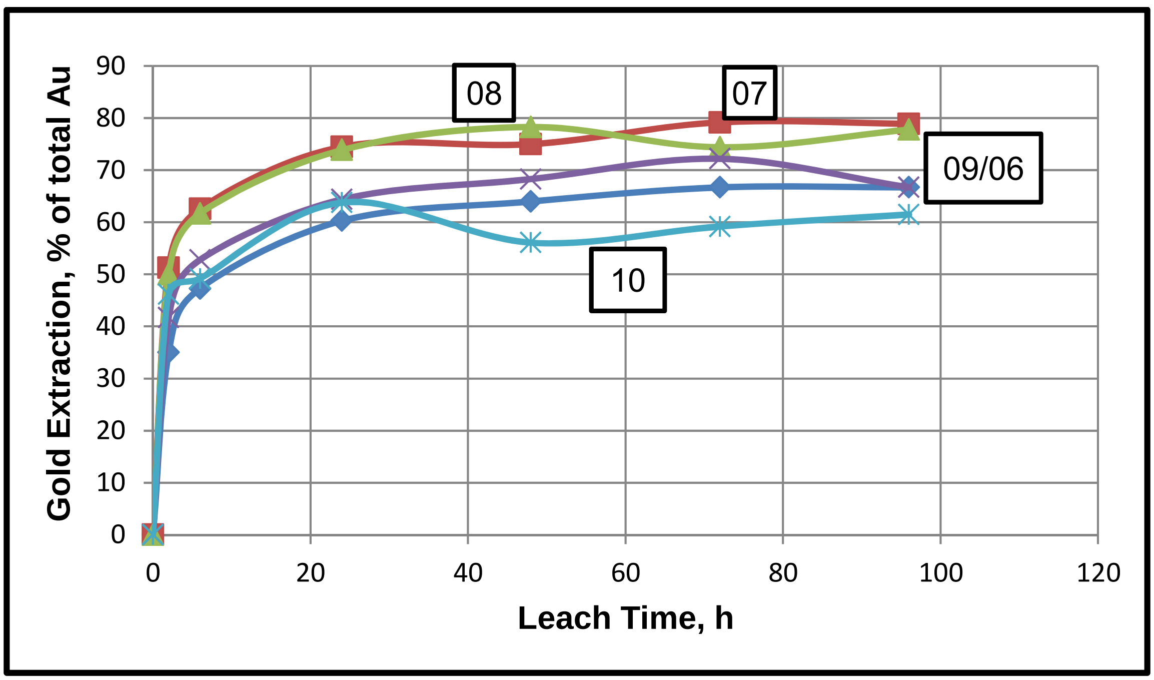

Figure 16.7: Gold Leach Curves for Composites TZOXBR06-10

The metallurgical results show that the Tizate Zone oxide material is generally amenable to cyanide leaching when crushed to a fine particle size. The 96-hour gold recovery averaged 68.8% in spite of one low-grade composite that achieved only 40% extraction. If this value is dropped from the data set, the average final extraction rises to 72%. These recoveries are lower than those achieved in bottle roll tests on the North-South samples. For these, recoveries ranged from 76 to 91% and averaged 81%. No studies have been completed to determine the nature of the unrecovered gold.

The gold recoveries do not correlate closely with head grade. There are three low-grade samples with back calculated gold head grades of 0.10 g/t. In one the recovery was only 40%, but in the other two it was 80%, the lowest and highest recoveries observed in the bottle roll program.

The initial gold leach rate was very rapid. Gold extraction ranged from 30 to 60% (average 46%) in the first two hours. These 2-hour extractions represented anywhere from 50 to 90% of the 96-hour gold recovery. After the first two hours, leaching continued in all tests, but at a slower rate, as shown in Figures 16.2 and 16.3. After 24 hours, three of the columns reached their maximum extraction level while the remaining columns continued to leach very slowly. Only TZOXBR05 exhibited a typical pseudo-parabolic leach curve, with significant extraction continuing when the test was terminated. The leach curves for all other tests showed a rapid initial rate, with a sharp break at two hours and a much slower and nearly linear rate thereafter.

Only 40% of the samples actually achieved their highest extraction levels at the end of the tests. All other tests reached their peak extraction levels at intermediate times of 24 to 72 hours, with some falloff in extraction thereafter. The average maximum gold extraction was 71%, as compared to the average 96-hour extraction of 68.8%.

In several cases the apparent declines in gold recovery were small and are probably a consequence of the way the tests were run. When the tests were interrupted for sampling, a 100-ml gold-bearing solution sample was removed for analysis. To maintain the solution balance, an equivalent volume of de-ionized (gold-free) water was added. If little or no gold was still leaching, this would have the effect of a 10% dilution. However, in three of the tests, the maximum extraction was reached in just 24 hours, with a sharp decrease in apparent recovery when sampled again at 48 hours. The later recoveries also failed to recover to the 24-hour maximums. This suggests there was some sort of gold re-adsorption mechanism at work, sometimes referred to as “preg robbing”. This possibility should be carefully evaluated in any further leach tests, as this behavior was not observed in the tests on the North and South Zone material.

Copper extraction was low but somewhat erratic. The average for all ten tests was 6.75%, with a range of 2.5 to 18.7%. Because the copper back calculated head grades vary by a factor of almost ten, both the percent copper extraction and the actual amount of copper extracted in grams should be considered. The latter ranged from 13 to 248 g/t Cu , with an average of 99 g. Clearly the percent copper extraction is independent of the head grade for Tizate Zone. This is shown by the horizontal trend line and very low correlation coefficient (~0.001) in Figure 16.4.

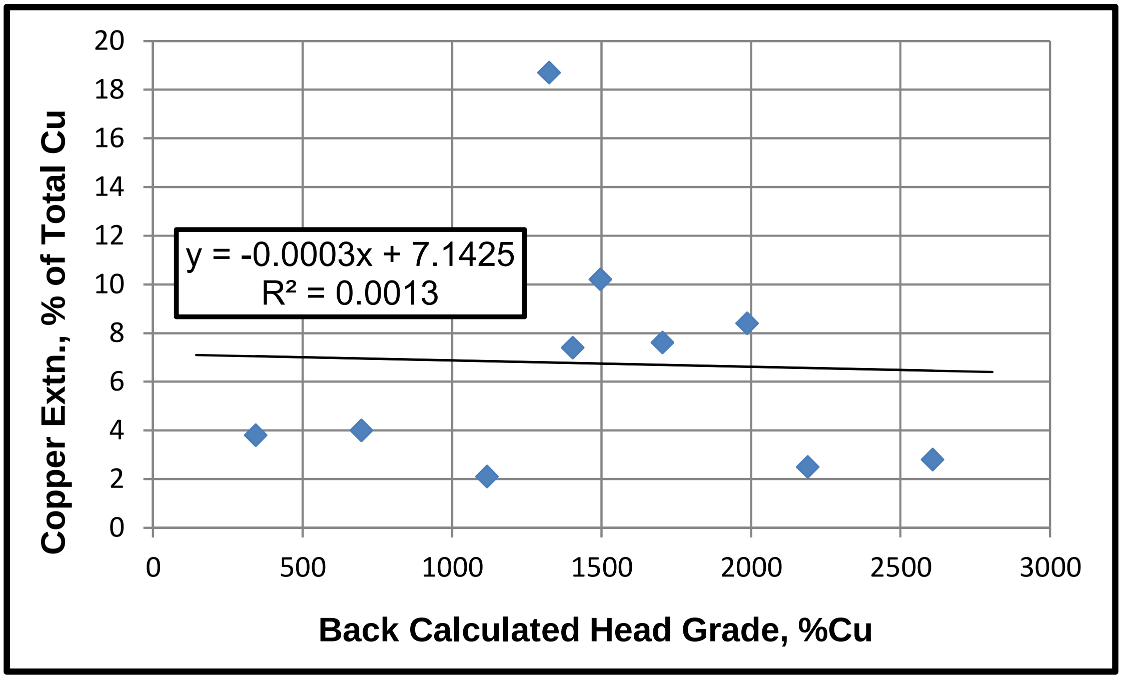

Figure 16.8: Percent Copper Extraction as a Function of the Back Calculated Head Grade

Overall, these results are similar to those obtained in the tests on the North and South Zone material. In the latter tests copper extraction averaged 6.0%, with an average of 152 g of copper extracted. The recovery ranged from 0.5 to 15.5% and the amount extracted varied from 7 to 782 g. Taken in total, the results suggest that the copper mineralogy varies throughout the oxide zones, being more soluble in some areas than others. If chrysocolla is the dominant copper mineral, cyanide consumption will be low as chrysocolla has very low cyanide solubility. Other oxide minerals have higher solubilities.

On a mass basis, the amount of copper extracted from the various zones at Tepal is typically at least an order of magnitude higher than the grams of gold and silver extracted. Thus, some type of technology such as SART will likely be needed to remove the copper that builds up and recover the cyanide from the copper cyanide complexes in the leach solution.

Reagent consumption was generally low, but somewhat variable. Consumption of NaCN averaged 0.33 kg/t, with a range of <0.07 to 0.75 kg/t. This is somewhat lower than was observed for the North-South Zones where the average was 0.57 kg/t. Because copper is the main cyanide consumer, two cyanide correlations were checked. One was cyanide consumption vs. copper head grade. Results are shown in Figure 16-5. Although there is a slight positive trend, the correlation coefficient is less than 0.1, indicating that the two variables are independent of each other. However, there is a very strong correlation between cyanide consumption and the amount of copper extracted. This is shown in Figure 16.6.

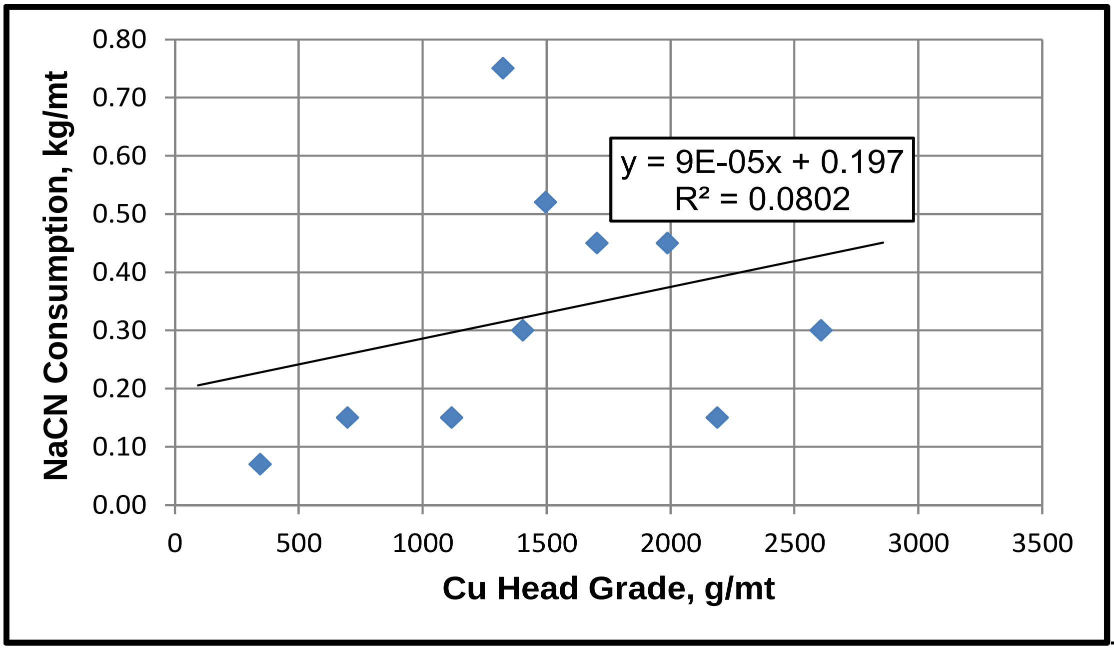

Figure 16.9: Relationship between the Copper Head Grade and Cyanide Consumption

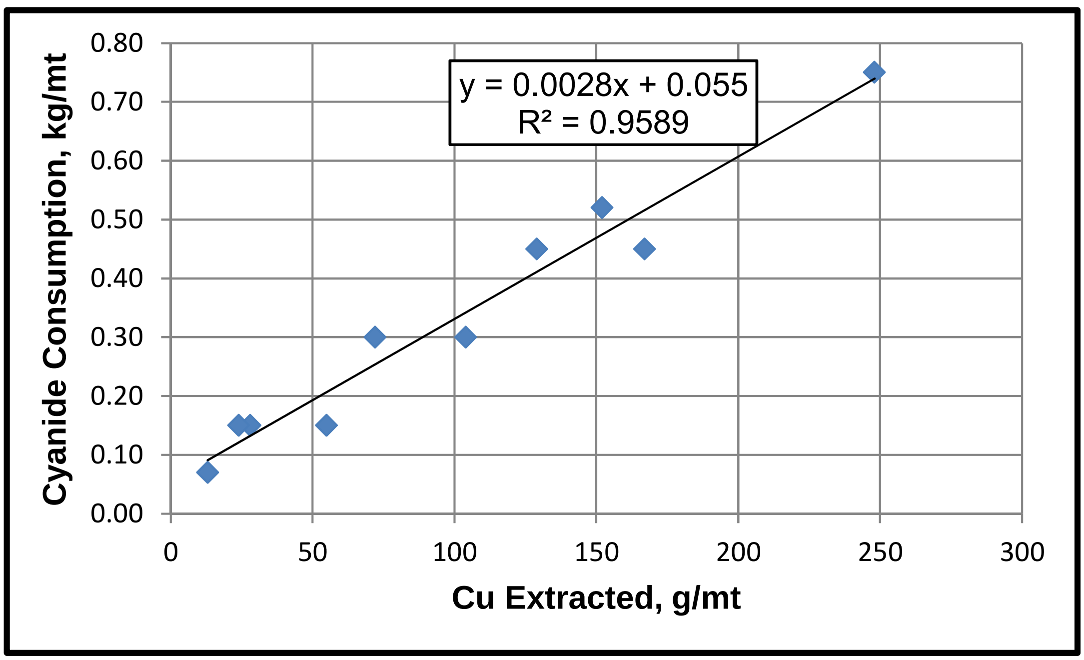

Figure 16.10: Correlation between Cyanide Consumption and the Amount of Copper Extracted

Lime consumption for the Tizate Zone samples averaged 4.4 kg/t. The range was 1.8 to 7.5 kg/t. Although a correlation between the natural pH and the amount of lime consumed to maintain a fixed pH of 11 might be expected, this does not appear to be the case. This relationship is plotted in Figure 16-7. As can be seen, the trend line is horizontal and the two parameters are independent of each other. The lime consumptions measured in the North-South Zone tests showed even more scatter (1.7 to 9.2 kg/t), but the average was only 3.9 kg/t.

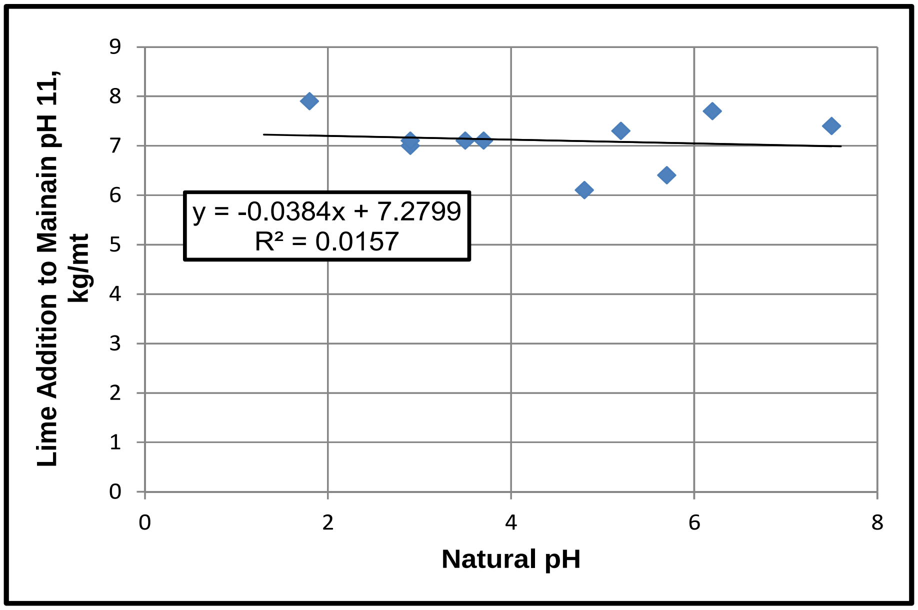

Figure 16.11: The Relationship between the Natural pH and the Required Lime Addition to Maintain a pH of 11

16.5 Metallurgical QA/QC

At the moment, there are no specific guide lines on metallurgical testing for NI 43-101 reports. However, QA/QC programs are just as critical to the success of the metallurgical program as they are in the drilling program and modeling of the resource. The QA/QC programs in place at both laboratories are addressed in the following sections.

16.5.1 G&T

G&T is an ISO 9001 certified laboratory. The ISO requirements cover equipment calibrations and operating protocols. G&T's QA/QC practices include the following:

For each test, the mass of test products must equal the mass of the initial feed, within 2 percent.

For each project, a comparative head assay table is prepared. This table compares the recalculated feed value for each test (based on individual stream assays and weights) with the initially measured head assay values. The recalculated values need to be within 5 to 15 percent of the measured value depending on the element being assayed. If they are outside this accepted error range, re-assays are conducted until results are within that range.

Commercially prepared standards are purchased with certified known concentrations. These standards are run with every set of samples to ensure the QC of the samples being assayed. The number of pulp standards applied for each element varies. For Tepal, they were as follows: Cu: 4 standards, Fe: 5 standards, S: 4 standards, Au: 3 standards.

16.5.2 MLI

MLI has an in-house QA/QC plan that encompasses four elements: a) Personnel training, b) Instrument calibration and maintenance, c) Instrument operation and d) Titrimetric testing.

Personnel selected for performing laboratory activities are given the instructions or training commensurate with the scope, complexity, or special nature of the activities.

All instruments including atomic absorption, pH metres, and probes (pH, ORP, etc) are calibrated and maintained using appropriate methods and standards to calibrate and verify satisfactory operations.

There are specific operating protocols for all instruments including atomic absorption equipment and all types of metres and probes.

Various titrimetric analytical procedures are performed at MLI in order to support the metallurgical tests. These QA/QC procedures apply to any of those methods, including free acid titrations. Prior to any analytical run, the procedures and reagents used are checked by titrating a selected standard. QC duplicate samples are titrated at a frequency of 5 samples per shift or 5% of the number of samples analyzed during a shift, whichever is greater. If results vary by greater than 5 percent, all samples analyzed during that shift are re-analyzed.

Specific to Tepal, all head and residue assays were run in triplicate. Either five samples or 5% of the solution analyses that were run in-house were also check assayed at an outside third-party laboratory. All sets of pulp assays run by outside laboratories included at least one standard, one blank and one replicate. If any of these had failed to check within specified limits, the entire set of samples would have been reassayed. Fortunately this did not occur with any of the Tepal samples.

Back calculated compositions were compared to head assays. If agreement had differed by more than 10%, assays would have been rerun. If this failed to resolve the discrepancy, then the test would have been repeated. Fortunately, such measures were not required, as assay agreement was excellent.

16.6 Conclusions

The results obtained from the metallurgical testwork programs undertaken at G&T and MLI lead to the conclusions enumerated below.

16.6.1 Sulphide Ore Processing

North and South Zones

The QA/QC procedures in place at G&T are more than adequate to assure the accuracy of the metallurgical results.

With one exception, back calculated and assays heads agreed closely, showing that there were no significant assaying problems affecting the flotation program.

Based on a single Bond ball mill grindability test conducted on NSX-1, the North Zone grinding work index was 19.8 kW-h/mt, which would rank the material as “hard”.

Following optimization studies on various parameters, including grind size, collectors and dosages, and pH levels, locked cycle testing showed that the sulphide ore responded well to conventional copper-gold technology. Material from the North Zone responded somewhat better than material from the South Zone. The optimum primary grind was 125 µm, regrinding to 25 µm for cleaning. The collector 3418A gave the best overall performance. The ore from the North Zone was little impacted by pH, but the South Zone material performed better at pH 11.

The North Zone locked cycle cleaner concentrate graded 27% Cu at 90% recovery and 33.8 g/t Au at 65% recovery. The South Zone cleaner concentrate assayed 26.1% Cu and 32.7 g/t Au. Metal recoveries dropped to 84% for copper and 52% for gold.

Final concentrate quality was excellent, with payable gold and silver and no impurities present at concentrations above threshold penalty levels. Silver recovery to concentrate was typically around 25%

Evaluation of the tailings showed that most of the unrecovered gold was associated with pyrite. However, a few particles of free gold were observed. Installation of a gravity trap on the tailings line should recover most of the free particles, marginally increasing overall gold recovery

Because most of the unrecovered gold was associated with pyrite, a pyrite concentrate was produced and a gravity concentrate was produced from the pyrite tailings. Gold grades were low in both products and cyanide leaching did not do a good job extracting the gold. As a result, further gold recovery from the rougher tailings does not appear to be economically viable.

Tizate Zone

- The master composite appears to be a representative sample of the sulphide portion of the Tizate Zone deposit. Material was drawn from shallow, intermediate and deep intervals on drill holes spread across the deposit. Although the samples used to prepare the master composite had a wide range of head grades, the average was close to the average for the deposit.
- Based on 13 rougher and cleaner tests, optimum flotation conditions included a 147 µm K80 primary grind size with lime at pH 11, rougher flotation at pH 11 using 3418A and MIBC as reagents, plus use of fuel oil during grinding to enhance molybdenum recovery. Regrinding was done for 15 minutes at pH 10 producing a regrind discharge of 16 µm, with cleaner flotation using the same reagents run at pH 10 or 11.
- The Tizate Zone sulfide material responds well to milling and flotation. Locked cycle testing with a cleaner float at pH 10 produced concentrate containing 24% Cu, 0.68% Mo, 248 g/t Ag and 14.6 g/t Au with corresponding recoveries of 85%, 71%, 55% and 66%.
- Raising the pH to 11 improved the grade and recovery for copper and silver, but reduced gold and molybdenum recovery.

- The Tizate Zone flotation results are generally on a par with earlier results from tests on the North and South Zones. Copper and gold recoveries are similar , as is the copper grade. However, the Tizate Zone gold grade is lower, reflecting the lower head grade.

16.6.2 Oxide Ore Processing

North and South Zones

The QA/QC procedures in place at MLI are more than adequate to assure the accuracy of the metallurgical results.

In all cases the back calculated and assay head grades agreed closely, indicating that no significant assay accountability issues affected the results. For the gold assays, the standard deviation was 0.02 g/t and the precision averaged 95%. For copper the results were even better, with an average precision of more than 97%.

Based on a single Bond ball mill grindability test conducted on N0XCL02, the grinding work index was 9.0 kW-h/mt, which would rank the material as “moderately soft”. Thus, crushing the oxide should require about half the power needed for crushing the sulphide ore.

Based on a single test conducted on NOXCL02, the abrasion index for the oxide was measured as 0.0245. Such a value would class the oxide as being nearly non-abrasive.

Eleven -1.7 mm samples spatially distributed across the deposit and covering the expected range of head grades were subjected to bottle roll cyanide leaching. On average, 81% of the gold, 21% of the silver and 6% of the copper were extracted in this small-scale variability test program. Gold recovery ranged from 70 to 91%, while copper extraction varied from 0.5 to 15.5%.

In the bottle roll program, cyanide consumption averaged 0.57 kg NaCN per tonne. The range was 0.15 to 2.08 kg/t and generally increased as copper extraction increased. Lime consumption averaged 3.9 kg/t, with a range of 1.7 to 9.2 kg/t.

Gold extraction was rapid in the bottle roll program, with most tests reaching 60% recovery in six hours, or less. One third of the samples were leached to exhaustion in less than 24 h and another third were leached to exhaustion in less than 72 h.

Both bottle roll and column leach tests were conducted on three composites of - 12.5 mm material taken from the north end of the North Zone, the south end of the North Zone and the South Zone. The composites were leached to exhaustion in all tests and the average gold extraction was 78% for both types of testing. The gold recovery range for the column tests was 72.5 to 86%. Average copper extractions were also similar, with 14% in the columns and 17% in the bottle rolls.

Average cyanide consumption was 1.59 kg/t in the columns vs. 1.41 kg/t in the bottle rolls.

Lime consumption in the columns was uncertain, as lime additions to the columns were too low and caustic additions were required to provide the alkalinity needed to achieve the desired pH levels.

In spite of the lime addition problems, the gold extraction rate in the column tests was rapid. In 10 to 28 days, the gold extractions reached 80% of the final values. In 16 to 38 days, extractions reached 90% of the final values. In less than 60 days, all three columns reached 98% of the final extractions. Never the less, these rates may be biased to the low side. Additional tests should be run with proper lime additions in order to confirm the gold leach kinetics.

Size distributions were determined on the column feed and residue for each composite. All three composites had similar size distributions, with about 80% of the material in the +1.7 mm fractions and 7 to 10% in the -150 µm fractions. The only significant upgrading was in the latter fractions, which contained 14 to 21% of the gold.

The -150 µm fines tended to skew the column results. Not only were the gold grades higher, but the gold recoveries averaged 91%. Virtually all coarser fractions had both head grades and recoveries that were below the average for their respective composite. It is not clear how the behaviour of the fines will affect the recovery when leaching a coarser crush size or ROM material.

On a mass basis (g/t), anywhere from 500 to 2,000 times as much copper was extracted as gold in the column tests. In addition, copper concentration in the leach solution reached as much as 2 g/L in a single 90-day leach cycle. Therefore, technology such as SART will be needed to remove copper from the leach solution and recover the cyanide for recycle.

Results of static acid/base accounting (ABA) tests showed that all three column residues would be classed as non-acid generating. As a result, no special measures should be required to control acidic drainage from the gold heaps following closure.

A split from composite NOXCL02 was subjected to rougher flotation after grinding to 146 µm. The flotation recovered only 52% of the gold and 14% of the copper. After regrinding to 13 µm, the concentrate was given a cyanide leach, which recovered 98% of the contained gold, giving an overall recovery of 50%. This is far less than the 78% average recovery in the column leach tests. In addition, cyanide consumption was high at 10.6 kg/t. Based on the added cost of grinding, the low recovery and the high cyanide consumption in flotation-plus-concentrate leaching, heap leaching the oxide ore appears to be the more attractive processing route.

Tizate Zone

- The Tizate Zone oxide material is generally amenable to cyanide leaching when crushed to a relatively fine size.
- The average head grade of the ten Tizate Zone composites was 0.24 g/t Au, 0.15% Cu and 1.7 g/t Ag . The grade was lower than the average of the North and South Zones previously tested under the same conditions. These two zones averaged 0.48 g/t Au, 0.22% Cu and 1.8 g/t Ag.
- For Tizate Zone there was a positive correlation between the gold and copper head assays.
- For Tizate Zone the gold recovery averaged 69% and was independent of the gold head grade. This recovery level is lower than the average gold recovery in the North-South samples, which was 81%.
- Gold extraction was very fast and averaged 46% during the first two hours of leaching. This represents about 70% of the final 96-hour extraction.

- The typical gold leach curve showed a near vertical segment for the first two hours, followed by a much slower nearly linear rise thereafter.
- In all, 60% of the tests reached their maximum gold extraction at an intermediate time and not at the end of the leach cycle.
- Three of the tests reached their peak extraction in just 24 hours, followed by a sharp drop thereafter. Such behavior suggests possible gold readsorption (preg robbing), a phenomenon not seen in the earlier North/South tests.
- Copper extraction was low but somewhat scattered, averaging about 7%. This is about the same as the copper extraction in the North-South tests.
- The variation in copper extraction suggests that the copper mineralization is variable in the Tizate Zone oxide capping.
- Copper extraction was independent of the copper head grade.
- Silver recoveries are subject to possible revision due to reassaying. However, the available results show the silver recovery was erratic, but averaged 39%. This was nearly twice the average for the North-South tests.
- NaCN consumption averaged 0.33 kg/t ore, less than the North-South average of 0.57 kg/t.
- Lime consumption for Tizate Zone averaged 4.4 kg/t ore, slightly more than the average for the North-South tests.
- The lime consumption was independent of the natural pH of the samples.

16.7 Recommendations for Further Metallurgical Testing

Recommendations for further development of the processing routes for both the sulphide and oxide ore types are listed below. These are based on the current understanding of the Tepal resource. However, some adjustment may be appropriate if the current drilling program expands the tonnage and/or grade of the deposit. The metallurgical testing program is estimated to cost $0.15M for heap leach testing and $0.5M for comminution and flotation testing.

The metallurgical recommendations are:

- The milling and flotation results are sufficiently encouraging to proceed with the next phase of the metallurgical testwork. Additional activities should be directed at the North, South and Tizate Zone. Activities should include the following:
  - A metallurgical drilling program on all three deposits to obtain material for a variability study on flotation response as related to location, depth, alteration, and grade.
  - Tests on composites that represent each quarter's mine output for the first three years of operation, with annual composites thereafter.
  - A comprehensive comminution study to define the crushing and grinding parameters, including abrasion indices, for each deposit.
  - Operation of a flotation pilot plant to produce sufficient concentrates and tailings for settling and filtration tests, plus tests on production of a molybdenite concentrate for the Tizate zone.

- The initial leach tests for all three zones are sufficiently encouraging that a more comprehensive metallurgical testwork program should be initiated using fresh drill core. Results should be suitable for supporting a prefeasibility-level study on Tepal.
- A variability study similar to that for the sulphide material should be undertaken for the oxide material.
- The new tests should include comminution and geotechnical studies, as well leach studies. The latter should include tests at different crush sizes, pH levels, leach flow rates, cyanide concentrations, etc.
- The additional studies should also include appropriate environmental testing.
- The possibility of a preg robbing constituent in the Tizate zone ore should be investigated.
- Diagnostic work should be undertaken to determine the nature of the gold remaining in the leach residues from both the Tizate and the North-South tests.

17 Mineral Resource and Mineral Reserve Estimates

The Tepal project contains no mineral reserve estimates that are compliant with NI 43-101. A mineral reserve, as defined in the CIM Standards and referenced in NI 43-101, means “the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined”.

The Mineral Resources in this report are reported in accordance with NI 43-101 and have been estimated in conformity with generally accepted CIM “Estimation and Mineral Resource and Mineral Reserve Best Practices” guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserves. The mineral resource statement was prepared by Dr. Gilles Arseneau, P.Geo. (APEGBC#23474), “independent qualified person” as defined in National Instrument 43- 101.

The geostatistical studies, variography and block validation were carried out by Marek Nowak P.Eng.

17.1 Historical Mineral Resource and Mineral Reserve Estimates

Previous mineral resource estimates for the project were prepared by ACA in 2008 and presented in a report prepared by SRK entitled Revised Tepal Project Economic Assessment Technical Report and dated February 18, 2011. Other previous historical estimates have been prepared for the property and are discussed in Section 5 of this report.

17.2 SRK Mineral Resource Update (March 2011)

17.2.1 Data Summary

Raw data incorporated in to this resource update study consists of all diamond drilling data collected by Geologix, Arian during 2007 and 2008, Teck and Hecla historical diamond drill data and geology data from one INCO drill hole.

Arian and Geologix have also collected weathering data and interpreted geological wireframe solids for the Tepal porphyry system delineated by drilling. This data has been forwarded to SRK, reviewed and modified where appropriate and used in the resource update study.

SRK has reviewed and discussed the sample collection methodologies adopted by Geologix and are satisfied that data collection methodologies are of a satisfactory standard.

A review of findings pertaining to input data are presented in the report sections below and issues regarding the suitability of this data for inclusion in current and future resource estimates discussed in the Interpretation, Conclusions and Recommendations section of this report.

17.2.2 Data Validation

Drill hole collar, assay, survey, geology, recovery and weathering data were presented as Excel spreadsheets files. These data files were checked and imported into Gemcom GEMs version 6.2.3 software and further validated using GEMs validation functions. Key fields within these critical drill hole database data files was validated for potential numeric and alpha-numeric errors. Data validation cross referencing Collar, Survey, assay and geology files was performed in GEMs to confirm drill hole depths, Inconsistent or missing sample/logging intervals and survey data.

No fatal errors were detected during data validation. Any missing intervals were accounted for by the selective sampling methodology adopted for the sampling of drill holes.

17.2.3 Input Data

Data selected for use in resource estimation for both the Tepal and Tizate deposit is contained in the drill hole database Tepal all GEMs drill hole database and summarised in Table 17.1.

Table 17.1: Tepal all GEMs Database Summary

MM Data Type	No of Records	Geologix Holes	Arian Holes	Teck Holes	Hecla Holes	INCO Holes	Comment
MM Database							
DH Collar	172	43	42	49	17	21	
DH Geology	NE						Geology data not supplied
DH Assay	15,365	5,839	3,530	4,505	1,491	0	
DH Survey	529	246	196	49	17	21	
DH Geotech	5,154	5,154	0	0	0	0	
Specific Gravity	40	19	21	0	0	0	
ICP	11,094	5,839	3,530	234	1,491	0	

Input data files, along with relevant wireframes representing the geological outlines of the mineralized zones were provided.

17.2.4 Classical Statistical Analysis

Descriptive statistical analysis of Tepal and Tizate assay data was undertaken on both the entire database a well as on the subset of the data that is contained within the mineralized wireframe. Specifically the analyses were carried out to determine the statistical distribution of the gold and copper values and to identify potential outliers that may require capping prior to block model and grade interpolation

Descriptive statistics (unrestricted) were generated for the all gold and copper assays and are presented in Table 17.2 and descriptive statistics of assays contained within the mineralized envelope are summarized in Table 17.3.

Table 17.2: Tepal Descriptive Statistics of All Samples

	Au (g/t)	Cu (ppm)	Length (m)
Valid cases	15365	15365	15365
Mean	0.20	1204.15	1.76
Variance	0.13	2882900.66	0.16
Std. Deviation	0.37	1697.91	0.41
Variation Coefficient	1.81	1.41	0.23
Minimum	0.00	0.00	0.20
Maximum	10.90	35400.00	5.95
1st percentile	0.00	5.00	0.70
5th percentile	0.00	20.00	1.00
10th percentile	0.01	40.00	1.00
25th percentile	0.02	100.00	1.52
Median	0.07	600.00	2.00
75th percentile	0.24	1795.00	2.00
90th percentile	0.55	3000.00	2.00
95th percentile	0.83	4000.00	2.00
99th percentile	1.66	7534.00	2.45

Table 17.3: Descriptive Statistics of Assays Contained Within the Mineralized Envelopes

	Au (g/t)	Cu (ppm)	Length (m)
Valid cases	6133	6133	6133
Mean	0.42	2420.90	1.71
Variance	0.20	4007281.44	0.20
Std. Deviation	0.45	2001.82	0.45
Variation Coefficient	1.06	0.83	0.26
Minimum	0.00	0.00	0.25
Maximum	8.15	35400.00	5.95
1st percentile	0.01	48.68	0.82
5th percentile	0.05	472.00	1.00
10th percentile	0.09	800.00	1.00
25th percentile	0.15	1340.00	1.52
Median	0.29	2000.00	2.00
75th percentile	0.53	2910.00	2.00
90th percentile	0.92	4296.00	2.00
95th percentile	1.27	5673.00	2.00
99th percentile	2.09	9862.80	2.50

17.2.5 Geological Domain Interpretation

Mineralization and Geology

Work by ACA indicates that a natural break exists in the log normal histograms population for gold at about 0.18 ppm Au and ACA considered this value as a natural boundary to gold mineralization. In a general sense, elevated gold grade is accompanied by elevated copper grades however this is not always the case, for this reason, the new geological model prepared by Geologix was based on a dollar equivalent value. Geological models were designed to include all assays that were greater than US$8.00 equivalent based on a price of US$900/oz for gold and US$2.75/lb for copper. This mineralized boundary was only used to model mineralization and is considered to represent the primary economic limits of the mineralization (Figure 17.1).

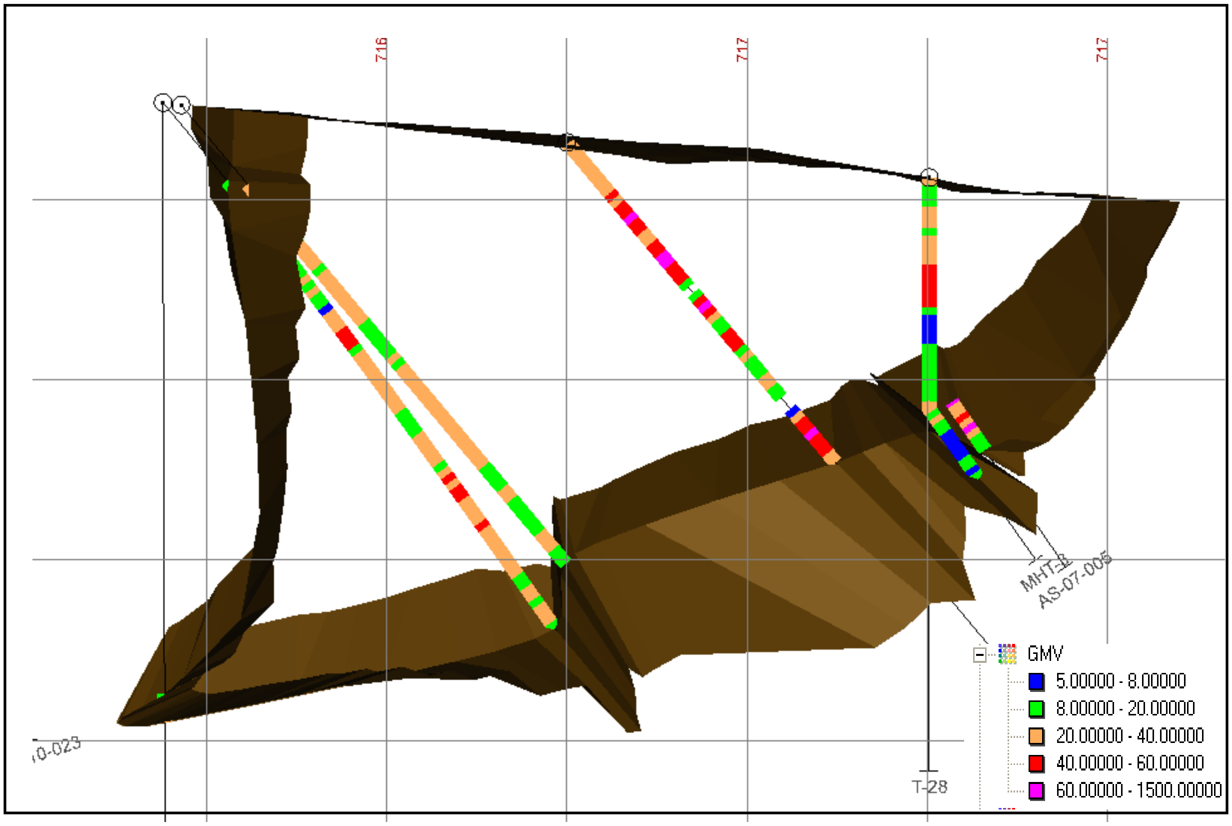

Figure 17.1: Cross Section 2,115,850N showing Geological Interpretation And Drill hole Dollar Equivalent Values

Note: Section is looking north and grid is 50 by 50 m

A review of geological interpretation and discussion with staff geologists suggests that the local geology and spatial features associated with the mineralization are well understood in a general sense, and controls on mineralization and the extent of structural controls at the deposit are also understood. SRK audited and validated the geological model prepared by Geologix and found it to be a reasonable interpretation of the mineralization at Tepal and Tizate.

SRK understand from data review and discussions with the Geologix geologists that the deposit geology is relatively simple and studies have determined that mineralization is intimately associated with Tonalite host rocks, quartz stockwork and brecciation, all easily identified and logged in core.

For the Tepal property three mineralized zones have been interpreted:

- The Tepal North Zone;
- The Tepal South Zone; and
- The Tizate Zone.

Within these defined zones, a total of six separate domains have been interpreted. The domains essentially separate oxide from fresh or sulphide mineralization for each of the mineralized zones.

Weathering Boundaries

Drill hole weathering data was use in interpreting the base of oxidation and the contact with fresh rock. A narrow transition zone occurs between the oxidized and fresh material but in general, the transition from oxide to fresh rock occurs over one or two metres. For this reason, SRK did not see the need to model a separate transition unit. The base of the oxide interval usually corresponds with the base of hematite mineralization and was used to create a Digital Terrain Model (“DTM”). The DTM was then used to divide the geology solid into oxide and fresh segments (Figure 17.2).

These weathering zones were used to flag the block model. Blocks that were at least 50% above the base of oxidization were flagged as oxide the blocks below the sulphide DTM were flagged as sulphide.

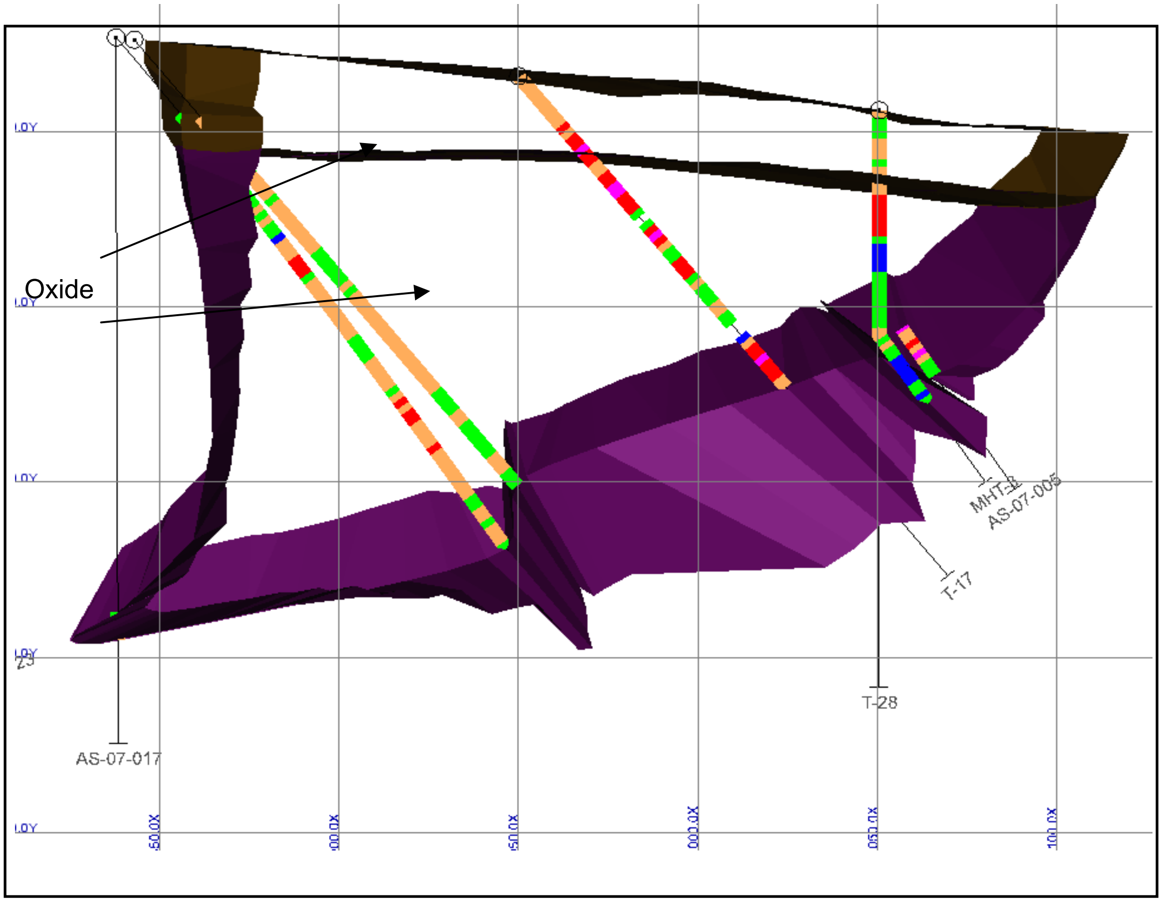

Figure 17.2: Cross Section 2,115,850N showing Oxide and Fresh Geological Domains

Note: Section is looking north and grid is 50 by 50 m

Capping

Top cut analysis was performed on mineralized domain raw gold and copper data prior to final block model grade interpolation. Top cut analysis is undertaken to assess the influence extreme grade outliers has on the sample population of each domain. Whilst extreme grades are real, their influence in interpolation may overstate the block grades in some parts of the deposits. Excel spreadsheets were prepared to examine the effects of a range of top cuts applied to raw data and the effect these have on the co-efficient of variation ("COV") and loss of data from the domain. Tepal and Tizate deposits were considered together for the purpose of top cut assessment.

After a review of gold and copper data, only minimal assay top cuts have been applied. Top cut limits were identified from inflection points on the cumulative frequency plots for both copper and gold in the Tepal and Tizate deposits, which denoted outlying high grade samples considered unrepresentative of the population. The limiting of anomalous high grades will ensure a more representative block model. Descriptive statistics were then generated for the capped data. Summary details are contained in Table 17.4.

Assay data for Tizate was also evaluated for silver and molybdenum, SRK decided not to cap silver and molybdenum for this study, SRK recommends that requirement for capping of silver and molybdenum be reviewed following the collection of additional data and before re-estimating mineral resources for Tizate.

Table 17.4: Tepal Top Cut Analysis Summary

Element	No of Samples	COV	Top Cut	COV (Cut)	% Data Cut	Metal Lost %
Au	1,692	1.06	4ppm	1.02 g/t	0.09	0.38
Cu	1,480	0.83	1.5%	0.76%	0.27	0.75

*Note: Metal lost is (mean uncapped-mean uncapped)/mean uncapped*100*

Composites

Prior to estimation, samples within the mineralized wireframes contained in the Tepal and Tizate drill hole assay files were composited to a standard length to reduce bias for geostatistical analysis and interpolation. The composite length was determined by considering the histogram for raw drill hole sample intervals and the average lengths of all the assay intervals within the mineralized zones. The histogram of drill hole sampling length shows the dominant sample interval length is 2m and has been chosen as the optimum composite length.

A composite assay file was created for samples within the domain wireframes for use as input data for block model interpolation. All composites that were less than 1 m in length were combined with the previous composite to assure that all composites were between 1 and 3 m in length.

Descriptive statistics were then generated for the composited data as summarized in Table 17-5 for Tepal and Table 17.6 for Tizate.

Table 17.5 Summary Statistical Data for The Tepal 2-M Composites

	Au (g/t)	Cu (%)	AuCAP (g/t)	CuCAP (%)
Valid cases	4033	4033	4033	4033
Mean	0.47	0.26	0.47	0.25
Variance	0.21	0.04	0.20	0.04
Std. Deviation	0.46	0.20	0.45	0.19
Variation Coefficient	0.99	0.78	0.95	0.74
Minimum	0.00	0.00	0.00	0.00
Maximum	6.89	3.11	3.98	1.50
1st percentile	0.01	0.01	0.01	0.01
5th percentile	0.06	0.06	0.06	0.06
10th percentile	0.10	0.09	0.10	0.09
25th percentile	0.18	0.14	0.18	0.14
Median	0.34	0.21	0.34	0.21
75th percentile	0.59	0.32	0.59	0.32
90th percentile	0.99	0.47	0.99	0.47
95th percentile	1.38	0.60	1.38	0.60
99th percentile	2.13	1.00	2.12	0.98

Table 17.6: Summary Statistical Data for The Tizate 2-M Composites

	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)	AuCAP (g/t)	CuCAP (%)
Valid cases	1025	1025	1025	1025	1025	1025
Mean	0.21	0.18	2.35	62.26	0.21	0.18
Variance	0.02	0.01	4.98	3222.76	0.02	0.01
Std. Deviation	0.15	0.08	2.23	56.77	0.15	0.08
Variation Coefficient	0.73	0.43	0.95	0.91	0.73	0.43
Minimum	0.00	0.00	0.03	0.50	0.00	0.00
Maximum	1.31	0.76	41.14	458.00	1.31	0.76
1st percentile	0.00	0.00	0.22	0.50	0.00	0.00
5th percentile	0.03	0.06	0.39	7.18	0.03	0.06
10th percentile	0.06	0.10	0.70	13.00	0.06	0.10
25th percentile	0.11	0.14	1.10	24.00	0.11	0.14
Median	0.18	0.18	1.78	48.03	0.18	0.18
75th percentile	0.27	0.22	3.10	81.92	0.27	0.22
90th percentile	0.39	0.27	4.66	128.21	0.39	0.27
95th percentile	0.49	0.31	5.66	171.26	0.49	0.31
99th percentile	0.78	0.42	8.41	306.64	0.78	0.42

17.2.6 Variography

Spatial data analysis was considered prior to block model grade estimation in an attempt to generate a series of correlograms that would define the directions of grade anisotropy and spatial continuity of gold and copper grades such that these correlogram parameters could be used as input parameters for grade estimation.

At the current drill spacing over the deposit there is sufficient sample data density within the Tepal North and South and Tizate to be able to reliably generate directional correlogram for gold and copper. Data are insufficient to generate robust correlograms for silver and molybdenum. Therefore, search range and orientation parameters used for silver and molybdenum were derived from data collected interpreted from the copper correlogram. Table 17.7 summarizes the strike orientation and dip orientation of the domain wireframes.

Table 17.7: Correlogram Parameters For Tepal And Tizate

Zone	Metal	Nugget C0	Sill C1/C2	Gemcom Rotations (RRR)			Ranges a1, a2		
				around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
Tepal	Au	0.15	0.15	0	-60	0	60	30	40
			0.70	0	-60	0	170	60	100
Tepal	Cu	0.12	0.47	0	-60	0	180	100	80
			0.41	0	-60	0	500	250	120
Tizate	Au	0.10	0.60	-45	45	0	25	18	18
			0.30	-45	45	0	170	70	120
Tizate	Cu	0.12	0.60	-45	45	0	100	100	50
			0.28	-45	45	0	300	300	150

17.2.7 Block Modelling

Empty Cell Block Model

An empty block model was created to cover the extents of mineralized wireframes at Tepal and a separate model was prepared for Tizate (Table 17.8).

A block size of 10 m × 10 m × 5 m was selected. The decrease in block size relative to the previous ACA model is a reflection of the additional in-fill drilling by Geologix thereby yielding a smaller drill mesh capable of supporting a small block size.

Table 17.8: Tepal and Tizate Block Model Extents

	Dimension (m)	Minimum (m)	Maximum (m)	Block size (m)	Number of blocks
Tepal	Easting	715,987.5	717,637.5	10	165
	Northing	2,115,137.5	2,117,487.5	10	235
	Elevation	50.0	770.0	5	144
Tizate	Easting	718,200	719,400	10	120
	Northing	2,116,200	2,117,200	10	100
	Elevation	70	770	5	140

The domain wireframes were then assigned to the block model file such that blocks that were at least 0.001% inside any given domain were assigned to that domain (Table 17.9). A percent bock model file was also generate to monitor the volume of each blocks contained within the wireframes. All blocks outside the wireframe model were then coded with 99 if they were 50% or more below the topographic surface or a 0 code if they were more than 50% above the surface topography.

Table 17.9: Block Model Rock Codes

Block Model Code	Domain
101	Tepal North Oxide
102	Tepal North sulphide
201	Tepal South Oxide
202	Tepal South Sulphide
301	Tizate Oxide
302	Tizate Sulphide
99	Waste Rock
0	Air

Grade Interpolation

Gold and copper grade were interpolated into the block models on respective domains for Tepal. In addition to gold and copper, silver and molybdenum grades were interpolated for Tizate. For interpolation both the block model and composite assay file was filtered by domain and blocks within each domain assigned an interpolated grade using only composite data falling within each domain.

For each domain, blocks were interpolated by ordinary kriging for gold and copper and by inverse distance weighted to power two (ID2) for silver and molybdenum. Interpolation was performed at different search radii, until all blocks within each domain had received an interpolated grade. The search distances were derived from the ranges derived from the correlogram analysis, geological model and deposit geometries, exploration data spacing and interpreted grade continuity. Interpreted geometries and search ellipse orientations for each modeled domain are tabulated in Table 17.10 below.

Table 17.10: Domain Geometries and Search Parameters

Estimator	Domain	Metal	Search Pass	Rotation (RRR) Deg			Range (m)			Number of Composite		Max Per hole
				Z	Y	Z	X	Y	Z	Min.	Max.	
OK	Tepal	Au	1	0	-60	0	60	30	40	5	12	3
OK	Tepal	Au	2	0	-60	0	170	60	100	5	12	3
OK	Tepal	Au	3	0	0	0	200	200	80	5	12	3
OK	Tepal	Cu	1	0	-60	0	85	35	60	5	12	3
OK	Tepal	Cu	2	-45	45	0	170	70	120	5	12	3
OK	Tepal	Cu	3	0	0	0	200	200	80	5	12	3
OK	Tizate	Au	1	-45	45	0	90	50	60	5	12	3
OK	Tizate	Au	2	-45	45	0	180	100	80	5	12	3
OK	Tizate	Au	3	0	0	0	200	200	80	5	12	3
OK	Tizate	Cu	1	-45	45	0	100	100	50	5	12	3
OK	Tizate	Cu	2	-45	45	0	150	150	200	5	12	3
OK	Tizate	Cu	3	0	0	0	200	200	80	5	12	3
ID2	Tizate	Ag	1	-45	45	0	100	100	50	5	12	3
ID2	Tizate	Ag	2	-45	45	0	150	150	200	5	12	3
ID2	Tizate	Ag	3	0	0	0	200	200	80	5	12	3
ID2	Tizate	Mo	1	-45	45	0	100	100	50	5	12	3
ID2	Tizate	Mo	2	-45	45	0	150	150	200	5	12	3
ID2	Tizate	Mo	3	0	0	0	200	200	80	5	12	3

Grade interpolation was carried out in multiple passes with each successive pass having a larger volume than the preceding pass and only block that were not interpolated during the previous pass were interpolated. Model cells were estimated using data from drill hole composites. The first search radius was selected to be equal the first structure of the correlogram. Blocks that were not estimated during the first interpolation run were used in the next interpolation run. The second search radius was set to the full range of the correlogram and the third and final search was set to fill most of the un-interpolated blocks within the wireframes representing the mineralized zones.

Data used to interpolate grade into the Tepal and Tizate block model contains varying sample spatial densities. To ensure that clustered sample groups did not preferentially inform block grades, interpolations included a restriction on the maximum number of samples per drill holes used in block grade estimation.

17.2.8 Mineral Resource Classification

Mineral resources were estimated in conformity with the generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines (2005). The Mineral Resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent resource estimates. The Mineral Resources may also be affected by subsequent assessments of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

Mineral reserves can only be estimated based on the results of an economic evaluation as part of a preliminary feasibility study or feasibility study. As a result, this study has no mineral reserve estimates. There is no certainty that all or any part of the Mineral Resources will be converted into a mineral reserve.

The Mineral Resources for the Tepal and Tizate deposit have been classified according to the "CIM Definition Standards for Mineral Resources and Mineral Reserves" (December, 2005) by Dr. Gilles Arseneau, P.Geo. (APEGBC#23474) with the assistance of Marek Nowak, P.Eng. (APEGBC#119958), both are independent Qualified Persons as this term is defined in National Instrument 43-101. Jim Robertson, of SRK, designed the Whittle shell considered for resource reporting.

The Tepal and Tizate deposits have been investigated by drilling at a spacing varying between 60 m to 100 m to a depth of approximately 300 m. SRK considers that the quantity and quality of the exploration data (confidence in the location and reliability of assaying results) acquired by Geologix and the previous workers to be reliable and therefore not a factor that would impact resource classification. The confidence in the underlying datasets supports classification of Indicated and Inferred Mineral Resources within the meaning of CIM Definition Standards.

However, there is insufficient information to confirm both the geological and grade continuity with the current level of sampling to support a Measured Mineral Resource classification within the meaning of CIM Definition Standards.

The estimated blocks were classified according to the confidence in interpretation of the mineralized zones, the reliability of continuity of gold and copper grades as defined by the correlogram, the number of data used to estimate a block grade and the average distance of the composite used in the estimate.

In order to classify mineralization as an Indicated Mineral Resource, the location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge." To satisfy this requirement, the following requirements were applied to the Tepal and Tizate mineral resource:

Blocks were classified as Indicated if at least five composites representing at least two drill holes were found within a search radii defined by the short range structure of the correlogram, the pass one search volume.

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

17.2.9 Model Validation

Global and local model validation was undertaken on the Tepal and Tizate block models prior to resource reporting.

Global Validation

In order to test for global bias, SRK estimated the mineral resources by ordinary kriging and carried out global validation by re-estimating the mineral resources using ID2 interpolation method. The ID2 model agreed well with the OK model and do not indicate the presence of any significant biases. For Tepal, the ID2 model reported 1.5% more total gold and 3.4% less total copper at the $US 5/t cut-off than the OK model and the Tizate ID2 model reports 1% less total gold and 1% less copper than the OK model at the $US 5/t cut-off.

SKR prepared swath plots for both gold and copper in order to compare the variability of the estimated grades as compared to the composite grades across the deposit. Figures 17.3 to 17.5 show the results of the gold swath plots for Tepal and Tizate. The swath plots for both copper and gold correlate very well with the original composited drill hole data.

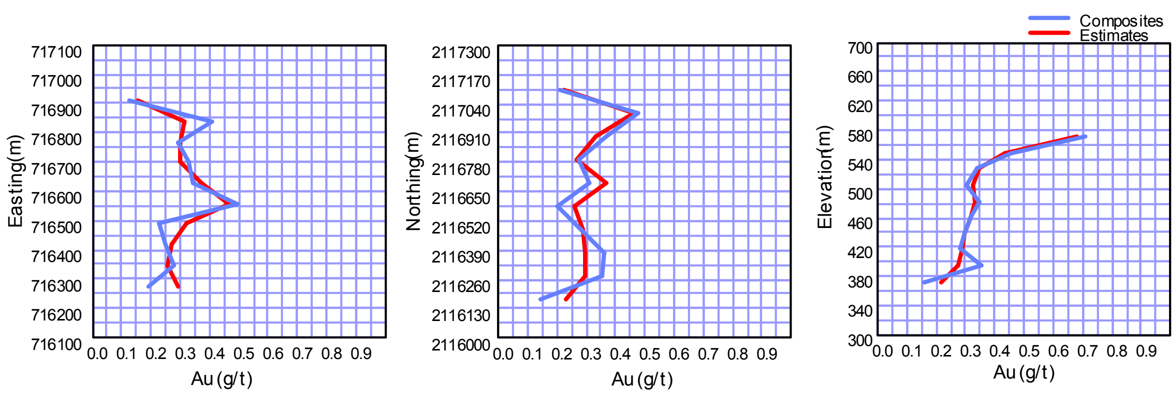

Figure 17.3: Tepal North Sulphide Declustered Average Gold Composites Compared to Estimated Gold Grades

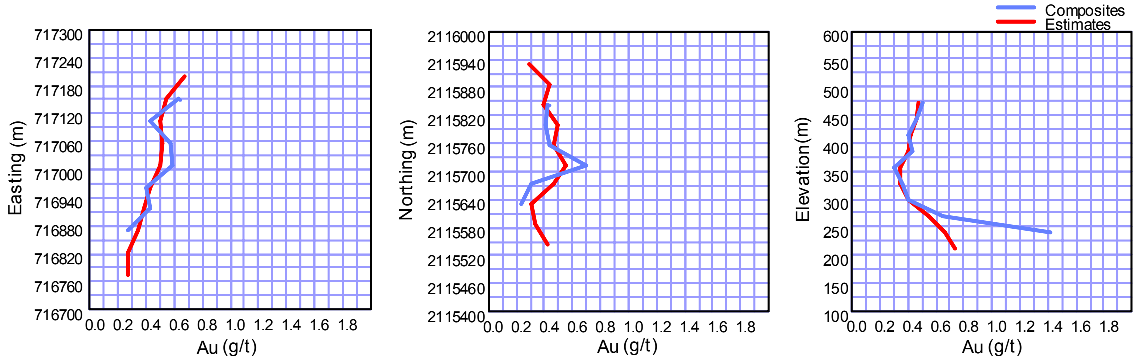

Figure 17.4: Tepal South Sulphide Declustered Average Gold Composites Compared to Estimated Gold Grades

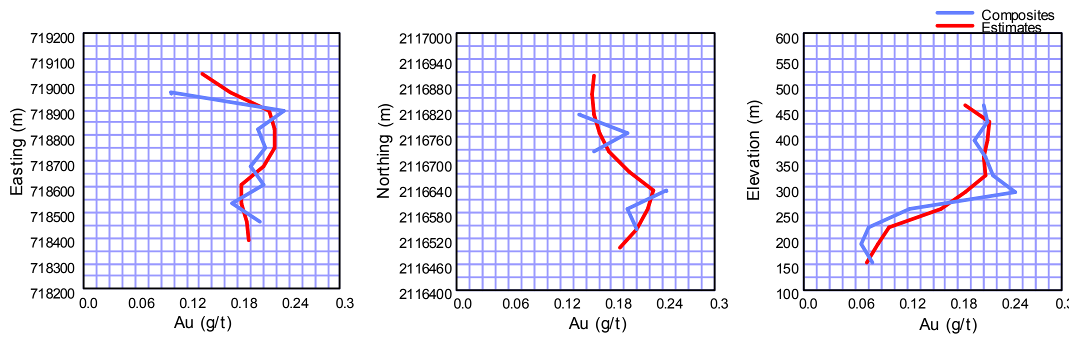

Figure 17.5: Tizate Sulphide Declustered Average Gold Composites Compared to Estimated Gold Grades

The development of modeling domains has been influenced by using a 'natural' cut-off of US$ 8/t equivalent to define mineralized envelopes. Composite grade data has then been used to calculate block grades within each domain. A comparison of the mean composite grade and mean estimated block grade has been undertaken to assess potential over/under estimating during interpolation. This validation is contained in Table 17.11.

Table 17.11: Gold and Copper Composite Mean versus Block Mean

Domain	Comp Mean Au (ppm)	Block Mean Au (ppm)	Comp Mean Cu (%)	Block Mean Cu (%)
Tepal North Oxide	0.474	0.355	0.246	0.213
Tepal South Oxide	0.447	0.415	0.274	0.226
Tepal North Fresh	0.405	0.348	0.217	0.239
Tepal South Fresh	0.509	0.476	0.234	0.232
Tizate Oxide	0.258	0.24	0.217	0.213
Tizate Fresh	0.205	0.20	0.182	0.179
Tepal Total	**0.470**	**0.392**	**0.256**	**0.233**
Tizate Total	**0.208**	**0.203**	**0.184**	**0.182**

A degree of smoothing of grade is inevitable when estimating block grades at the current data spacing of the deposit. However the mean of domain grades compare favourably to the mean of input composite grades used to estimate blocks.

The Tepal oxide domains show a marked decrease in mean grade relative to the input mean composite grade. The decrease in mean grade during interpolation can be attributed to the smaller volume represented by the oxide material when compared to the fresh material. The smaller volume makes meaningful comparison of the means more difficult because of the high degree of variability in smaller populations.

17.2.10 Composites versus Block Estimates

Figures 17.6 to Figure 17.9 show a comparison of the local “well-informed” estimated block grades with average composite grades contained within those blocks in the Tepal and Tizate deposits.

On average, the estimated block grades are almost identical to the composite assays for the intervals above 0.1 g/t to 1.0 g/t for gold and 0.1 to 1.0% for copper. In addition, the estimated block grade estimates are smoother than the assay grades. This is indicated by the thick white line. The thick white line that runs through the middle of the cloud is the result of a piece-wise linear regression smoother. Generally, at lower concentrations, the estimates are higher, and at higher concentrations they are lower.

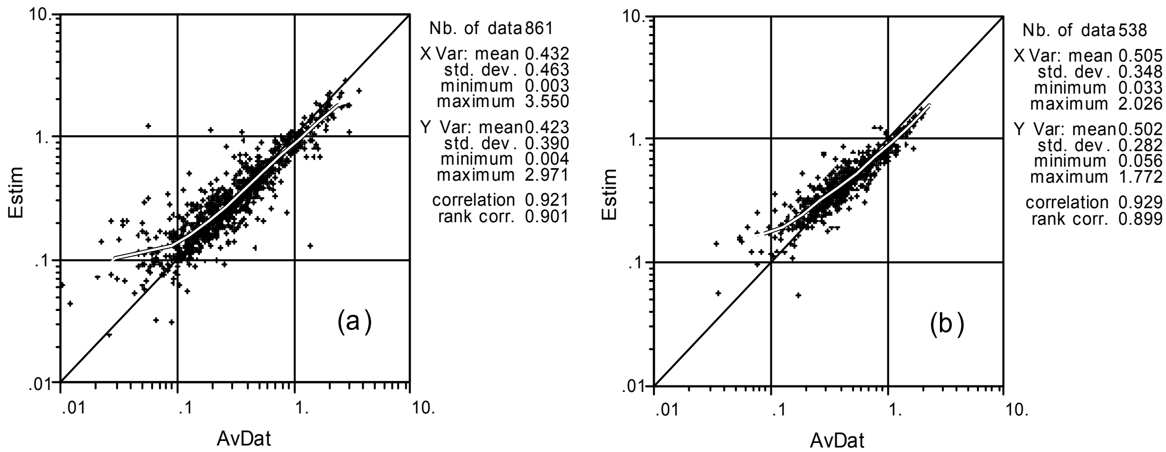

Figure 17.6: Comparison Of Fresh Rock Au Block Estimates With Composites (a) Tepal North (b) Tepal South

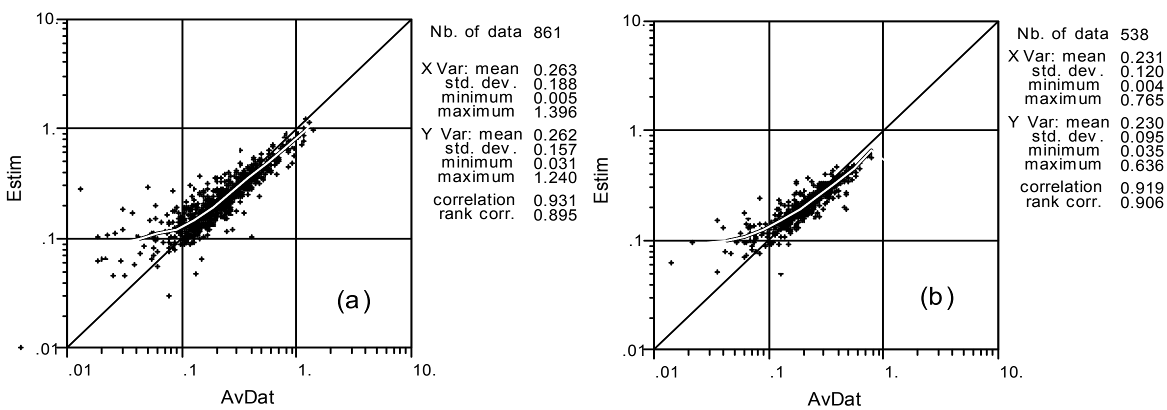

Figure 17.7: Comparison of fresh rock Cu block estimates with composites (a) Tepal North (b) Tepal South

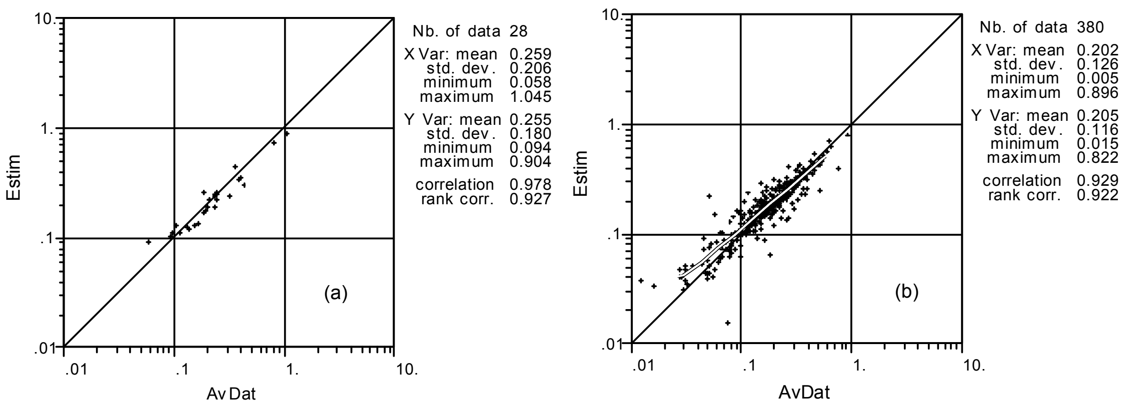

Figure 17.8: Comparison of Tizate Au block estimates with composites (a) oxide (b) fresh rock

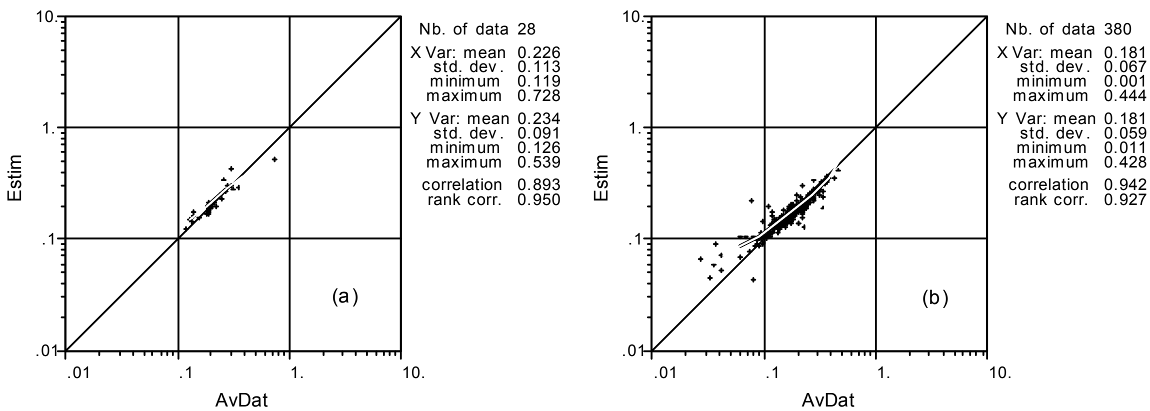

Figure 17.9 Comparison of Tizate Cu block estimates with composite (a) oxide (b) fresh rock

Local Validation

On completion of the modeling, the block model was displayed in 2-D slices along with composite drill hole data in order to determine if the block grades had honoured the general sense of composite drill hole grades, that is to say, that high grade blocks are located around high sample grades, and vice versa (Figure 17.10 and Figure 17.11).

A degree of smoothing is apparent in all linear block model estimations and is to be expected but on the whole block grades correlate very well with input composite sample grades.

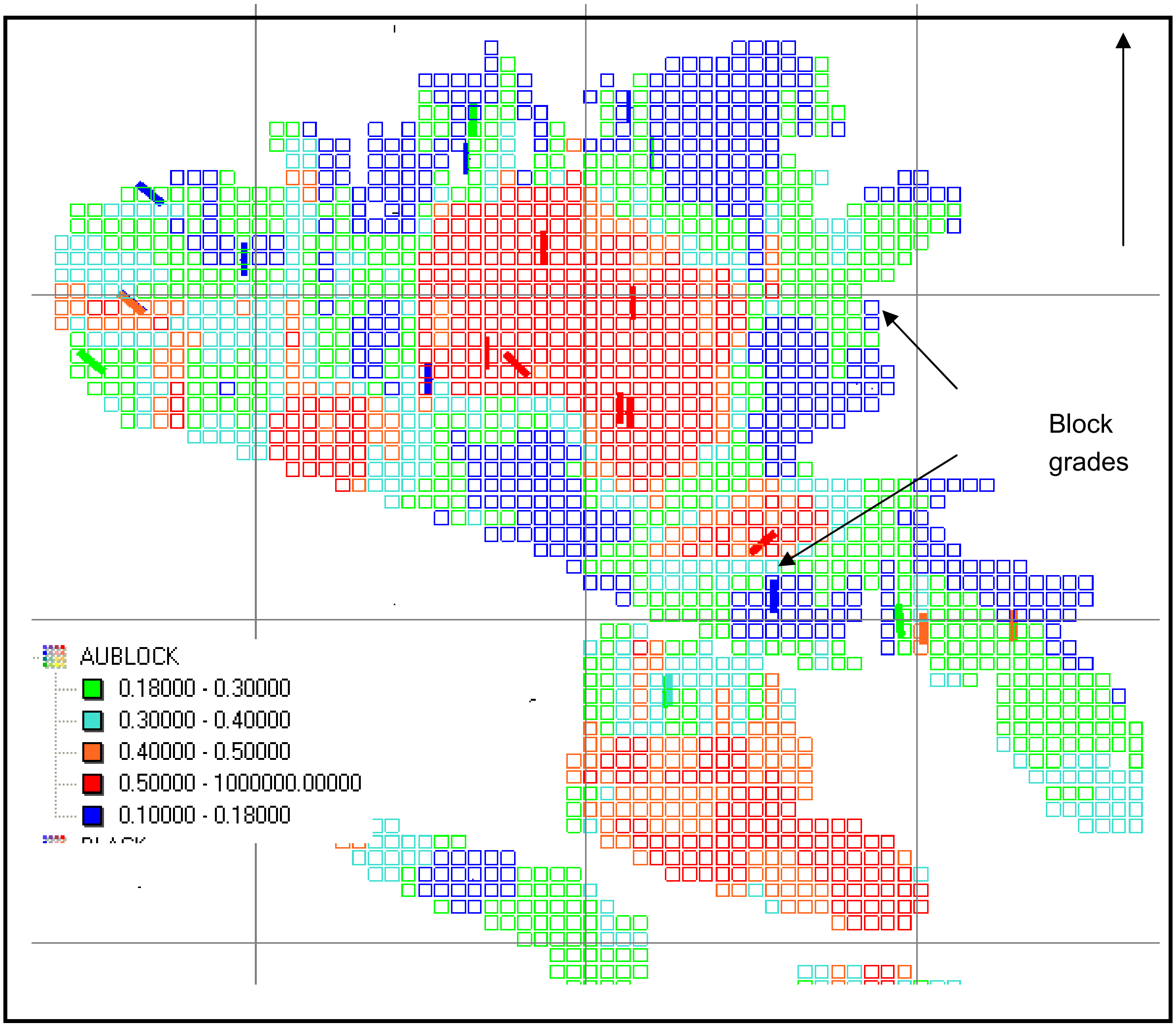

Figure 17.10: Tepal Level 46, Comparison Of Drill Hole Composites And Estimated Block Grades For Gold

Note: block model blocks are 10 by 10m grid is 200 by 200m

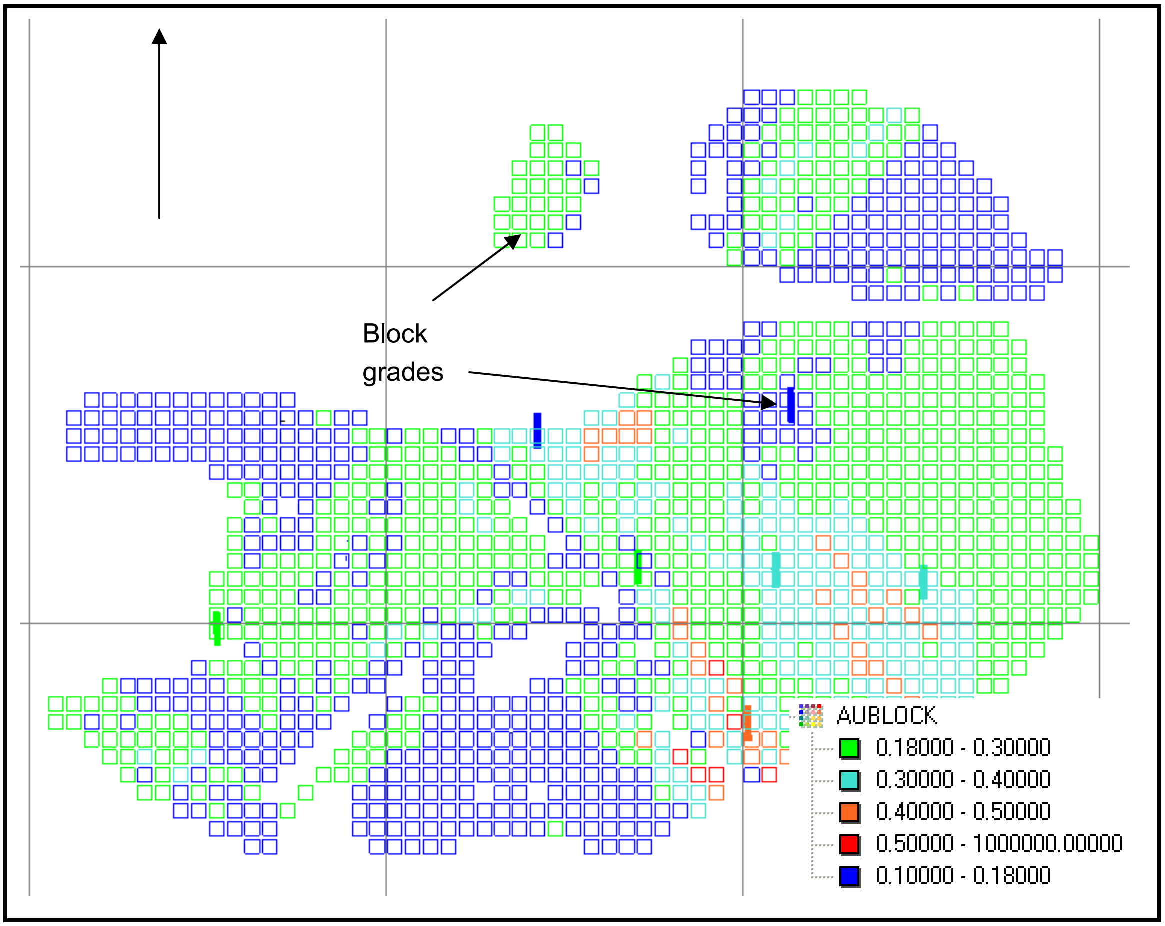

Figure 17.11: Tizate Level 81, Comparison Of Drill Hole Composite Grades And Interpolated Block Grades For Gold

Note: block model blocks are 10 by 10m grid is 200 by 200m

17.2.11 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005) defines a mineral resource as:

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

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries.

SRK considers that the copper-gold mineralization evaluated in the Tepal and Tizate deposits are amenable for open pit extraction.

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

SRK selected to report the mineral resources at a cut-off grade of US$ 5.00 equivalent. The cut-off grade is selected to represent a break even mining costs at an open pit type operation of the size envisaged for Tepal and Tizate. Based on the above, SRK estimated that the Tepal and Tizate deposits contained 57.8 million tonnes of Indicated mineral resources grading 0.42 g/t Au and 0.24% Cu. The deposits contained an additional 93.2 million tonnes grading 0.28 g/t Au and 0.20 %Cu classified as Inferred mineral resource. The mineral resources were capped at 4 g/t Au and 1.5% Cu. Table 17.12 summarizes the mineral resources for each deposit.

Table 17.12: Mineral resource statement*, Tepal and Tizate deposit, Tepal property, Tepalcatepec Mexico, SRK Consulting (Canada) Inc, March 16, 2011

Deposit	Class	Quantity (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)
Tepal	Indicated	46.5	0.47	0.26	NE	NE
Tepal	Inferred	47.5	0.35	0.22	NE	NE
Tizate	Indicated	11.3	0.23	0.19	2.24	67
Tizate	Inferred	45.7	0.20	0.18	2.33	62
Total	Indicated	57.8	0.42	0.24	NE	NE
Total	Inferred	93.2	0.28	0.20	NE	NE

** Reported at a cut-off of US$ 5.00 within a Whittle pit shell. Gold grades were caped to 4 g/t and copper grades were capped to 1.5%. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All numbers have been rounded to reflect the relative accuracy of the estimates. NE indicates that the grades were not estimated.*

17.2.12 Cut-off grade sensitivity

The mineral resources at Tepal and Tizate were evaluated for their sensitivity to the selection of the cut-off grade. Figure 17.12 demonstrates the sensitivity of the indicated mineral resources to cut-off grade for the Tepal deposit and Figure 17.13 demonstrates the sensitivity of the inferred mineral resources.

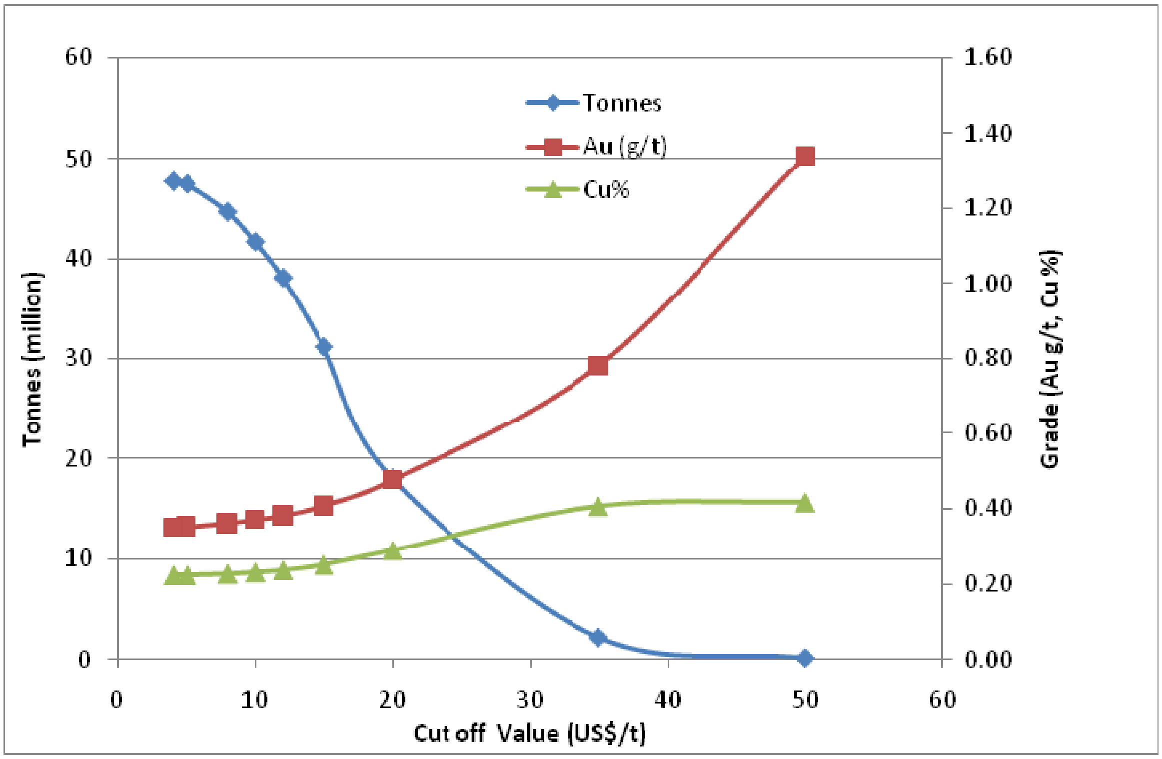

Figure 17.12: Grade Tonnage Curve for Tepal Indicated Mineral Resource

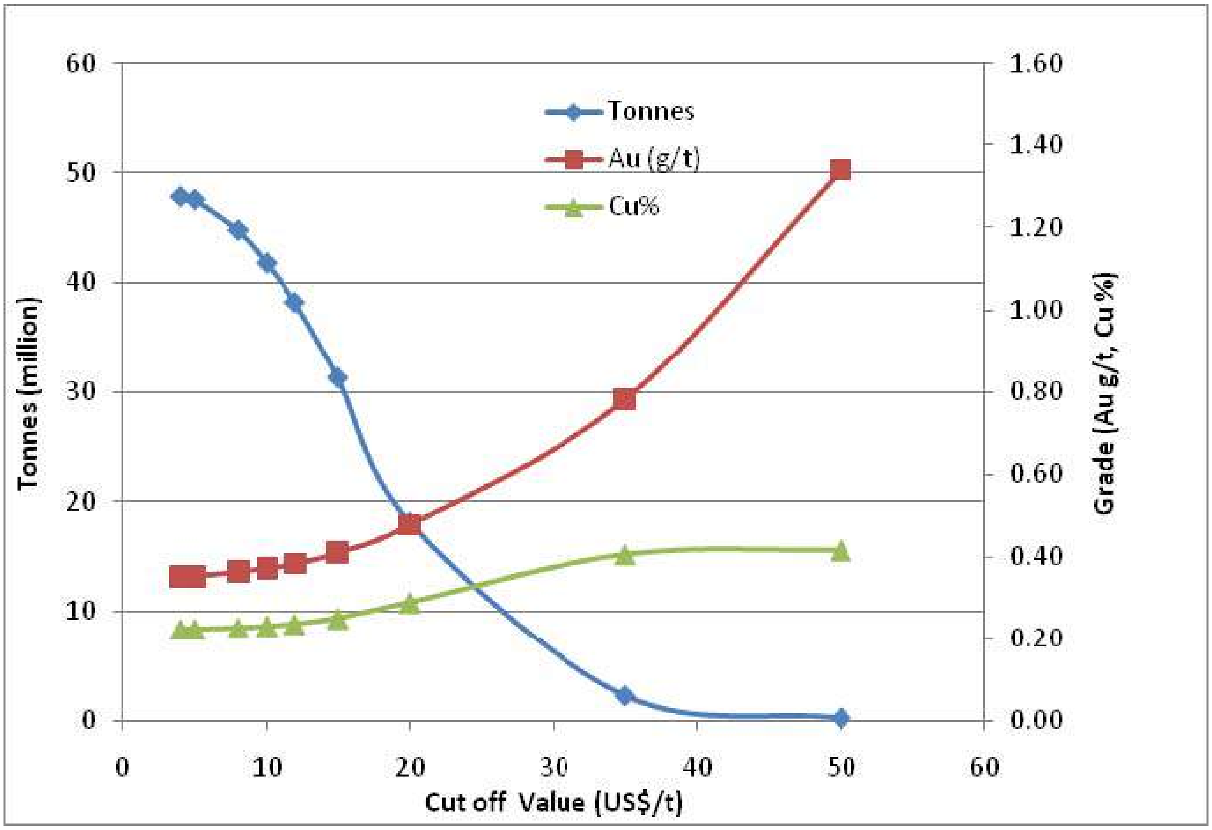

Figure 17.13: Grade Tonnage Curve For Tepal Inferred Mineral Resource

As can be seen from the figures, the deposits are very sensitive to cut-off grade. The sharp decrease in tonnage above the US$10/t cut-off is in part due to the development of modeling domains based on the 'natural' cut-off of US$8/t equivalent. This modeling criteria assured that most of the material enclosed within the wireframes was at least above US$8/t cut-off, therefore, the tonnage within the wireframes is not expected to vary much below the US$8/t cut-off.

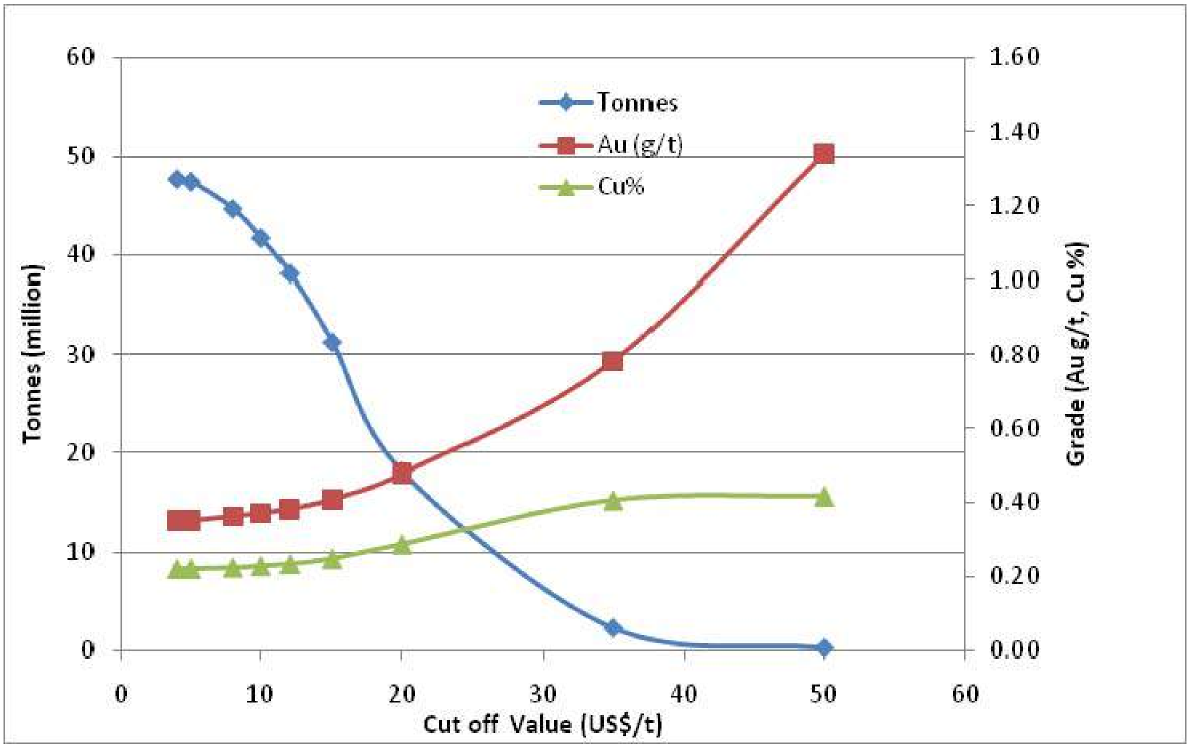

Figure 17.14: Grade Tonnage Curve For Tizate Indicated Mineral Resource

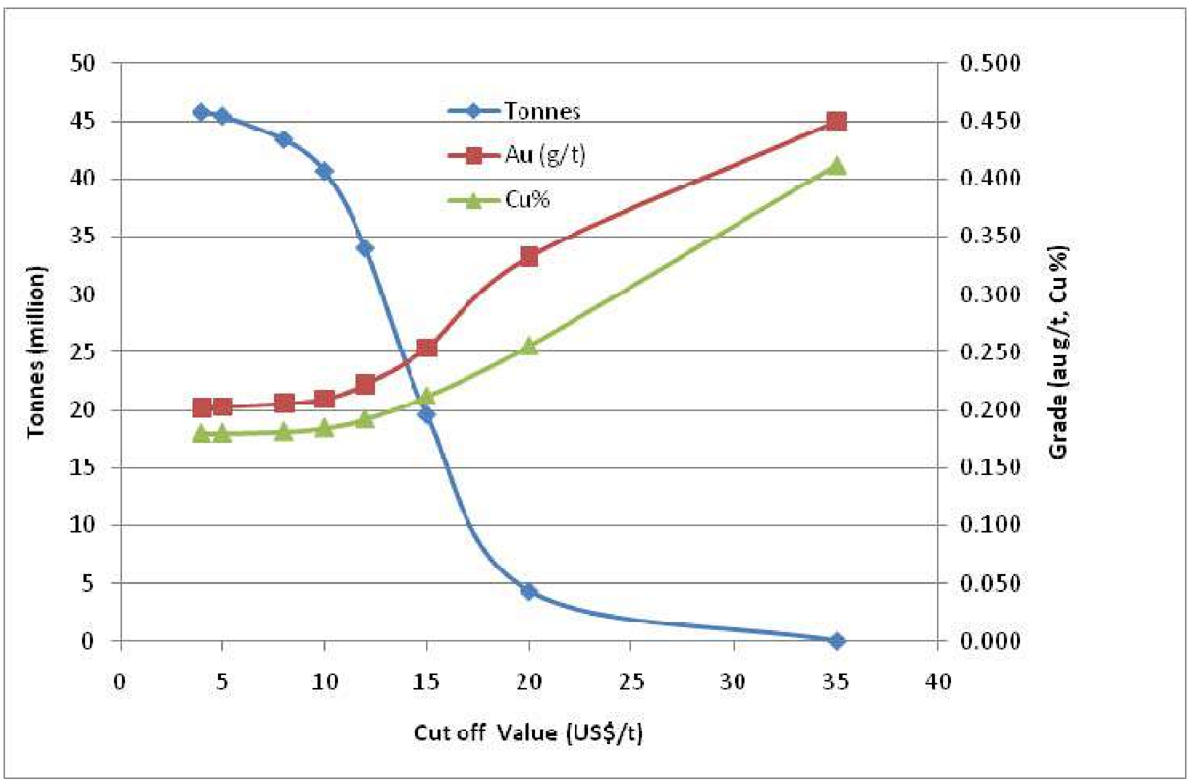

Figure 17.15: Grade Tonnage Curve for Tizate Inferred Mineral Resource

18 Other Relevant Data and Information

18.1 Geotechnical Information

18.1.1 Slope Design Review

SRK completed a scoping level review of available geotechnical and structural data for the purposes of open pit slope design. This review was based on available diamond drill core (onsite core review, core photo review, and core recovery and Rock Quality Designation (“RQD”) data), and 3D surfaces and solids. All data has been provided to SRK by Geologix. More details of the slope design review are presented in Appendix 1.

Structural Information

Fault structures within the planned Tepal open pits have been provided as 3D surfaces for the North, South and Tizate Zones. In the North and South Zone s these are currently interpreted as largely sub-vertical structures and are not likely to have a major impact on slope stability. In the Tizate pit the major structure dips into the slope wall.

The currently modelled structures are considered to be at a PEA/scoping level assessment, and would need to be evaluated in more detail for a pre-feasibility design.

Seismicity Potential

The Tepal property is located in a high seismic hazard zone. Within this zone the peak ground acceleration is more significant at the coast and reduces somewhat as you move inland towards the Tepal site. Based on available seismogenic data, peak ground accelerations, with a 500 year return period is in the range 4.6 to 5.6 m/s^2. This should be considered during planning and costing for the various facilities for open pit operations (waste dumps, tailings dams etc.).

Drill Core Review

<u>North and South Zone Oxide Surface</u>

A 3D surface representing the base of the oxide zone has been reviewed by SRK. Drill core photos show generally weak ground conditions throughout the oxide zone. In places this weak zone is interpreted to extend to 110m depth and beneath the currently modelled surface down into what may be termed the ‘mixed zone’. This needs to be verified in the core or through additional drilling.

At Tizate the upper oxidized zone is variable, but generally the upper weaker zone in the west and east domain is in the order of 50m in depth. In some cases in the East Domain the weaker oxidized zone can be as deep as 75m

<u>Drill Hole RQD</u>

Down hole RQD has been collected for most recent diamond drill holes at the Tepal project. The RQD for both the North and South Zones beneath the oxide zone shows improving rock mass quality with depth. Figure 18.1 shows down hole RQD data in relation to the planned North and South Zone open pit shells (left and right respectively). At Tizate, calibration logging from photographs indicated that the RQD where slightly higher than that predicted from site logging. This was likely a result of some mechanical damage being included in the RQD estimate. In general, the rock mass is weaker at the top, but is of variable strength with depth.

Slope recommendations for the North and South Zone have been made based on RQD data and core photo reviews, separated into oxide and fresh rock (beneath oxide zone) lithologies. For the Tizate Zone, Rock Mass Rating (RMR) values where estimated and used for the slope angle derivation.

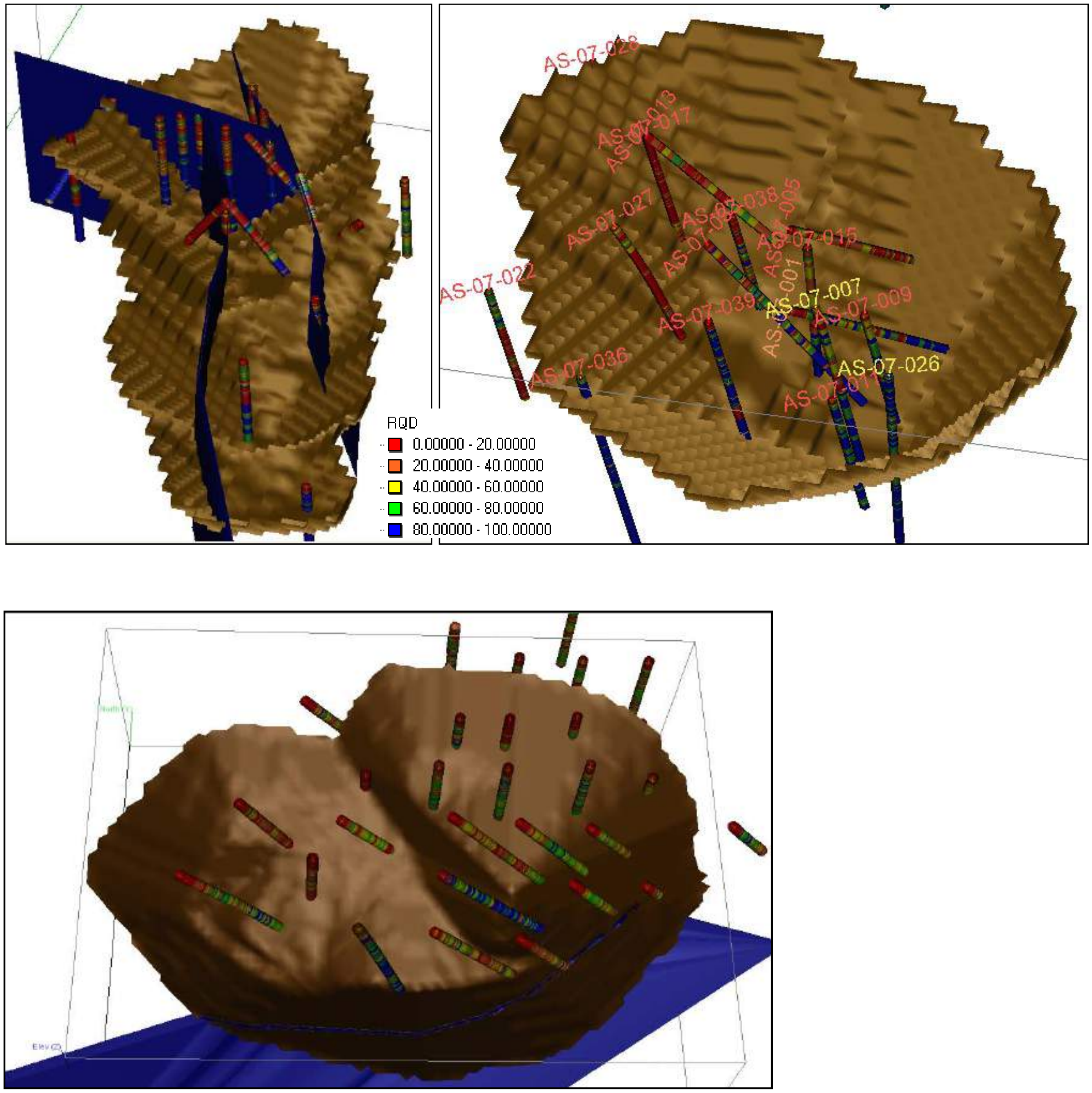

Figure 18.1: Down-hole RQD Data – North Zone (left) South Zone (right), Tizate (below) – no to scale

Slope Angle Recommendations

Table 18.1 and Figure 18.2 present the slope angle recommendations for the North and South Zone open pits.

Table 18.1: Slope Angle Recommendations

Open Pit	Sector	Oxide Zone		Fresh Rock	Comments
		Height	Inter Ramp Angle (°)		
North Zone	North East	60	40	50	Assumes oxidation reduces in thickness towards the slope areas
	North West	90	40	50	North of 2116600N
	South	20	40	50	South of 2116600N
South Zone	North			40	Possibility to increase IRA to 45° for a 50m height to accommodate a ramp
	South			50	Possibility to increase IRA to 55° for a 50m height to accommodate a ramp
Tizate	West	50	40	48	Maximum overall slope angle for a 200m heights is 43˚
	East	50	40	52	Maximum overall slope angle for a 200m slope height is 46˚

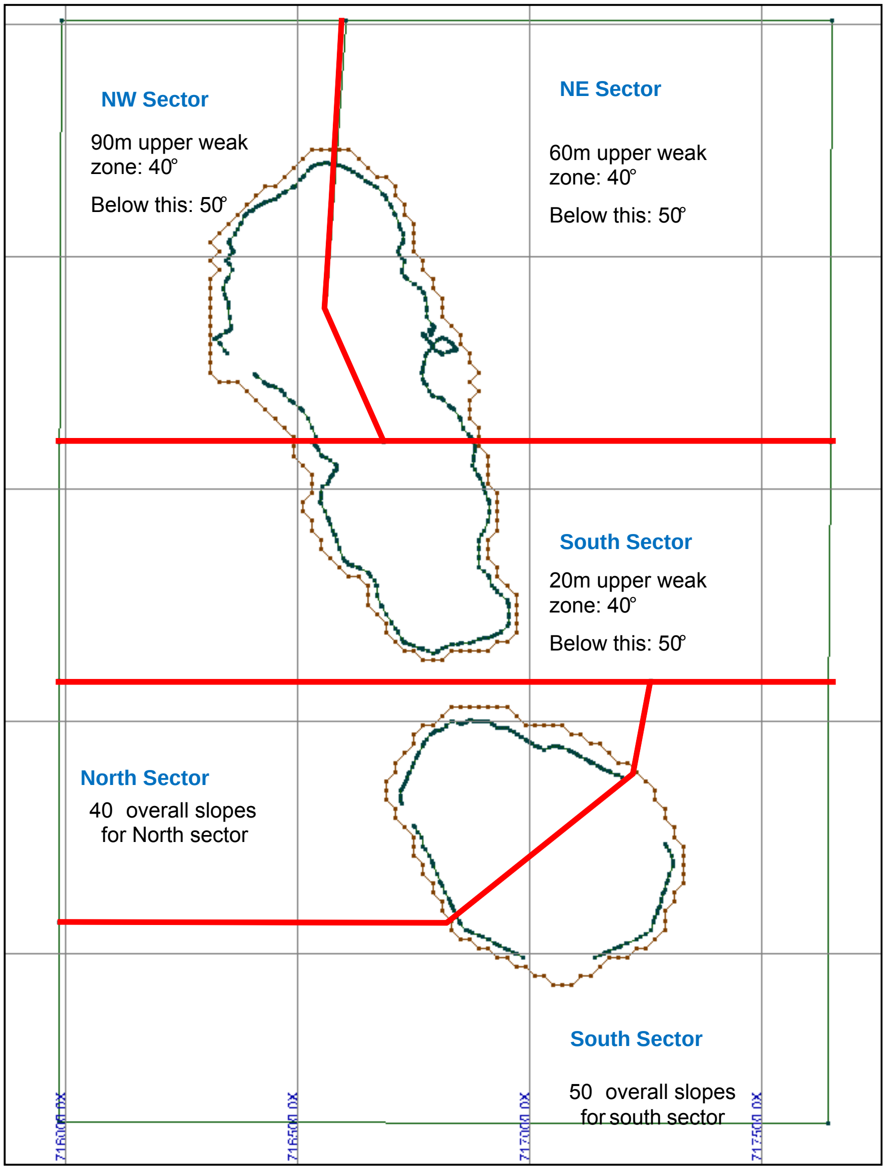

Figure 18.2: Summary of Slope Angle Recommendations for the North and South Zones

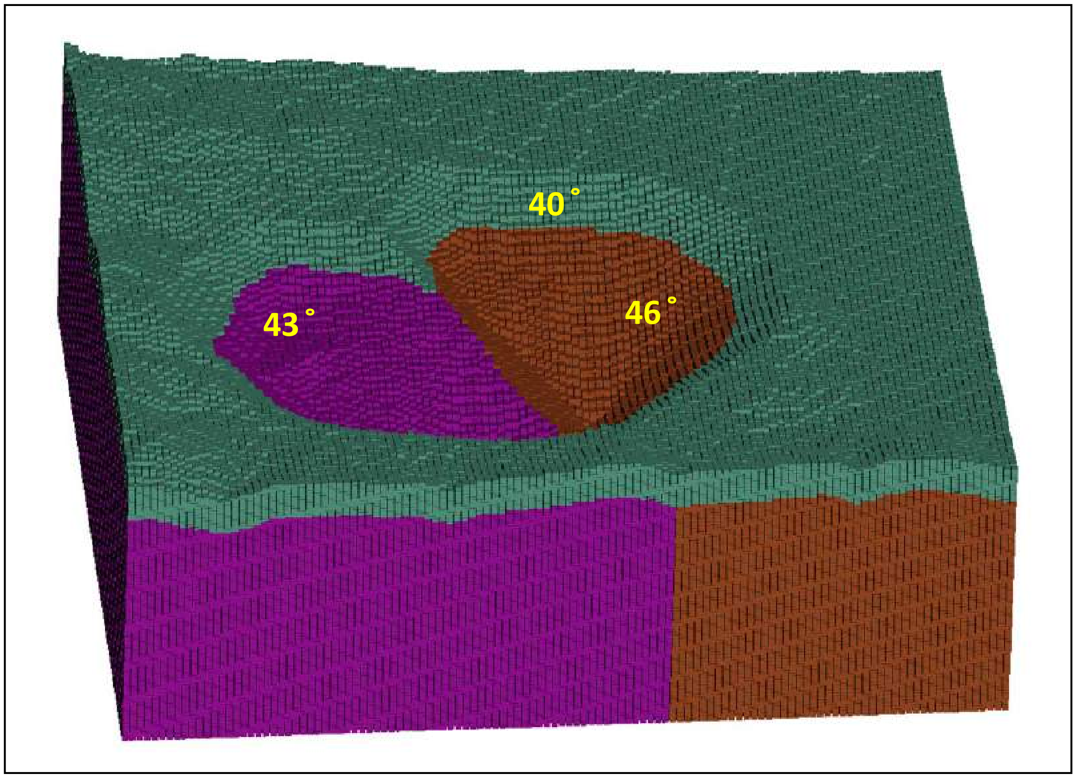

Figure 18.3: Slope angle domains for the Tizate pit as used in the Whittle optimization.

19 Additional Requirements for Technical Reports on Development Properties and Production Properties

19.1 Mining Operations

19.1.1 Whittle™ Pit Optimization

Net Smelter Return Model

The 3D mineral resource block models as developed by SRK were used as the basis for deriving the economic pit limits for the Tepal and Tizate deposits. A number of calculations were performed on the models in order to determine the net smelter return ("NSR") of each individual block. These parameters are summarized in Table 19.1.

The NSR calculations considered the following factors:

- Mineralized Zone grades (Cu, Au, Ag), thus taking into account the variability in the precious metal content of the deposit (on a whole block basis);
- Ore type (oxide or sulphide);
- Process recoveries for both flotation and heap leach;
- Operating costs;
- Contained metal in concentrate;
- Deductions and Payable Metal Value;
- Metal prices;
- Freight costs (trucking, rail, shipping, insurance);
- Smelting and refining charges (TC/RC); and
- Royalty charges.

Table 19.1: NSR Parameters Used in the Whittle™ Optimization Model

Item	Unit	Flotation	Heap	Comments
Exchange Rate	US$:C$	1.10	1.10	
Metal Prices				
Copper	$/lb	2.75	2.75	
Gold	$/oz	1000	1000	
Silver	$/oz	16.00	16.00	
Tepal Recovery				
Copper	%	87.4	14.3	
Gold	%	60.7	78.4	
Tizate Recovery				
Copper	%	85.1	6.8	
Gold	%	50.0	68.8	
Silver	%	54.7	38.9	
Cu Concentrate Grade – Tepal				
Copper	%	25.1	70	
Gold	g/t	Variable	Variable	
Cu Concentrate Grade – Tizate				
Copper	%	28.0	70	
Gold	g/t	Variable	Variable	
Silver	g/t	Variable	Variable	
Moisture content	%	8.0	8.0*	*SART Concentrate
Operating Costs				
Mining cost	$/t rock	1.35	1.35	Based on diesel fuel cost of US$0.68/l
Milling cost	$/t ore	4.30	4.31	
G&A/Sustaining Capital	$/t ore	0.68	0.68	
Royalties	%	2.5	2.5	Percentage of NSR
Off site costs				
Cu concentrate TC	$/dmt	50.00	50.00*	*SART Concentrate
Cu Refining	$/pay lb	0.05	0.05*	*SART Concentrate
Au refining	$/pay oz	5.50	5.50*	*SART Concentrate
Ag refining	$/pay oz	0.35	0.35*	*SART Concentrate
Transport to smelter	$/wmt	33.00	33.00*	*SART Concentrate
Re-handling (Truck to	$/wmt	3.30	3.30*	*SART Concentrate
Insurance	$/wmt	1.00	1.00*	*SART Concentrate
Ocean Freight	$/wmt	0.00	0.00	Ship to Mexican Smelter (San Luis de
Smelter Payables for Cu Concentrate				
Copper deduction	unit	0	0	
Payable Copper	%	97	97*	*SART Concentrate
Payable Gold	%	98	98	
Payable Silver	%	97.5	98	
Mine Parameters				
Mining Recovery	%	100	100	
Grade factor	%	95	95	
Production capacity	Mt/yr	8.0	3.0	Mill feed tonnage
Economics				
Discount Rate	%	5.0	5.0	

19.1.2 Economic Pit Limit

The ultimate economic pit limits were based on a Whittle™ pit optimization evaluation of the resources in the NSR models. This evaluation included the aforementioned NSR calculations as well as geotechnical parameters and mining/milling costs. The economic pit limits included indicated and 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 will be upgraded to a higher resource category.

19.1.3 Cut-off Grade

The base case economic parameters mentioned above were used to calculate NSR cut-off grades for the Tepal and Tizate deposits. The incremental cut-off grade incorporates mining dilution and all operating costs except mining. This cut-off is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the Whittle™ optimization. The incremental NSR cut-off of $5.23/t for flotation, and $5.24/t for heap leach material, was applied to all of the mineral resource estimates that follow for both Tepal and Tizate.

19.1.4 Optimization Parameters and Results

The geotechnical parameters as well as mining, milling, G&A and power costs are summarized in Table 19.2 for both flotation and heap leach for the Tepal and Tizate Deposits. The estimated projected topography as of early 2010 was used as the starting surface for the pit optimization.

A series of Whittle™ pit shells were generated based on varying revenue factors. The results were analyzed with pit shells chosen as the basis for further design work and preliminary phase designs.

Table 19.2: Operating Costs and Geotechnical Parameters Used for Pit Optimization

Parameter	Unit	Flotation	Heap Leach
Waste Mining OPEX	US$/waste tonne	1.35	1.35
Mineralized Zone Mining OPEX	US$/mill feed tonne	1.35	1.35
Processing, G&A, and Sustaining Capital OPEX	US$/milled tonne	4.98	4.99
Overall Pit Slope Angles w/ Ramps			
North Pit	degrees	50	40
South Pit (north portion)	degrees	40-45	40-45
South Pit (south portion)	degrees	50-55	50-55
Tizate – upper 50 m	degrees	40	40
Tizate SE sector – lower	degrees	46	46
Tizate all other - lower	degrees	43	43

The resources within the various pit shells were generated from the following 3-D block model items:

- Block centroid coordinates;
- Copper grade;
- Gold grade;
- Silver Grades (Tizate only);
- Resource category (indicated, inferred);
- Rock code;
- Topography percentage; and
- Specific gravity.

The results of the Whittle™ pit optimization evaluation on the Tepal deposit for varying revenue factors values (Whittle™ shell 36 is revenue factor 1.0) are summarized in Table 19.3, as well as, Figure 19.1 to Figure 19.3, for indicated and inferred resources.

The Tizate optimization results are summarized in Table 19.4 and Figures 19.4 to 19.6.

Table 19.3: Whittle™ Pit Optimization Results – Tepal

Final	Revenue	Mine	MILL Diluted	HEAP Diluted	TOTAL Diluted	TOTAL Diluted Grades			Waste	Strip	Total	NPV Best	NPV Worst	Incr.	Incr.	Incr.	NPV best	NPV worst
Pit	Factor	Life	(ktonnes)	(ktonnes)	(ktonnes)	Au (g/t)	Cu	NSR (US$/t)	(ktonnes)	Ratio	(ktonnes)	$ disc	$ disc	ktonne ore	ktonne waste	ratio	incr. $ disc	incr. $ disc
1	0.30	1.7	6,424	4,331	10,754	0.68	0.34	24.13	1,198	0.11	11,952	176,571,388	176,571,388					
2	0.32	1.8	7,808	4,884	12,692	0.64	0.34	23.39	1,686	0.13	14,378	197,884,802	197,218,310	1,938	488	0.25	21,313,414	20,646,922
3	0.34	2.2	11,012	5,544	16,556	0.61	0.32	22.45	3,113	0.19	19,669	240,370,381	239,529,588	3,864	1,427	0.37	42,485,579	42,311,278
4	0.36	2.8	15,517	6,515	22,032	0.57	0.31	21.35	5,141	0.23	27,173	293,179,063	290,784,949	5,476	2,028	0.37	52,808,682	51,255,361
5	0.38	3.2	18,382	7,010	25,392	0.55	0.30	20.81	6,421	0.25	31,813	322,499,705	319,406,618	3,360	1,280	0.38	29,320,642	28,621,669
6	0.40	3.6	21,795	7,517	29,312	0.53	0.29	20.24	7,968	0.27	37,279	353,976,391	349,835,629	3,920	1,547	0.39	31,476,686	30,429,011
7	0.42	4.0	25,262	8,328	33,589	0.51	0.28	19.80	11,237	0.33	44,826	386,128,203	381,133,340	4,277	3,269	0.76	32,151,812	31,297,711
8	0.44	4.2	26,919	8,615	35,534	0.50	0.28	19.57	12,207	0.34	47,741	399,619,439	394,185,011	1,945	970	0.50	13,491,236	13,051,671
9	0.46	4.7	30,500	8,988	39,488	0.49	0.27	19.20	15,290	0.39	54,779	426,141,899	417,410,934	3,954	3,083	0.78	26,522,460	23,225,923
10	0.48	5.0	33,186	9,451	42,637	0.48	0.27	18.90	17,755	0.42	60,393	444,585,329	434,939,775	3,149	2,465	0.78	18,443,430	17,528,841
11	0.50	5.8	39,500	10,503	50,003	0.46	0.26	18.18	22,000	0.44	72,002	482,792,794	464,998,325	7,366	4,245	0.58	38,207,465	30,058,550
12	0.52	6.2	42,793	10,843	53,636	0.45	0.26	17.96	25,734	0.48	79,370	501,772,047	480,839,624	3,633	3,734	1.03	18,979,253	15,841,299
13	0.54	6.6	46,114	11,059	57,172	0.44	0.25	17.73	29,277	0.51	86,449	518,212,116	493,633,078	3,536	3,543	1.00	16,440,069	12,793,454
14	0.56	7.1	49,762	11,504	61,267	0.44	0.25	17.50	34,205	0.56	95,472	536,388,100	505,672,482	4,095	4,928	1.20	18,175,984	12,039,404
15	0.58	7.4	51,876	11,643	63,519	0.43	0.25	17.38	37,259	0.59	100,778	545,764,931	513,227,616	2,252	3,054	1.36	9,376,831	7,555,134
16	0.60	7.7	54,485	11,945	66,431	0.43	0.25	17.23	41,450	0.62	107,880	556,683,238	522,743,434	2,912	4,191	1.44	10,918,307	9,515,818
17	0.62	7.9	56,218	11,964	68,183	0.42	0.24	17.14	44,215	0.65	112,398	562,811,358	527,547,570	1,752	2,765	1.58	6,128,120	4,804,136
18	0.64	8.2	58,412	12,002	70,414	0.42	0.24	17.04	48,277	0.69	118,691	570,298,681	532,798,320	2,231	4,062	1.82	7,487,323	5,250,750
19	0.66	8.5	60,724	12,053	72,778	0.42	0.24	16.92	52,409	0.72	125,186	577,401,592	537,680,490	2,364	4,132	1.75	7,102,911	4,882,170
20	0.68	8.6	62,018	12,105	74,123	0.41	0.24	16.87	55,688	0.75	129,811	581,408,630	540,734,261	1,345	3,279	2.44	4,007,038	3,053,771
21	0.70	9.8	71,431	12,353	83,784	0.41	0.24	16.83	91,704	1.09	175,488	611,581,462	564,870,326	9,661	36,016	3.73	30,172,832	24,136,065
22	0.72	10.0	73,051	12,409	85,460	0.41	0.24	16.78	96,162	1.13	181,622	615,263,378	567,075,928	1,676	4,458	2.66	3,681,916	2,205,602
23	0.74	10.2	74,867	12,425	87,292	0.41	0.24	16.72	101,664	1.16	188,956	619,483,455	568,908,453	1,832	5,502	3.00	4,220,077	1,832,525
24	0.76	10.4	75,907	12,483	88,389	0.41	0.23	16.68	104,881	1.19	193,270	621,716,507	570,082,843	1,097	3,217	2.93	2,233,052	1,174,390
25	0.78	10.5	76,947	12,504	89,450	0.41	0.23	16.68	109,927	1.23	199,377	624,099,335	571,433,367	1,061	5,046	4.76	2,382,828	1,350,524
26	0.80	10.6	77,531	12,518	90,049	0.41	0.23	16.65	111,072	1.23	201,121	624,923,148	571,489,978	599	1,145	1.91	823,813	56,611
27	0.82	10.7	78,610	12,537	91,147	0.41	0.23	16.61	114,929	1.26	206,076	626,559,046	571,678,003	1,098	3,857	3.51	1,635,898	188,025
28	**0.84**	**10.8**	**79,685**	**12,552**	**92,237**	**0.41**	**0.23**	**16.59**	**119,713**	**1.30**	**211,950**	**628,108,317**	**571,872,573**	**1,090**	**4,784**	**4.39**	**1,549,271**	**194,570**
29	0.86	10.9	80,224	12,554	92,778	0.41	0.23	16.57	121,710	1.31	214,488	628,707,814	571,810,817	541	1,997	3.69	599,497	-61,756
30	0.88	11.0	80,896	12,584	93,480	0.41	0.23	16.54	124,086	1.33	217,566	629,302,056	571,226,593	702	2,376	3.38	594,242	-584,224
31	0.90	11.1	81,795	12,605	94,401	0.40	0.23	16.49	126,499	1.34	220,899	629,937,928	570,043,110	921	2,413	2.62	635,872	-1,183,483
32	0.92	11.2	82,260	12,610	94,869	0.40	0.23	16.48	129,152	1.36	224,021	630,330,077	569,768,589	468	2,653	5.67	392,149	-274,521
33	0.94	11.2	82,602	12,630	95,233	0.40	0.23	16.46	130,202	1.37	225,434	630,484,230	569,092,405	364	1,050	2.88	154,153	-676,184
34	0.96	11.2	83,027	12,637	95,663	0.40	0.23	16.47	133,780	1.40	229,444	630,694,723	568,575,468	430	3,578	8.32	210,493	-516,937
35	0.98	11.3	83,634	12,656	96,291	0.40	0.23	16.44	136,093	1.41	232,383	630,822,059	567,508,047	628	2,313	3.68	127,336	-1,067,421
36	1.00	11.4	83,926	12,662	96,588	0.40	0.23	16.42	137,185	1.42	233,772	630,838,374	567,008,681	297	1,092	3.68	16,315	-499,366
37	1.02	11.4	84,283	12,666	96,949	0.40	0.23	16.41	139,012	1.43	235,961	630,809,411	566,322,483	361	1,827	5.06	-28,963	-686,198
38	1.04	11.4	84,430	12,676	97,105	0.40	0.23	16.41	139,903	1.44	237,008	630,774,240	566,060,834	156	891	5.71	-35,171	-261,649
39	1.06	11.5	84,943	12,681	97,624	0.40	0.23	16.38	142,355	1.46	239,980	630,583,464	564,842,612	519	2,452	4.72	-190,776	-1,218,222
40	1.08	11.5	85,315	12,690	98,005	0.40	0.23	16.38	144,990	1.48	242,995	630,362,724	563,854,852	381	2,635	6.92	-220,740	-987,760
41	1.10	11.6	85,616	12,691	98,306	0.40	0.23	16.36	146,211	1.49	244,518	630,193,751	563,122,438	301	1,221	4.06	-168,973	-732,414
42	1.12	11.6	85,976	12,696	98,672	0.40	0.23	16.34	148,066	1.50	246,739	629,914,817	561,960,805	366	1,855	5.07	-278,934	-1,161,633
43	1.14	11.6	86,101	12,707	98,808	0.40	0.23	16.34	148,771	1.51	247,579	629,796,812	561,545,124	136	705	5.18	-118,005	-415,681
44	1.16	11.7	86,540	12,712	99,252	0.40	0.23	16.31	150,969	1.52	250,222	629,361,046	560,060,481	444	2,198	4.95	-435,766	-1,484,643
45	1.18	11.7	86,814	12,717	99,530	0.40	0.23	16.31	153,348	1.54	252,878	628,941,956	559,100,687	278	2,379	8.56	-419,090	-959,794
46	1.20	11.8	87,059	12,723	99,782	0.40	0.23	16.31	155,367	1.56	255,150	628,548,068	558,214,974	252	2,019	8.01	-393,888	-885,713
47	1.22	11.8	87,084	12,724	99,808	0.40	0.23	16.30	155,429	1.56	255,237	628,523,983	558,134,292	26	62	2.38	-24,085	-80,682
48	1.24	11.8	87,210	12,729	99,939	0.40	0.23	16.30	156,173	1.56	256,112	628,333,899	557,588,058	131	744	5.68	-190,084	-546,234
49	1.26	11.8	87,493	12,731	100,224	0.40	0.23	16.28	157,878	1.58	258,102	627,872,505	556,352,030	285	1,705	5.98	-461,394	-1,236,028
50	1.28	11.8	87,682	12,736	100,418	0.40	0.23	16.27	158,955	1.58	259,374	627,548,888	555,567,292	194	1,077	5.55	-323,617	-784,738
51	1.30	11.8	87,759	12,737	100,496	0.40	0.23	16.27	159,546	1.59	260,042	627,384,445	555,174,200	78	591	7.58	-164,443	-393,092
52	1.32	11.9	87,926	12,739	100,665	0.40	0.23	16.26	160,658	1.60	261,322	627,044,054	554,341,456	169	1,112	6.58	-340,391	-832,744
53	1.34	11.9	88,087	12,739	100,826	0.40	0.23	16.26	162,182	1.61	263,008	626,607,920	553,537,444	161	1,524	9.47	-436,134	-804,012
54	1.36	11.9	88,146	12,740	100,886	0.40	0.23	16.26	162,868	1.61	263,754	626,419,052	553,223,789	60	686	11.43	-188,868	-313,655
55	1.38	11.9	88,259	12,742	101,001	0.40	0.23	16.26	164,128	1.63	265,129	626,049,448	552,429,800	115	1,260	10.96	-369,604	-793,989

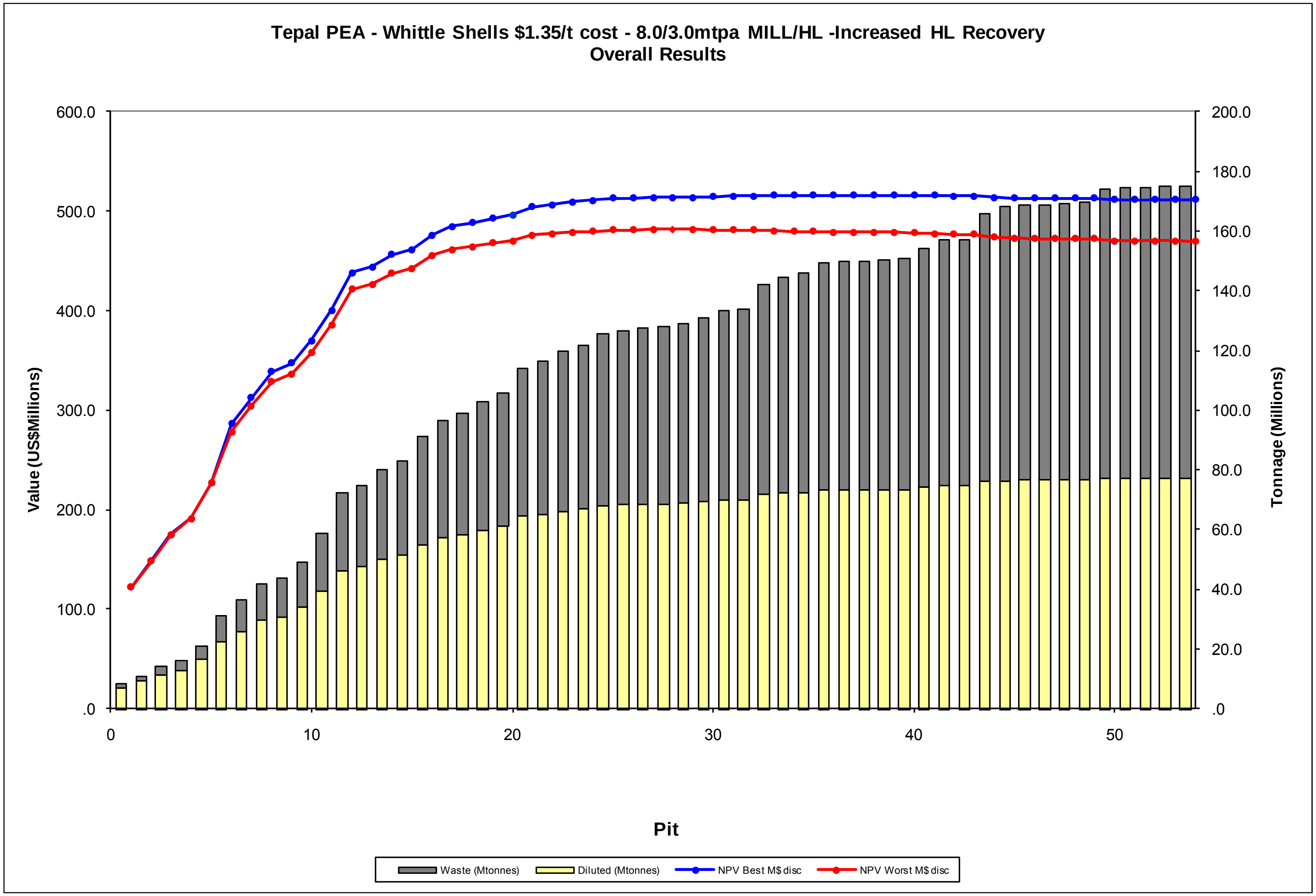

Figure 19.1: Whittle™ Pit Optimization Results – Tepal

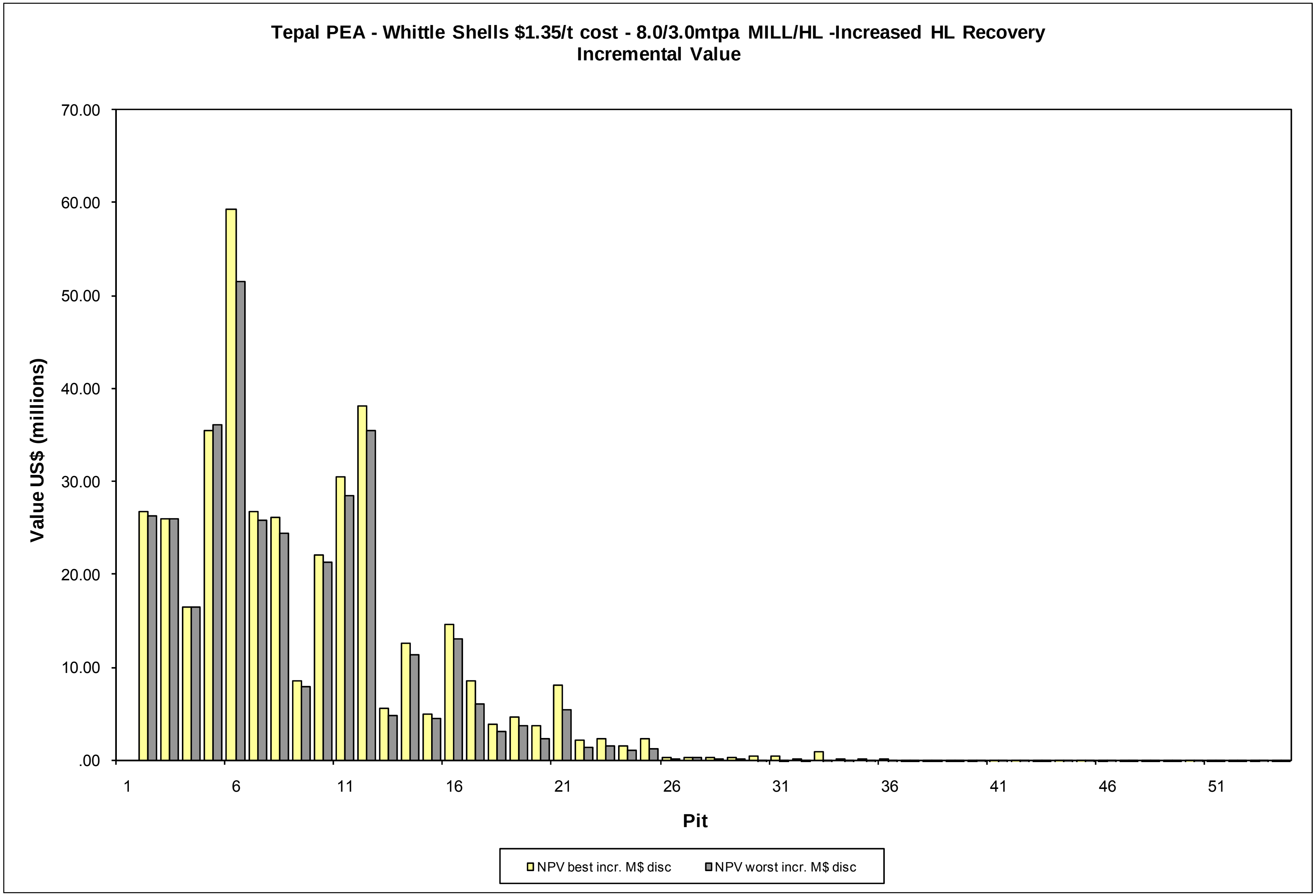

Figure 19.2: Incremental Whittle™ Value Results - Tepal

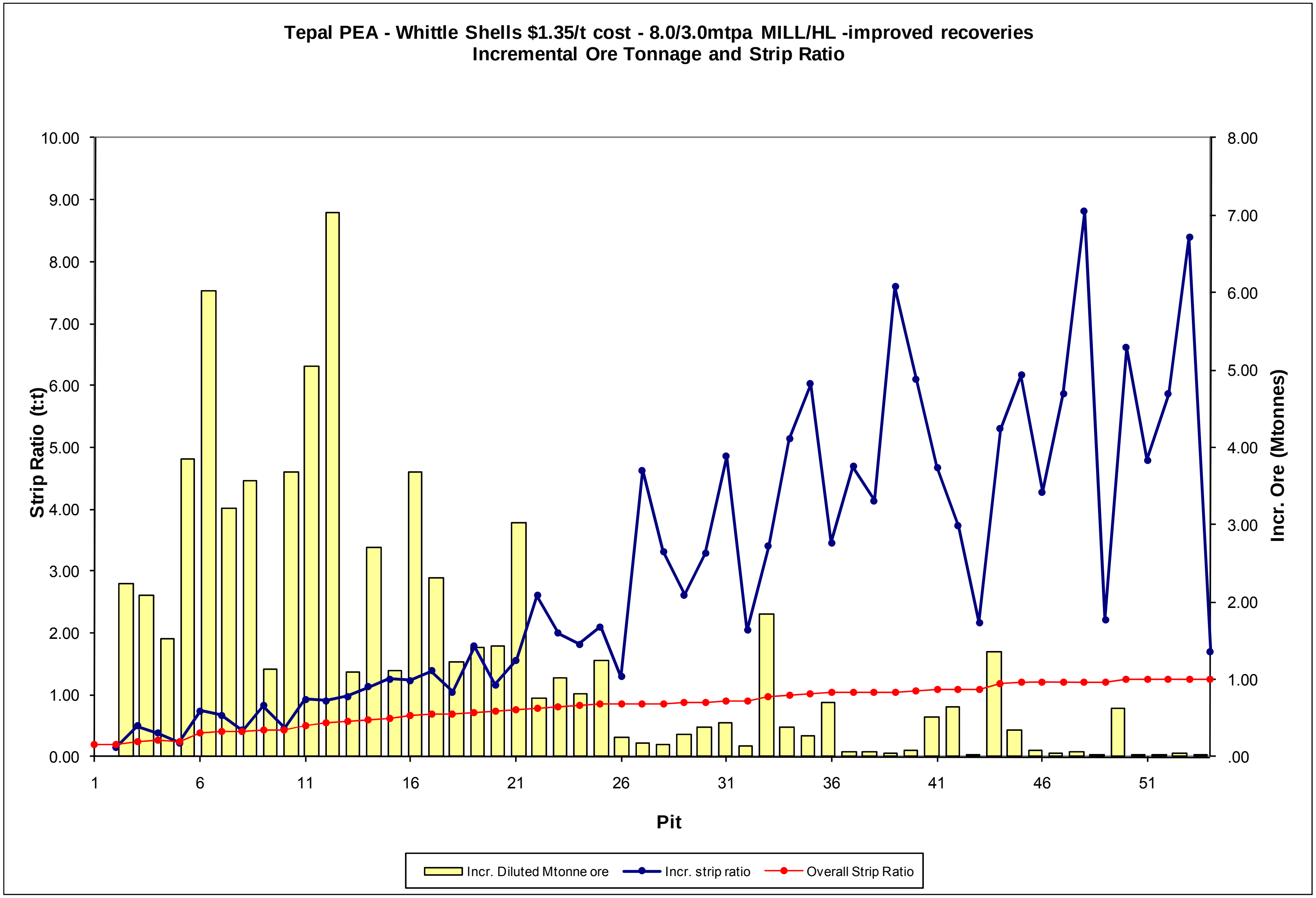

Figure 19.3: Incremental Whittle™ Tonnage Results - Tepal

Table 19.4: **Whittle™ Pit Optimization Results – Tizate**

Final Pit	Revenue Factor	Mine Life	MILL Diluted (ktonnes)	HEAP Diluted (ktonnes)	TOTAL Diluted (ktonnes)	TOTAL Diluted Grades Ag (g/t)	Au (g/t)	Cu (%)	Mo (%)	NSR (US$/t)	Waste (ktonnes)	Strip Ratio	Total (ktonnes)	Total CF (US$)	NPV Best $ disc	NPV Worst $ disc	Incr. Diluted ktonne ore	Incr. ktonne waste	Incr. strip ratio	NPV best incr. $ disc	NPV worst incr. $ disc
1	0.36	0.0	0	3	3	2.05	0.85	0.27	0.001	18.37	0	0.11	3	35,723	35,721	35,721					
2	0.38	0.0	0	12	12	2.02	0.81	0.27	0.001	17.60	1	0.11	13	133,253	133,227	133,227	9	1	0.11	97,506	97,506
3	0.40	0.0	7	40	48	1.99	0.74	0.25	0.002	16.72	2	0.05	50	490,856	490,536	490,536	36	1	0.03	357,309	357,309
4	0.42	0.0	45	113	157	2.07	0.66	0.24	0.002	15.74	3	0.02	160	1,474,907	1,472,205	1,472,205	109	1	0.01	981,669	981,669
5	0.44	0.1	215	261	476	2.24	0.55	0.22	0.004	14.69	9	0.02	485	3,966,630	3,949,849	3,949,849	319	6	0.02	2,477,644	2,477,644
6	0.46	0.1	386	357	744	2.29	0.50	0.22	0.005	14.11	13	0.02	757	5,767,209	5,733,780	5,733,780	268	4	0.01	1,783,931	1,783,931
7	0.48	0.2	945	549	1,494	2.29	0.43	0.21	0.006	13.58	93	0.06	1,588	10,700,776	10,605,666	10,605,666	750	80	0.11	4,871,886	4,871,886
8	0.50	0.2	1,126	605	1,731	2.31	0.42	0.21	0.006	13.50	143	0.08	1,874	12,217,140	12,097,519	12,097,519	237	50	0.21	1,491,853	1,491,853
9	0.52	0.2	1,841	748	2,589	2.44	0.38	0.21	0.006	13.22	367	0.14	2,956	17,330,816	17,121,182	17,121,182	858	224	0.26	5,023,663	5,023,663
10	0.54	0.4	2,816	946	3,762	2.51	0.34	0.21	0.006	12.78	545	0.14	4,307	23,504,227	23,103,942	23,103,942	1,173	178	0.15	5,982,760	5,982,760
11	0.56	0.5	4,257	1,228	5,486	2.97	0.30	0.21	0.007	12.88	1,988	0.36	7,474	33,222,125	32,370,619	32,370,619	1,724	1,443	0.84	9,266,677	9,266,677
12	0.58	0.9	7,546	1,349	8,896	2.56	0.29	0.20	0.006	12.50	3,371	0.38	12,267	50,315,815	48,052,560	48,052,560	3,410	1,383	0.41	15,681,941	15,681,941
13	0.60	1.3	10,068	1,418	11,486	2.52	0.28	0.20	0.006	12.36	4,859	0.42	16,345	62,693,601	59,567,488	59,536,085	2,590	1,488	0.57	11,514,928	11,483,525
14	0.62	1.6	12,695	1,450	14,145	2.44	0.26	0.20	0.006	12.20	6,302	0.45	20,447	74,561,295	70,374,898	70,184,396	2,659	1,443	0.54	10,807,410	10,648,311
15	0.64	2.6	21,127	1,592	22,719	2.27	0.24	0.19	0.006	11.75	10,261	0.45	32,980	109,356,660	100,811,847	99,855,322	8,574	3,959	0.46	30,436,949	29,670,926
16	0.66	3.0	23,622	1,645	25,268	2.22	0.23	0.19	0.006	11.66	11,712	0.46	36,979	118,941,953	108,784,588	107,460,609	2,549	1,451	0.57	7,972,741	7,605,287
17	0.68	3.3	26,116	1,675	27,791	2.19	0.23	0.19	0.006	11.57	13,343	0.48	41,134	127,726,080	116,208,808	114,563,615	2,523	1,631	0.65	7,424,220	7,103,006
18	0.70	3.5	28,350	1,691	30,041	2.16	0.23	0.19	0.007	11.50	14,973	0.50	45,014	135,073,655	122,289,927	120,241,117	2,250	1,630	0.72	6,081,119	5,677,502
19	0.72	3.9	31,207	1,707	32,914	2.13	0.22	0.19	0.007	11.42	17,546	0.53	50,460	143,835,379	129,300,818	126,606,066	2,873	2,573	0.90	7,010,891	6,364,949
20	0.74	4.3	34,455	1,710	36,165	2.09	0.22	0.18	0.007	11.32	20,445	0.57	56,611	152,762,898	136,436,599	133,076,435	3,251	2,899	0.89	7,135,781	6,470,369
21	0.76	4.5	35,919	1,714	37,632	2.09	0.22	0.18	0.007	11.28	22,110	0.59	59,743	156,535,454	139,402,103	135,732,810	1,467	1,665	1.13	2,965,504	2,656,375
22	0.78	4.7	37,695	1,714	39,410	2.09	0.22	0.18	0.007	11.24	24,287	0.62	63,696	160,782,521	142,666,276	138,527,612	1,778	2,177	1.22	3,264,173	2,794,802
23	0.80	4.9	39,394	1,715	41,110	2.09	0.22	0.18	0.006	11.19	26,359	0.64	67,469	164,345,155	145,328,543	140,697,638	1,700	2,072	1.22	2,662,267	2,170,026
24	0.82	5.1	40,740	1,718	42,459	2.09	0.21	0.18	0.006	11.15	27,903	0.66	70,361	166,805,205	147,187,598	142,241,104	1,349	1,544	1.14	1,859,055	1,543,466
25	0.84	5.3	42,535	1,719	44,253	2.10	0.21	0.18	0.006	11.11	30,919	0.70	75,172	169,994,557	149,627,123	144,223,834	1,794	3,016	1.68	2,439,525	1,982,730
26	0.86	5.7	45,801	1,724	47,525	2.15	0.21	0.18	0.006	11.03	36,073	0.76	83,598	174,793,782	153,170,419	146,836,626	3,272	5,154	1.58	3,543,296	2,612,792
27	0.88	5.9	46,997	1,724	48,721	2.15	0.21	0.18	0.006	11.01	38,391	0.79	87,112	176,407,988	154,324,829	147,550,222	1,196	2,318	1.94	1,154,410	713,596
28	0.90	6.0	47,814	1,724	49,539	2.16	0.21	0.18	0.006	10.99	39,800	0.80	89,338	177,288,353	154,937,369	147,849,139	818	1,409	1.72	612,540	298,917
29	0.92	6.1	48,496	1,724	50,220	2.15	0.21	0.18	0.006	10.98	41,190	0.82	91,411	177,892,890	155,373,478	147,998,168	681	1,390	2.04	436,109	149,029
30	0.94	6.2	49,509	1,724	51,234	2.16	0.21	0.18	0.006	10.96	43,438	0.85	94,671	178,612,155	155,897,063	148,129,779	1,014	2,248	2.22	523,585	131,611
31	0.96	6.3	50,168	1,724	51,892	2.16	0.21	0.18	0.006	10.94	44,756	0.86	96,648	178,947,841	156,136,821	148,113,541	658	1,318	2.00	239,758	-16,238
32	0.98	6.3	50,541	1,724	52,265	2.16	0.21	0.18	0.006	10.93	45,606	0.87	97,872	179,044,755	156,203,253	148,011,935	373	850	2.28	66,432	-101,606
33	1.00	6.4	51,569	1,724	53,293	2.16	0.21	0.18	0.006	10.91	48,212	0.90	101,505	179,111,251	156,238,280	147,555,545	1,028	2,606	2.54	35,027	-456,390
34	1.02	6.5	52,013	1,724	53,738	2.17	0.21	0.18	0.006	10.91	49,578	0.92	103,316	179,080,053	156,209,605	147,262,478	445	1,366	3.07	-28,675	-293,067
35	1.04	6.5	52,223	1,725	53,947	2.17	0.21	0.18	0.006	10.90	50,138	0.93	104,085	179,022,110	156,164,695	147,092,884	209	560	2.68	-44,910	-169,594
36	1.06	6.6	52,808	1,725	54,533	2.17	0.21	0.18	0.006	10.89	51,933	0.95	106,466	178,704,650	155,927,073	146,513,790	586	1,795	3.06	-237,622	-579,094
37	1.08	6.6	53,127	1,725	54,852	2.18	0.20	0.18	0.006	10.89	52,865	0.96	107,717	178,480,378	155,761,194	146,168,076	319	932	2.92	-165,879	-345,714
38	1.10	6.7	53,360	1,725	55,085	2.18	0.20	0.18	0.006	10.88	53,522	0.97	108,607	178,291,512	155,622,344	145,882,748	233	657	2.82	-138,850	-285,328
39	1.12	6.7	53,394	1,725	55,119	2.18	0.20	0.18	0.006	10.88	53,623	0.97	108,741	178,255,608	155,596,096	145,836,851	34	101	2.97	-26,248	-45,897
40	1.14	6.7	53,838	1,725	55,563	2.18	0.20	0.18	0.006	10.87	55,160	0.99	110,723	177,691,633	155,185,754	145,150,228	444	1,537	3.46	-410,342	-686,623
41	1.16	6.7	53,914	1,725	55,639	2.18	0.20	0.18	0.006	10.87	55,514	1.00	111,153	177,565,000	155,094,075	144,970,906	76	354	4.66	-91,679	-179,322
42	1.18	6.8	54,074	1,725	55,799	2.18	0.20	0.18	0.006	10.86	56,054	1.00	111,853	177,321,751	154,918,130	144,695,994	160	540	3.38	-175,945	-274,912
43	1.20	6.8	54,534	1,725	56,259	2.19	0.20	0.18	0.006	10.85	57,780	1.03	114,039	176,465,391	154,301,603	143,744,961	460	1,726	3.75	-616,527	-951,033
44	1.22	6.8	54,630	1,725	56,355	2.19	0.20	0.18	0.006	10.85	58,160	1.03	114,515	176,267,963	154,159,963	143,520,229	96	380	3.96	-141,640	-224,732
45	1.24	6.8	54,685	1,725	56,410	2.19	0.20	0.18	0.006	10.85	58,366	1.03	114,776	176,150,700	154,075,916	143,393,953	55	206	3.75	-84,047	-126,276
46	1.26	6.8	54,762	1,725	56,487	2.19	0.20	0.18	0.006	10.85	58,655	1.04	115,142	175,974,960	153,950,056	143,224,576	77	289	3.75	-125,860	-169,377
47	1.28	6.9	55,152	1,725	56,876	2.19	0.20	0.18	0.006	10.84	60,206	1.06	117,082	175,002,375	153,255,483	142,209,869	389	1,551	3.99	-694,573	-1,014,707
48	1.30	6.9	55,192	1,725	56,917	2.19	0.20	0.18	0.006	10.84	60,407	1.06	117,324	174,882,680	153,170,192	142,077,429	41	201	4.90	-85,291	-132,440
49	1.32	6.9	55,226	1,725	56,950	2.19	0.20	0.18	0.006	10.83	60,552	1.06	117,502	174,785,875	153,101,245	141,993,240	33	145	4.39	-68,947	-84,189
50	1.34	6.9	55,476	1,725	57,200	2.20	0.20	0.18	0.006	10.83	61,600	1.08	118,800	174,050,255	152,578,206	141,264,186	250	1,048	4.19	-523,039	-729,054
51	1.36	7.0	55,632	1,725	57,357	2.20	0.20	0.18	0.006	10.82	62,273	1.09	119,630	173,557,239	152,228,435	140,780,102	157	673	4.29	-349,771	-484,084
52	1.38	7.0	55,742	1,725	57,467	2.20	0.20	0.18	0.006	10.82	62,799	1.09	120,266	173,172,490	151,955,787	140,413,354	110	526	4.78	-272,648	-366,748
53	1.40	7.0	55,773	1,725	57,497	2.20	0.20	0.18	0.006	10.82	62,962	1.10	120,459	173,058,050	151,874,730	140,292,898	30	163	5.43	-81,057	-120,456

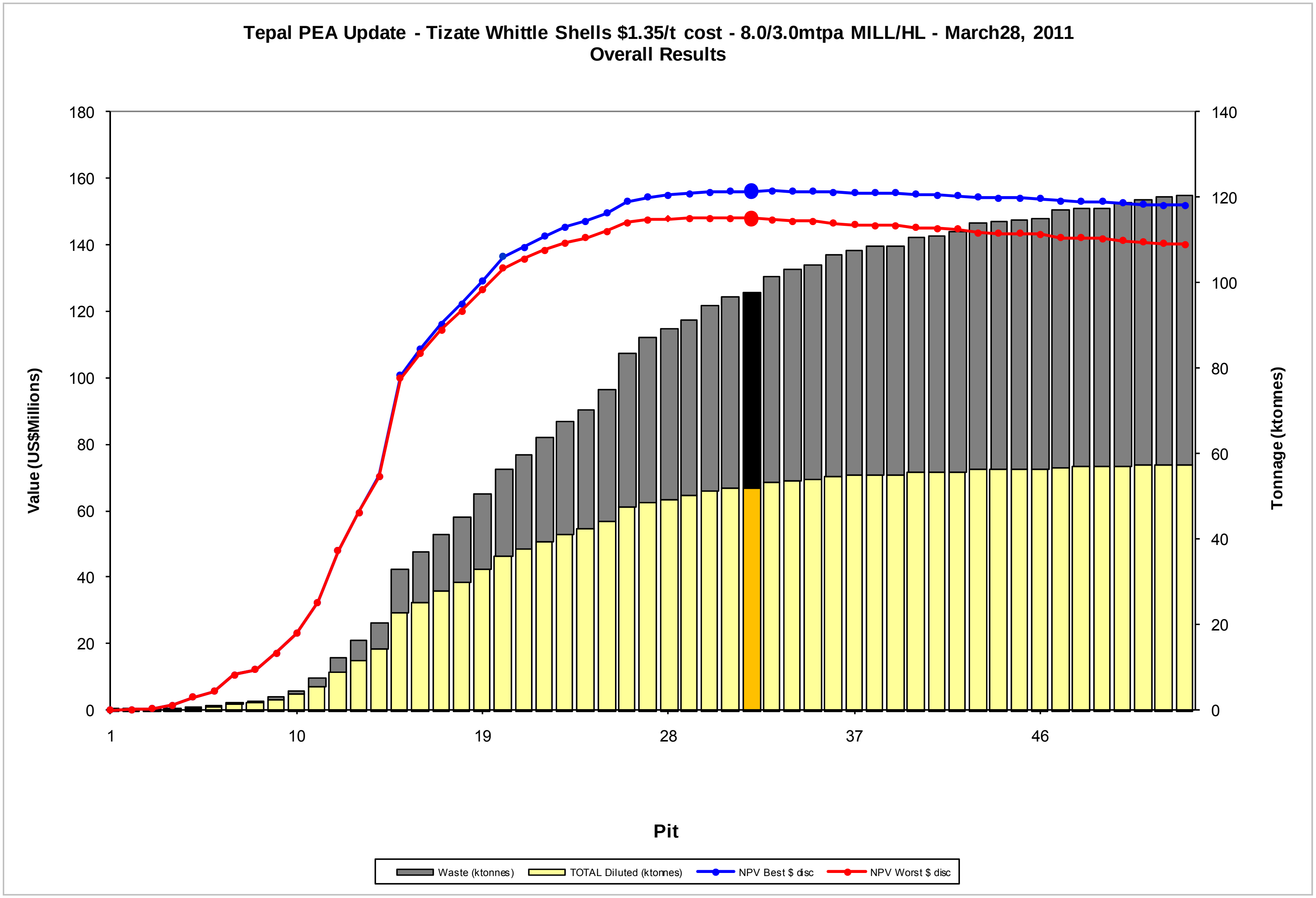

Figure 19.4: **Whittle™ Pit Optimization Results - Tizate**

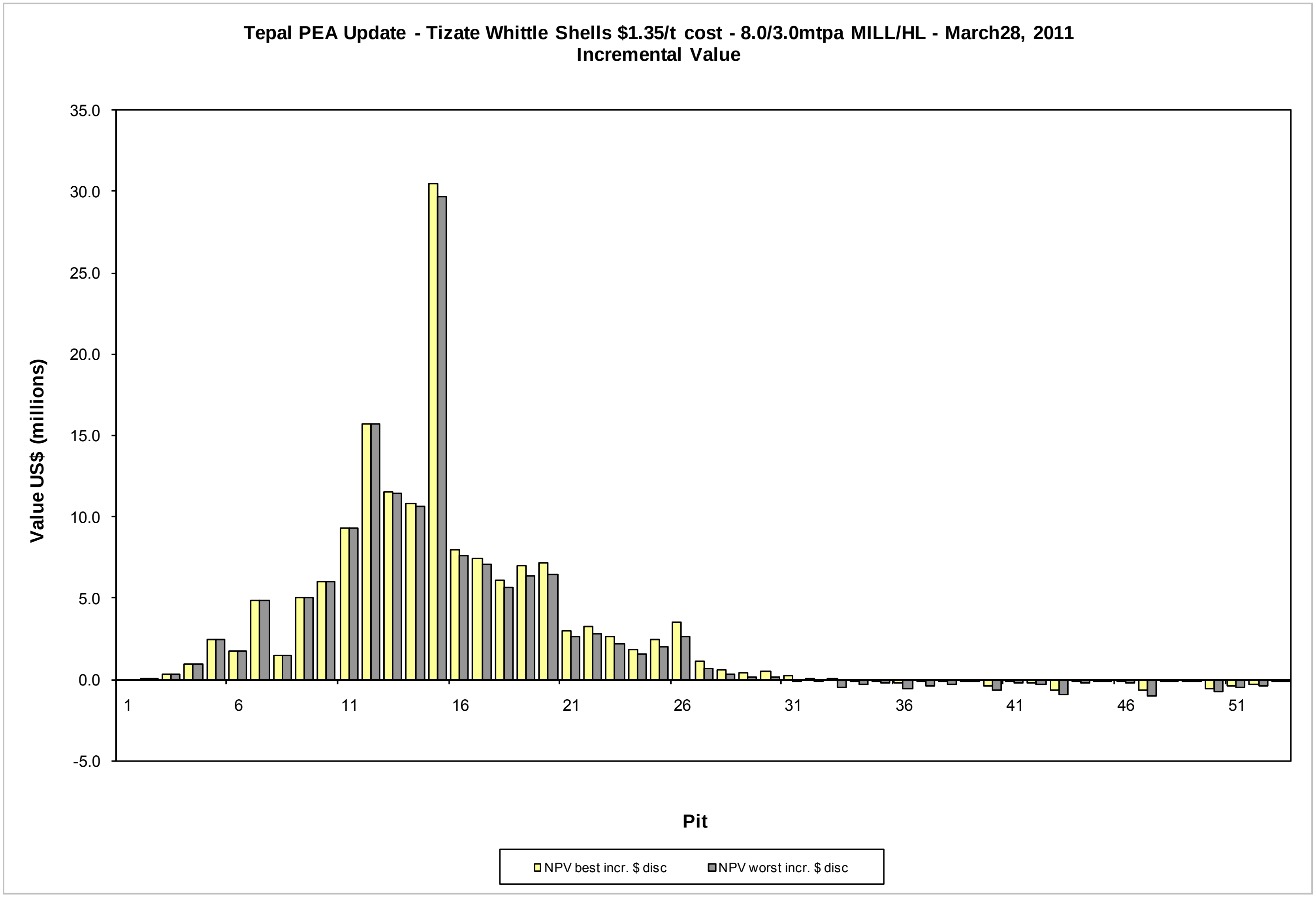

Figure 19.5: **Incremental Whittle™ Value Results – Tizate**

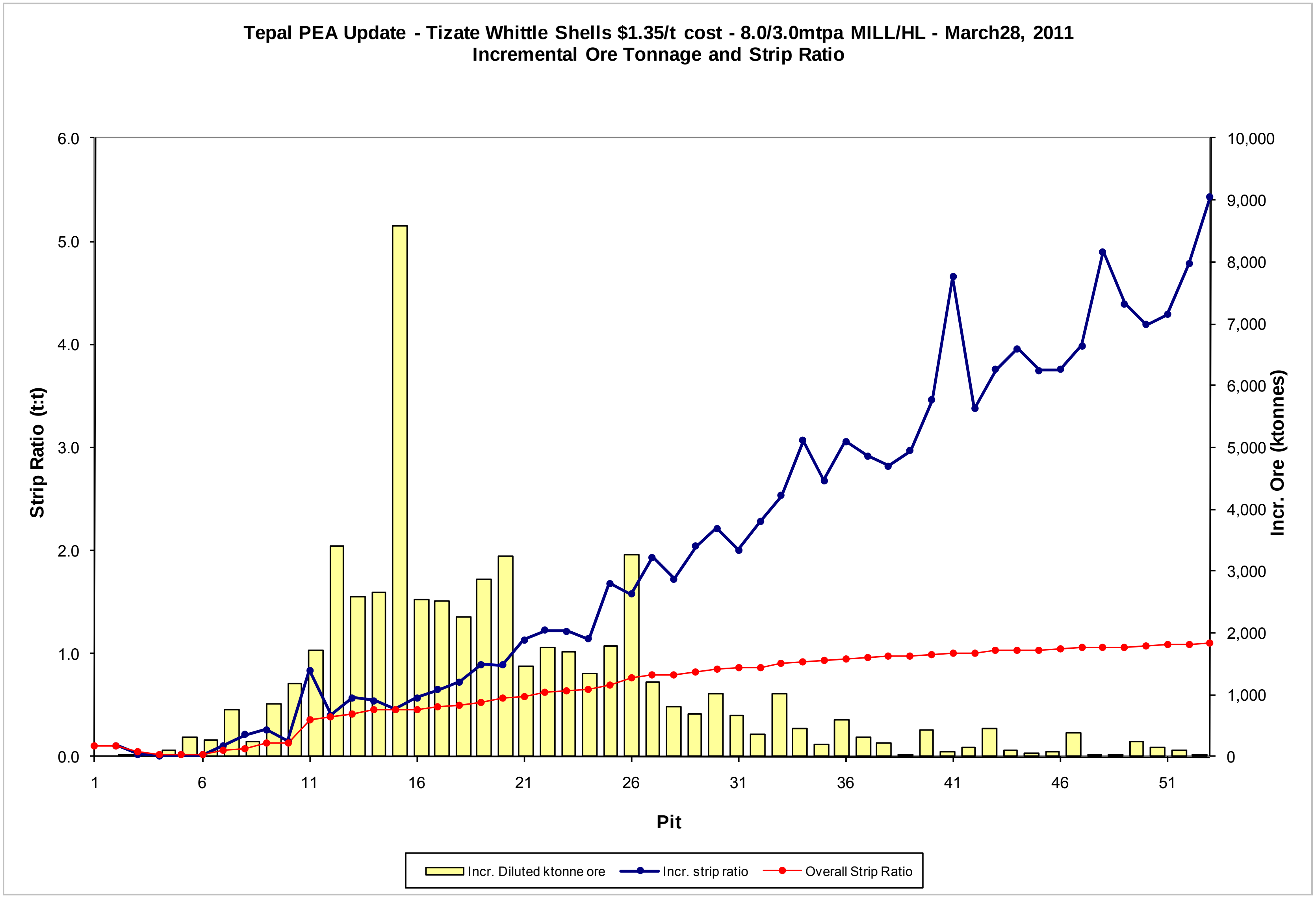

Figure 19.6: **Incremental Whittle™ Tonnage Results - Tizate**

For the Tepal deposits, shells beyond 28 add mineralized rock and waste tonnages to the overall pit but have higher incremental strip ratios with minimal effects on the overall NPV.

To better determine the optimum Whittle™ shell on which to base the pit phasing and scheduling, and to gain a better understanding of the deposit, the shells were analyzed in a preliminary schedule. The schedule assumed a maximum milling capacity of 8.0 Mt/yr for flotation and 3.0 Mt/yr for heap leach. No stockpiles were used in the analysis and no capital costs were added. Both best case (mine out pit 1, the smallest pit, and then mine out each subsequent pit shell from the top down, before starting the next pit shell) and a worst case (mine each bench completely to final limits before starting next bench) scenarios were analyzed. The shells were each scheduled at varying revenue factors (0.3 through to 1.4 of base case) to produce a series of nested pit with the NPV results shown in Figures 19.1 through Figure 19.3.

Based on the analysis of the Whittle™ pit shells and preliminary schedule, Whittle™ pit shell 28 was chosen as the base case shell for further pit phasing and scheduling for the Tepal deposit. A similar analysis was conducted for the Tizate optimization and in this case, pit shell 32 was chosen as the base case shell.

Table 19.5 and Table 19.6 summarizes the tonnages and grades contained within the shell limits (using the incremental cut-off grade of $5.23/t for flotation, and $5.24/t for heap leach material, and a dilution factor of 5%) for both the Tepal and Tizate deposits.

A typical long section (looking west) of the Tepal deposit is shown in Figure 19.7 with existing ground, selected Whittle™ shell, and NSR value block model outlines shown. Figure 19.8 is a typical section through the Tizate deposit (looking North West).

Table 19.5: Resources Extracted in LOM Plan by Classification

Category	Oxide							Sulphide							Total						
	Quantity (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Cont.Au (koz)	Cont. Cu (Mlb)	Cont. Ag (koz)	Quantity (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Cont.Au (koz)	Cont. Cu (Mlb)	Cont. Ag (koz)	Quantity (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Cont. Au (koz)	Cont. Cu (Mlbs)	Cont. Ag (koz)
Tepal																					
Indicated	4.6	0.51	0.24		75	25		41.0	0.46	0.25		602	226		45.6	0.46	0.25		677	251	
Inferred	7.9	0.35	0.20		89	36		38.6	0.36	0.22		442	188		46.5	0.35	0.22		530	223	
Total Tepal	**12.5**	**0.41**	**0.22**		**164**	**61**		**79.6**	**0.41**	**0.24**		**1,043**	**413**		**92.1**	**0.41**	**0.23**		**1,207**	**474**	
Tizate																					
Indicated	0.3	0.30	0.20	2.32	3	1	25	11.3	0.22	0.19	2.12	81	46	771	11.6	0.22	0.19	2.13	84	48	796
Inferred	1.4	0.32	0.21	2.66	15	6	119	39.2	0.20	0.17	2.16	249	150	2717	40.6	0.20	0.17	2.17	263	156	2,837
Total Tizate	**1.7**	**0.32**	**0.21**	**2.59**	**18**	**8**	**145**	**50.5**	**0.20**	**0.18**	**2.15**	**329**	**196**	**3488**	**52.2**	**0.21**	**0.18**	**2.16**	**347**	**204**	**3,633**

Cont. = Contained

Table 19.6: Material by Type

Material	Destination	Tonnage (Mt)
Sulphide Material	Mil	130.2
Oxide Material	Heap Leach	14.3
Waste Rock	WRF	165.3
Total Material		309.9

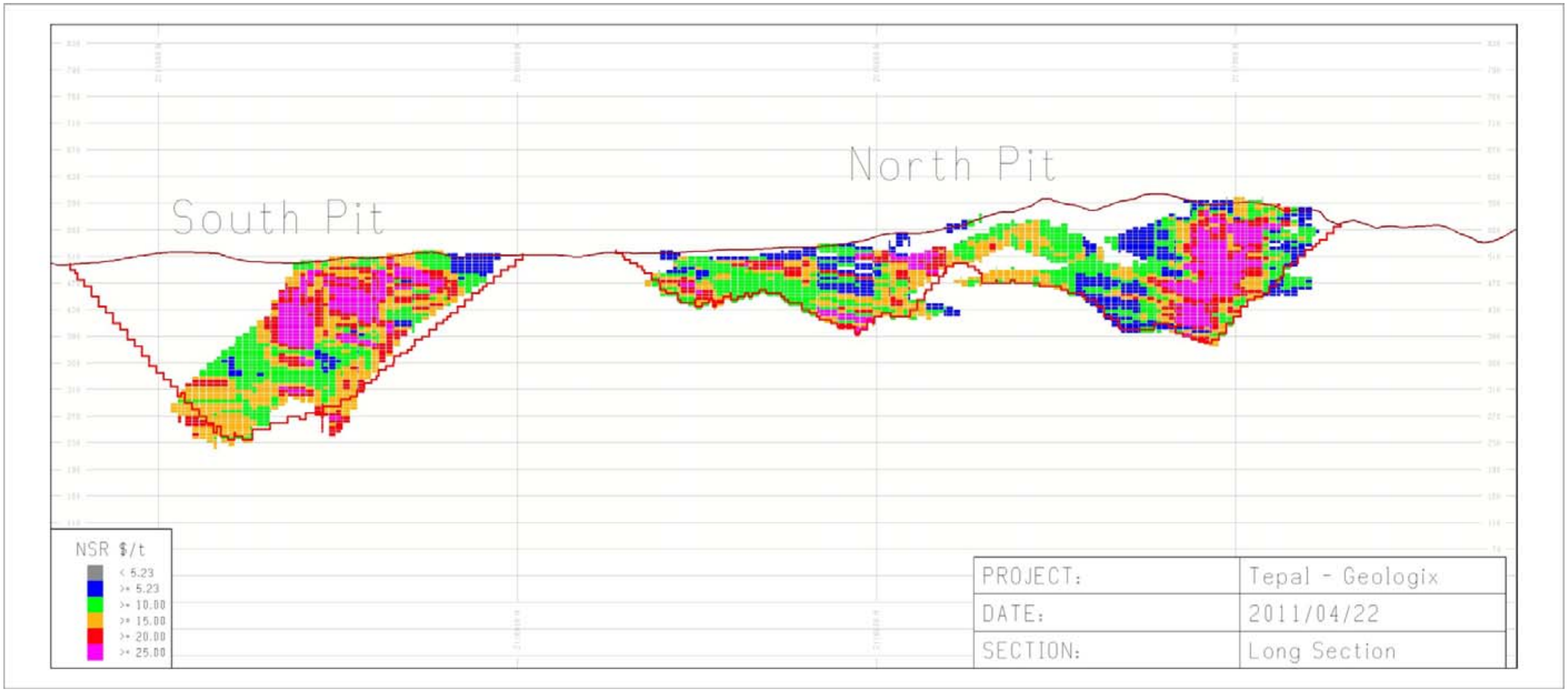

Figure 19.7: Typical Longitudinal Section (looking west) - Tepal

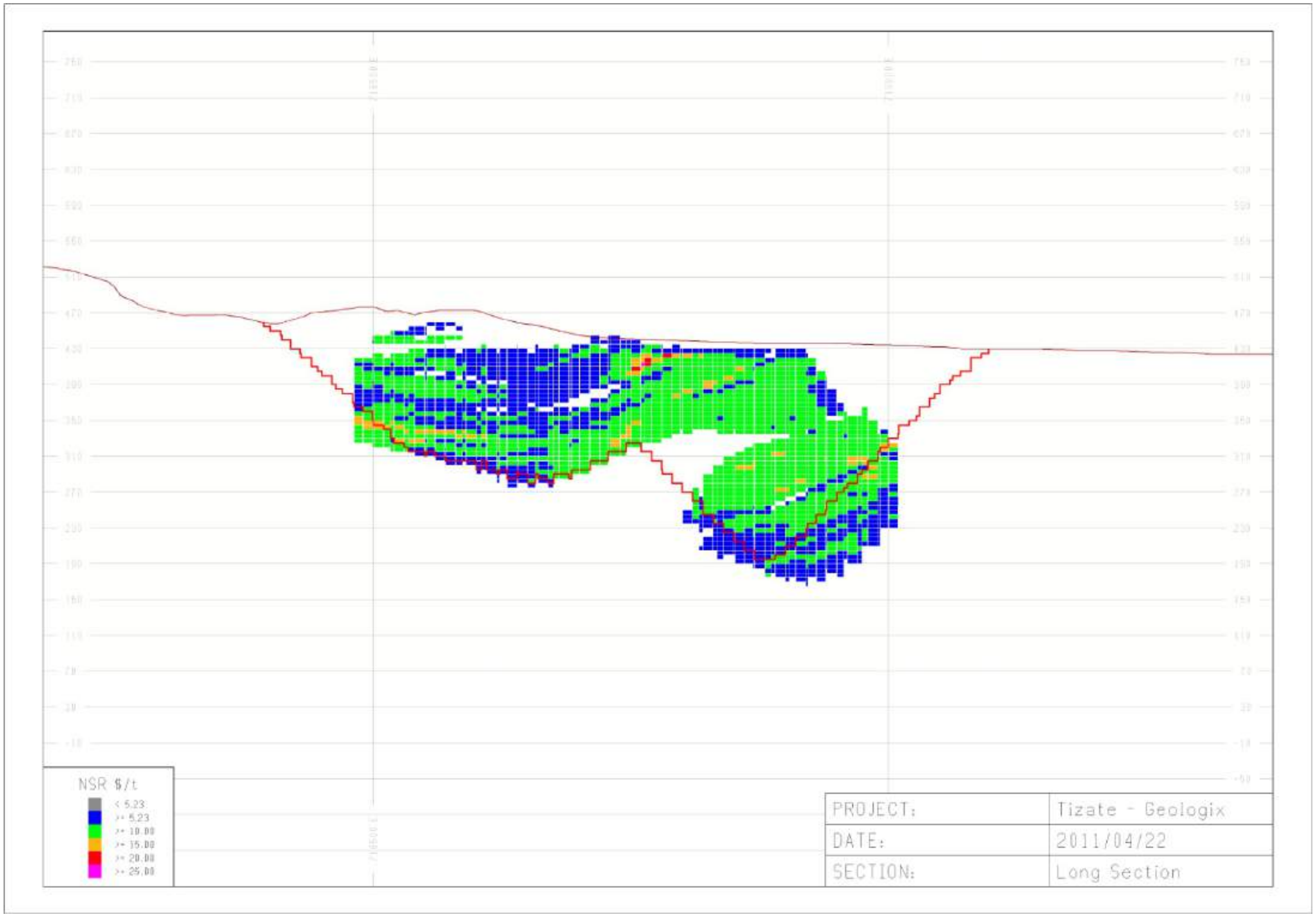

Figure 19.8: Typical Longitudinal Section (looking north west) - Tizate

19.2 Mine Design

Mine planning for the Tepal deposit was conducted using a combination of software, including Mintec Inc. MineSight™, Gemcom GEMS™, and Whittle™. The base 3-D block models, along with subsequent NSR modeling were analyzed using GEMS™. The phase design and production scheduling was undertaken with the use of MineSight™ and Whittle™ software.

Preliminary pit designs for both North and South Pits and Tizate, along with the associated pit phasing, were then based on the Whittle™ shell analysis described in this report. Preliminary waste dumps were then designed to account for the material produced in each mining phase and pit.

Whittle™ pit shell 28 was chosen as the base case pit design for the Tepal deposit, while shell 32 was chosen for the Tizate deposit. Figure 19.9 represents an isometric view of the pit designs for the Tepal base case Whittle™ shell with the Tizate pit design shown in Figure 19.10.

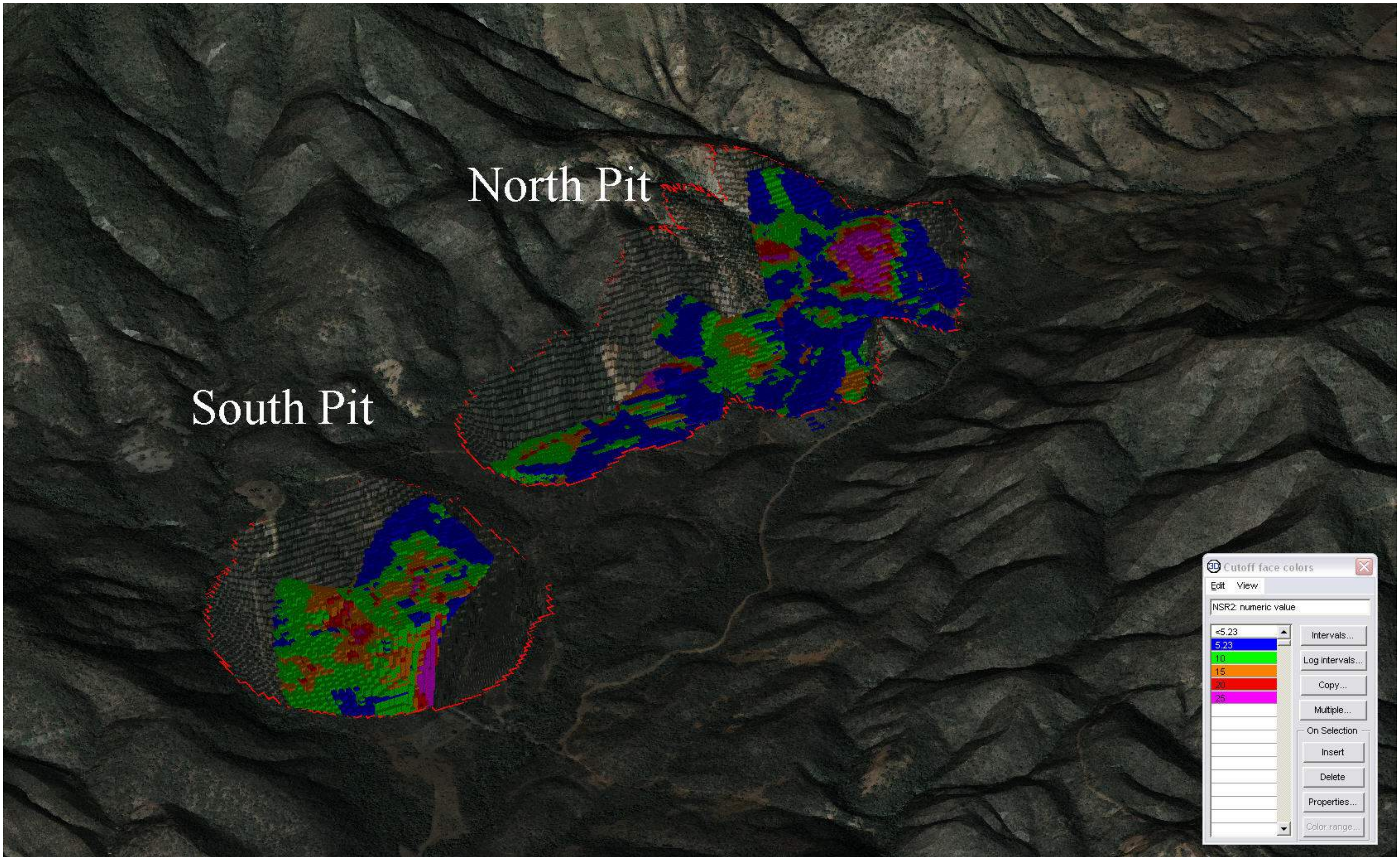

Figure 19.9: Preliminary Pit Designs - Tepal Deposit looking North-West

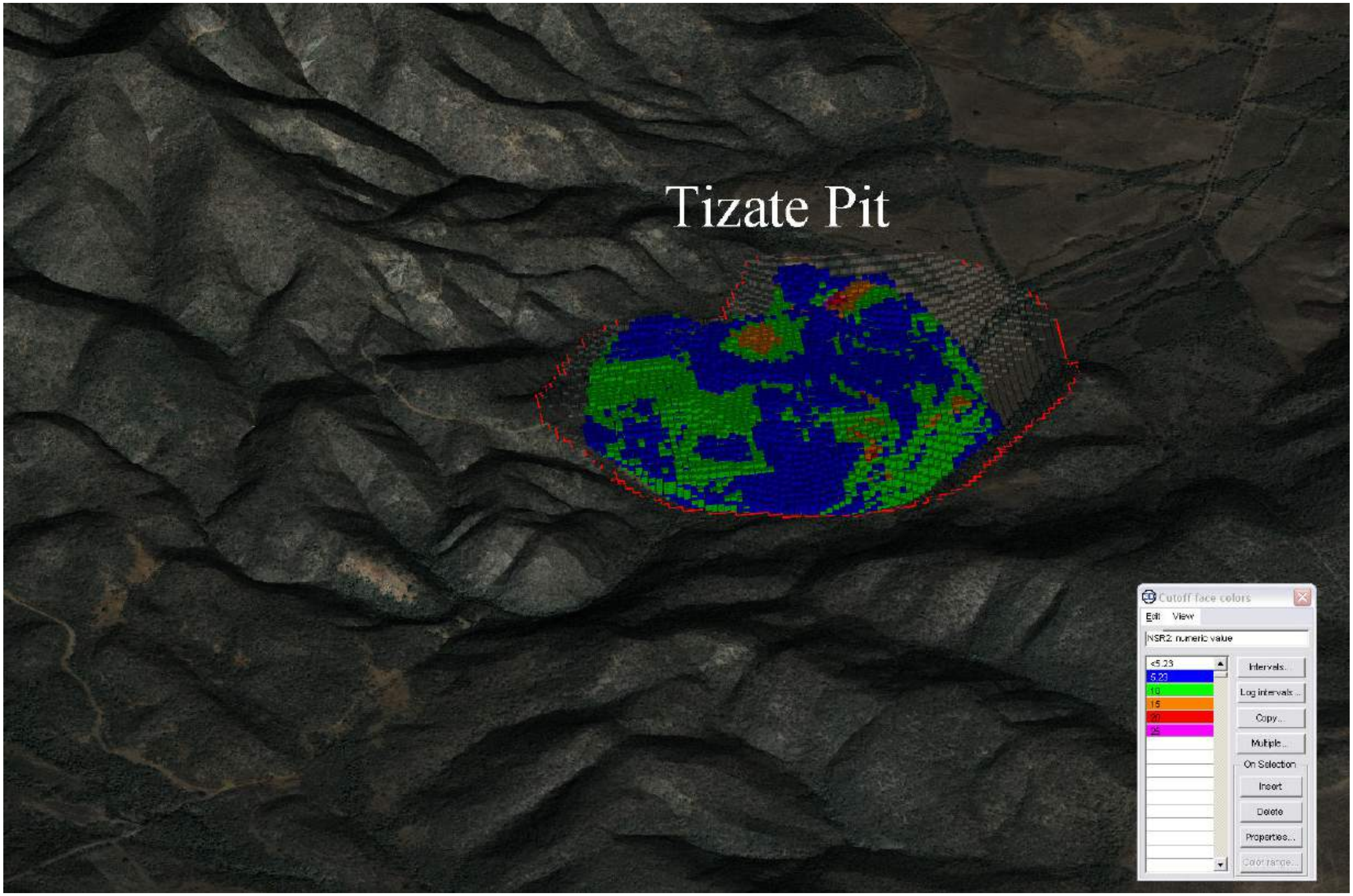

Figure 19.10: Preliminary Pit Designs – Tizate looking North East

Mine Operation

The open pit mining activities for the Tepal pits were assumed to be primarily undertaken by the owner as the basis for this preliminary economic assessment. The unit rate used in the Whittle™ optimization was $1.35 of material mined for pit and dump operations, road maintenance, mine supervision and technical services. The cost estimate was built from first principles and based on experience of similar sized open pit operations. In order to minimize capital requirements, a small contractor truck fleet was assumed for the initial mining of the Tizate pit, and the operating costs adjusted accordingly.

Equipment

The major owner mining equipment requirements are indicated in Table 19.7 and are based on similar sized open pit operations. The proposed plant processing rate of 8.0 Mtpa and 3.0 Mtpa heap leach operation was used, along with deposit and pit geometry constraints, to estimate the mining equipment fleet needed. The fleet has an estimated maximum capacity of 75,000 t/d total material, which will be sufficient for the life-of-mine plan. An additional four contract haul trucks (100 t) will be required during the initial mining of the Tizate deposit.

Table 19.7: Major Open Pit Mining Equipment

Equipment Type	No. of units
Cat D10-class Dozer	2
Cat D9-class Dozer	2
Diesel, 13-cu-yd Front Shovel	2
Cat 992, 14-cu-yd Wheel Loader	1
Cat 988, 8.5-cu-yd Wheel Loader	1
Cat 777, 100-ton Haul Truck	10
Cat 16H-class Grader	2
Cat 14H-class Grader	1
Cat 824H-class Rubber Tire Grader	2
9.88” dia. Rotary, Crawler Drill	2
6.5” dia. Rotary, Crawler Drill	2
3.5” dia. Hydraulic Track Drill	1
16-cu-yd Scraper	1

Unit Operations

The 9.88" diameter drill performs the majority of the production drilling in the mine, with the 6.5" diameter drills primarily used in ore production. The hydraulic drill with a 3.5" diameter bit is to be used for secondary blasting requirements and may be used on the tighter spaced patterns required for pit development blasts. The main loading and haulage fleet consists of Cat 777- 100 t haul trucks, which are loaded primarily with the diesel 13 yd^3 front shovel or the Cat 992, 14 yd^3 wheel loaders, depending on pit conditions.

As pit conditions dictate, the Cat D10 and Cat D9 dozers are used to rip and push material to the excavators, as well as maintaining the waste dumps and heap leach pad.

The additional equipment listed in Table 19.7 will be used to maintain and build access roads, and to meet various site facility requirements, (including coarse mill feed stockpile maintenance, heap leach pad maintenance, and further exploration development).

The work schedule is based on two twelve hour shifts, seven days a week, 365 days per year.

19.2.1 Production Schedule

Mine Sequence/Phasing

The base case Whittle™ pit shell 28 for the Tepal model was divided into a North and South Pit. The pits for both Tepal and Tizate were further divided into a number of phases for the mine plan development to maximize the grade in the early years, reduce the pre-stripping requirements in the early years, provide required oxide production for the heap leach process and keep the process plant at full production capacity per period.

North Pit, South Pit and Tizate were divided so that each pit contained three phases. The pit tonnages and associated grades and metal recoveries of the Tepal and Tizate pits are summarized in Table 19.8.

Table 19.8: Phase Tonnages and Grades

Category	Oxide							Sulphide							Total Resource							Waste	
	Quantity (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Contained Au (koz)	Contained Cu (Mlb)	Contained Ag (koz)	Quantity (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Contained Au (koz)	Contained Cu (Mlb)	Contained Ag (koz)	Quantity (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Contained Au (koz)	Contained Cu (Mlb)	Contained Ag (koz)	Quantity (Mt)	Strip Ratio
Tepal				-			-				-			-				-			-		
North Pit	9.5	0.41	0.22	-	124	46	-	46.3	0.37	0.24	-	547	249	-	55.8	0.37	0.24	-	671	295	-	45	0.8
South Pit	3.1	0.41	0.22	-	40	15	-	33.4	0.46	0.22	-	496	165	-	36.4	0.46	0.22	-	536	179	-	74	2.0
Total Tepal	**12.5**	**0.41**	**0.22**	**-**	**164**	**61**	**-**	**79.6**	**0.41**	**0.24**	**-**	**1,043**	**413**	**-**	**92.1**	**0.41**	**0.23**	**-**	**1,207**	**474**	**-**	**120**	**1.3**
Total Tizate	**1.7**	**0.32**	**0.21**	**2.59**	**18**	**8**	**145**	**50.5**	**0.20**	**0.18**	**2.15**	**329**	**196**	**3488**	**52.2**	**0.21**	**0.18**	**2.16**	**347**	**204**	**3,633**	**46**	**0.9**

Figure 19.11 further summarizes the phase designs, with the phase layout shown in isometric view for the Tepal North and South pits, while Figure 19.11 illustrates the Tizate phase designs and ultimate pit outlines.

The Tepal pit phases were based on the Whittle™ pit shells 04 and 14. The North pit waste will be placed into a waste rock facility ("WRF") to the north of the final pit limits, while the majority of the South pit waste will be placed in a WRF to the west of the pit. All oxide material will be placed on the heap leach pad to the east of the pit, while sulphide material will be hauled to the primary crusher to the south east of the pit.

The Tizate pit phases were based on pit shells 14 and 20 with all waste generated from Tizate to be placed in a WRF to the south east of the pit. All oxide will be placed on the heap leach pad near the Tepal pits, while sulphide material will be hauled to the primary crusher.

Figure 19.12 provides an overall site plan of the Tepal project, outlining the proposed Tepal and Tizate pits, various WRF's, Heap Leach pad, process facilities, and TMF.

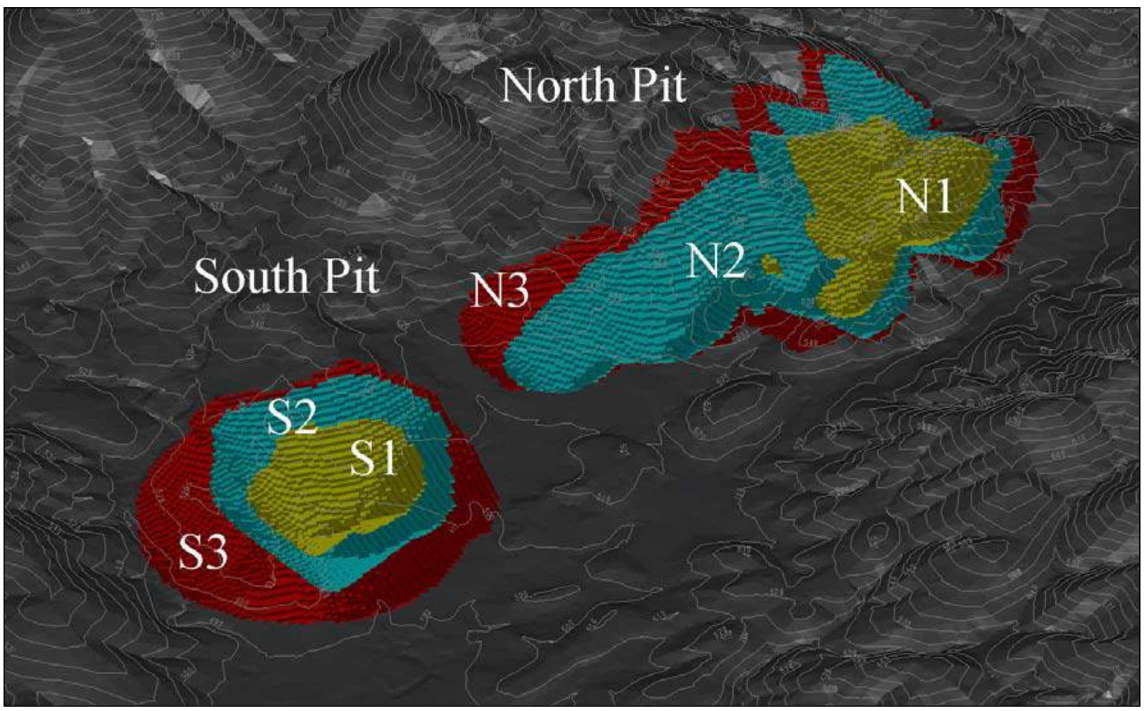

Figure 19.11: **Tepal Phase Design in Isometric View (looking NW)**

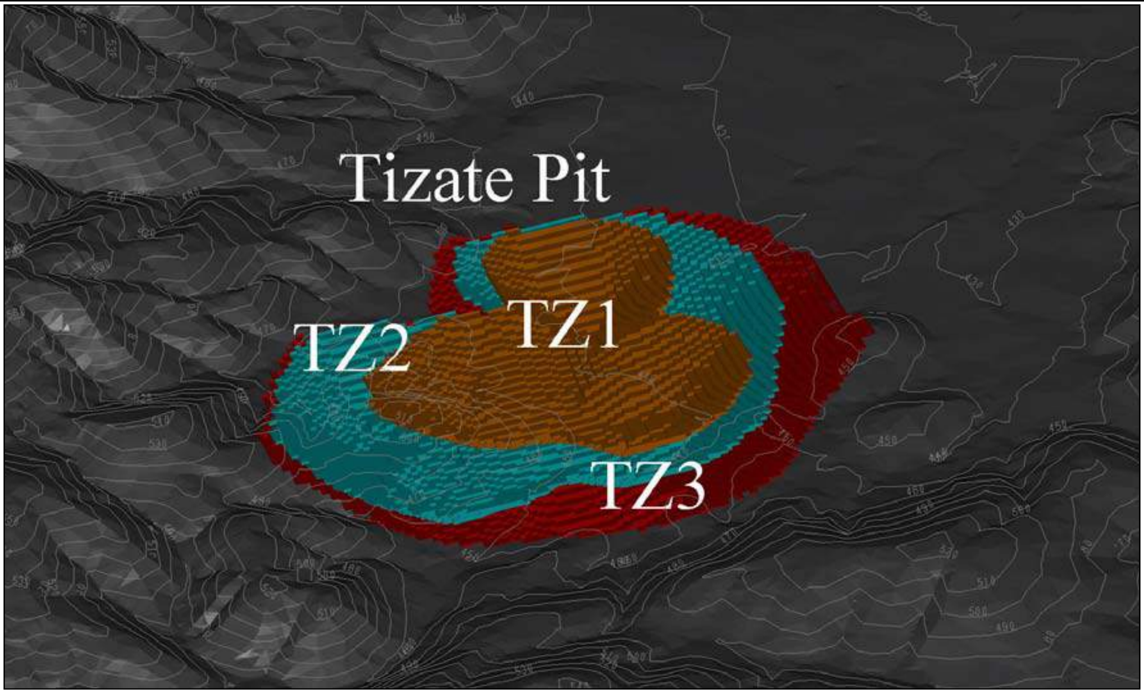

Figure 19.12: **Tizate Phase Design in Isometric View (looking NE)**

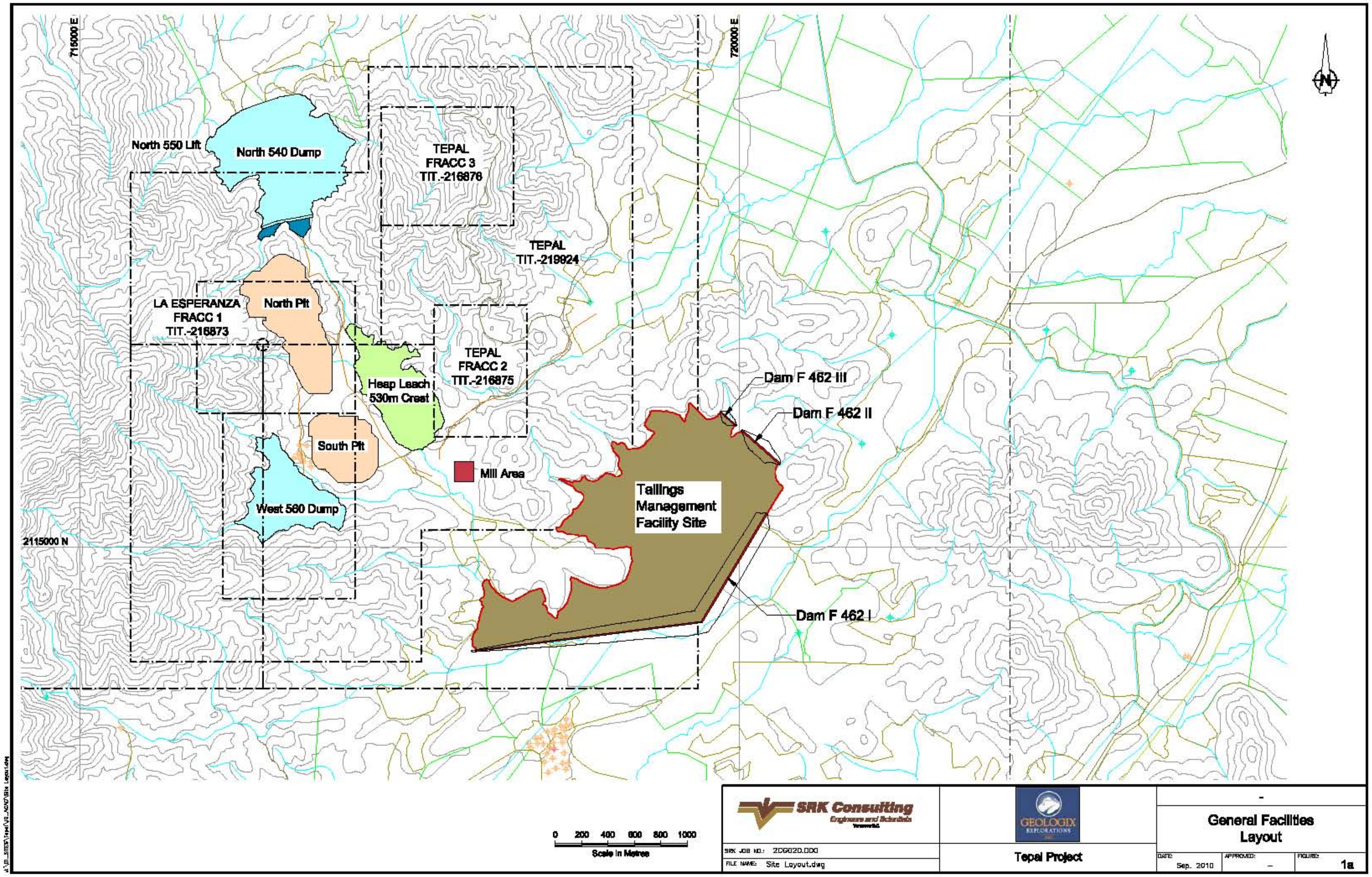

Figure 19.13: Tepal Overall Site Plan

Mine Production Schedule

The production schedule for the Tepal and Tizate deposit models was developed with the aid of Whittle™ and MineSight™ software, and incorporated the various phases mentioned above.

The proposed plant processing rate of 8.0 Mtpa along with the proposed 3.0 Mtpa heap leach was used, along with deposit and pit geometry constraints, to estimate the mining equipment fleet needed. The fleet has an estimated maximum capacity of 75,000 tpd total material, which will be sufficient for the life-of-mine plan. The plant throughput was planned at 8.0 Mtpa of sulphide material, with an additional 3.0 Mtpa Heap Leach capacity. Due to limited pre-stripping requirements, with the oxide material near surface, Year 1 represents the commencement of heap leach processing. The maximum planned amount of total material to be moved is 73,000 t/d. The average total mining rate was planned to be 47,000 t/d from both the Tepal and Tizate deposits. Indicated and inferred resources were used in the LOM plan, with inferred resources representing 60% of the material mined and processed. The resources calculated included an external dilution of 5%.

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 will be upgraded to a higher resource category.

Table 19.9 is a summary of total material movement by year for the mine production schedule

Table 19.9: Proposed Production Schedule

Section	Item	Unit	Year 1 - 19 Total	Y E A R																		
				1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
MATERIAL SCHEDULE																						
Mining Total	Operating Days	*days*	**6,617**	365	365	366	365	365	365	366	365	365	365	365	365	365	365	365	365	365	365	45
Mining Total	Waste	Mt	**165.3**	1.91	2.53	9.54	9.36	11.82	16.26	12.90	16.00	14.00	14.61	15.94	4.92	14.30	6.83	4.89	3.39	3.66	2.50	-
	Oxide Ore	Mt	**14.3**	0.75	3.00	3.00	3.00	2.18	2.35	-	-	-	-	-	-	-	-	-	-	-	-	-
	Sulphide Ore	Mt	**130.2**	-	0.99	7.97	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	1.28
	Total Mining	**Mt**	**309.9**	**2.7**	**6.5**	**20.5**	**20.4**	**22.0**	**26.6**	**20.9**	**24.0**	**22.0**	**22.6**	**23.9**	**12.9**	**22.3**	**14.8**	**12.9**	**11.4**	**11.7**	**10.5**	**1.3**
	Strip Ratio	waste:ore	**1.14**	2.5	0.6	0.9	0.9	1.2	1.6	1.6	2.0	1.8	1.8	2.0	0.6	1.8	0.9	0.6	0.4	0.5	0.3	-
	Daily Production	t/day	**46,827**	7,279	17,863	56,030	55,767	60,274	72,893	57,112	65,734	60,274	61,948	65,586	35,384	61,093	40,625	35,307	31,195	31,953	28,773	28,489
North and South Pit	Waste	Mt	**119.7**	1.9	2.5	9.5	9.4	11.8	6.2	12.9	16.0	14.0	14.6	15.9	4.9	0.0						
Flotation	Flotation Circuit Feed	Mt	**79.7**		1.0	8.0	8.0	8.0	4.7	8.0	8.0	8.0	8.0	8.0	8.0	2.1						
	Cu head grade	%Cu	**0.23**		0.46	0.34	0.25	0.24	0.22	0.22	0.21	0.21	0.20	0.20	0.22	0.25						
	Au head grade	g/t Au	**0.41**		0.76	0.53	0.41	0.35	0.36	0.38	0.39	0.38	0.35	0.35	0.43	0.70						
	Ag head grade	g/t Ag	**-**																			
Heap Leach	HL Feed	Mt	**12.55**	0.7	3.0	3.0	3.0	2.2	0.6													
	Cu head grade	%Cu	**0.22**	0.28	0.27	0.22	0.20	0.17	0.18													
	Au head grade	g/t Au	**0.41**	0.72	0.49	0.36	0.40	0.29	0.28													
	Ag head grade	g/t Ag	**-**																			
Tizate Pit	Waste	Mt	**45.6**						10.1							14.3	6.8	4.9	3.4	3.7	2.5	
Flotation	Flotation Circuit Feed	Mt	**50.5**						3.3							5.9	8.0	8.0	8.0	8.0	8.0	1.3
	Cu head grade	%Cu	**0.18**						0.19							0.18	0.18	0.17	0.17	0.17	0.18	0.18
	Au head grade	g/t Au	**0.20**						0.23							0.19	0.22	0.21	0.21	0.21	0.18	0.11
	Ag head grade	g/t Ag	**2.15**						2.21							1.83	1.97	1.87	1.82	2.23	2.79	3.75
Heap Leach	HL Feed	Mt	**1.73**						1.7													
	Cu head grade	%Cu	**0.20**						0.20													
	Au head grade	g/t Au	**0.32**						0.32													
	Ag head grade	g/t Ag	**2.60**						2.60													

The Tepal and Tizate deposits will produce a total of 14.3 Mt of oxide heap leach feed, 130.2 Mt of mill sulphide feed and 165.3 Mt of waste (1.14:1 overall strip ratio) over a 19 year mine operating life. The current life of mine ("LOM") plan focuses on achieving consistent heap leach and mill feed production rates, mining of higher grade material early in schedule, and balancing grade and strip ratios. No blending of stockpiled material has been included in this preliminary schedule

Figure 19.14 and 19.15 summarize pit tonnages, strip ratios and grades by period.

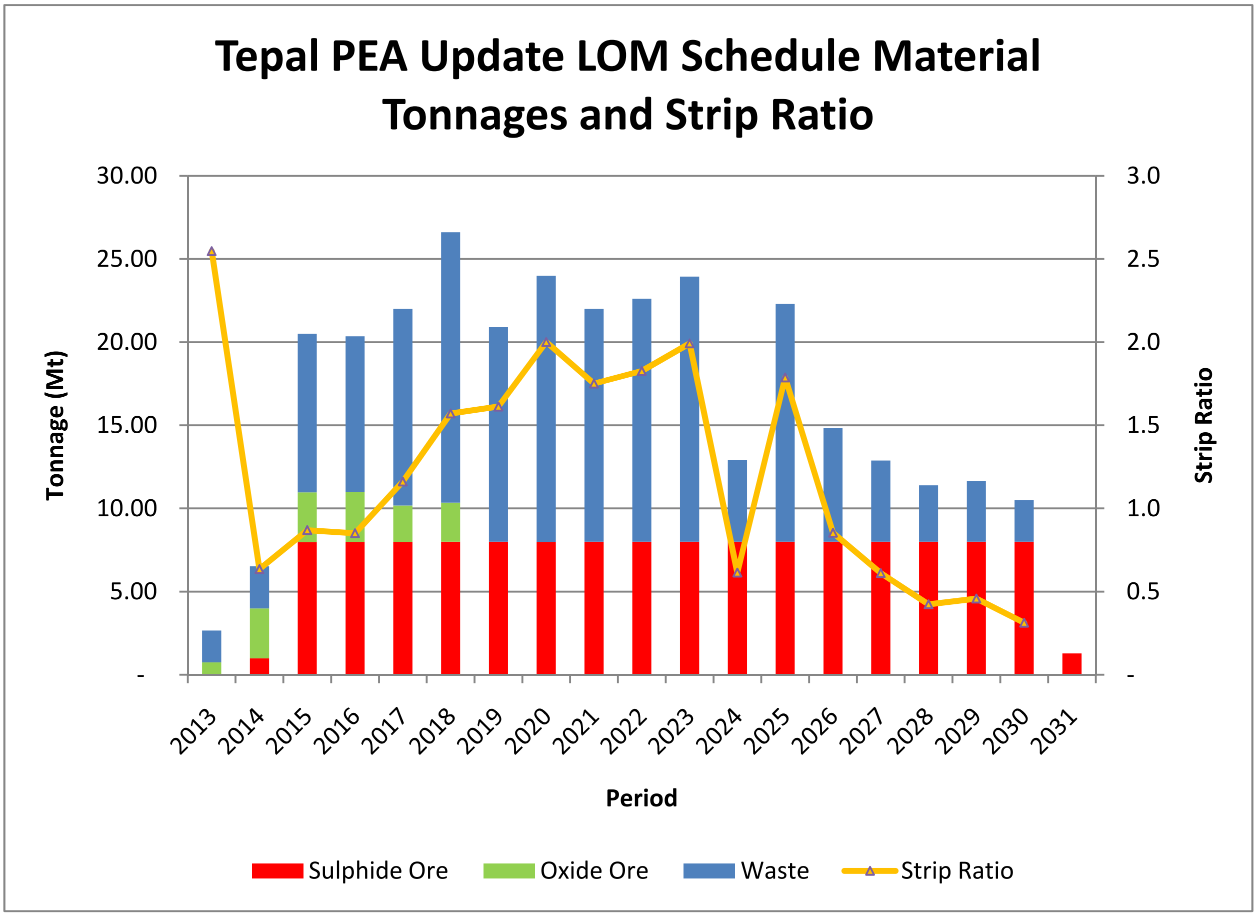

Figure 19.14: Material Tonnages and Strip Ratio

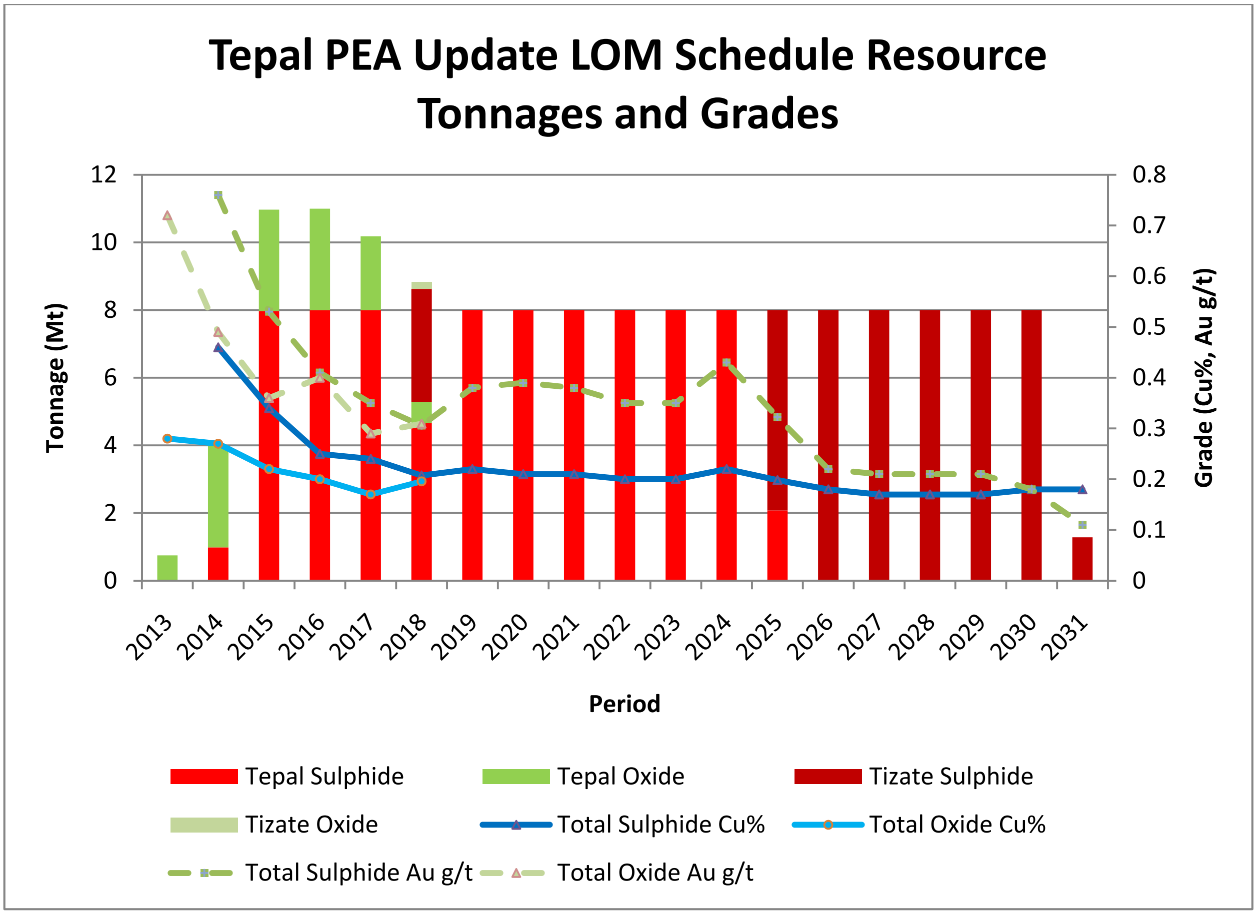

Figure 19.15: Period Resource Tonnages and Grade

To further illustrate the progression of mining of the Tepal and Tizate deposit Figures 19.16 through to 19.24 provide a snapshot of the pit configurations at the end of various periods.

The Tepal and Tizate deposits provide maximum returns when the various pit phases are mined concurrently. This also allows for the Oxide material to be delivered to the Heap Leach pad during the first half of the mine life at the targeted 3.0 Mtpa. The North and South pits, as well as the Tizate deposit, are mined out in a series of push-backs. The mining fleet was selected based on the need for this flexibility and mobility.

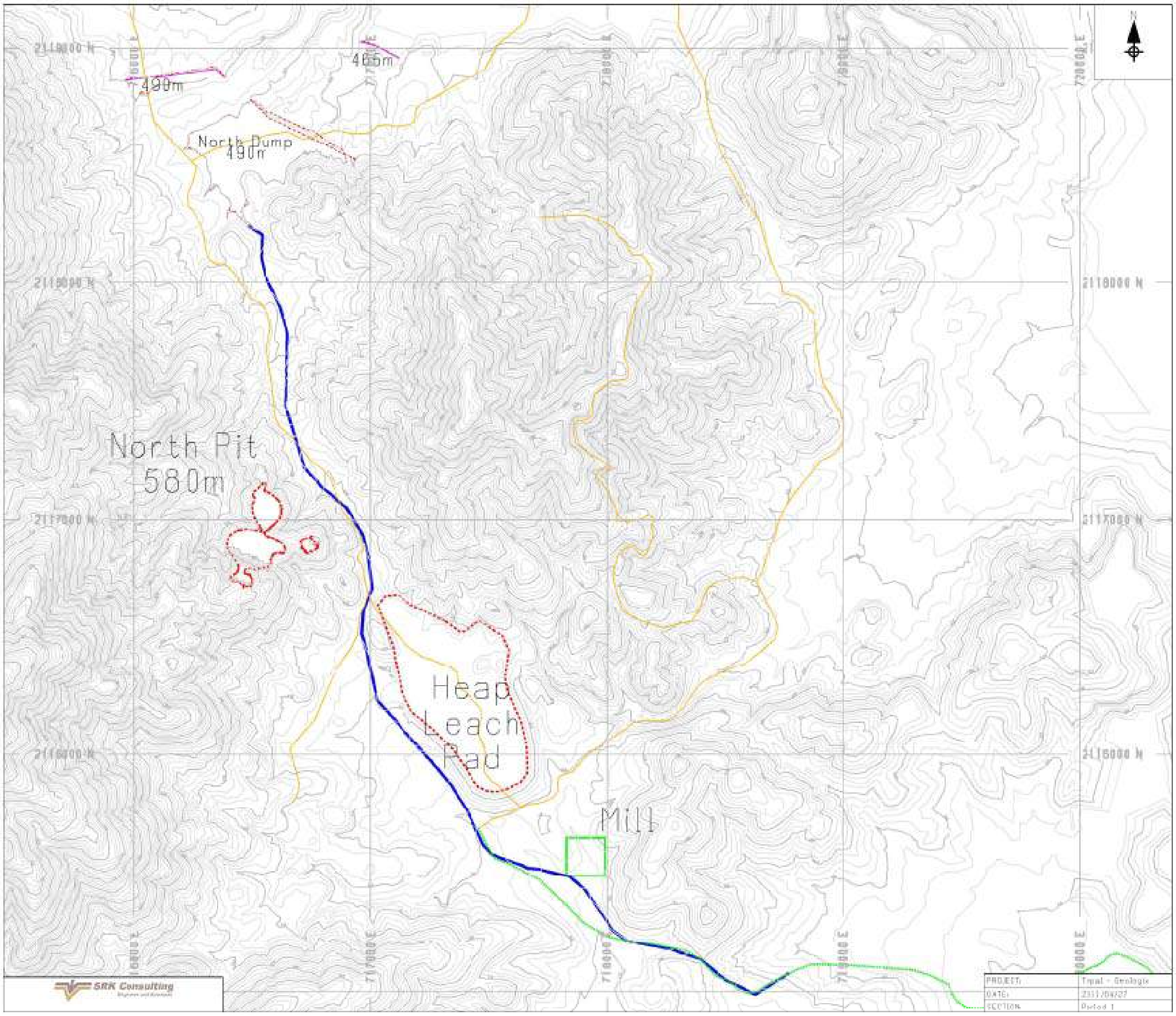

Figure 19.16: End of Year 1

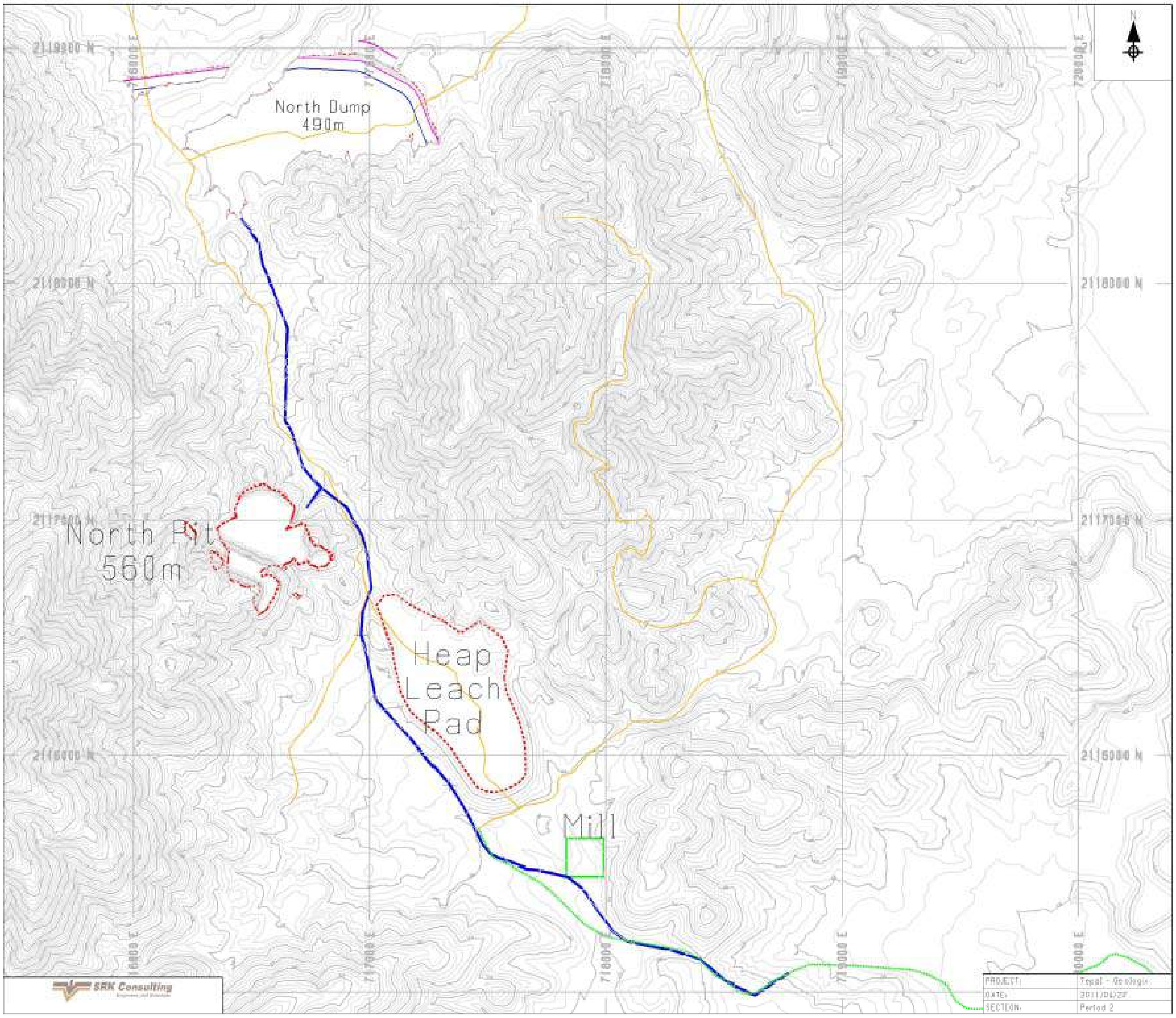

Figure 19.17: End of Year 2

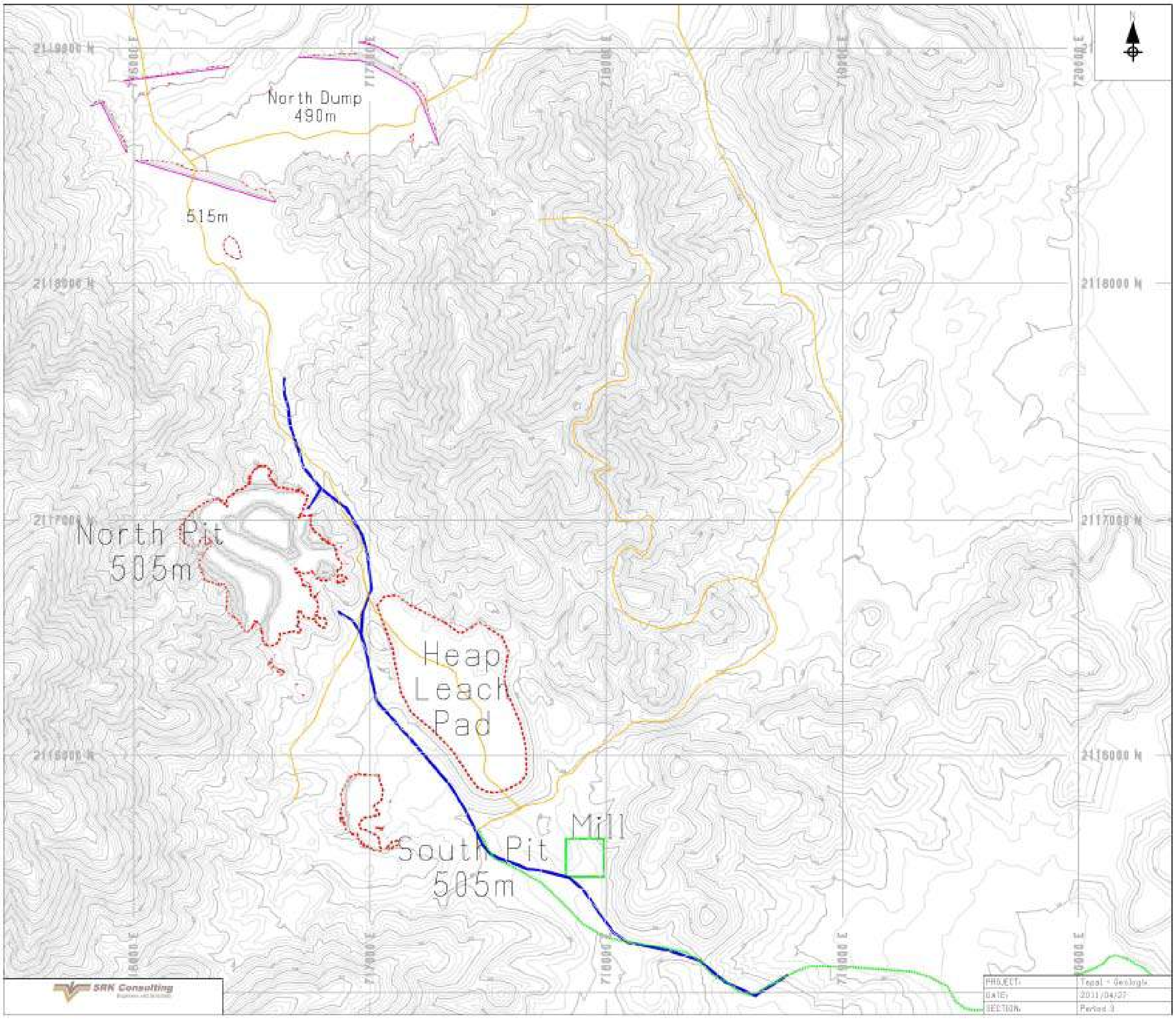

Figure 19.18: End of Year 3

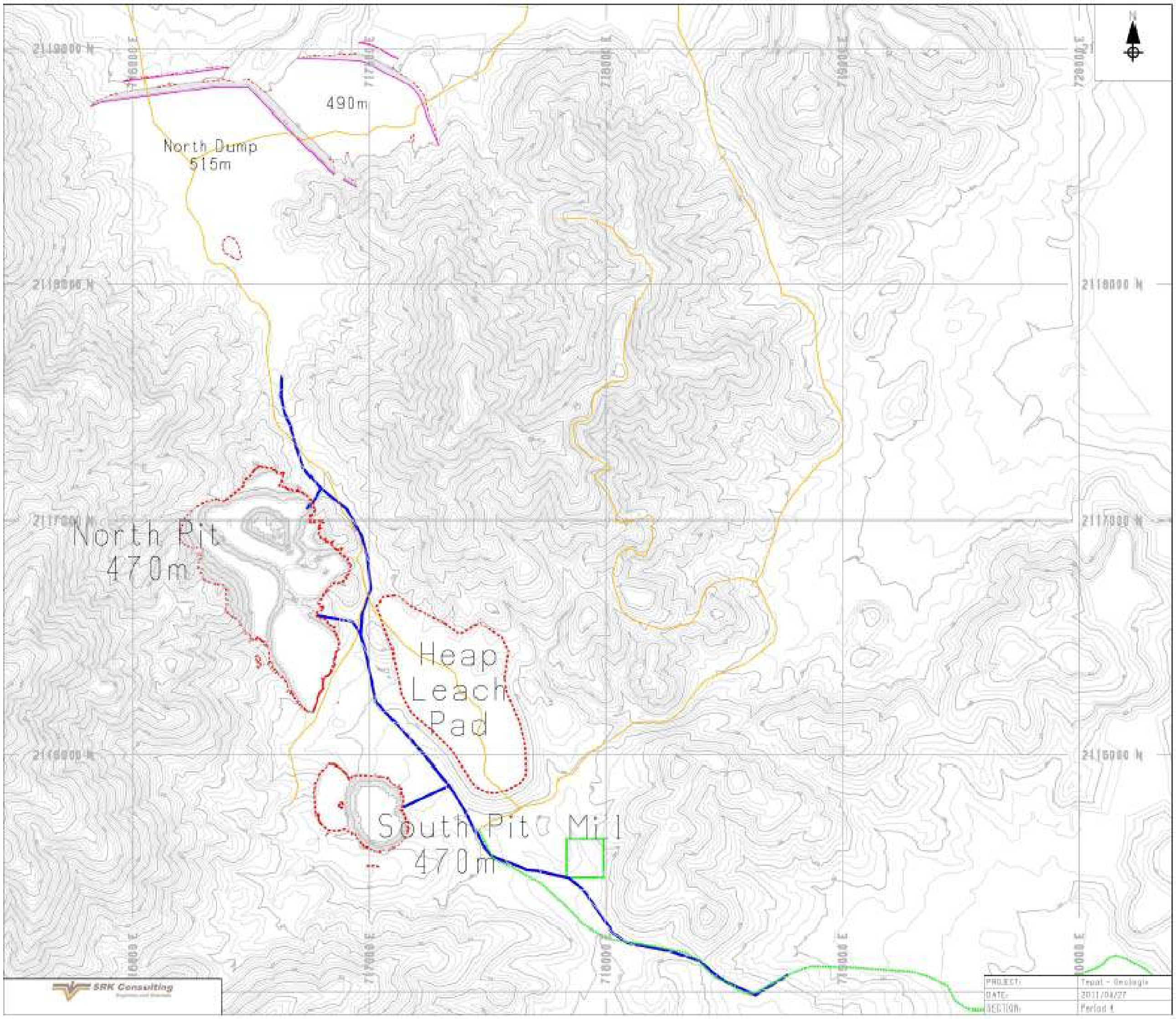

Figure 19.19: End of Year 4

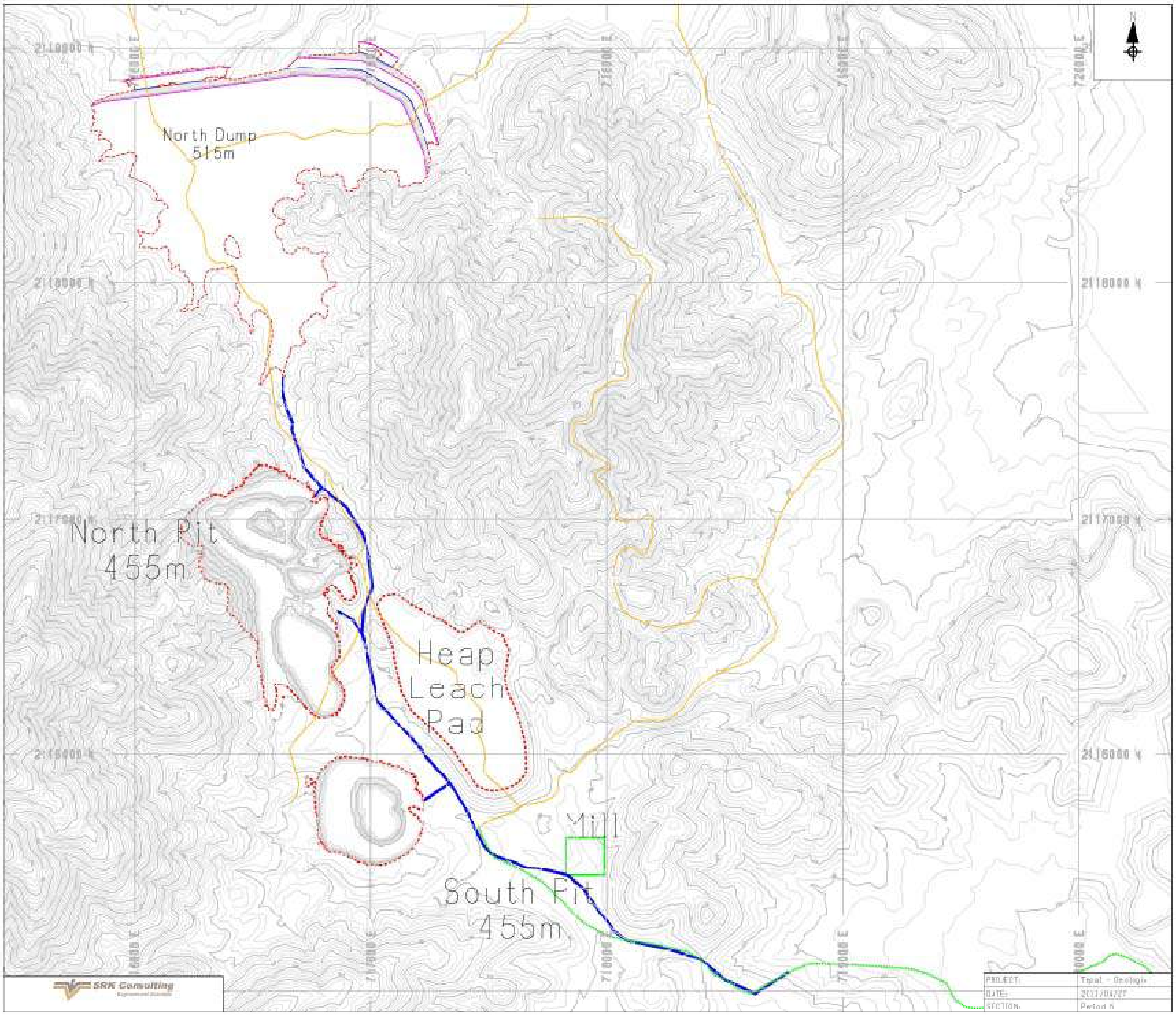

Figure 19.20: End of Year 5

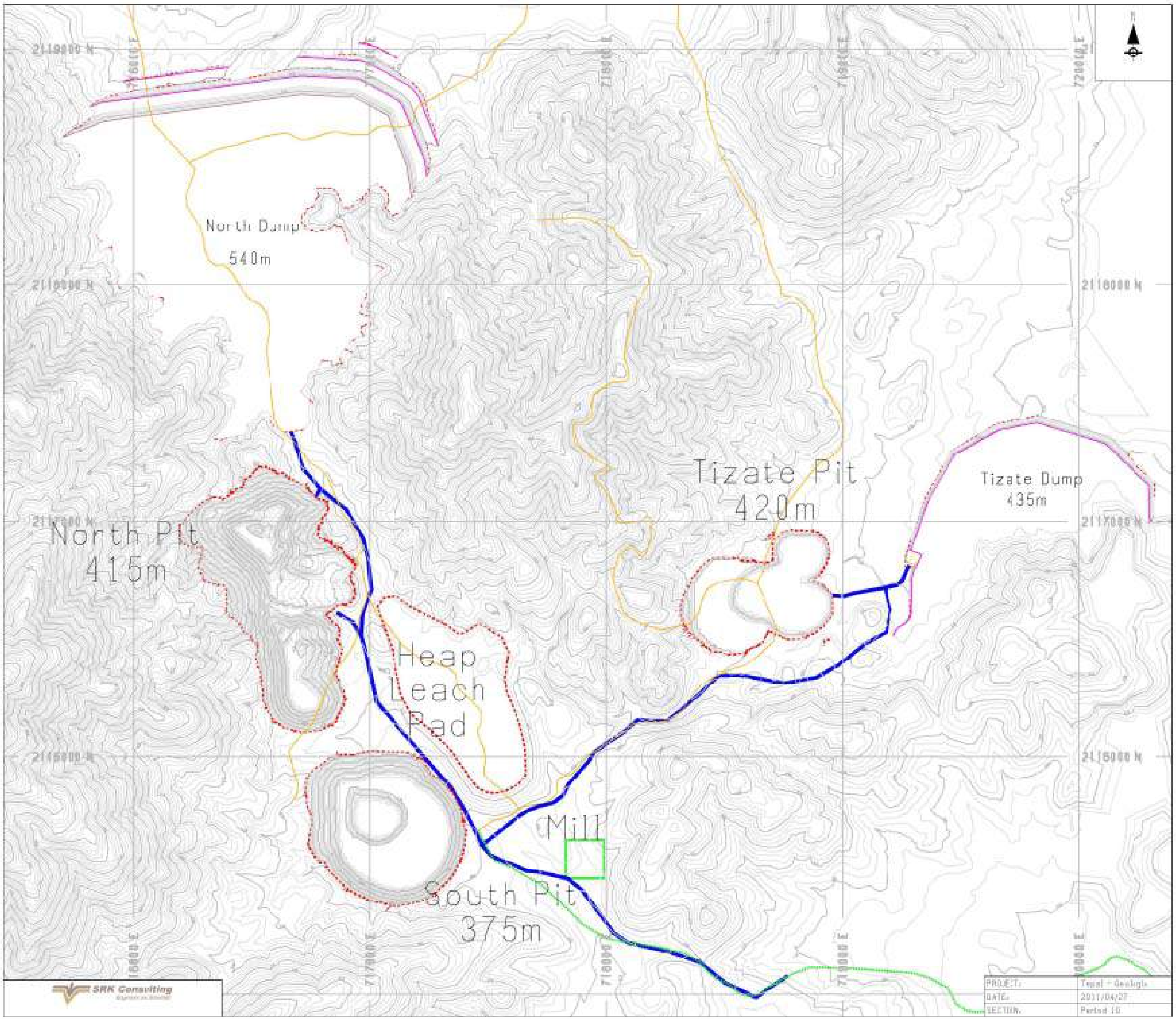

Figure 19.21: End of Year 6

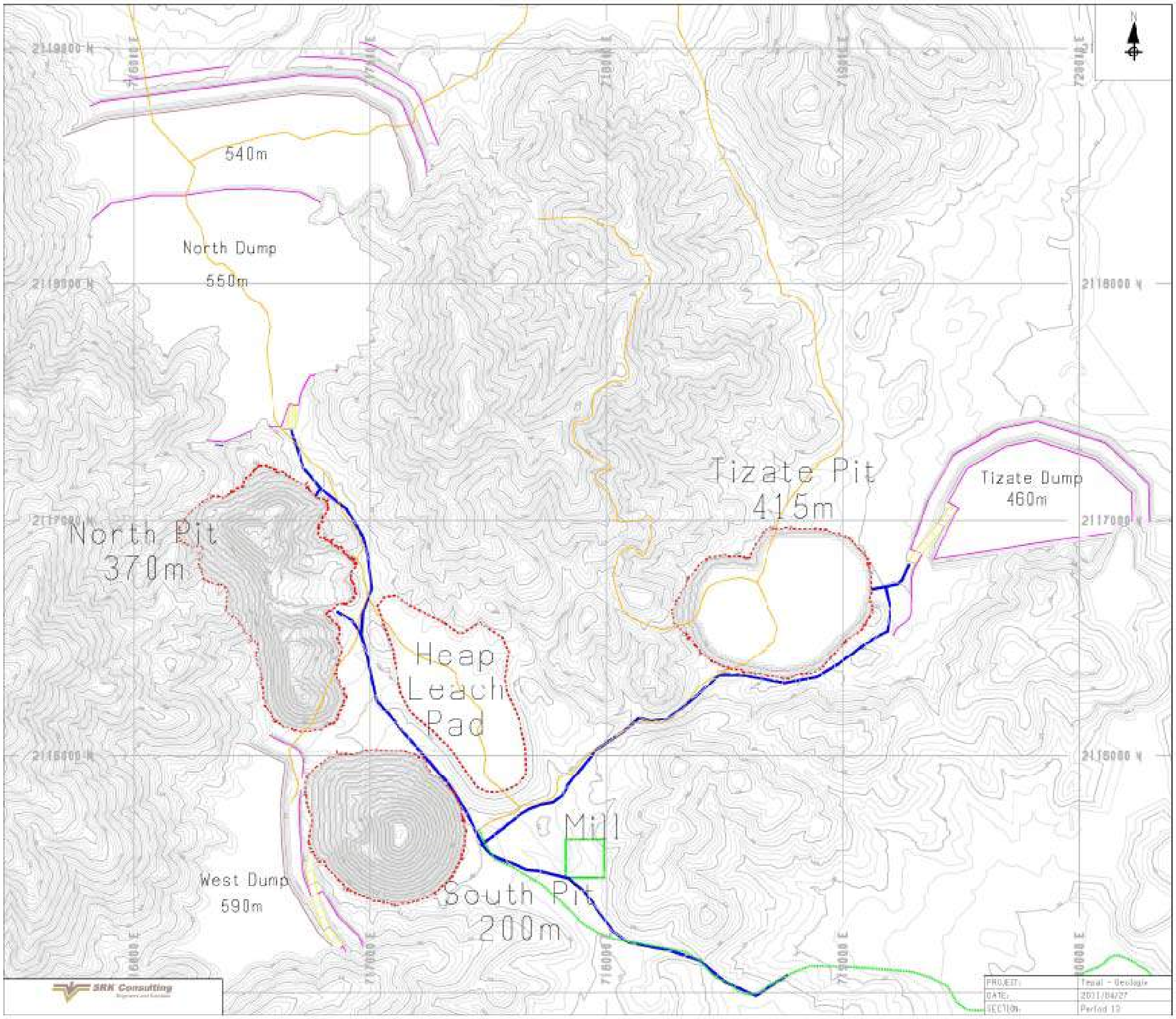

Figure 19.22: End of Year 7

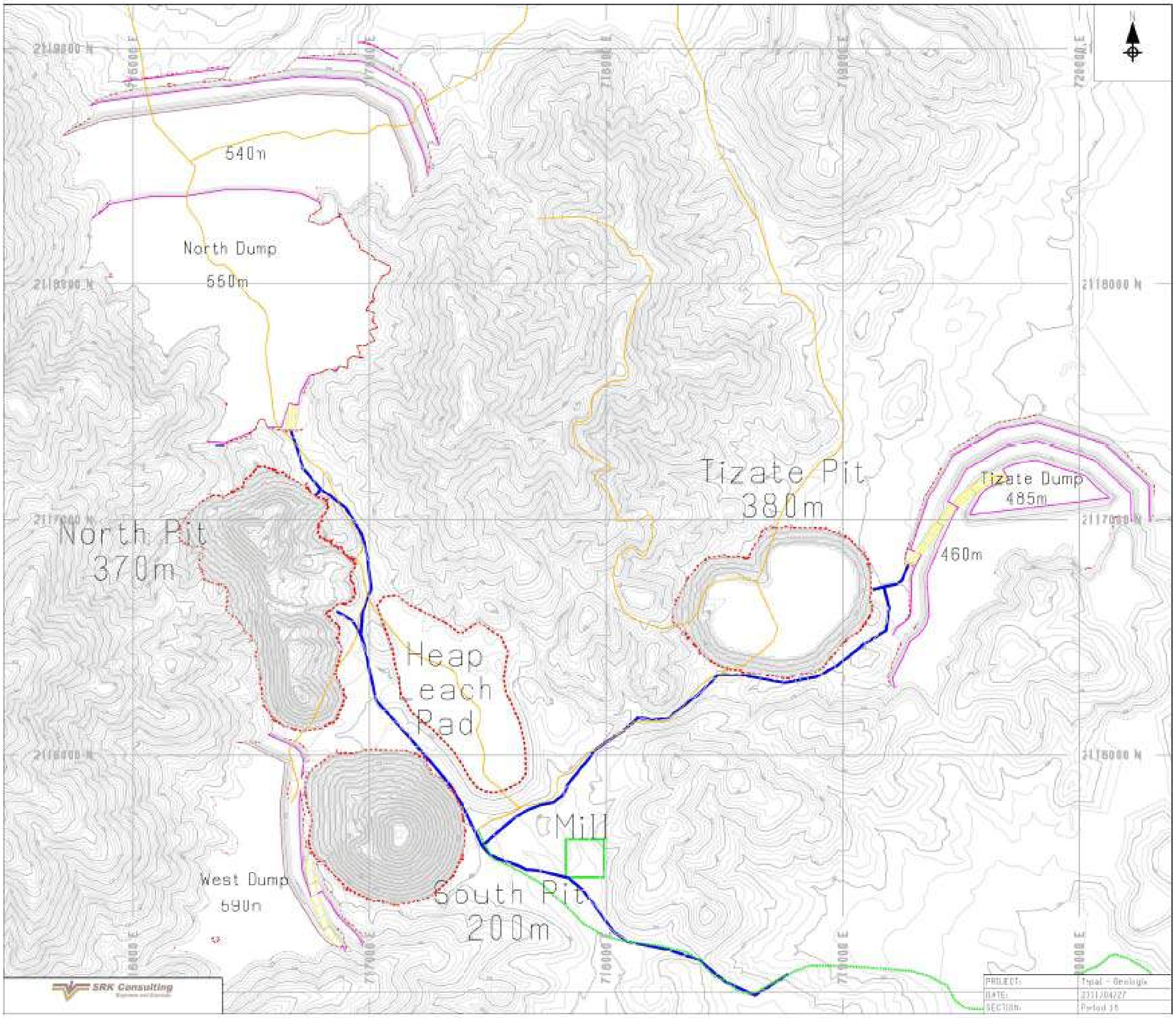

Figure 19.23: End of Year 8

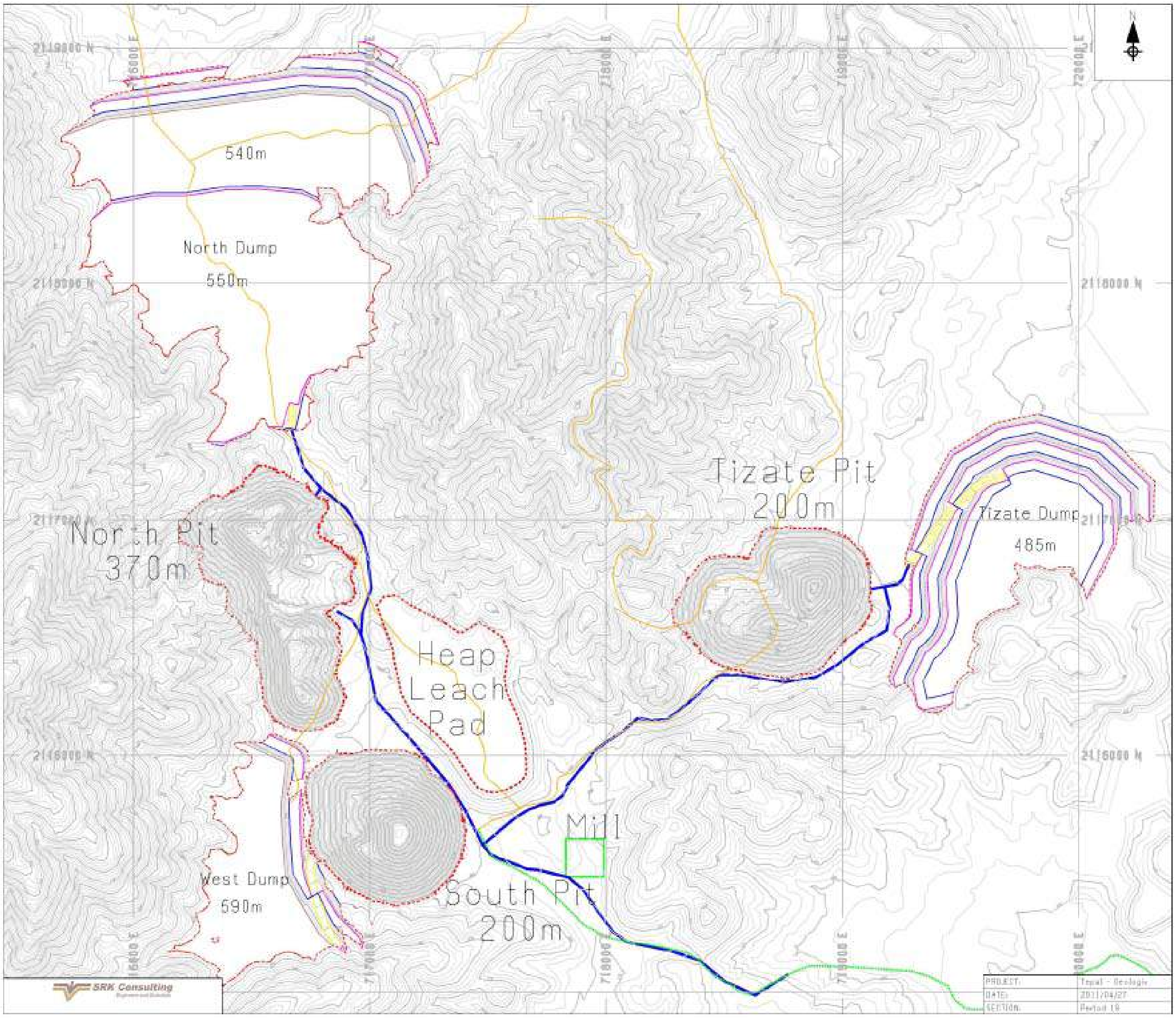

Figure 19.24: Final Pit Configuration

Pit Development

- Year 1: Development of Tepal deposit commences with mining of both North and South pits for a total of 1.9 Mt of waste. A total of 0.7Mt of oxide is mined and delivered to heap leach pad. No sulphide ore is mined in the period.
- Year 2: Both oxide and sulphide ore are mined from the Tepal deposits. The 3.0 Mtpa target of oxide is attained while sulphide production is 1.0 Mt. Oxide gold head grade is 0.49 g/t Au, while sulphide gold grade is 0.76 g/t Au with copper head grades of 0.27 Cu% and 0.46 Cu% respectively. A total of 2.5 Mt of waste rock is produced at a mined strip ratio of 0.6:1 (waste:ore).

- Year 3: Sulphide production reaches target of 8.0 mtpa and oxide remains at targeted 3.0 Mtpa. Total waste mined from the North and South pits is 9.5 Mt for a strip ratio of 0.9:1.
- Years 4-5: Mining continues in North and South pits with oxide material nearing depletion and 5.2 Mt sent to heap leach. Sulphide produced at 8.0 Mtpa target. Stripping of push backs increases waste mined to 21.2 Mt at a strip ratio of 1:1. Production rates reach 60,000 t/d total material.
- Year 6: Last remaining amount (2.4 Mt) of oxide produced from North and South pits and mining commences in Tizate in order to maintain heap leach feed. Sulphide production maintained at 8.0 Mtpa. The mining rate reaches maximum of 73,000 t/day.
- Years 7-12: Mining of North and South pits continue with Tizate pit idle over this time period. Only sulphide material produced at steady state of 8.0 Mtpa. Waste mining averages 13,000 t/d at an average strip ratio of 1.6. Average grades are 0.21% Cu and 0.38 g/t Au.
- Year 13: Mining of North and South pits is completed and mining in Tizate recommences.
- Years 14-18: Tizate pit feeds sulphide ore at 8.0 Mtpa at average grades of 0.17% Cu, 0.21 g/t Au and 2.14 g/t Ag. Total material mined averages 34,000 t/d at an average strip ratio of 0.5:1.
- Year 19: Mining completed in Tizate with final 1.3 Mt of sulphide fed to the mill.

19.3 Waste Management Facilities

19.3.1 Waste Rock Facilities (“WRF”)

The waste rock facilities Are planned to be located adjacent to the final pit limits for both the Tepal and Tizate deposits. A North and West WRF have been designed for the Tepal deposits, along with a dump to the south east of the Tizate pit. Due to the pit and deposit geometry along with the LOM schedule and the benefit of using the mined out pits for tailings disposal, the potential for backfilling into previously mined out areas is limited and has not been utilized in this study.

The West WRF and Tizate WRF will be built in a series of lifts in a “bottom-up” approach in order to maximize stability. The WRFs will be constructed by placing material at its natural angle of repose (approximately 1.5H:1V) with safety berms spaced at regular intervals (25 to 50m lifts) to allow for a final reclaimed slopes of 3:1.

The North WRF will take advantage of the existing natural topography to the north of the ultimate pit limits and will be built in two lifts (540 m elevation and 550 m elevation). The dump will be advanced to the north at the 540 m elevation with standard end dumping. Two 5 metre lifts will then be added once the ultimate limits are reached.

The North WRF is designed to contain 108 Mt of waste, the West WRF has a design capacity of 14 Mt, while the Tizate WRF is designed at 52 Mt.

Tailings Management Facility ("TMF")

Several options were researched for the location of the TMF. (See Figure 19.25) Costs and attributes of each option are documented in Appendix 2. .

Required Tailings Capacity

Mining will be from three open pits (North, South and Tizate). Oxide ore will be heap leached on site and sulphide ores will be processed through a conventional flotation plant on site. The total quantity of sulphide ore to be processed from the North/South pits is 79.7 Mt, and 50.5 Mt for the Tizate pit, for a total combined tailings capacity requirement of 130.2 Mt. Complete tailings testing have not been carried out to date; however, assuming a relatively coarse grind, a final in-place density of 1.5 t/m^3 has been assumed. Therefore, a total tailings capacity of 86.8 Mm^3 is required.

Mine life is 19 years. Sulphide ore production commences in year 2 at 1 Mt/yr, after which it will ramp up to 8 Mt/yr for the next 16 years, followed by a final year's production rate of 1.3 Mt.

TMF Site Alternatives

Geologix confirmed that Sites A, E and F were their preferred TMF sites. Based on the revised tailings capacity, SRK concluded that Site F would not have sufficient capacity for the required tailings. Since a side-hill impoundment in a seismically active area is not desirable, SRK eliminated that alternative from further analysis. Site E remains an option; however, doubling the required capacity makes the impoundment efficiency very low, unless dry-stacking is considered. Due to high operational costs, this has not been considered at this time.

Site A was subsequently selected as the preferred tailings disposal site. Three variants of this site were evaluated as follows:

- Option 1: All 130.2 Mt of tailings at Site A.
- Option 2: Assume 1/3 of the Tizate sulphide ore tailings can be deposited back into a pit, with the remaining 113.4 Mt of tailings going to Site A.
- Option 3: Assume all of the Tizate sulphide ore tailings can be deposited back into a pit, with the remaining 79.7 Mt of tailings going to Site A.

TMF Design

Site A is located in a shallow and wide valley, which will require construction of the three dams (Dam I, Dam II and Dam III) in order to retain the required volume of tailings. The main dam (Dam I) is about 4 km northeast from the proposed mill site. The key metrics of the dams associated with each option evaluated is presented in Table 19.10.

Table 19.10: Key Metrics for the Site A TMF Dam Options

Option	Required Capacity	Dam ID	Full Supply Level (masl)	Maximum Crest Elevation (masl)	Maximum Crest Height (m)	Maximum Crest Length (m)	3D Surface Area of Basin at Maximum Crest Elevation (m^2)
1	130.2Mt (86.8 Mm^3)	Dam I	441	443	43	1,490	7,796,603
		Dam II			37	1,610	
		Dam III			10	408	
2	113.4 Mt (75.6 Mm^3)	Dam I	439	441	41	1,462	5,535,692
		Dam II			35	1,594	
		Dam III			8	365	
3	79.7 Mt (53.1 Mm^3)	Dam I	434	436	36	1,424	4,218,415
		Dam II			30	1,543	
		Dam III			3	-	

Construction of a conventional retaining dam (earthen dam) will be expensive, especially considering the waste rock is potentially acid generating (PAG); therefore, the bulk of the construction materials will have to be sourced from locally developed quarries. The available preliminary tailings data suggests that there may be a significant coarse fraction, which may make it possible to construct containment dams using cyclones. The conceptual design presented here is therefore based on an assumed starter dam, using locally developed quarries and upstream raising (to be confirmed later with a proper stability assessment) of the dams and cyclone tailings in increments of 2 to 3 m. Downstream construction may ultimately be required because of the high seismicity of the site and the lack of data pertaining to foundation conditions and tailings properties.

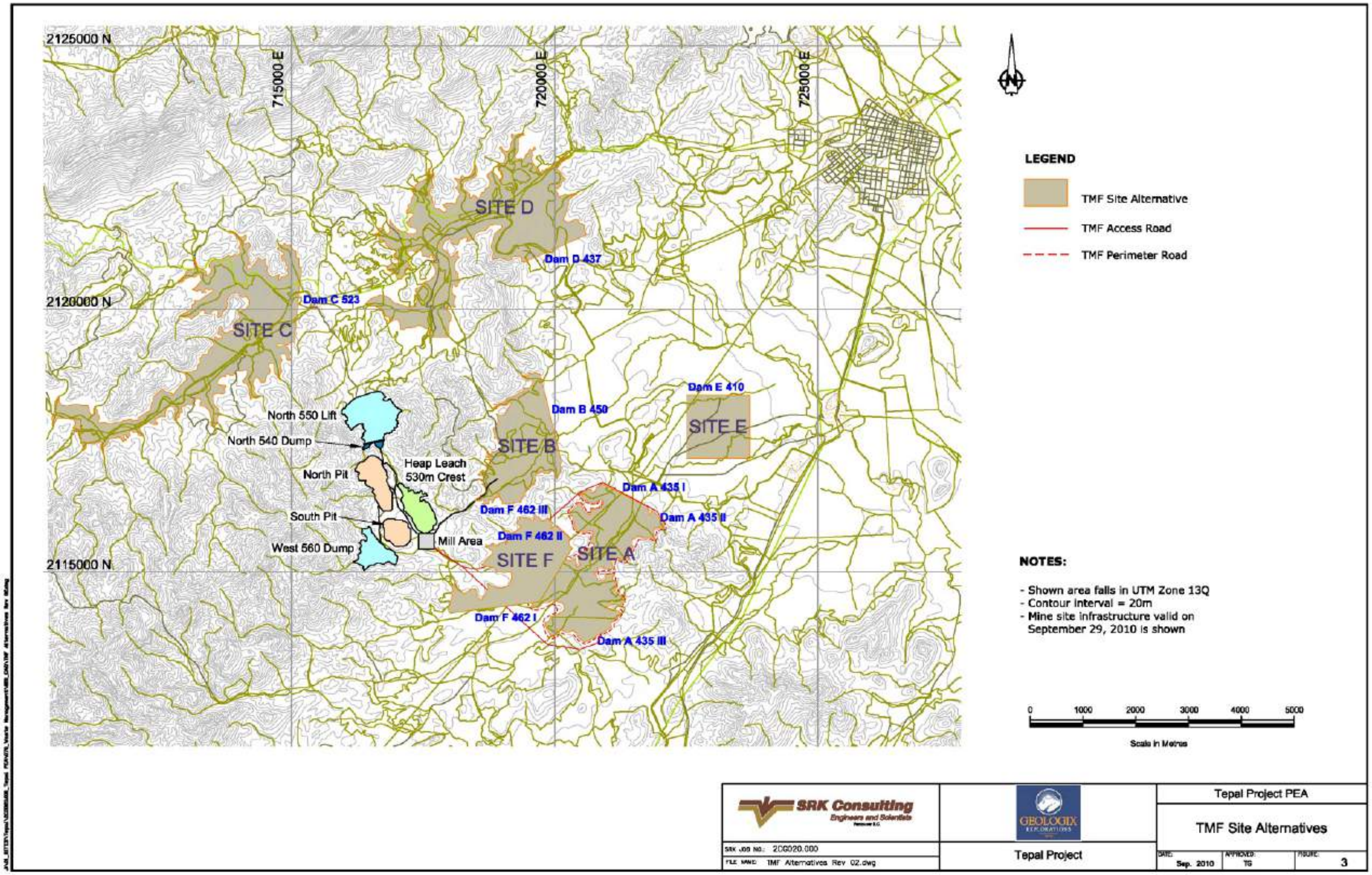

Figure 19.25: TMF Site Alternatives

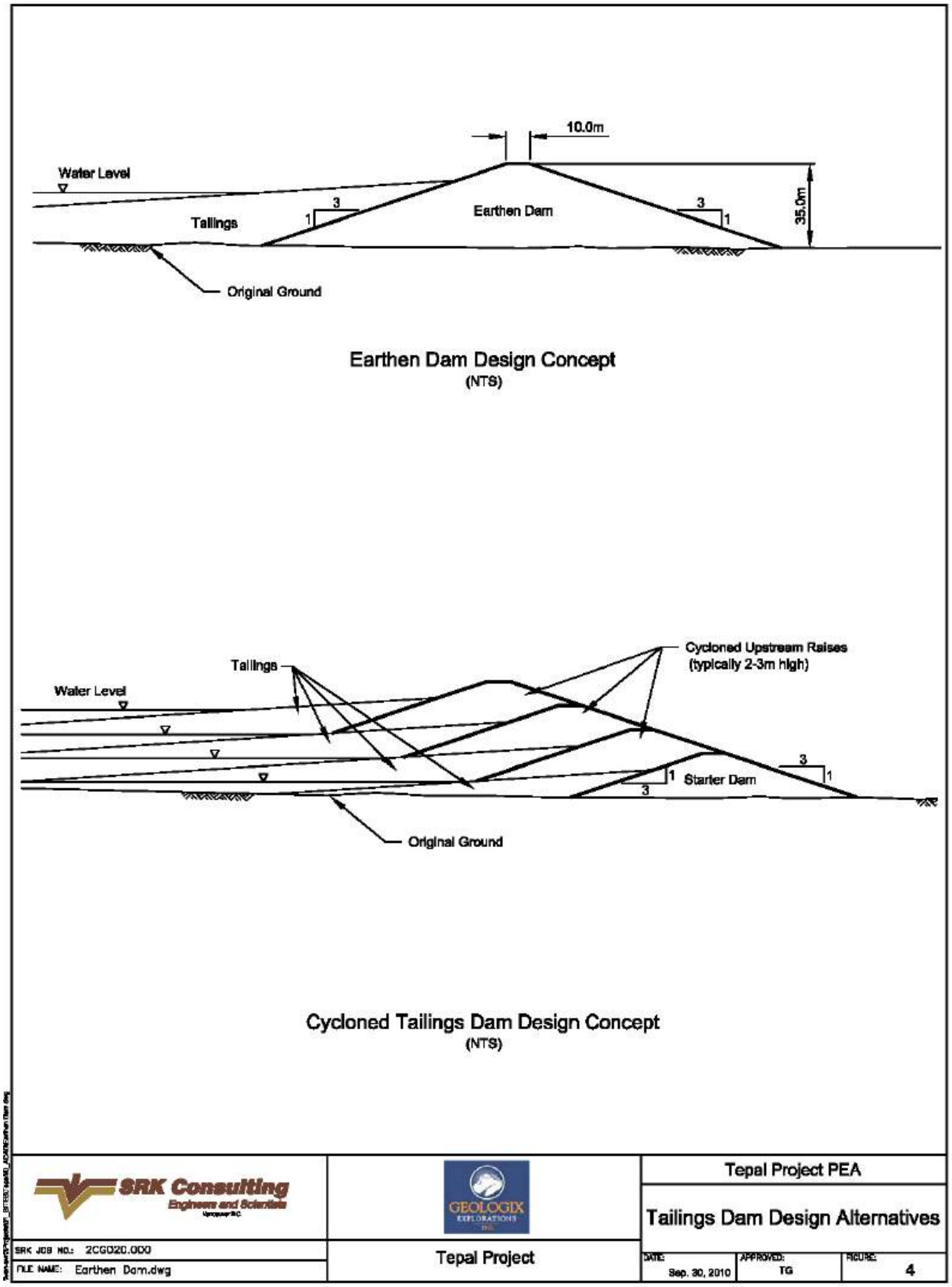

Figure 19.26: Tailings Dam Design Alternatives

19.4 Recoverability

Recovery estimates are shown in Table 19.11 and are based on metallurgical test results discussed in detail in Section 16 of the Priesmeyer report. Only the recovery of gold and copper were considered in the study.

Based on preliminary test work, metallurgical recovers for the North and South pits were estimated to be 87.4% and 60.7% respectively for copper and gold recovery in the sulphide flotation circuit. Heap leach/SART recoveries are estimated to be 14.3% and 78.4% for copper and gold, respectively for a crushed product which was the option selected for this study. No silver recovery was included in the North and South zones.

Preliminary test work on the Tizate zone mineralization yielded recoveries of 85.3%, 66.2% and 55.5% for copper, gold and silver respectively in the sulphide flotation circuit. Oxide heap leach and SART recoveries for Tizate material were estimated to be 6.8% for copper, 68.8% for gold and 38.9% for silver. Table 19.11 shows the recovery and concentrate grade assumptions for the project.

Table 19.11: Metallurgical Recovery Assumptions

		North and South		Tizate	
Item	**Unit**	**Sulphide Flotation**	**Oxide Heap Leach/SART**	**Sulphide Flotation**	**Oxide Heap Leach/SART**
Recovery					
Copper	%	87.4	14.3	85.3	6.8
Gold	%	60.7	78.4	66.2	68.8
Silver	%	0.0	0.0	55.5	38.9
Cu Concentrate Grade (Flotation and SART concentrate)					
Copper	%	25.1	70.0	24.2	70.0
Gold	g/t	variable with Cu	variable with Cu	variable with Cu	variable with Cu
Silver	g/t	variable with Cu	variable with Cu	variable with Cu	variable with Cu

19.5 Markets

It was assumed that the Tepal flotation and SART concentrates would be sent the San Luis de Potosi smelter, or similar, located in Mexico. The concentrates are envisioned to contain about 70% Cu and over 30 g/t Au. According to the preliminary testwork analyses completed to date, the concentrates are not expected to contain deleterious elements. Transportation would be trucked about 74 km to the smelter and re-loaded onto a rail system for final transport to the San Luis de Potosi smelter, a distance of less than 700 km. Standard industry smelting terms were used in the economic analysis and are shown in the operating cost section.

Planned annual concentrate production is shown in Table 19.12.

Table 19.12: Planned Annual Concentrate Production

Parameter	Unit	Total	Year																		
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Flotation Concentrate Grade	% Cu	25.1		25.1	25.1	25.1	25.1	24.7	25.1	25.1	25.1	25.1	25.1	25.1	24.4	24.2	24.2	24.2	24.2	24.2	24.2
	Au g/t	27.5		28.8	27.2	28.6	25.4	26.5	30.1	32.4	31.5	30.5	30.5	34.1	30.0	23.0	23.2	23.2	23.2	18.8	11.5
	Ag g/t	125.2		0.0	0.0	0.0	0.0	29.4	0.0	0.0	0.0	0.0	0.0	0.0	80.0	172.3	173.2	168.6	206.5	244.1	328.0
Flotation Concentrate Tonnes	Dry t	964,808	0	15,825	94,345	69,615	66,831	58,081	61,284	58,470	58,484	55,713	55,713	61,284	55,701	50,757	47,937	47,937	47,937	50,757	8,134
	Wet t*	1,041,992	0	17,091	101,893	75,185	72,177	62,728	66,187	63,147	63,163	60,170	60,170	66,187	60,157	54,818	51,772	51,772	51,772	54,818	8,785
Flotation Concentrate Contained Metal	Mlb Cu	528	0		8.8	52.2	38.5	37.0	31.7	33.9	32.4	32.4	30.8	30.8	33.9	30.0	27.1	25.6	25.6	25.6	27.1
	t Cu	239,296	0		3,972	23,681	17,473	16,774	14,360	15,382	14,676	14,680	13,984	13,984	15,382	13,610	12,283	11,601	11,601	11,601	12,283
	oz Au	853,332	0		14,655	82,434	63,994	54,629	49,543	59,334	60,865	59,319	54,650	54,650	67,141	53,719	37,464	35,761	35,761	35,761	30,652
	oz Ag	1,808,608	0						54,871							143,211	281,248	266,971	259,833	318,367	398,315
SART Concentrate Grade	% Cu	70	70	70	70	70	70	70													
SART Concentrate Tonnes	Dry t	5,992	428	1,655	1,348	1,226	757	578													
	Wet t*	6,471	463	1,787	1,456	1,324	817	624													
SART Concentrate Contained Metal	Mlb Cu	9.2	0.7	2.6	2.1	1.9	1.2	0.9													
	t Cu	4,194	300	1,158	944	858	530	405													

**Assumes 8% moisture in concentrate*

Gold and silver, in the form of doré, would be produced from the heap leach operation and is planned to be transported and sold to a refinery. Annual planned doré production is shown in Table 19.13.

Table 19.13: Annual Estimated Doré Production

Product	Unit	Total	Year					
			1	2	3	4	5	6
Doré from heap leach	oz Au	**140,746**	13,595	37,057	27,226	30,251	15,930	16,687
	oz Ag	**3,427**	-	-	-	-	-	3,427

As would be expected at this early project stage, Geologix currently does not have any smelting or refining contracts in place.

19.6 Contracts

As the project is still at an early stage, there are currently no mining, concentrating, smelting, refining, transportation, handling, sales and hedging contracts or arrangements.

19.7 Environmental Considerations

The Environmental Baseline (EBL) refers to the collection and generation of a preliminary inventory of the environmental background conditions for the Tepal Project. The survey, at this preliminary stage, has been conducted during the dry season (May 2010) and rainy season (October 2010), representing the main annual variations in the area (seasonal fluctuations).

Additional environmental work should be directed mainly for pre mining monitoring and gap analysis (specific areas for mine development).

19.7.1 Physical Environment

Meteorology and Air Quality

The Tepalcatepec River basin is characterized by a warm sub-humid climate on its central portion; the eastern and northern portions present semi-warm sub-humid and temperate-humid climates. The annual average temperature for the region is 26.61º C with an average annual precipitation rate of 860.37 mm and an evapo-transpiration rate ranging from 600 to 700 mm. The area is susceptible to dust generation, especially during the driest period of the year (January through May).

Tailings

The Tepal project is located within a moderately high seismic region (Zone D, frequent quakes and ground acceleration velocities may exceed 70% of the gravity caused by seismic activity) and within the area of cyclonic influence of the Pacific Ocean, falling under the following classification for the design of tailings impoundments (G.II-Sg.4-C.19-SR.D, according to Mexican Standard NOM-141-SEMARNAT):

This classification (II-4-10) implies the following environmental design conditions:

- Construction method for downstream design (rock dike, homogeneous filters dike, classified and compacted sands);
- Conventional tailings dam;
- Static stability analysis (fixed element);
- Seismic stability analysis (fixed element);
- Requires the installation of piezometers;
- Installation of surface control points;
- Installation of seismograph; and
- Drainage control by portal section.

Additional potential infrastructure might include:

- Waste rock dumps;
- Process plant (structure and foundations);
- Slope design;
- Open-pit mine walls;
- Drainage control systems; and
- Ancillary facilities.

Dust Control

At present, the sources of dust are mainly of natural origin, activities related to extensive cattle grazing, traffic on unpaved roads, and slash and burn practices.

There are several areas considered as zones of high contribution of dust and low capacity to retain the generated dusts. This situation can be aggravated during subsequent stages of exploration, especially during site preparation and construction. All of these situations can be mitigated by the implementation of appropriate measures; such as: watering of roads, handling of wet material, setting speed limits and speed reducers, and the assessment of potential addition of soil stabilizers on dirt roads.

Hydrology and Sedimentation

The project is located on the western part of the Rio Tepalcatepec basin. The basin’s main rivers are Tepalcatepec or Río Grande and the El Marquez River. The Tepalcatepec River is important for the agricultural activities (second largest irrigation district in Mexico) within its reach as well as for providing flow for the El Infiernillo hydro-electrical dam.

At the Tepal site, only seasonal streams have been identified, there are no relevant hydraulic structures such as deep wells or channels. There is however, one stream flowing from the site, which crosses the irrigation channels and pours into the irrigation district and special control measures should be installed in future stages for runoff and sediment control.

Water Reservoirs

The hydroelectric dam Plan de Apatzingán controls the runoff from the Tepalcatepec River, stores water for the Irrigation District 097 Lazaro Cardenas and generates electricity. It is located 25 kms to the northeast of the Tepal Site. Also on the basin, of the Otates River, the Los Olivos water dam was constructed to expand the irrigation district mentioned above. Los Olivos is the most important artificial water body related to the study area (approximately 9 kms NNE from the Tepal site). Surface water is not available for new concessions (surface water is compromised for agricultural purposes).

Water Quality

A total of 7 water samples were collected in May and October 2010 at the Tepal site and the surrounding project area, on shallow wells, irrigation channels, and streams as a pre-mining development reference. Water quality in the area, currently does not present any evidence of serious issues. It has only minor issues on solid contents, hardness and alkalinity.

Rustic water sources, such as La Estanzuela and La Cienega are relatively exposed to environmental and human influences, resulting in unreliable water quality (sediments and potential biological pollution). La Estanzuela is mainly affected by the immediate households and low level in relation to stream flows that may affect the rustic underground reservoir. La Cienega has a shallow well, affected mainly from pollution from livestock in the area and rustic protection.

From a pre mining perspective, the total metals are relatively low, a characteristic that should be monitored and maintained in the following years, especially during operation and weathering of geological materials (tailings and waste rock).

Hydrogeology

The project stands between the provinces Eje Neovolcanico, and the Sierra Madre del Sur (transition area), on the south-western portion of Michoacán. The main hydrogeological unit in the region is the Apatzingan aquifer.

The region presents a diverse geology composed mainly of rhyolites, rhyolitic tuffs, andesites, and intrusives such as granites. These function, to some degree, as an impermeable barrier for underground flow. But on the surface, they preserve faulting and fracturing that serve as recharge paths towards the main valley (Tepalcatepec Depression).

The Tepalcatepec valley, according to CONAGUA, presents a horizontal underground flow of about 55 million m^3 per year. The water availability for the aquifer is positive and new water concessions are viable according to the official CONAGUA water balance (2010).

19.7.2 Biological Environment

The use of land for livestock, in an area with limited livestock capacities (such as the area of study), represents heavy pressure on the ecosystem, causing loss of plant cover, soil compaction, fragmentation of the ecosystem and increased risk of forest fires.

Furthermore, the area has traditionally been used for hunting; which is practiced mostly for subsistence purposes. The areas used for hunting purposes correspond to sites with topographic constraints that impede the advance of farming practices. The most hunted mammals are opossum and deer. Bird hunting was not detected and the locals are in disagreement with this practice, especially the west Mexican Chachalaca (diminished population).

The occurrence of fauna at the area is quite diverse, especially for large mammals that use ravines, canyons and high ground as shelter sites. On the other hand, the distribution of birds appears homogeneous, but certain preferences do take place; depending on the species, based on feeding and reproductive biology.

Vegetation

Tropical Deciduous Forest, Pine-Oak forest and Tropical Subdeciduous Forest are the vegetation associations that exist in the basin (CONABIO-INEGI). Tropical Deciduous Forest is the most common type of vegetation. The Tepal area is dominated by valleys and hill systems, in which a heterogeneous mosaic of plant species from the genera Acacia, Cordia and Amphypterigium exist. This has only changed in the areas transformed for agriculture and livestock ranches. The vegetation mosaic in the region also includes the main land use which is agriculture and seasonal crops. The main irrigated farmland is Irrigation District 097 Lazaro Cardenas, one of the most important districts in the country.

The conservation of the Deciduous forest resources at the Tepal area is considered moderate, ranging from well preserved sites, to areas affected by forest fires used to clear land for agricultural and livestock purposes, more markedly towards ejido grounds.

The area that presented the highest affectations is located to the northeast of the study area, towards La Estanzuela. This is an area where fires, erosion and degradation have had a greater effect on the land, allowing the colonization of huisache (Acacia) and induced grassland. The lower strata have been removed by cattle, impeding the growth of some species by natural regeneration. Most of the forest degradation is caused by livestock and human influence (clearing) and thus represents a higher fire risk during the dry season November-May.

In the Tepal study area, a total of 54 species in 33 families of plants, were identified. Three of the species that were detected are described as, of difficult regeneration and one of local interest. Cordia eleagnoides is a plant of local interest; it is used for construction wood and domestic firewood. The species Stenocerous queretaroensis, Cephalocerous senilis, Mammillaria beneckei, are considered as difficult to regenerate due to habitat requirements and seed dispersal conditions.

Only one species was found under a protected category (NOM-059-SEMARNAT-2001), the Cephalocereus senile (“old man” cactus).

No existence of rare species has been reported. This may be due to the gradual alteration of the microclimate and constant pressure on the vegetation. The study area borders with the agricultural frontier, this makes the tropical deciduous forest subject to frequent human disturbances (fire, over grazing, trampling, compaction, and soil loss). The highest degradation degree for the vegetation is towards the north, mostly by pressures from La Estanzuela and the expansion of the agricultural frontier, though the areas to the south and east present indications of similar trends).

Mine development plans usually result in the loss of forest land and the mineralization seldom coincides with ecological criteria. This, along with the close proximity of towns warrants for the consideration of compensatory measures from early mine planning stages, measures such as selection of potential buffer zones around the potential Tepal Project infrastructure.

Reptiles and Amphibians

To classify the herpetological fauna, a direct and unrestricted search method was used. The method, in general terms, consists in daily and nocturnal walks, directing the search to the areas that may have a high probability for harbouring amphibians and reptiles (rock piles, canyons, high humidity areas, fallen trees, wells, holes, bark, crevices, etc.). The intensive field inspections and site evaluations have resulted in the identification of 17 species of reptiles; 5 of these species are protected by NOM-059-SEMARNAT-2010 and one of these (rattle snake) is also mentioned in Appendix III from Convention for International Trade of Endangered Flora and Fauna Species (CITES).

The study area is characterized by two seasonal periods: rain (June-October) and drought (November-May). In the case of amphibians, the field activities and inspections need to be executed during the rainy season in order to accurately register the presence and dynamics in regard to their natural surroundings. Currently, the Tepal Project contemplates a potential amphibian species inventory, pending validation.

Mammals

In order to detect mammal presence within and around the study area, line transects extending 10 km were implemented using existing rural and secondary dirt roads (old exploration roads). In addition to the transects, exhaustive searches were performed for the detection of signs and tracks on hoof paths and streams in order to complete the inventory for existing species.

During the field inspections, a total of 13 species of mammal were identified in 6 orders and 9 families, 7 species were carnivores. It is important to note an indirect puma registry at the study area as a personal reference from the local community; this coincides with bibliographic registries that should be confirmed in the future.

For the identified mammal species, none is currently under a protection status by NOM-059-SEMARNAT-2001 or by the Convention for International Trade of Endangered Flora and Fauna Species (CITES).

Birds

To determine the presence and/or absence of bird fauna, intensive searches and monitoring stations were used. The tours included cool mornings, high temperature times and sunset. Limited bird activity was detected during the hottest hours of the day (approx. from 2:00 to 4:00 PM).

A total of 30 bird species were detected at the Tepal area in 21 families and 10 orders. The best represented families are Columbidae, Cuculidae and Icteridae, with 3 species each.

The Tepal Project site is not located within an important bird conservation area (AICA). It is important to note that there is presence of migratory birds that use the Tepal area in their route; some individuals stay during their migration period. The bird inventory presents three species mentioned by one of the CITES appendixes.

19.7.3 Socio-Economic Context

Because of their proximity to the Tepal Project, four locations were selected for this study: Tepalcatepec, Colomotitlan, La Estanzuela and La Cienega. Tepalcatepec is the main population centre, with 14,598 people; it is the least marginal of the four, having better health, education and living conditions. Colomotitlan presents a medium margination index while La Cienega and La Estanzuela present higher levels.

At state level, 43.45% of the population does not have basic education (grammar school); Tepalcatepec has a higher education level while the three rural towns have lower education levels in comparison with the state average.

The Tepalcatepec region is one the largest irrigation districts (over 50,000 Ha) in Mexico and most of the productive activities are centered on agriculture (irrigation and seasonal) and livestock. Because of this, little attention has been paid to the potential mining development of the region. A few examples of mining activities have taken place and may raise potential concerns.

The social climate and its effects on the mining industry must be taken into account from early mine planning stages. Examples from other mining projects and operations, especially in Michoacán serve as a general guideline. Geologix is responsible for obtaining the social license, a process that requires consideration of current authorities, groups, community and the irrigation district.

Archaeology and Cultural Heritage

The main indigenous languages in the region are Purépecha and Zapoteco. Approximately 54 people speak some indigenous language this represents less than 1% of the population of the area (51 of them also speak Spanish).

Sites of Historical and Archaeological Significance

There are some architectural monuments in the Tepalcatepec municipality, such as the San Francisco Parrish, municipal building, portals and the Melchor Ocampo Avenue. Other significant sites are the Los Olivos Dam, Chilatan Dam and Las Jacarandas Hacienda.

There are no records regarding patrimonial investigations at the Tepal area, however, the property is located within the estimated limits of the ancient Tarasco Empire. For this reason, and taking into account that Tepalcatepec was considered an administrative centre in the XVI century; Geologix should apply for a land liberation permit from the National Institute of Anthropology and History (INAH).

Land Use

The land uses and activities within the municipality are represented by: agriculture, ranching and commerce. Industrial activities are limited to Tepalcatepec.

In the case of the project site, land use is limited by the slopes, steep topography, soil depth, erosion, access and low population density.

The areas restricted by topography, corresponding to the more remote western and northeastern portions of the project area are sites adequate for wildlife and forest lands, traditionally used for hunting (sport and subsistence) by the local community.

Special care and control measures will need to be taken into account during the wet season, when cattle are brought to the central areas near the exploration sites.

19.7.4 Permitting

The Tepal Project, in terms of permitting will need to consider the following environmental procedures:

- Exploration activities are currently (2010-early 2011) considered to be within applicable Mexican exploration standards and in the event of potential exceedances, Geologix Inc. will proceed to file the appropriate environmental impact and forest report to SEMARNAT for potential expansion of exploration surfaces (drilling, pads, roads, etc.);
- Preparation of the Environmental Impact Statement (MIA document) for Environmental Impact Authorizations);
- Preparation of a Technical Report for Forest Land Use Modification (ETJ document), needed prior to forest clearing. The submittal if this report implies land tenure of the legal right granted from the land owner to modify the land use to mining; and
- Preparation of a Risk Assessment (ER report) in the event that the intended process involves cyanide in amounts that exceed SEMARNAT criteria (over 1 kg of CN requires an ER).

19.7.5 Other Potential Environmental Activities for Pre-mining Stages

The project will need to expand reference environmental data on several issues, in the months prior to mine development, including the following:

- Dust monitoring (site, perimeter and nearby villages)
- Continuation of water monitoring activities
- Installation of water monitoring equipment (piezometers and monitoring wells)
- Consideration of an appropriate buffer zone in relation to mining infrastructure, conservation goals and community

19.7.6 Summary Conclusions

The present environmental baseline has expanded to 3,217 ha as a reference inventory, covering the 1,406 ha of the Tepal mining concessions and micro/nano basins of direct influence. The following are the general environmental conclusions:

- The project is located in the vicinity of land routes suitable for the operation of a mining project, however, locally; the road system is rudimentary and requires an important work of access in the event of major mining related activities
- The Tepal concession are located on surface land belonging to the Tepalcuatita Ranch, private land and ejido lands, implying potential displacement of productive activities (cattle ranching and seasonal agriculture) and closing rural roads recently used by the local community (travel to and forth the highway and La Estanzuela)
- The Environmental Baseline (LBA) covers 200% more surface area than the footprint of the project (3,217 ha studied versus 1,406 ha of the current Tepal mining concession), this allows for a better understanding of the local environmental system and future consideration for the preliminary mine development plan
- Once a conceptual mining development plan is prepared, new areas for the expansion of environmental inventories should be contemplated in order to include potential new sites of interest (mining infrastructure)
- The main components that have been considered for the establishment of this area of study correspond to the area of geological interest (mining concessions), the possible development of open pit mining, areas suitable for the establishment of a process plant, associated infrastructure and the construction of an access road dedicated to the mining unit, that connects the project to the East (towards the state highway); as well and the hydrological micro / nano basins of direct influence from the project
- Water quality at the Tepal Project is considered a key item in regard to potential areas of opportunity for community support and consideration of pre-mining parameters (cyanide, metals, etc.)

- The additional environmental monitoring activities are, at this moment, focused towards pre-mine stages, development and gap analysis in regard to specific infrastructure and environmental design/management

Geologix is required to prepare and submit to SEMARNAT different environmental reports (MIA, ETJ, ER) for environmental impact authorizations prior to site preparation and construction for operation permits, land use modification, risk assessment, among others. Overall environmental permitting in Michoacán can take from 6 month to 1 year with land tenure usually being the most sensitive issue in delaying the permitting process.

The current environmental baseline information indicates that there are no environmental “fatal flaws” identified for the proposed Tepal Project. The extent of habitat degradation in the area as well as the surrounding conservation status (heterogeneous mosaic), current land use and local trends do suggest the need for an integrated and careful environmental management policy and program in order to ensure that the mine site activities can coexist with the local communities.

19.8 Taxes

The engineering economic model developed for Tepal for this report does consider taxation and, therefore, the information provided in this section is only for general information. Detailed tax calculations are typically very complex and take into account many factors of a corporation’s entire financial performance and not just the results of an individual operation. In addition, the type of project financing will also influence the outcome of the after-tax financial analysis.

The recent passing of the 2010 Mexican tax reform bill has increased corporate income tax from 28% to 30% for 2010 and 2012, 29% in 2013 and back to 28% in 2014.

A valued added tax (“IVA”) of 16% is due to the government on goods and services but is generally refundable by the Mexican government. Mexican law also has a provision for a profit sharing tax paid to employees. The tax rate is 10% of company profit after tax.

An NSR royalty of 2.5% was assumed for the economic analysis.

19.9 Capital and Operating Cost Estimates

The open pit mining activities for the Tepal and Tizate pits were assumed to be primarily undertaken by the owner as the basis for this preliminary economic assessment. The cost estimate was built from first principles, along with input from Geologix, as well as SRK experience of similar sized open pit operations. Equipment efficiency was estimated based on Tepal conditions (e.g. haul routes for each phase). In order to minimize capital requirements, a small contractor truck fleet was assumed for the initial mining of the Tizate pit, and the operating costs adjusted accordingly.

Local labour rates (for operating, maintenance, and supervision/technical personnel) and estimates on diesel fuel pricing were taken into consideration for the mining cost estimate.

Open pit mining costs were estimated to be $1.36/t material mined or $2.92/t processed (includes both oxide and sulphide material), for pit and dump operations, road maintenance, mine supervision and technical services. Table 19.14 summarizes the mining operating cost by function.

Table 19.14: Mine Operating Cost Estimate by Function

Open Pit Function	$/t mined
Drill	$0.13
Blast	$0.36
Load	$0.13
Haul	$0.36
Roads/Dumps/Support Equipment	$0.24
General Mine/Maintenance	$0.06
Supervision/Technical	$0.08
Total	**$1.36**

Processing Cost Estimate

Operating costs for the processing plant are summarized in Table 19.15. Operating costs for the two leach options are summarized in Table 19.16. The crushed ore leach was the option selected to be used in the economic model. Labour and supervision costs were built up from detailed manning charts and Mexican wage rate information. Power costs were built up from estimates of installed power and a cost of US$0.0942/kWh. Consumables costs were based on estimated reagent cost and usage, wear items, and maintenance supplies. The processing costs include concentrate transportation to the port. No operating cost contingency was included.

Table 19.15: Summary of Operating Costs for the Mill Circuit

Operating Area	M$/year	$/t[1]
Consumables	17.2	2.15
Power	13.7	1.71
Labour	3.5	0.44
Total Operating Costs	34.4	4.30

Note 1. Based on 8,000,000 tonne/year throughput

Table 19.16: Summary of Operating Costs for the Leach Circuit

Operating Area	Crushed Ore Leach[2]		ROM Ore Leach	
	M$/year	$/t[1]	M$/year	$/t[1]
Ore Re-handling	0.2	0.05	0.0	0
Operating Labour	0.5	0.15	0.2	0.07
Staff/Supervision	0.5	0.15	0.3	0.10
Reagents	10.0	3.20	10.0	3.20
Electric Power	1.6	0.44	0.6	0.20
Mobile Equipment	0.2	0.06	0.2	0.06
SART	0.9	0.26	0.9	0.30
Total Operating Costs	13.8	4.31	11.3	3.63

Note 1 Based on 3,120,000 tonnes of leach ore per year

Note 2 Option used in the economic analysis

General and Administration Cost Estimate

G&A costs were estimated to be $0.68/tonne of heap leach and mill feed material.

Off-site Costs

The following off-site costs and smelter terms were estimated and used in the economic analysis.

- Copper concentrate treatment charge: $50.00/dmt
- Copper refining charge: $ 0.05/ payable lb
- Gold refining charge: $ 5.50/ payable oz
- Concentrate transport cost: $37.30/wmt
- Copper payable in Cu concentrate: 97% with no deductions
- Gold payable in Cu concentrate: 98% with no deductions
- Gold payable in doré: 100% with no deductions
- Royalty: 2.5% of net smelter return

19.9.1 Capital Cost Estimate

Summary

Capital costs for the project were developed from a mix of first principles, reference projects, and experience. The annual capital costs by major category are shown in Table 19.17.

Table 19.17: Capital Cost Estimate Summary

Category	Unit	Total	Year					
			-2	-1	1	2	3 to 18	19
Mining Equipment	M$	77.6		24.0	5.2	23.0	25.4	
Roads and General Infrastructure	M$	15.7		15.7				
Electrical Power Line and Generators	M$	14.2		14.2				
Flotation Process Plant	M$	124.0		24.0	100.0			
Heap Leach Pad and Facility	M$	17.3		17.3				
Tailings Management Facility	M$	19.9		10.0	9.9			
Owners Costs	M$	9.6		4.1	5.5			
EPCM	M$	28.2		12.2	16.0			
Closure	M$	9.0						9.0
Contingency (10%)	M$	31.5		12.1	13.7	2.3	2.5	0.9
Working Capital	M$	0			3.4			-3.4
Total Capital Cost	M$	**346.7**		**133.5**	**153.7**	**25.3**	**28.0**	**6.5**

Mine Equipment

Mine equipment capital costs (including sustaining and replacement costs) were developed using productivity factors for production equipment and SRK experience for ancillary equipment and are summarized in Table 19.18 below. Unit costs are based on budgetary manufacturer quotes.

Table 19.18: Mine Capital Cost Estimate

Item	Unit	Quantity	Total Cost
Primary			
Crawler-Mounted, Rotary Tri-Cone, 9.875-in Dia.	M$	4	10.00
Crawler-Mounted, Rotary Tri-Cone, 6.5-in Dia.	M$	2	3.40
Diesel, 13-cu-yd Front Shovel	M$	2	5.00
Diesel 14-cu-yd Wheel Loader	M$	2	3.20
100-ton class Haul Truck	M$	14	21.25
D10-class 17.3' blade	M$	4	5.60
D9-class 15.8' blade	M$	4	3.52
824H-class 13.8' blade	M$	4	2.54
16H-class 16' blade	M$	4	3.44
14H-class 14' blade	M$	2	1.20
HD325-7R(40ton) 35m3 9000 gallon	M$	3	1.58
Subtotal Primary	M$		60.73
Ancillary			
ANFO/Slurry Truck, 12-ton	M$	1	0.20
Stemming truck, 15-ton	M$	1	0.09
Powder Truck, 1-ton	M$	1	0.07
AN Storage Bin, 60-ton	M$	1	0.05
Powder magazine, 24-ton	M$	1	0.05
Cap magazine, 3.6-ton	M$	1	0.01
Excavator (backhoe), 4 cu-yd	M$	1	0.48
Haul Truck (road constr), 35-ton	M$	3	1.53
Backhoe/Loader, 1.4 cu-yd	M$	1	0.15
Portable Aggregate Plant,30 tph	M$	1	0.30
All-terrain Crane, 60-ton	M$	1	0.63
Transporter w/Tractor, 100-ton	M$	1	0.40
Fuel truck, 5000-gal	M$	1	0.28
Lube/Service Truck	M$	1	0.32
Mechanic Field Service Truck	M$	6	1.08
Off-Road tire handling Truck	M$	1	0.35
Wheel Loader 8.5-cu-yd	M$	1	0.80
16 cu-yd Scraper	M$	1	0.59
Light Plant, 6-kW	M$	10	0.20
Pickup Truck, 0.75-ton, 4-WD	M$	20	1.00
Crew Van, 1-ton, 4-WD	M$	10	0.55
Mobile Radio, installed	M$	96	0.07
Subtotal Ancillary	M$		9.18
Miscellaneous			
Shop Equipment	M$	1	0.75
Engineering & Office Equip plus Software	M$	1	0.50
Radio Communications System + GPS	M$	1	0.50
Subtotal Miscellaneous	M$		1.75
Total Equipment & Misc.	M$		71.66
Spares, Inventory, Contingency	M$		5.96
TOTAL MINE CAPITAL, Pre-Tax	M$		77.62

Infrastructure and Power

Infrastructure to provide access to the site (roads and a bridge) and on-site infrastructure were developed from a combination of factored costs and budgetary quotes. Diesel generator costs were based on a budgetary quote. The infrastructure and power cost estimates are shown in Table 19.19.

Table 19.19: Infrastructure and Electrical Power Capital Estimate

Item	Cost Estimate (M$)
Mine Haul Roads	2.5
Access Road (Hwy to site)	0.7
Access Road Canal Bridge	0.5
Buildings	4.0
Sewage and Waste Water	0.5
General	0.5
Mine Light Industrial Area	5.5
Mine Ancillary Facilities	1.5
Total Infrastructure Costs	15.7
Power Line from Tepalcatepec	3.0
Power Generators	11.2
Total Power Costs	14.2
Total Infrastructure and Power Costs	29.9

Flotation Process Plant

There are two types of process facilities envisioned to treat the Tepal ore types. The sulphide ore requires crushing, grinding and flotation to produce a concentrate for sale to a toll smelter. The oxide ore will be stacked on lined pads and then leached with a dilute cyanide solution to extract the gold. The ore may either be crushed and screened or treated in Run-of-Mine (“ROM”) condition. The gold and any silver are then recovered from the pregnant leach solution (“PLS”) in a carbon adsorption (“ADR”) plant. Due to the presence of high levels of cyanide soluble copper in the oxide ore, a Sulfidation-Acidification-Recycling-Thickening (“SART”) plant will be needed to remove the copper and recover and recycle the cyanide.

The initial capital costs for each type of process plant are summarized in Tables 19.20 and 19.21. These costs are drawn from a variety of sources including vendor budgetary quotations, equipment cost data bases and bench marking against similar Mexican projects. The summary tables exclude engineering, procurement and construction management fees, owner’s costs and working capital.

Table 19.20: Flotation Capital Cost Estimate

Area	Cost Estimate (M$)
General Plant	5.7
Water Supply	4.4
Sulphide Primary Crushing	13.6
Sulphide Primary Grinding	45.1
Sulphide Flotation	21.3
Sulphide Concentrate Thickening and Filtration	9.3
Sulphide Tailings Thickening and Disposal	4.1
Sulphide Reagents	3.0
Sulphide Services	6.4
Mobilisation and Demobilisation	2.6
Temporary Facilities	2.6
Commissioning	0.6
Vendor Representatives	0.7
First Fills and Spares	4.5
Total	**124.0**

Table 19.21: Heap Leach – SART Facility Cost Estimate

Area	Cost Estimate (M$)
Crushing & Screening	3.0
Stacker, Conveyor & Lime Silo	1.3
Leach Pad & Ponds	7.4
Leach Pumps	0.2
SART Plant	2.0
ADR Plant	2.0
Loaded Carbon	0.5
Yard Facilities	0.2
Heavy Mobile Equipment	0.8
Total Direct Costs	**17.3**

Tailings Management Facility

The following primary assumptions have been used in preparing the cost estimate:

- A starter dam (berm) will be constructed using material from a locally developed quarry, assumed to be within 5 km from the dam site.
- The starter dams are 3 m high with a 10 m crest width and 3H:1V side slopes.
- Upstream dam raises will be completed with cyclone tailings.

- Unit rates have been calculated using first principles assuming North American Equipment rates for an independent contractor to complete the work (inclusive of fuel, profit, maintenance and insurance).
- The capital costs exclude the operational cycloning cost which is about $0.28/tonne to be added to operating cost (for in pit portion this cost does not apply).
- Since the tailings may be PAG, a base liner may be required. To evaluate the sensitivity of this on the capital, two scenarios were evaluated for each option: one with a liner and one without.
- In the case where no liner is required, all capital will be spent pre-mining, with the exception of the tailings thickener (for in-pit disposal).
- If a liner is required, the costs of the liner may be amortized however the extent was not investigated.
- Capital costs of the tailings pipeline and pumps are excluded. Costs of the cyclones are included.

The TMF costs are shown in Table 19.22.

Reclamation

Reclamation/closure costs were estimated using unit rates ($\$/m^2$) based on other similar Mexican projects. It was assumed 1 m of cover material would be used for the heap leach and tailings areas. Water treatment was assumed not to be required at closure. This has not been confirmed with testing. Reclamation costs are shown in Table 19.23.

It was assumed that building and equipment removal would be paid for by the salvage value.

Table 19.22: Reclamation Closure Cost Estimate

Item	Cost Estimate (M$)
Ground preparation	1.4
Heap Leach pad cover (1 m)	0.4
Tailings cover (1 m)	5.2
Re-vegetation	2.0
Total Reclamation Costs	9.0

Owner's Costs

Owner's costs prior to the production decision on the project have been excluded. These costs would normally include preliminary and final feasibility studies (including the related field work), definition diamond drilling, environment and social impact assessments, permit applications, corporate office expenses, camp expenses, insurance, property taxes, etc. Owner's costs, once a project go/no go decision is made, were given an allowance of 5% of the capital costs of infrastructure, process plant, heap leach facility and tailings management facility. Owner's costs total $9.5 M.

EPCM and Contingency

Engineering, procurement and construction management costs were estimated at approximately 15% of capital costs for infrastructure, process plant, heap leach facility and tailings management facility. The total EPCM cost was estimated to be $28.2 M. It is the intention of Geologix to conduct some independent EPCM work which has been included in the owner's costs.

A 10% contingency allowance was applied to all capital costs. A total contingency estimate of $31.5 M was used in the capital cost.

Working Capital

A working capital allowance equivalent to 1/3 of the operating costs in the first ($3.4M) was used in Year 1. The working capital was recouped during the last production year.

19.10 Economic Analysis

The economic analysis described in this report provides only a preliminary overview of the project economics based on broad, factored assumptions. The mineral resources used in the LOM plan and economic analysis include no measured resources, 57.2 Mt (40%) of indicated resources and 87.1 Mt (60%) of 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 will be upgraded to a higher resource category. Based on this, there is no certainty that the results of this preliminary economic assessment will be realized.

19.10.1 Assumptions

Simplified earnings before interest, taxation, depreciation and amortization ("EBITDA") analyses were compiled for three cases using varying copper, gold and silver prices. For each case the mill feed tonnes were held constant and the metal prices were varied only in the economic model. The base case (Case B) metal prices were used for Whittle optimization and mine planning were $2.75/lb Cu, $1,000/oz Au and $16.00/oz Ag. The metal prices used in the economic model for the three cases are shown in Table 19.24.

Table 19.23 Metal Prices by Case

Case	Copper Price ($/lb)	Gold Price ($/oz)	Silver Price ($/oz)
Case A	2.75	900	16.00
Case B (base case used for mine design)	2.75	1,000	16.00
Case C	3.50	1,200	16.00

Common assumptions to all cases included:

- 5% discount rate (“DR”) for net present value (“NPV”) calculation;
- 100% equity financing as per guidance by Geologix;
- Exclusion of all pre-development costs as per guidance by Geologix (e.g., exploration and resource definition costs, engineering field work and studies costs, environmental baseline studies costs, etc.);
- Exclusion of all duties and taxes (a brief description of Mexican taxes is included in Section 19.7);
- 2.5% royalty on net smelter return;
- All 2011 costs were assumed to be sunk costs with analysis beginning in 2012 (Year 0).

19.10.2 Results

Table 19.25 summarizes the key economic results for each case.

Table 19.24: LOM Key Economic Results

Parameter	Unit	Results
Case A		
EBITDA $NPV_{0\%}$	M$	653
EBITDA $NPV_{5\%}$	M$	347
EBITDA IRR	%	20
EBITDA payback period	Production years	4.5
Case B		
EBITDA $NPV_{0\%}$	M$	749
EBITDA $NPV_{5\%}$	M$	412
EBITDA IRR	%	22
EBITDA payback period	Production years	4
Case C		
EBITDA $NPV_{0\%}$	M$	1,320
EBITDA $NPV_{5\%}$	M$	786
EBITDA IRR	%	34
EBITDA payback period	Production years	3

The Case B economic model is shown in Table 19.27.

For all cases, gold and copper contribute approximately 40% and 60% respectfully to the project revenue. Silver contributes less than 1% of total revenue.

19.10.3 Break-even Metal Prices

Table 19.26 shows ranges of gold and copper prices that, when combined, result in a break-even situation or an $NPV_{5\%}$ of $0. For example, with a gold price of $1,000/oz the project requires a copper price of $1.49/lb to break even.

Table 19.25: Combined Break-even ($0 $NPV_{5\%}$) Copper and Gold Prices

Copper Price ($/lb)	Gold Price ($/oz)
1.49	1,000
1.69	900
1.89	800
2.08	700
2.00	740
2.25	615
2.50	485
2.75	360
3.00	235

Table 19.27 Case B Economic Model

SECTION	ITEM	UNIT	Year -1 to 19 Total		Y E A R																		
				-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
MATERIAL SCHEDULE																							
Mining Total	Operating Days	days	6,617		365	365	366	365	365	365	366	365	365	365	365	365	365	365	365	365	365	365	45
	Waste	Mt	165.3		1.91	2.53	9.54	9.36	11.82	16.26	12.90	16.00	14.00	14.61	15.94	4.92	14.30	6.83	4.89	3.39	3.66	2.50	-
	Oxide Ore	Mt	14.3		0.75	3.00	3.00	3.00	2.18	2.35	-	-	-	-	-	-	-	-	-	-	-	-	-
	Sulphide Ore	Mt	130.2		-	0.99	7.97	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	1.28
	Total Mining	Mt	309.9		2.7	6.5	20.5	20.4	22.0	26.6	20.9	24.0	22.0	22.6	23.9	12.9	22.3	14.8	12.9	11.4	11.7	10.5	1.3
	Strip Ratio	t waste:t ore	1.14		2.5	0.6	0.9	0.9	1.2	1.6	1.6	2.0	1.8	1.8	2.0	0.6	1.8	0.9	0.6	0.4	0.5	0.3	-
	Daily Production	t/day	46,827		7,279	17,863	56,030	55,767	60,274	72,893	57,112	65,734	60,274	61,948	65,586	35,384	61,093	40,625	35,307	31,195	31,953	28,773	28,489
North and South Pit	Waste	Mt	119.7		1.9	2.5	9.5	9.4	11.8	6.2	12.9	16.0	14.0	14.6	15.9	4.9	0.0						
Flotation	Flotation Circuit Feed	Mt	79.7			1.0	8.0	8.0	8.0	4.7	8.0	8.0	8.0	8.0	8.0	8.0	2.1						
	Cu head grade	%Cu	0.23			0.46	0.34	0.25	0.24	0.22	0.22	0.21	0.21	0.20	0.20	0.22	0.25						
	Au head grade	g/t Au	0.41			0.76	0.53	0.41	0.35	0.36	0.38	0.39	0.38	0.35	0.35	0.43	0.70						
	Ag head grade	g/t Ag	-																				
Heap Leach	HL Feed	Mt	12.55		0.7	3.0	3.0	3.0	2.2	0.6													
	Cu head grade	%Cu	0.22		0.28	0.27	0.22	0.20	0.17	0.18													
	Au head grade	g/t Au	0.41		0.72	0.49	0.36	0.40	0.29	0.28													
	Ag head grade	g/t Ag	-																				
Tizate Pit	Waste	Mt	45.6							10.1							14.3	6.8	4.9	3.4	3.7	2.5	
Flotation	Flotation Circuit Feed	Mt	50.5							3.3							5.9	8.0	8.0	8.0	8.0	8.0	1.3
	Cu head grade	%Cu	0.18							0.19							0.18	0.18	0.17	0.17	0.17	0.18	0.18
	Au head grade	g/t Au	0.20							0.23							0.19	0.22	0.21	0.21	0.21	0.18	0.11
	Ag head grade	g/t Ag	2.15							2.21							1.83	1.97	1.87	1.82	2.23	2.79	3.75
Heap Leach	HL Feed	Mt	1.73							1.7													
	Cu head grade	%Cu	0.20							0.20													
	Au head grade	g/t Au	0.32							0.32													
	Ag head grade	g/t Ag	2.60							2.60													
TOTAL	Operating Days	days	6,572			365	366	365	365	365	366	365	365	365	365	365	365	365	365	365	365	365	365
	Daily Mill Feed Rate	t/day	19,815			2,707	21,773	21,910	21,910	21,915	21,858	21,907	21,912	21,918	21,918	21,918	21,918	21,918	21,918	21,918	21,918	21,918	3,512
Flotation	Flotation Circuit Feed	Mt	130.2			1.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	1.3
	Cu head grade	%Cu	0.21			0.46	0.34	0.25	0.24	0.21	0.22	0.21	0.21	0.20	0.20	0.22	0.20	0.18	0.17	0.17	0.17	0.18	0.18
	Au head grade	g/t Au	0.31			0.76	0.53	0.41	0.35	0.31	0.38	0.39	0.38	0.35	0.35	0.43	0.32	0.22	0.21	0.21	0.21	0.18	0.11
	Ag head grade	g/t Ag	1.89							0.92							1.35	1.97	1.87	1.82	2.23	2.79	3.75
Heap Leach	Operating Days	days	2,190		365	365	365	365	365	365													
	Daily Heap Leach Rate	t/day	6,520		2,052	8,219	8,219	8,219	5,970	6,438													
	Heap Leach Feed Tonnes	Mt	14.3		0.7	3.0	3.0	3.0	2.2	2.4													
	Cu head grade	%Cu	0.20		0.28	0.27	0.22	0.20	0.17	0.20													
	Au head grade	g/t Au	0.31		0.72	0.49	0.36	0.40	0.29	0.31													
	Ag head grdae	g/t Ag	1.91							1.91													
Combined Flotation+Heap	Combined Feed Tonnes	Mt	144.5		0.75	3.99	10.97	11.00	10.18	10.35	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	1.28
	Cu head grade	%Cu	0.21		0.28	0.32	0.31	0.24	0.23	0.20	0.22	0.21	0.21	0.20	0.20	0.22	0.20	0.18	0.17	0.17	0.17	0.18	0.18
	Au head grade	g/t Au	0.33		0.72	0.56	0.48	0.41	0.34	0.31	0.38	0.39	0.38	0.35	0.35	0.43	0.32	0.22	0.21	0.21	0.21	0.18	0.11
	Ag head grade	g/t Ag	1.89							1.15							1.35	1.97	1.87	1.82	2.23	2.79	3.75
FLOTATION CIRCUIT																							
Combined Recovery	Cu recovery	% of Cu	86.6			87.4	87.4	87.4	87.4	86.5	87.4	87.4	87.4	87.4	87.4	87.4	85.8	85.3	85.3	85.3	85.3	85.3	85.3
	Au recovery	% of Au	62.8			60.7	60.7	60.7	60.7	63.0	60.7	60.7	60.7	60.7	60.7	60.7	64.8	66.2	66.2	66.2	66.2	66.2	66.2
	Ag recovery	% of Ag	48.97			-	-	-	-	23.1	-	-	-	-	-	-	41.1	55.5	55.5	55.5	55.5	55.5	55.5
Combined Conc. Grade	Cu grade of concentrate	% Cu	25.1			25.10	25.10	25.10	25.10	24.72	25.10	25.10	25.10	25.10	25.10	25.10	24.43	24.20	24.20	24.20	24.20	24.20	24.20
	Au grade of concentrate	g/dmt Au	27.5			28.80	27.17	28.59	25.42	26.53	30.11	32.37	31.54	30.51	30.51	34.07	29.99	22.95	23.20	23.20	23.20	18.78	11.48
	Ag grade of concentrate	g/dmt Ag	125.15			0.00	0.00	0.00	0.00	29.38	0.00	0.00	0.00	0.00	0.00	0.00	79.96	172.33	173.20	168.57	206.55	244.06	328.03
	Moisture content	$\%H_2O$	8.0			8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0
Combined Conc. Tonnes	Cu Conc. Produced - Dry	dmt	964,808			15,825	94,345	69,615	66,831	58,081	61,284	58,470	58,484	55,713	55,713	61,284	55,701	50,757	47,937	47,937	47,937	50,757	8,134
	Cu Conc. Produced - Wet	wmt	1,041,992			17,091	101,893	75,185	72,177	62,728	66,187	63,147	63,163	60,170	60,170	66,187	60,157	54,818	51,772	51,772	51,772	54,818	8,785
	Cu Conc. % of Feed	% dmt	0.74			1.60	1.18	0.87	0.84	0.73	0.77	0.73	0.73	0.70	0.70	0.77	0.70	0.63	0.60	0.60	0.60	0.63	0.63
Combined Flot. Conc. Metal	Cu in Cu flotation concentrate	Mlb Cu	528			8.8	52.2	38.5	37.0	31.7	33.9	32.4	32.4	30.8	30.8	33.9	30.0	27.1	25.6	25.6	25.6	27.1	4.3
		tonnes Cu	239,296			3,972	23,681	17,473	16,774	14,360	15,382	14,676	14,680	13,984	13,984	15,382	13,610	12,283	11,601	11,601	11,601	12,283	1,968
	Au in Cu flot conc.	oz Au	853,332			14,655	82,434	63,994	54,629	49,543	59,334	60,865	59,319	54,650	54,650	67,141	53,719	37,464	35,761	35,761	35,761	30,652	3,002
	Ag in Cu flot conc.	oz Ag	1,808,608			-	-	-	-	54,871	-	-	-	-	-	-	143,211	281,248	266,971	259,833	318,367	398,315	85,793
HEAP LEACH /SART RECOVERY																							
Combined Recovery	SART Cu recovery	% of Cu	13.4		14.3	14.3	14.3	14.3	14.3	8.8													
	Leach Au recovery	% of Au	77.2		78.4	78.4	78.4	78.4	78.4	71.4													
	Leach Ag recovery	% of Ag	28.6		-	-	-	-	-	28.6													
SART Conc. Grade	SART Cu concentrate grade	% Cu	70.0		70.0	70.0	70.0	70.0	70.0	70.0													
	Moisture content	$\%H_2O$	8.0		8.0	8.0	8.0	8.0	8.0	8.0													
Combined SART Conc. Tonnes	Cu Conc. Produced - Dry	dmt	5,992		428	1,655	1,348	1,226	757	578													
	Cu Conc. Produced - Wet	wmt	6,471		463	1,787	1,456	1,324	817	624													
	Cu Conc. % of Feed	% dmt	0.04		0.06	0.06	0.04	0.04	0.03	0.02													
Combined SART Conc. Metal	Cu in SART concentrate	Mlb Cu	9.2		0.7	2.6	2.1	1.9	1.2	0.9													
		tonnes Cu	4,194		300	1,158	944	858	530	405													
Combined Leached Metal	Au in dore	oz Au	140,746		13,595	37,057	27,226	30,251	15,930	16,687													
	Ag in dore	oz Ag	3,427		-	-	-	-	-	3,427													
NET SMELTER RETURN																							
Metal Price	Cu Price	US$/lb	2.75		2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75
	Au Price	US$/oz	1,000		1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000
	Ag Price	US$/oz	16		16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16
Payable Metal Copper	Total Cu in flot. and SART conc.	Mlb	536.8		0.7	11.3	54.3	40.4	38.1	32.6	33.9	32.4	32.4	30.8	30.8	33.9	30.0	27.1	25.6	25.6	25.6	27.1	4.3
	Cu Deduction	%	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu Payable	%	97.0		97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0	97.0
	Payable Cu	Mlb	520.7		0.6	11.0	52.7	39.2	37.0	31.6	32.9	31.4	31.4	29.9	29.9	32.9	29.1	26.3	24.8	24.8	24.8	26.3	4.2
		tonnes	236,186		291	4,977	23,886	17,781	16,785	14,322	14,921	14,236	14,239	13,564	13,564	14,921	13,202	11,915	11,253	11,253	11,253	11,915	1,909
Gold	Total Au in dore and Cu conc.	oz	994,078		13,595	51,713	109,660	94,245	70,559	66,231	59,334	60,865	59,319	54,650	54,650	67,141	53,719	37,464	35,761	35,761	35,761	30,652	3,002
	Flotation Au Payable	%	-		98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0	98.0
	Flotation Au Payable	oz	836,265		-	14,362	80,786	62,714	53,536	48,553	58,147	59,647	58,133	53,557	53,557	65,798	52,644	36,714	35,046	35,046	35,046	30,039	2,942
	Dore Au Payable	%	-		100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Dore Au Payable	oz	140,746		13,595	37,057	27,226	30,251	15,930	16,687	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total Au Payable	oz	977,011		13,595	51,420	108,011	92,965	69,466	65,240	58,147	59,647	58,133	53,557	53,557	65,798	52,644	36,714	35,046	35,046	35,046	30,039	2,942
Silver	Total Ag in dore and Cu conc.	oz	1,812,035		-	-	-	-	-	58,297	-	-	-	-	-	-	143,211	281,248	266,971	259,833	318,367	398,315	85,793
	Flotation Ag Payable	%	5.0		97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5	97.5
	Flotation Ag Payable	oz	1,763,393		-	-	-	-	-	53,499	-	-	-	-	-	-	139,631	274,216	260,297	253,337	310,407	388,357	83,648
	Dore Ag Payable	%	-		100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Dore Ag Payable	oz	3,427		-	-	-	-	-	3,427	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total Ag Payable	oz	1,766,820		-	-	-	-	-	56,926	-	-	-	-	-	-	139,631	274,216	260,297	253,337	310,407	388,357	83,648
Smelter Payables	Cu Payables from Smelter/Refinery	M$	1,432		2	30	145	108	102	87	90	86	86	82	82	90	80	72	68	68	68	72	12
	Au Revenue from Smelter	M$	977		14	51	108	93	69	65	58	60	58	54	54	66	53	37	35	35	35	30	3
	Ag Revenue from Smelter/Refinery	M$	28		-	-	-	-	-	1	-	-	-	-	-	-	2	4	4	4	5	6	1
	Revenue from Smelter	M$	2,437		15.4	81.6	252.8	200.8	171.2	153.0	148.6	146.0	144.5	135.8	135.8	156.3	134.9	113.3	107.4	107.3	108.2	108.5	15.9
Offsite Costs	Total Conc. transport costs	M$	39.4		0.02	0.71	3.88	2.87	2.74	2.38	2.48	2.37	2.37	2.26	2.26	2.48	2.26	2.06	1.94	1.94	1.94	2.06	0.33
	Treatment Charge Cu Concentrate	M$	48.5		0.02	0.87	4.78	3.54	3.38	2.93	3.06	2.92	2.92	2.79	2.79	3.06	2.79	2.54	2.40	2.40	2.40	2.54	0.41
	Refining charge Cu	M$	26.0		0.03	0.55	2.63	1.96	1.85	1.58	1.64	1.57	1.57	1.50	1.50	1.64	1.46	1.31	1.24	1.24	1.24	1.31	0.21
	Refining charge Au	M$	5.4		0.07	0.28	0.59	0.51	0.38	0.36	0.32	0.33	0.32	0.29	0.29	0.36	0.29	0.20	0.19	0.19	0.19	0.17	0.02
	Refining charge Ag	M$	0.6		-	-	-	-	-	0.02	-	-	-	-	-	-	0.05	0.10	0.09	0.09	0.11	0.14	0.03
	Subtotal TC/RC	M$	80.6		0.13	1.71	8.01	6.01	5.61	4.89	5.03	4.82	4.81	4.58	4.58	5.07	4.58	4.15	3.92	3.92	3.94	4.15	0.66
	Cu TC/RC and transport	M$	113.9		0.07	2.13	11.30	8.37	7.97	6.89	7.19	6.86	6.86	6.54	6.54	7.19	6.50	5.91	5.58	5.58	5.58	5.91	0.95
	Au TC/RC and transport	M$	5.4		0.07	0.28	0.59	0.51	0.38	0.36	0.32	0.33	0.32	0.29	0.29	0.36	0.29	0.20	0.19	0.19	0.19	0.17	0.02
	Au TC/RC and transport	M$	0.6		-	-	-	-	-	0.02	-	-	-	-	-	-	0.05	0.10	0.09	0.09	0.11	0.14	0.03
	Cu NSR Contribution	M$	1,318.0		1.7	28.0	133.5	99.4	93.8	79.9	83.3	79.4	79.5	75.7	75.7	83.3	73.5	66.3	62.6	62.6	62.6	66.3	10.6
	Au NSR Contribution	M$	971.6		13.5	51.1	107.4	92.5	69.1	64.9	57.8	59.3	57.8	53.3	53.3	65.4	52.4	36.5	34.9	34.9	34.9	29.9	2.9
	Au NSR Contribution	M$	27.7		-	-	-	-	-	0.9	-	-	-	-	-	-	2.2	4.3	4.1	4.0	4.9	6.1	1.3
	NSR (excluding royalties)	M$	2,317		15.2	79.2	240.9	191.9	162.9	145.7	141.1	138.8	137.3	129.0	129.0	148.7	128.1	107.1	101.6	101.5	102.4	102.3	14.9
	Cu Royalties	M$	32.9		0.0	0.7	3.3	2.5	2.3	2.0	2.1	2.0	2.0	1.9	1.9	2.1	1.8	1.7	1.6	1.6	1.6	1.7	0.3
	Au Royalties	M$	24.3		0.3	1.3	2.7	2.3	1.7	1.6	1.4	1.5	1.4	1.3	1.3	1.6	1.3	0.9	0.9	0.9	0.9	0.7	0.1
	Ag Royalties	M$	0.7		-	-	-	-	-	0.0	-	-	-	-	-	-	0.1	0.1	0.1	0.1	0.1	0.2	0.0
	Total Royalties (2.5%)	M$	57.9		0.4	2.0	6.0	4.8	4.1	3.6	3.5	3.5	3.4	3.2	3.2	3.7	3.2	2.7	2.5	2.5	2.6	2.6	0.4
	Offsite Costs (including royalties)	M$	177.9		0.53	4.39	17.91	13.68	12.42	10.91	11.04	10.66	10.62	10.06	10.06	11.27	10.04	8.89	8.40	8.40	8.44	8.77	1.36
	Copper NSR (inc. royalties)	M$	1,285.0		1.7	27.3	130.2	96.9	91.4	77.9	81.2	77.5	77.5	73.8	73.8	81.2	71.7	64.7	61.1	61.1	61.1	64.7	10.4
	Gold NSR (inc. royalties)	M$	947.3		13.2	49.9	104.7	90.1	67.4	63.3	56.4	57.8	56.4	51.9	51.9	63.8	51.0	35.6	34.0	34.0	34.0	29.1	2.9
	Silver NSR (inc. royalties)	M$	27.0		-	-	-	-	-	0.9	-	-	-	-	-	-	2.1	4.2	4.0	3.9	4.7	5.9	1.3
Net Smelter Return	TOTAL NSR (including royalties)	M$	2,259		14.8	77.2	234.9	187.1	158.8	142.1	137.6	135.3	133.8	125.7	125.7	145.0	124.9	104.5	99.0	98.9	99.8	99.7	14.5
OPERATING COST																							
Unit OPEX	Mining	$/t mined	1.36		2.45	1.56	1.24	1.29	1.27	1.25	1.39	1.34	1.29	1.28	1.28	1.49	1.37	1.37	1.37	1.58	1.58	1.58	1.58
		$/t ore	2.92		8.70	2.54	2.32	2.39	2.74	3.21	3.64	4.02	3.56	3.63	3.82	2.40	3.83	2.54	2.21	2.25	2.31	2.08	1.58
	Flotation process	$/t milled	4.30		4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30	4.30
	Heap Leach/SART	$/t leached	4.31		4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31	4.31
	Tailings deposition (N&S milled tonnes or	$/t milled	0.15		0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28
	G&A	$/t milled/leached	0.68		0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68	0.68
Total OPEX	Mining	M$	421.8		6.5	10.1	25.4	26.3	27.9	33.2	29.1	32.1	28.5	29.1	30.5	19.2	30.6	20.4	17.7	18.0	18.5	16.6	2.0
	Flotation process	M$	560.0		-	4.2	34.3	34.4	34.4	34.4	34.4	34.4	34.4	34.4	34.4	34.4	34.4	34.4	34.4	34.4	34.4	34.4	5.5
	Heap Leach/SART	M$	61.5		3.2	12.9	12.9	12.9	9.4	10.1	-	-	-	-	-	-	-	-	-	-	-	-	-
	Tailings deposition (N&S milled tonnes o	M$	22.3		-	0.3	2.2	2.2	2.2	1.3	2.2	2.2	2.2	2.2	2.2	2.2	0.6	-	-	-	-	-	-
	G&A and Tailings Deposition	M$	98.3		0.5	2.7	7.5	7.5	6.9	7.0	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	0.9
1.00	Total OPEX	M$	1,164		10.3	30.3	82.3	83.3	80.8	86.1	71.2	74.2	70.5	71.1	72.6	61.3	71.0	60.2	57.5	57.9	58.3	56.5	8.4
	Unit OPEX per t processed	$/t processed	8.05		13.69	7.60	7.50	7.58	7.94	8.32	8.90	9.28	8.82	8.89	9.08	7.66	8.88	7.52	7.19	7.23	7.29	7.06	6.56
Cost/Payable Metal	Unit OPEX per Cu equivalent	$/lb Eq. Cu payable	1.31		1.84	1.02	0.90	1.14	1.30	1.55	1.32	1.40	1.34	1.44	1.47	1.08	1.45	1.46	1.47	1.48	1.48	1.43	1.46
	Unit OPEX per Au equivalent	$/oz Eq. Au payable	478		668	372	326	415	472	563	479	508	488	524	535	392	527	531	536	539	539	520	531
NET OPERATING INCOME		M$	1,096		5	47	153	104	78	56	66	61	63	55	53	84	54	44	41	41	42	43	6
CAPITAL COST																							
	Mining equipment fleet	M$	77.6	24.0	5.2	23.0					6.8	6.9	8.4	3.3									
	Roads and Mining Infrastructure	M$	15.7	15.7																			
	Electrical power line and generators	M$	14.2	14.2																			
	Process plant	M$	124.0	24.0	100.0																		
	Heap Leach Pad	M$	17.3	17.3																			
	Tailings Management Facility	M$	19.9	10.0	9.9																		
	Owners Costs	M$	9.6	4.1	5.5																		
	EPCM	M$	28.2	12.2	16.0																		
	Closure	M$	9.0																				9.0
	Contingency	M$	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%
	Contingency	M$	31.5	12.1	13.7	2.3	-	-	-	-	0.7	0.7	0.8	0.3	-	-	-	-	-	-	-	-	0.9
	Working CAPEX	M$	-		3.4																		- 3.4
1.00	TOTAL CAPITAL COST	M$	346.9	133.5	153.7	25.3	-	-	-	-	7.4	7.6	9.3	3.6	-	-	-	-	-	-	-	-	6.5
EBITDA																							
	EBITDA	M$	749	(134)	(149)	22	153	104	78	56	59	53	54	51	53	84	54	44	41	41	42	43	(0)
5.0%	Discounted EBITDA	M$	412	(134)	(142)	20	132	85	61	42	42	36	35	31	31	47	29	22	20	19	18	18	(0)
	Discounted Cumulative EBITDA	M$		(134)	(276)	(256)	(124)	(39)	22	64	106	142	177	208	239	286	315	337	357	376	394	412	412

19.10.4 Sensitivity Analysis

Sensitivity analyses were conducted for each case by individually modifying the capital cost, operating cost, metal price and grade up and down by 20% to show the sensitivity of the EBITDA net present value using a 5% discount rate (“$NPV_{5\%}$”).

The results of the sensitivity analyses show that the project is most sensitive to metal price and mill feed grade. For Case B, a 20% increase in gold and copper price (or mill feed grade) leads to a $311M (75%) increase in pre-tax $NPV_{5\%}$ from $412 M to $723 M. The converse occurs if the metal price or mill feed grade drops by 20%, the pre-tax $NPV_{5\%}$ drops from $412 M to $100 M.

Operating costs are the next most sensitive parameter. In the base case, a 20% increase in operating costs reduces the pre-tax $NPV_{5\%}$ by $149 M (43%). For capital costs, a 20% increase results in a $64 M (18%) drop in $NPV_{5\%}$.

The Case B economic model shows that the project breaks even if *both* the capital costs and operating costs are increased by 40% from the base case estimates.

A summary of the sensitivity analysis is shown in Table 19.28 and Figure 19.27.

Table 19.27: Sensitivity Analysis Results

Case	Variable	EBITDA $NPV_{5\%}$ (M$)		
		-20% Variance	0% Variance	20% Variance
Case A	Capital Cost	412	347	283
	Operating Cost	497	347	198
	Metal Price or Grade	49	347	646
Case B (Base Case)	Capital Cost	477	412	347
	Operating Cost	561	412	262
	Metal Price or Grade	100	412	723
Case C	Capital Cost	851	786	721
	Operating Cost	935	786	636
	Metal Price or Grade	400	786	1172

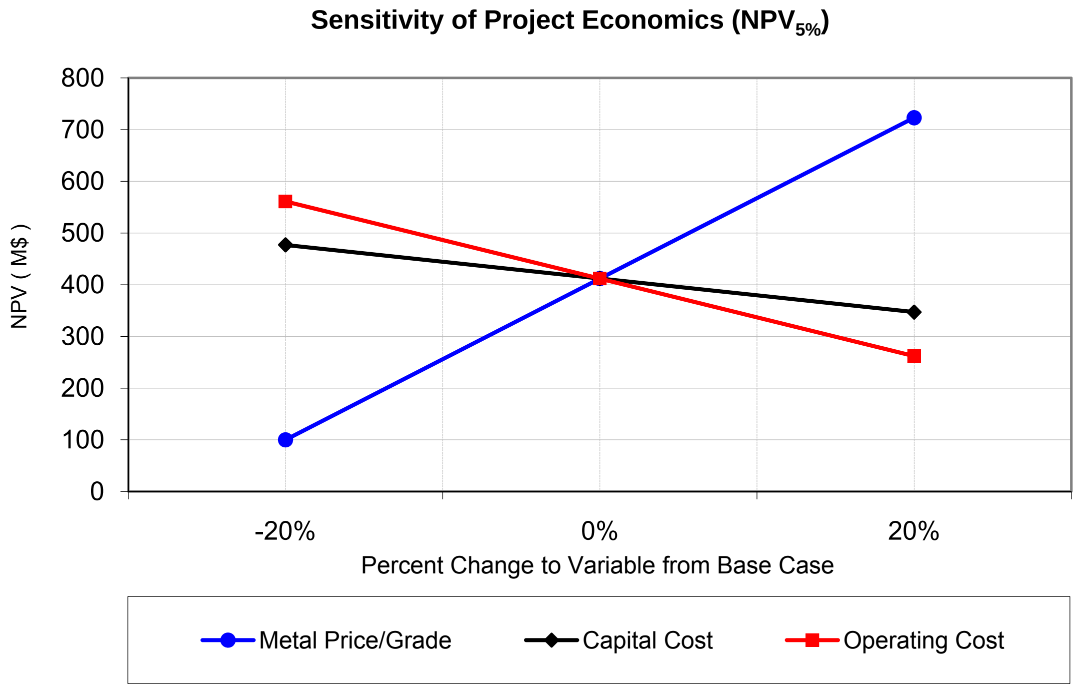

Figure 19.27: Case B Sensitivity Analysis Results

Further EBITDA NPV and IRR sensitivities were run for a variety of gold and copper prices with the results shown in Table 19.29.

Table 19.28: $NPV_{5\%}$ and IRR Results for Varying Metal Prices

Gold Price ($/oz)	Copper Price ($/lb)				
	2.50	2.75	3.00	3.25	3.50
5% Discount Rate EBITDA Net Present Value (M$)					
800	201	283	365	447	528
900	266	348	429	511	593
1,000	330	412	494	575	657
1,100	395	476	558	640	722
1,200	459	541	623	704	786
EBITDA Internal Rate of Return (%)					
800	14	17	20	23	25
900	17	20	23	25	28
1,000	20	22	25	28	30
1,100	22	25	27	30	32
1,200	25	27	30	32	34

Base Case

19.11 Payback Period

Payback period for the base case at a discount rate of 0% is approximately four years after commissioning. The payback estimates do not include capital expenditures prior to construction.

19.12 Mine Life

The life of mine is a little over 18 production years based on a flotation circuit capacity of 8.0 Mt/y (22,000 t/d) and a heap leach capacity of 3.0 Mt/y (8,200 t/d). The first year of production includes only heap leach operations with the flotation circuit starting up in the second year. The heap leach concludes operation after six years while the flotation plant continues to operate until the end of the mine life. One year of pre-production construction including infrastructure and heap leach facilities was assumed. The flotation plant was assumed to be completed at the end of the first year of heap leach operation.

20 Interpretation and Conclusions

Industry standard mining, process design, construction methods and economic evaluation practices have been used to assess the Tepal Project. The conclusion reached is that there is adequate geological and other pertinent data available to generate a PA.

Based on current knowledge and assumptions, the results of this study show that the project has positive economics (within the very preliminary parameters of a PA) and should be advanced to the next level of study by conducting the work indicated in the recommendations section.

As with almost all mining ventures, there are a large number of risks and opportunities that can influence the outcome of the Tepal project. Most of these risks and opportunities are based on a lack of scientific information (test results, drill results, etc.) or the lack of control over external drivers (metal price, exchange rates, etc.). The following section identifies the most significant potential risks and opportunities currently identified for the Tepal project, almost all of which are common to mining projects at this stage of study.

Subsequent higher-level engineering studies will need to further refine these risks and opportunities, identify new ones and define mitigation or opportunity implementation plans.

While a significant amount of information is still required to do a complete assessment, at this point there do not appear to be any fatal flaws for the project.

The study achieved its original objective of providing a preliminary review of the potential economic viability of the Tepal project.

20.1 Risks

As with most early-stage projects there are a multitude of risks that could influence the economic potential of the project. Many of these risks are based on lack of knowledge and can be managed with appropriate engineering and additional studies. External risks are beyond the control of project proponents and are much harder to anticipate and mitigate although, in many instances, some risk reduction can be achieved. Tables 20.1 and 20.2 identify some of the more internal and external significant project risks, potential severity and possible mitigation approaches.

Table 20.1: Internal Project Risks

Risk	Explanation	Potential Impact	Possible Risk Mitigation
Process Costs and Recoveries	Flotation recoveries are largely based on results from just two composites, one from the North Zone and one from the South Zone.	If life-of-mine recovery of copper or gold is lower than projected, project economics would be negatively impacted.	Conduct a flotation variability study to determine how material from different areas responds and the average copper and gold recoveries are likely to be.
	The oxide ore contains significant cyanide-soluble copper, which leaches along with the gold.	The copper leaching increases cyanide consumption and complicates gold recovery, raising processing costs.	In conjunction with the next column leach program, initiate tests to determine the operating parameters for a SART plant to recover the copper and regenerate the cyanide.
	Gold recovery by leaching has only been determined on ore with a 12.5 mm (1/2-in.) top size or less.	If gold recovery from coarser material proves to be lower than projected, a ROM ore leach may not be economically viable.	Column leach tests covering a wider range of top sizes should be initiated to guide selection of the optimum top size for the leach material.
Ability to Acquire Water	The region of the property is classified as a Warm-Dry Forest and the sources of water for the operation have not been well defined. However, the project is located on the immediate margin of the Tepalcatepec Basin, one of the largest water basins in Mexico	Failure to secure an adequate water supply could reduce the size of the operation and impact economics due to possible competition with agricultural usage in the project area.	Investigations on water sources need to continue and be documented in the next level of study. The design of water conservation measures in the plant will assist in the reduction of demand for water during the Dry period
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs would be an important element of success	As shown in the sensitivity analysis, an increase in CAPEX and/or OPEX would have a negative effect on the project economics.	Further cost accuracy with the next level of study as well as the active investigation of potential cost-reduction measures
Permit Acquisition	The ability to secure a mining permit is of paramount importance.	Failure to secure a mining permit would stop the project.	The development of close relationship with the communities and government along with a thorough ESIA and a project design that gives appropriate consideration to the environment and local people is required.

Project Development Schedule	The project development schedule as shown in the production schedule and economic model is very aggressive and would require permitting, financing and further studies to continue as planned and have no major issues arise.	A change in schedule would alter the project economics once project construction and mining commences.	If the schedule is to be followed Geologix needs to immediately embark on a PFS and the associated full field program including definition drilling
Water Management and Geochemistry	It has been assumed that the waste dumps and tailings ponds do not need to be lined and acid rock drainage and metal leaching ("ARD/ML") will not be a problem. No water treatment facility was budgeted.	If ARD/ML testing indicates that that geochemistry will be an issue the liners may have to be placed under the TMF and/or the WRFs. This would add CAPEX and OPEX costs to the project..	Adequate testing of tailings and waste materials needs to be done to determine if there is an ARD/ML issue.
Inability to upgrade inferred resources to measured or indicated	The PA mine plan uses roughly 60% inferred resources which cannot be used at a higher level of study	If none of the inferred resources can be upgraded to indicated then the mineable quantity would be about half of what is presented here and would likely make the project uneconomic	A well planned definition drilling campaign, renewed geostatistical analysis and resource estimation needs to be undertaken to determine the amount of inferred resource that can be converted
TMF Location and Stability	The TMF site needs to be fully engineered to ensure it is in an appropriate location and be able to sustain an appropriate seismic event.	The TMF location may have to change if underfoot conditions are not suitable. This may lead to increased TMF construction and/or operating costs.	The TMF could be moved to a different site or its design changed to improve stability. Various types of land use requires need to be assessed prior to the final site determination.
Inclusion of pre-development costs	All predevelopment costs such as exploration and resource definition costs, engineering field work and studies costs, environmental baseline studies costs, etc. were not included in the economic evaluation.	Considerable expenditures must be undertaken to advance the project to the next level. Typical pre-development expenditures for this size of project would be in the tens of millions of dollars and would reduce the overall project NPV if they were included.	Estimate the pre-development costs at the next level of study and assess their impact on the project economics.
Inclusion of taxation and financing costs	No taxation of financing costs were included in the project economic evaluation.	Taxation and financing costs will reduce the NPV of the project.	Include taxation and financing costs in the next level of study.

Table 20.2: External Project Risks

Risk	Explanation	Potential Outcome	Possible Risk Mitigation
Metal prices	Gold and copper prices have a significant impact on the economic viability of the project.	In the base case, a 20% drop in copper and gold prices reduces the EBITDA $NPV_{5\%}$ by about 75%.	Current strong demand for copper and gold make it possible to forward sell production to reduce the risk of metal price volatility. This can be done for all or a portion of production.
Regional Political Stability	Mexico in the past has enjoyed a fairly stable mining environment. Should this situation change , the project could be impacted	Potential for increased costs.	Close involvement and communication with local governments and increased security measures may be advantageous.
Earthquakes	The project is located in a seismically active area which could impact the stability of infrastructure, open pits and building	A significant earthquake could create a number of problems for the site from power failure to destruction of buildings, equipment and infrastructure. The current TMF design is the most susceptible to seismic activity of the options reviewed.	Appropriate design locations and standards must be adhered to should the project reach the construction phase to ensure all design work and building practices reasonably consider the potential impact of an earthquake.
Securing Finance	The project will require a JV partner, purchase from a larger producing company or extensive bank financing (or a combination of the above).	Failure to secure funding could slow the project or stop its development altogether	Continued value-adding field work including additional resource development and technical studies as well as developing a financing plan if the project continues to develop are needed
Recruiting Experienced Professionals for the development and operating teams	The selection of appropriate, experienced people for the project will be important to its success	The inability of the company to retain a skilled development and operating team could have a negative impact on project timing, costs and overall success	The early search for the ideal people would be required along with appropriate compensation and benefits

20.2 Opportunities

Table 20.3: Project Opportunities

Opportunity	Explanation	Potential Benefit
Metal prices	Gold and copper prices have a significant impact on the economic viability of the project.	In the base case, every 1% increase in copper and gold price increase the EBITDA $NPV_{5\%}$ by about \$15M.
Exploration potential	Favourable exploration potential in the area could increase resources and might have a positive impact on the project mineral resources	Increased resources would lead to a potentially better project economics if they could be converted to reserves in the future. The more economic tonnes available to mine the better the project economics would be as total revenues would increase, potentially without adding more capital cost.
Inclusion of silver in North and South Zone resource estimate	Silver contribution from the North and South deposits has not been taken into account in this study.	The accounting of the silver contained in the North and South zones may add incremental value to the project.
Inclusion of molybdenum recovery from the Tizate deposit	No economic value of Mo from the Tizate zone was included in this study.	The recovery of Mo from the Tizate deposit may add an incremental benefit to the project economics.
Mill throughput optimization	Based on the mine plan presented in this report, the project has an 18 year life which may not provide optimal project economics.	Increasing the mill throughput may lower operating costs and potentially add tonnes to the mine plan from lower operating costs. An increased plant capacity would increase capital costs but these may be offset by lower operating costs.

20.3 Recommendations

20.3.1 General Recommendations

- As per the Howe 2009 recommendations, a drilling program should be undertaken to improve the quality and reliability of future resources estimates and develop additional resources for the project;
- At the current drill spacing over the deposit, continuous mineralized zones are shown to be continuous, however there can be significant grade variability within the Tepal North and South zones and further infill drilling is warranted both to provide additional sample data to facilitate more meaningful geostatistical analysis and to upgrade currently defined inferred resources to indicated resources.
- Ensure logging procedures are maintained during Phase 2 activities so as to have consistency with Phase 1 practices.
- Develop the delineation of the weathering profile over the deposit in order to more reliably domain the geological model into zones of oxide, mixed and sulphide material.

- Following Phase 2 activities, the characteristics of gold and copper grade distribution should be assessed in the light of new data, and modelled separately if required.
- Implement the practise of orientated drill core for improved geotechnical and structural logging measurements, particularly as controls on mineralization are structural. Consistency of geotechnical measurements is improved with the use of the orientation reference line. A system such as EzyMark provides a reliable easy to use means of obtaining oriented drill core.
- Ensure non biased core sampling through routine submittal of same half of core, achievable through use of orientation reference line.
- Develop the use of QA/QC samples, ensuring that adequate field duplicates and CRMs are submitted.
- Continued bulk density determination of half core samples to build up the density database for use in future estimations.
- Multi-element grade domain modelling for improved single element domain geostatistical analysis and restricted grade interpolation.
- Improved geological modelling to include the interpretation of host geology, breccia, stockwork and alteration zones to domain assay data for improved geostatistical analysis and wireframe restricted grade interpolation.
- The cost of the resource definition drilling is estimated to be $4.0M and require approximately 22,500 m of drilling.
- PFS – This phase is contingent upon the conversion of a large percentage of Inferred resources to Indicated or Measured categories. The estimated cost of the PFS, including field work but excluding metallurgical testing and resource definition drilling detailed elsewhere in this section is expected to be $1.5M.;
- Continued work on the environmental baseline study.

20.4 Recommendations for Geotech Work

- As the project geotechnical evaluation is upgraded, it will be important to develop 3D models for both structural geology and alteration. The alteration model solids should be developed such that it includes the type of alteration and the intensities of those alteration types.
- It will be beneficial to future studies if the level of geotechnical data measures from the exploration and infill drilling programs is increased. Parameters such as Intact Rock Strength (IRS) and joint counts should be included. The IRS measurements should include estimates of IRS (hard), IRS (weak) and percentage weak.

21 Illustrations

All illustrations are included in the body of the report and in the Appendices.

22 Acronyms, Abbreviations and Definitions

Distance	
µm	micron (micrometre)
mm	millimetre
cm	centimetre
m	metre
km	km
” or in	inch
’ or ft	foot
Area	
ac	acre
ha	hectare
Time	
s	second
m or min	minute
h or hr	hour
d	day
y or yr	year
Volume	
l	litre
usg	US gallon
lcm	loose cubic metre
bcm	bank cubic metre
Mbcm	million bcm
Mass	
kg	kilogram
g	gram
t	metric tonne
Kt	kilotonne
lb	pound
Mt	megatonne
oz	troy ounce
wmt	wet metric tonne
dmt	dry metric tonne
Pressure	
psi	pounds per square inch
Pa	pascal

Unit Prefixes	
µ	micro (one millionth)
m	milli (one thousandth)
c	centi (one hundredth)
d	deci (one tenth)
k or K	kilo (one thousand)
M	Mega (one million)
G	Giga (one trillion)
Temperature	
oC	degree Celsius (Centigrade)
oF	degree Fahrenheit
Misc.	
Btu or BTU	British Thermal Unit
Ø	diameter
r	radius
hp	horsepower
s.g.	specific gravity
masl	metres above sea level
elev	elevation above sea level
Rates and Ratios	
p or /	per
mph	miles per hour
cfm	cubic feet per minute
usgpm	United States gallon per minute
tph	tonnes per hour
tpd	tonnes per day
mtpa	million tonnes per annum
ppm	parts per million
ppb	parts per billion
Acronyms	
SRK	SRK Consulting (Canada) Inc.
CIM	Canadian Institute of Mining
NI 43-101	National Instrument 43-101
ABA	acid- base accounting
AP	acid potential
NP	neutralization potential

kPa	kilopascal
MPa	megapascal
Elements and Compounds	
Au	gold
Ag	silver
As	arsenic
Cu	copper
Fe	iron
Mo	molybdenum
Pb	lead
S	sulphur
Zn	zinc
CN	cyanide
NaCN	sodium cyanide
Electricity	
kW	kilowatt
kWh	kilowatt hour
V	volt
W	watt
Ω	ohm
A	ampere

ML/ARD	metal leaching/ acid rock drainage
PAG	potentially acid generating
non-PAG	non-potentially acid generating
RC	reverse circulation
DD / DDH	diamond drill / diamond drill hole
IP	induced polarization
HL	heap leach
COG	cut off grade
NSR	net smelter return
NPV	net present value
LOM	life of mine
EBITDA	earnings before interest, taxation, depreciation and amortization
IRR	internal rate of return
DR	discount rate
PEA	preliminary economic assessment
PFS	preliminary feasibility study
FS	feasibility study
Conversion Factors	
1 tonne	2,204.6 lb
1 troy ounce	31.1035

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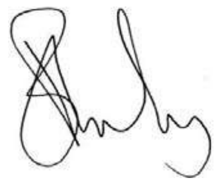
24 Date and Signature Page

This technical report was written by the Qualified Persons listed below. The effective date of this technical report April 29, 2011.

Qualified Person	Signature	Date
Bruce Murphy		April 29, 2011
Dino Pilotto, P.Eng.		April 29, 2011
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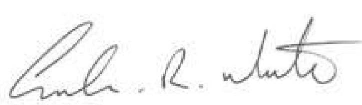
Gilles Arseneau		April 29, 2011

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CERTIFICATE OF QUALIFIED PERSON

Bruce Murphy, FSAIMM

I, Bruce Murphy, a Fellow of the South African Institute of Mining and Metallurgy, am employed as a Principal Consultant – Rock Mechanics with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Revised Tepal Project Preliminary Economic Assessment Technical Report - Tepal and Tizate Deposits" submitted on April 29, 2011.

I am a Fellow of the South African Institute of mining and Metallurgy. I graduated with a MSc.Eng (Mining) degree from the University Witwatersrand, in May 1996.

I have been involved in mining since 1990 and have practised my profession continuously since then. I have been involved in mining operations, mining related rock mechanics and consulting covering a wide range of mineral commodities in Africa, South America North America and Asia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have visited the Tepal site on 9th and 10th July 2010

I am responsible for the Slope Design Review and Section 18 of "Revised Tepal Project Preliminary Economic Assessment Technical Report – Tepal and Tizate Deposits", submitted on April 29, 2011.

I am independent of Geologix Explorations Inc. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Tepal Project since July 2010 doing the Slope Design Review.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Bruce Murphy, FSAIMM Dated: April 29, 2011

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Dino Pilotto, P.Eng.

I, Dino Pilotto, am a Professional Engineer, employed as a Principal Consultant - Mining with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Revised Tepal Project Preliminary Economic Assessment Technical Report – Tepal and Tizate Deposits" submitted on April 29, 2011 with an effective date of April 29, 2011.

I am a member of the Association of Professional Engineers and Geoscientists of Saskatchewan and Alberta. I graduated with a B.A.Sc. (Mining & Mineral Process Engineering) from the University of British Columbia in May 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, and Africa.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have visited the Tepal site on July 9th and 10th, 2010.

I am responsible for Sections 19.1, 19.2 and the mining part of 19.9 of "Revised Tepal Project Preliminary Economic Assessment Technical Report – Tepal and Tizate Deposits", submitted on April 29, 2011 with an effective date of April 29, 2011.

I am independent of Geologix Explorations Inc. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Tepal Project since 2010 participating in various independent studies.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

PROFESSIONAL ENGINEER
D.G. PILOTTO
MEMBER 14782
2011-04-29
YR. MN. DAY
SASKATCHEWAN

Dino Pilotto, P.Eng. Dated: April 29, 2011

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Europe	Vancouver	604.681.4196	Fort Collins	970.407.8302
North America	Yellowknife	867.445.8670	Reno	775.828.6800
South America			Tucson	520.544.3688

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vancouver@srk.com
www.srk.com

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CERTIFICATE OF QUALIFIED PERSON

Gilles Arseneau, Ph.D., P. Geo.

I, Gilles Arseneau, am a Professional Geoscientist, employed as an Associate Geological Consultant with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Revised Tepal Project Preliminary Economic Assessment Technical Report, Tepal and Tizate Deposits" submitted on April 29, 2011.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I graduated with B.Sc. in Geology from the University of New Brunswick, 1979; a M.Sc. in Geology from the University of Western Ontario, 1984 and a Ph.D. in Geology from the Colorado School of Mines, 1995

I have been involved in mining since 1979 and have practised my profession continuously since 1995. I have been involved with exploration projects and consulting covering a wide range of mineral commodities in Africa, South America North America and Asia including deposits similar to Tepal in Mexico and South America. I have over ten years experience in resource estimation using Gemcom software.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have visited the Tepal site on March 12 and 13, 2011.

I am responsible for the Sections 6 to 13, Section 14.2 and Section 17 of "Revised Tepal Project Preliminary Economic Assessment Technical Report, Tepal and Tizate Deposits", submitted on April 29, 2011.

I am independent of Geologix Explorations Inc. as independence is described by Section 1.4 of NI 43-101.

I have not previously been involved with the Tepal Project.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Gilles Arseneau, Ph.D., P.Geo.

Dated: April 29, 2011

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CERTIFICATE OF QUALIFIED PERSON

Gordon Doerksen, P.Eng.

I, Gordon Doerksen, am a Professional Engineer, employed as a Principal Consultant - Mining with SRK Consulting (Canada) Inc.

This certificate applies to the technical report titled "Revised Tepal Project Preliminary Assessment Technical Report – Tepal and Tizate Deposits" submitted and effective on April 29, 2011.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia. I graduated with a BS (Mining) degree from Montana College of Mineral Science and Technology in May 1990.

I have been involved in mining since 1985 and have practised my profession continuously since 1990. I have been involved in mining operations, mine engineering and consulting covering a wide range of mineral commodities in Africa, South America North America and Asia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I have not visited the Tepal site.

I am responsible for the Executive Summary and Sections 1 to 5, 15, 19.3, 19.5 to 19.8, 19.10 to 19.12 and 20 to 24 of "Revised Tepal Project Preliminary Assessment Technical Report – Tepal and Tizate Deposits", submitted and effective April 29, 2011.

I am independent of Geologix Explorations Inc. as independence is described by Section 1.4 of NI 43-101.

I previously worked on SRK's first preliminary Assessment of the Tepal project.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.
As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

PROFESSIONAL
PROVINCE OF
G. E. DOERKSEN
32273
BRITISH COLUMBIA
ENGINEER

Gordon Doerksen, P.Eng.

Dated: April 29, 2011

ce bc

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Australia	Toronto	416.601.1445	Elko	775.753.4151
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North America	Yellowknife	867.445.8670	Reno	775.828.6800
South America			Tucson	520.544.3688

Hydrometal, Inc.
(EIN 20-2908823)
Providing Specialized Metallurgical Services to the Worldwide Minerals Industry

CERTIFICATE OF QUALIFIED PERSON

W. Joseph Schlitt, P.Eng., Q.P.

I, W. Joseph Schlitt, am a Professional Engineer, employed as President of Hydrometal, Inc.

This certificate applies to the technical report titled "Revised Tepal Project Preliminary Economic Assessment Technical Report – Tepal and Tizate Deposits" submitted on April 29, 2011.

I am a member of the Mining & Metallurgical Society of America, with Qualified Professional registration in Metallurgy No. 01003QP. I a also a registered member of the Society for Mining, Metallurgy & Exploration and am a Registered Professional Engineer. I graduated with a BS (Metallurgical Engineering) degree from Carnegie Institute of Technology in 1964. I also graduated with a PhD (Metallurgy) degree from the Pennsylvania State University, College of Earth & Mineral Sciences, in 1968.

I have been involved with the minerals industry since 1968 and have practiced my profession continuously since then. I have been involved in mining, mineral processing and metal production covering copper, gold/silver and other nonferrous metals on a world-wide basis.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure of Mineral Projects* (NI 43-101).

I visited the Tepal site on July 9, 10 and 11, 2010. I have also visited both the laboratories doing the testwork.

I am responsible for Section 16 in its entirety, plus Sections 14.1 and 19.4 of "Revised Tepal Project Preliminary Economic Assessment Technical Report – Tepal and Tizate Deposits", submitted on April 29, 2011.

I am independent of Geologix Explorations Inc. as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Tepal Project since September 2009 as the project metallurgist.

I have read National Instrument 43-101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

MMSA
No. 01003QP

W. Joseph Schlitt, P.Eng., Q.P.

Dated: April 29, 2011

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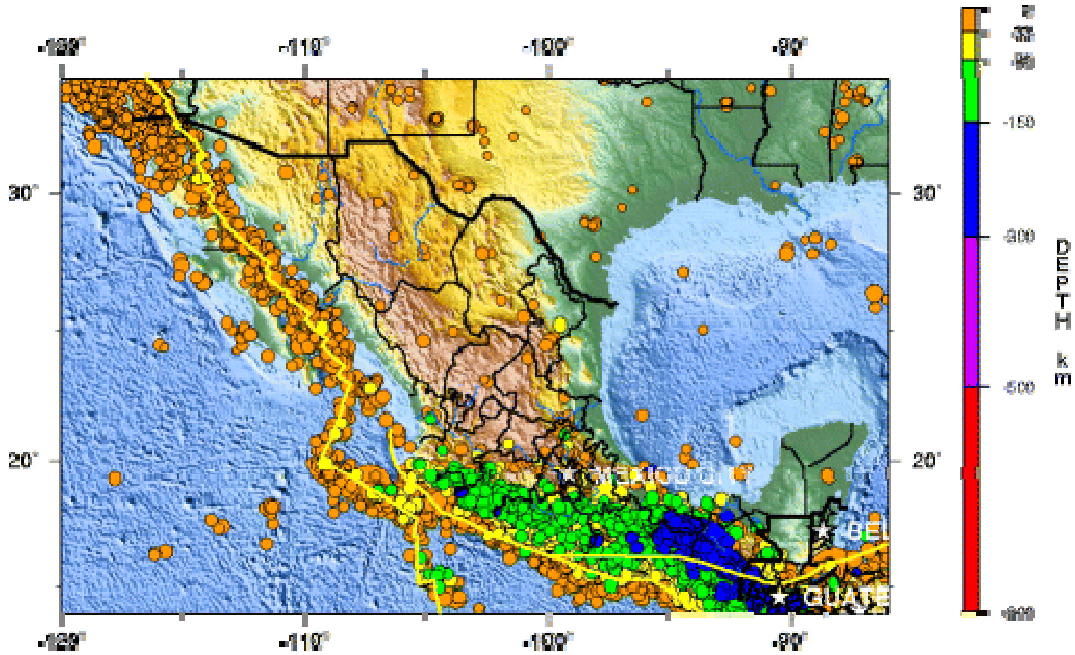
Appendix A: Slope Recommendations

Tepal Property – Slope Design Review

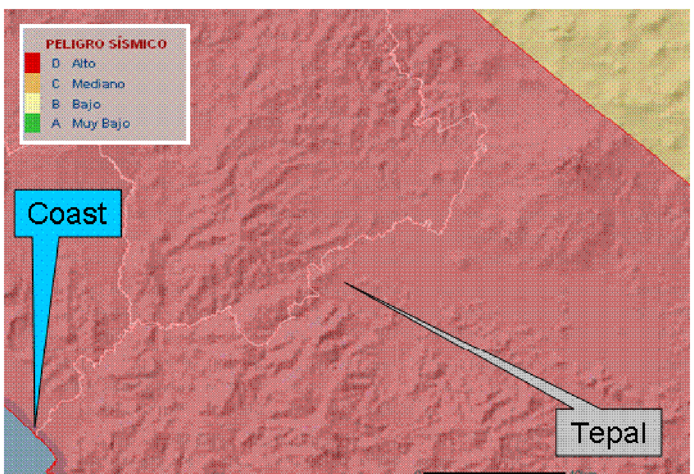
April 2011

Seismicity Potential

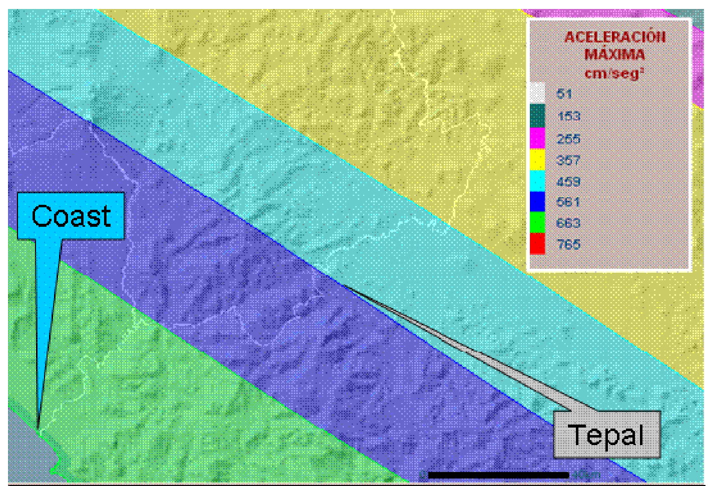
The Tepal property is in a high seismogenic zone and this should considered when planning and costing the various facilities



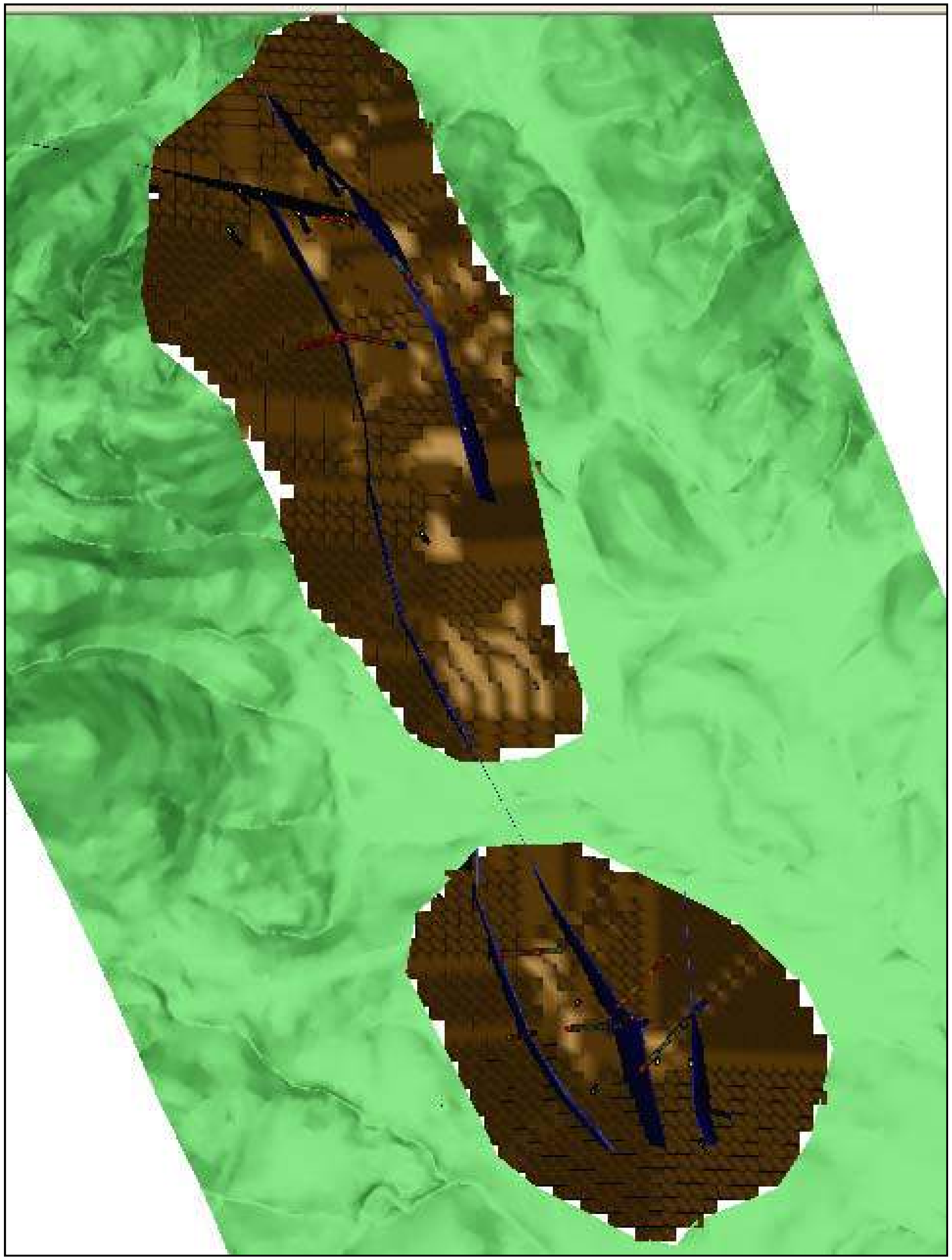
Seismicity of Mexico, 1990 - 2006



Zone	Hazard description	Hazard level	Seismic Factor
D	Large and frequent seisms, ground acceleration >70% of gravity	High	0.8



Id	Number	Return period (years)	Structure period	Max. Acceleration cm/seg²
1	5	500	0.05	459
2	6	500	0.05	561



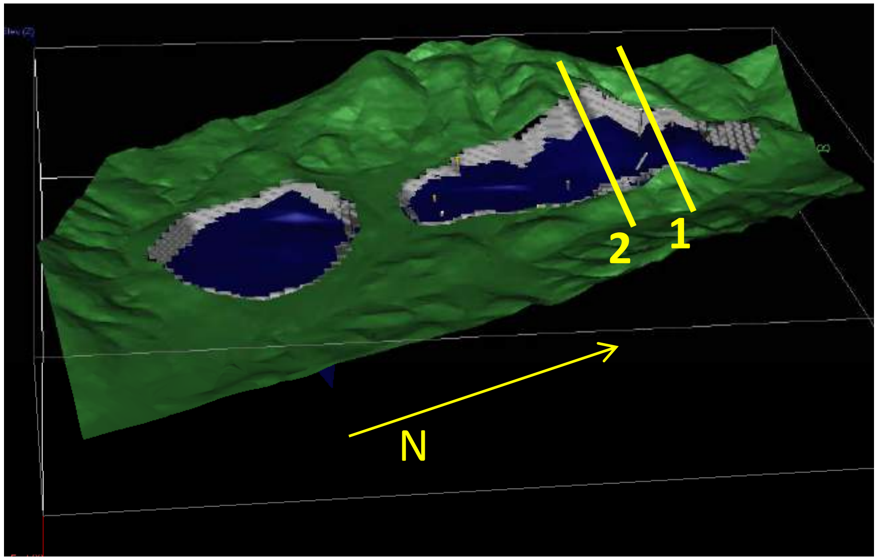
Interpreted Fault Structures Within the North and South Zone

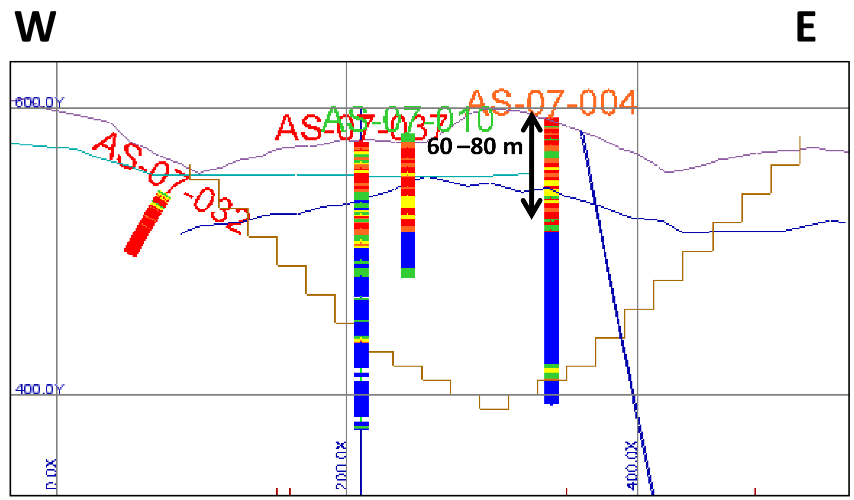
Currently interpreted as largely sub-vertical structures which will likely not have a major impact on slope stability.

These would need to be evaluated in detail at a pre-feasibility level study

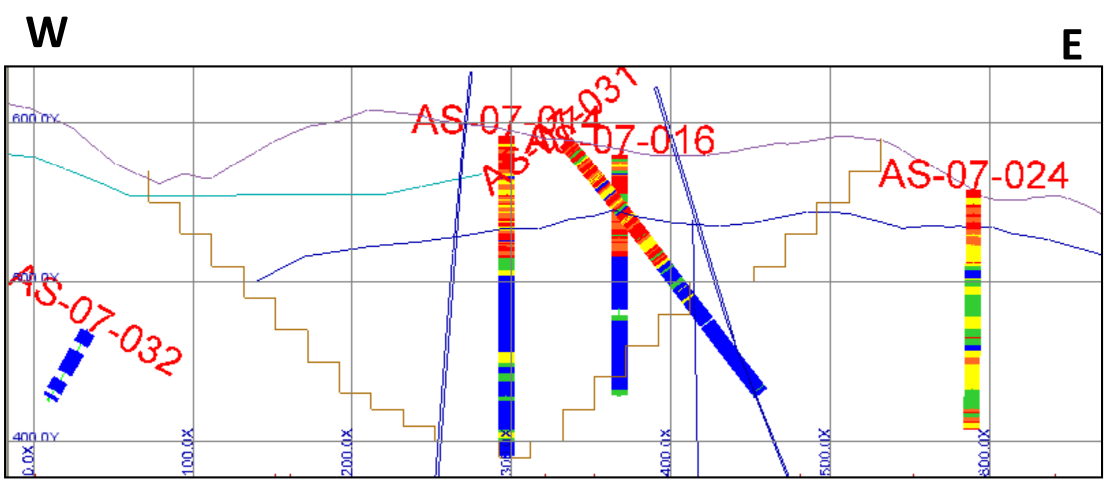
Oxide Zone Surface

The oxide zone generally shown weak ground conditions. These extend below the existing surface down into what may be termed the mixed zone



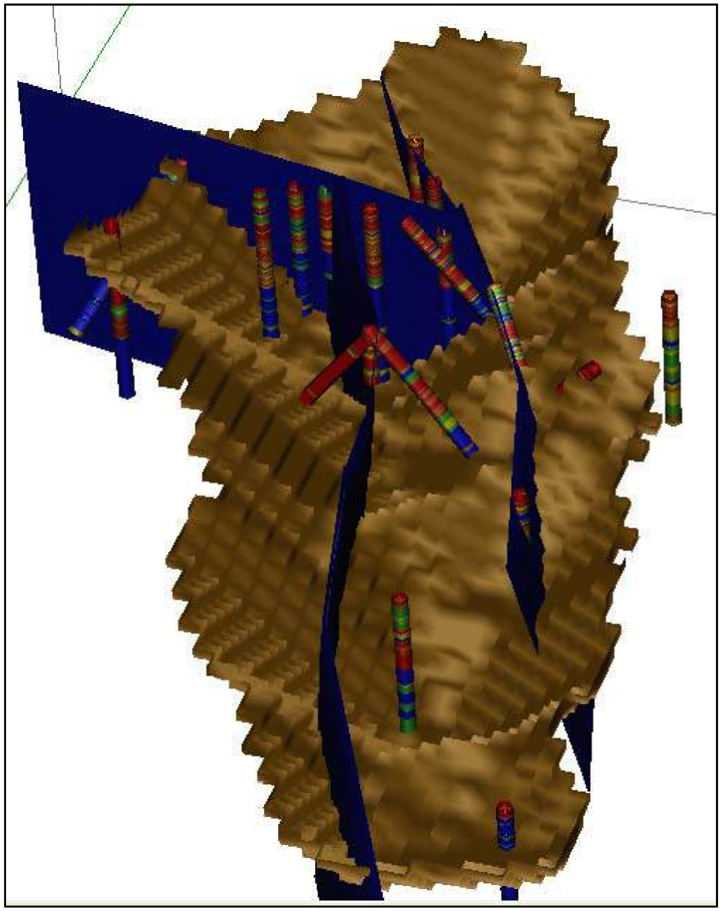


EW Section 1



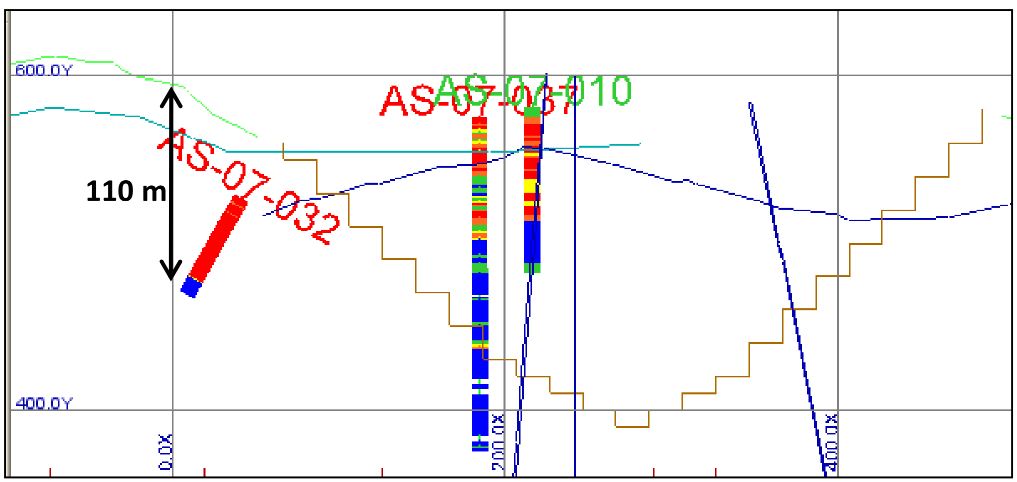
EW Section 2

North Zone Drill Hole RQD



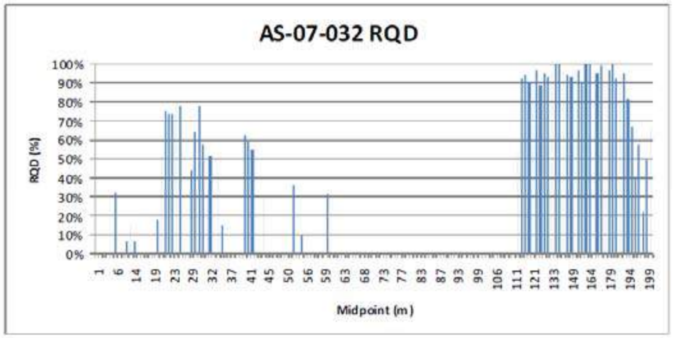
0.00000	20.00000	
20.00000	40.00000	
40.00000	60.00000	
60.00000	80.00000	
80.00000	100.00000	

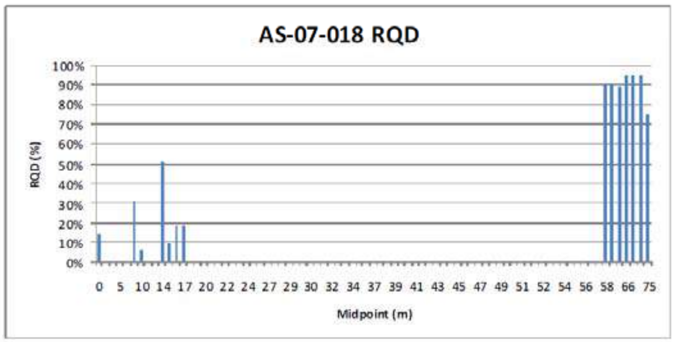
North View of the North zone showing the down-hole RQD

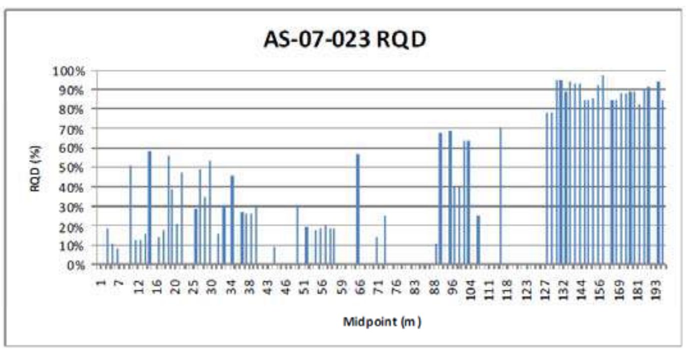


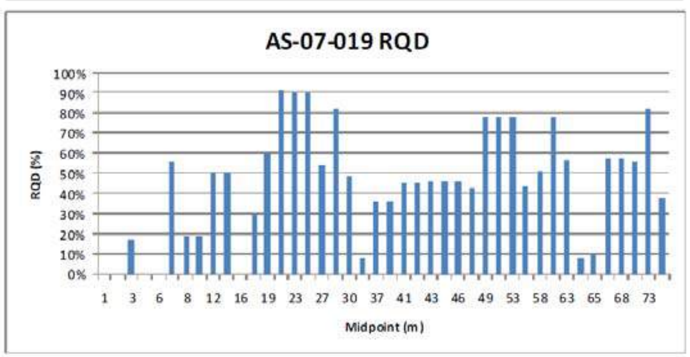
E-W Section through AS07-032 showing the deep alteration

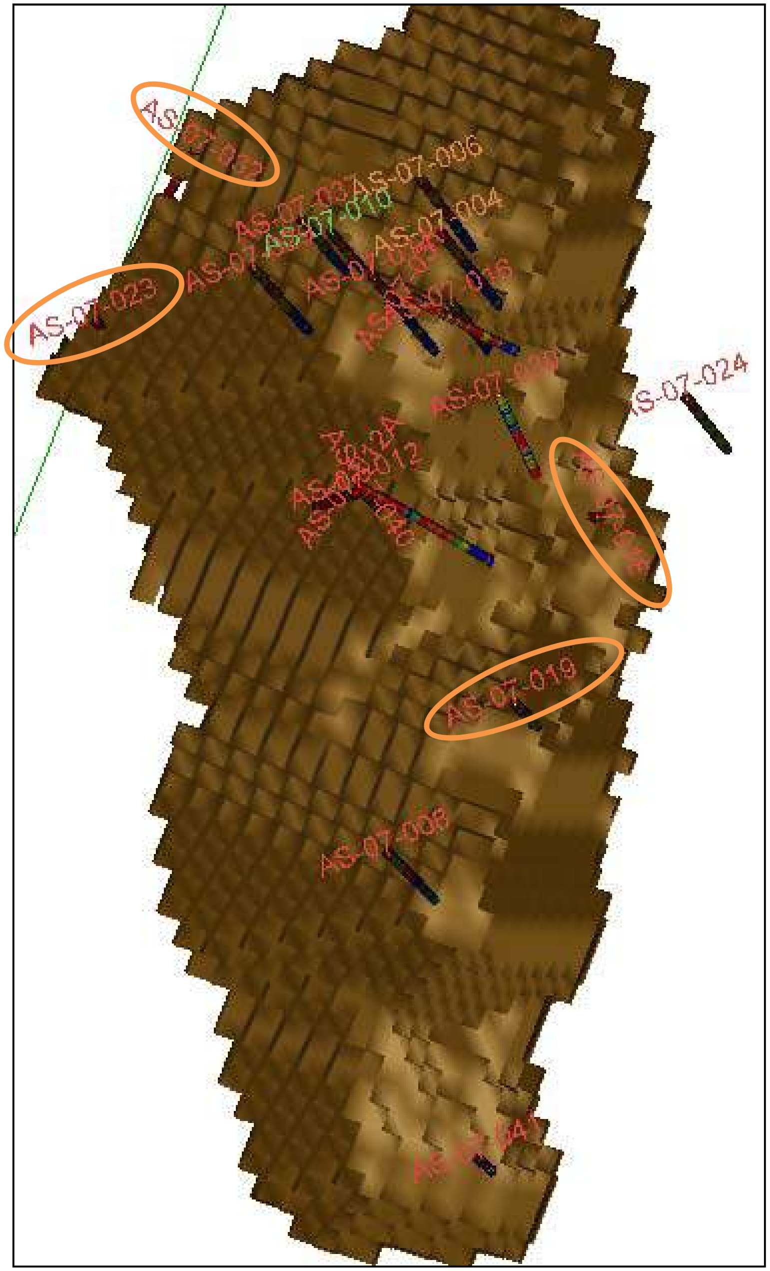
North Zone Drill Hole RQD



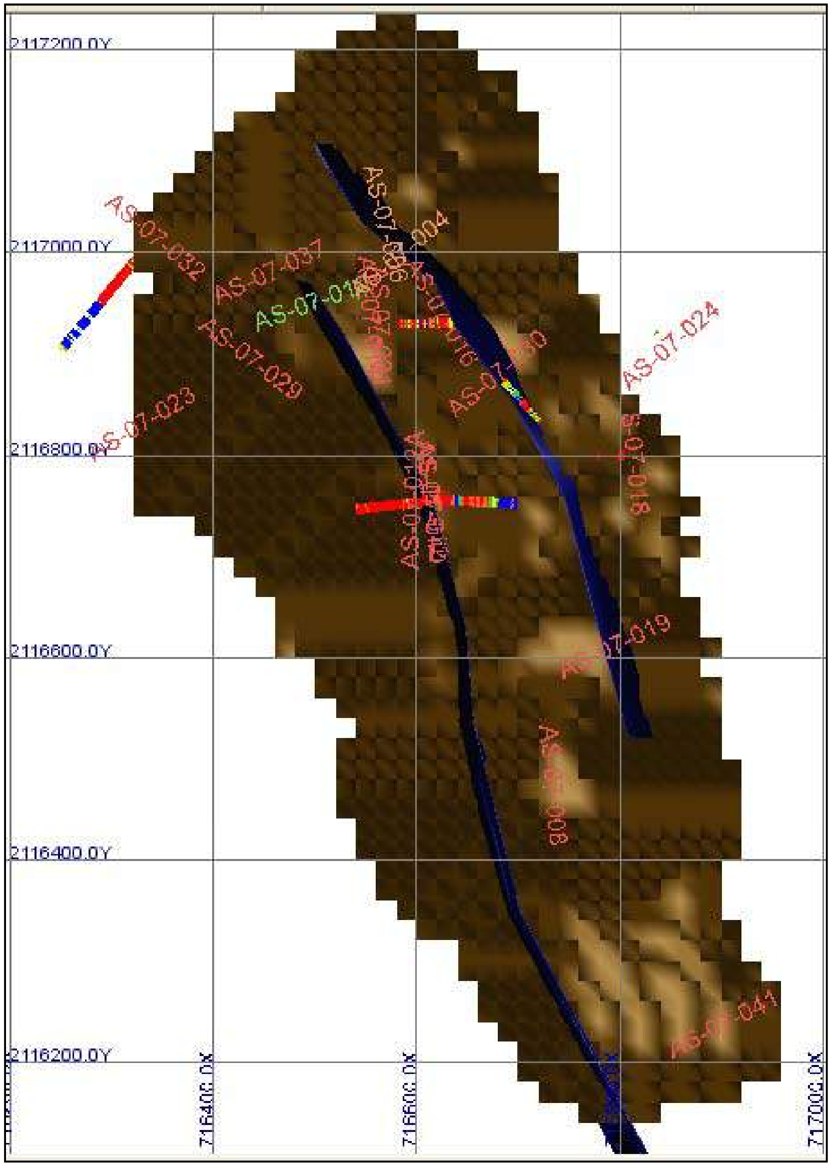








North Zone Slope Angle Recommendations



More of the oxidation and alteration appears to be focused in the north sector of this pit, associated with the better mineralized zone. AS-07-032 shows poor rock mass conditions down to ~110m.

North East Sector
For a scoping level evaluation the assumption is the oxidation will reduce in thickness towards the slope areas. Thus in general the upper 60 m of the pit slope areas will be at 40° and the slope in the better rock mass conditions will be 50°

North West Sector
For the west slope, north of the 2116 600 the upper west weak zone should be taken down to 90 m

South Sector
South of 2 116 600 the upper weak zone can be reduced to 20 m in the area of the slopes.

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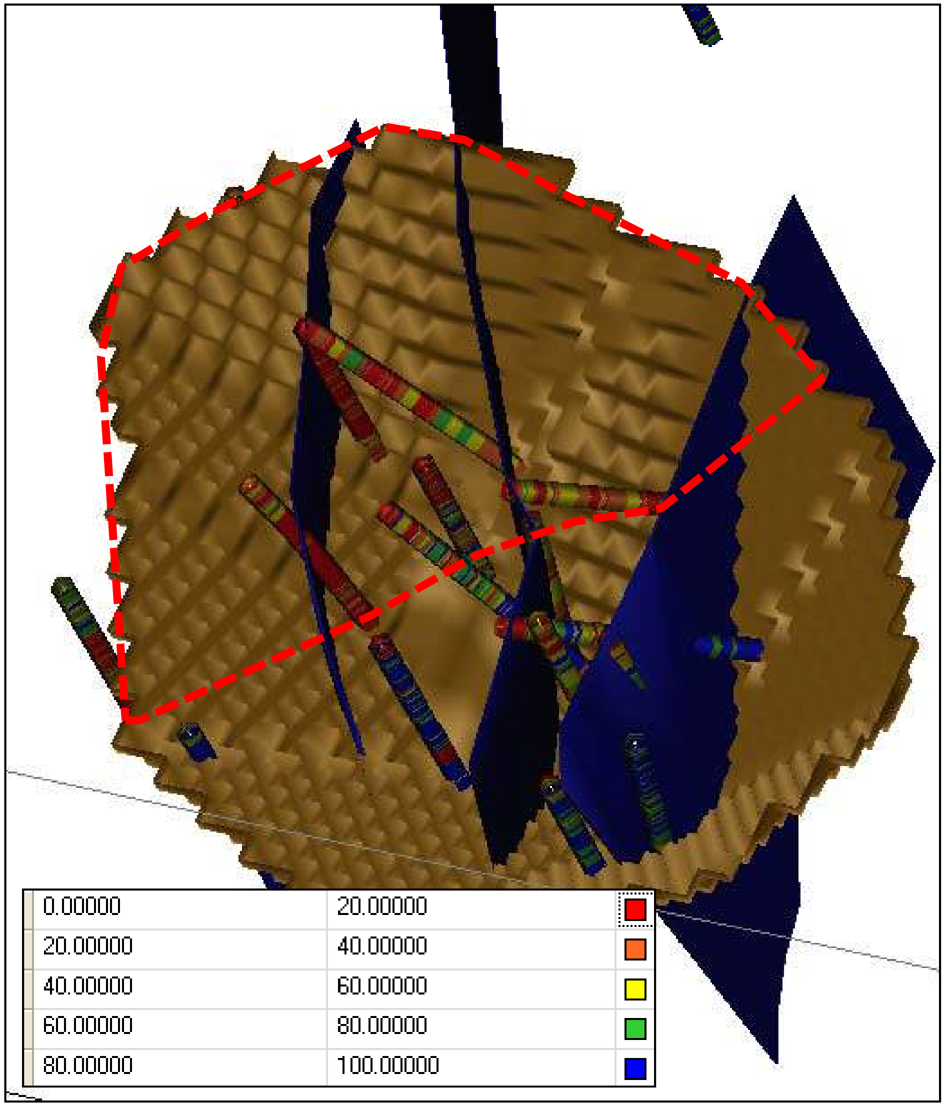
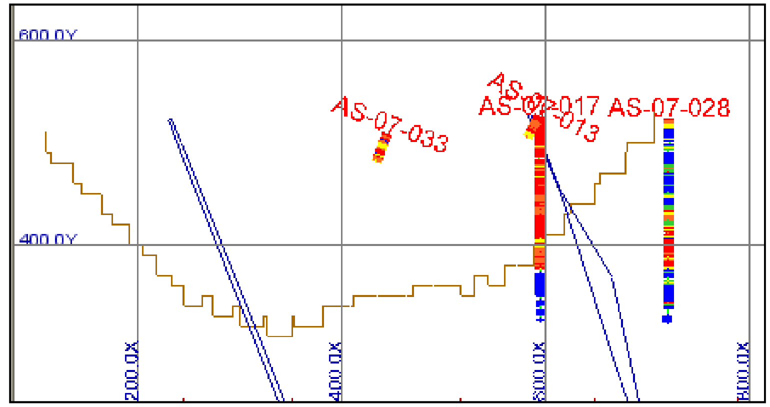
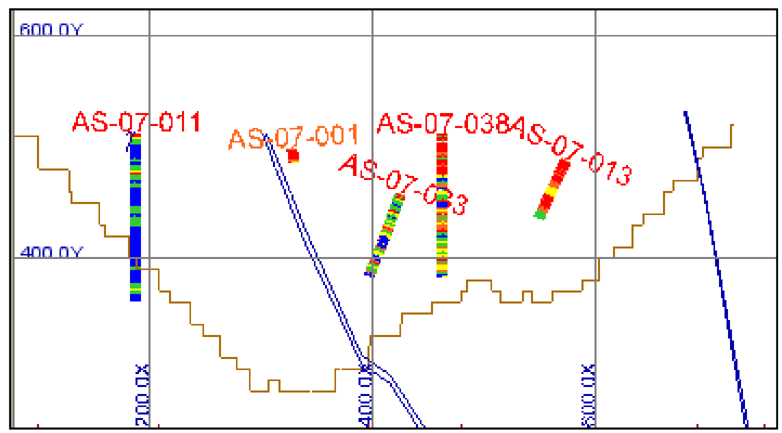
South Zone Drill Hole RQD



North View of the South Zone showing the down-hole RQD and the weak zone on the north of the proposed pit.

SN Sections





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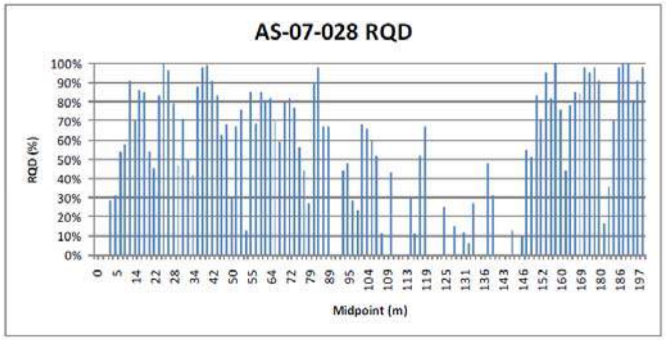
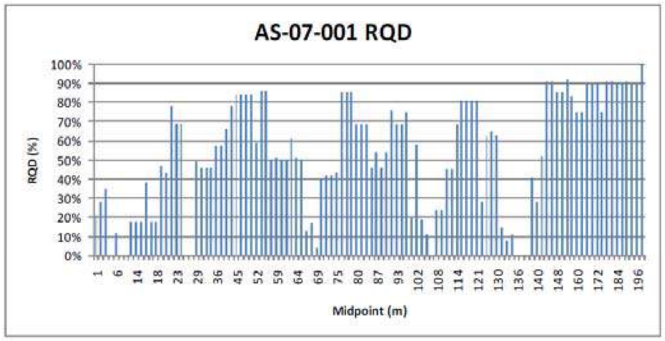
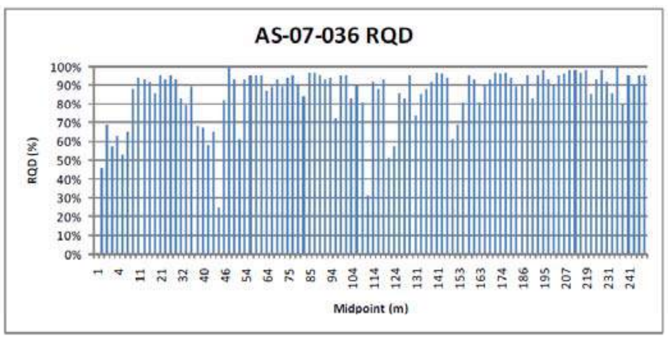
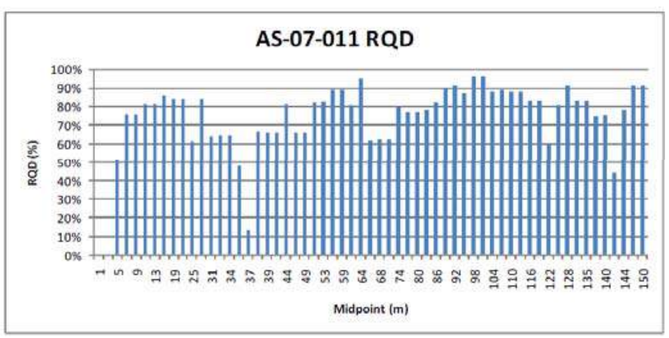
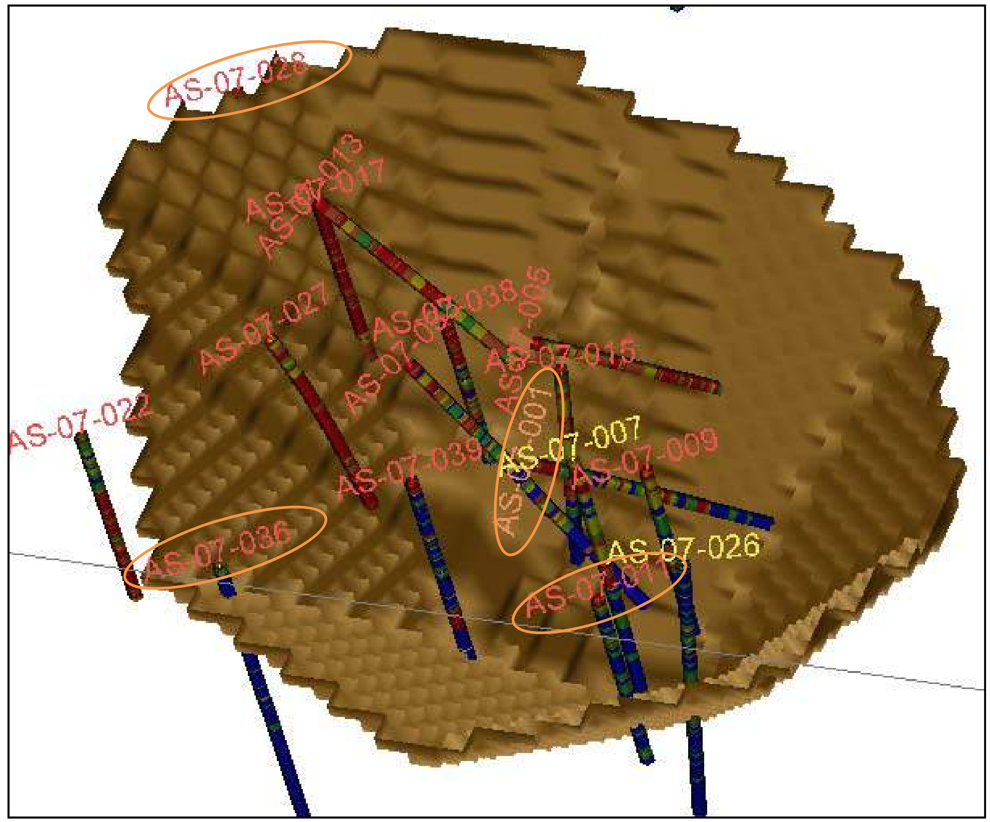
South Zone Drill Hole RQD











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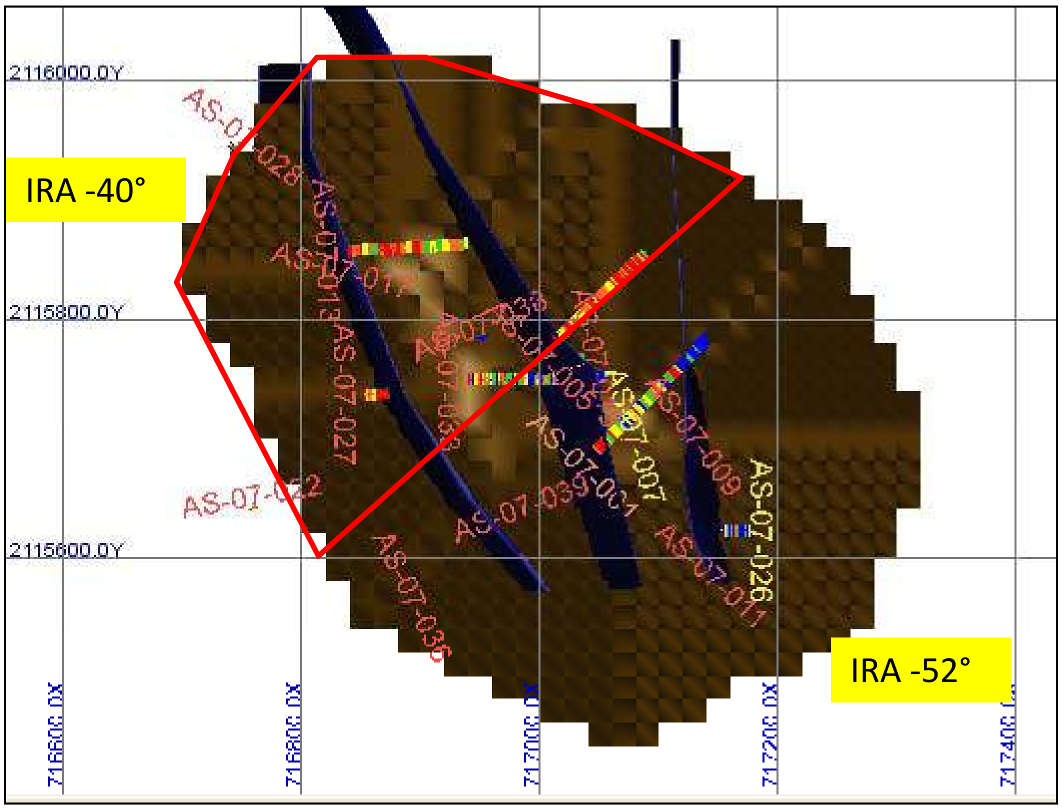
South Zone Slope Angle Recommendations

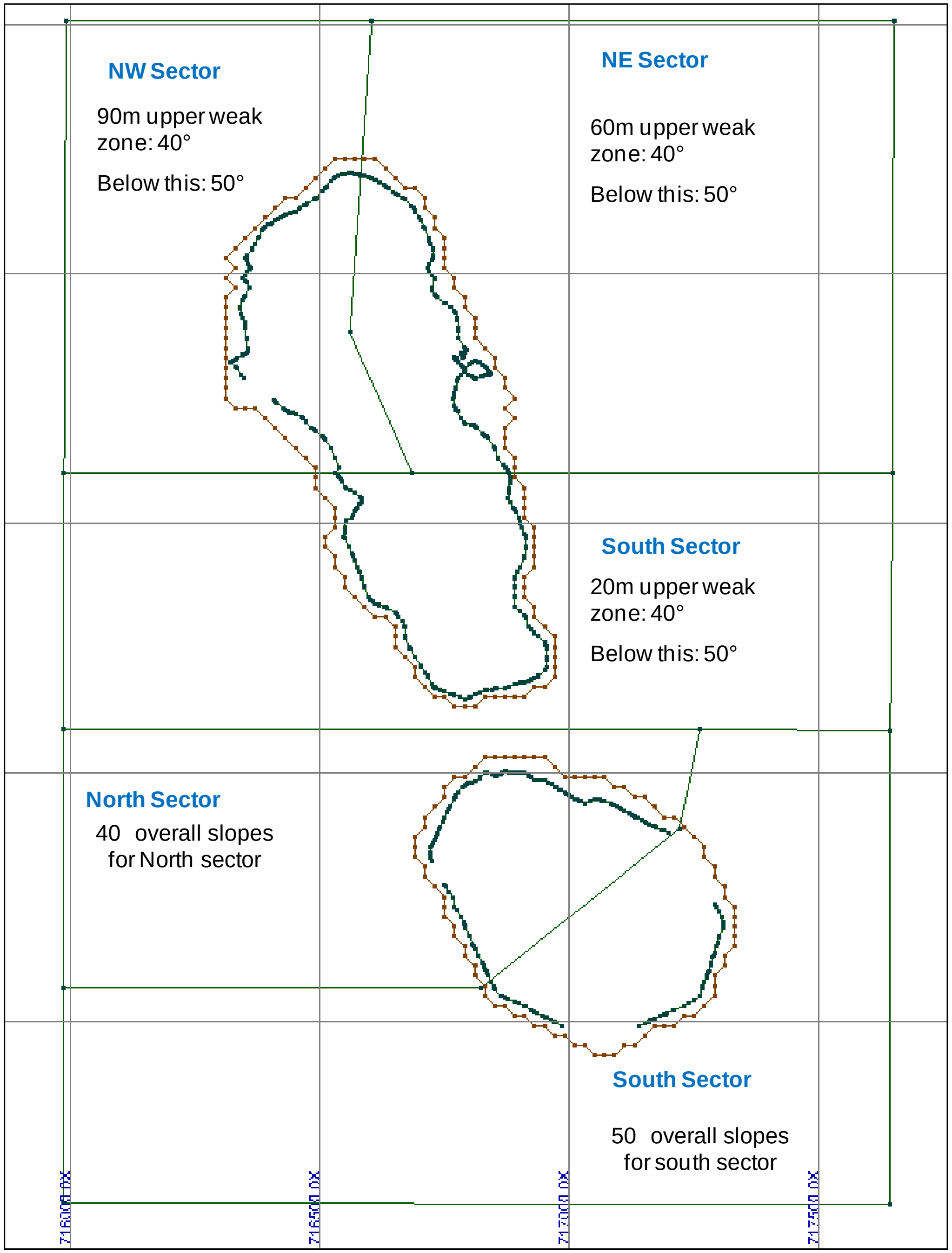


AS07-28 does show a strength increase in the upper area, but lower down in weak again

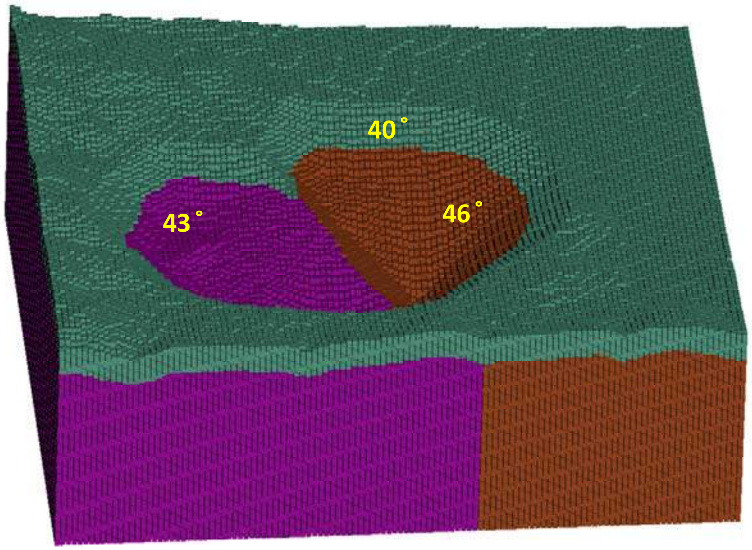
North Sector
Maximum overall slope on the north sector is to be a maximum of 40° . There may be a possibility of increasing the IRA over limited stack heights to 45° for 50 m height to accommodate a ramp.

South Sector
Maximum overall slope angle for the south sector is to be 50°. There may be a possibility of increasing the IRA over limited stack heights to 55° for 50 m height to accommodate a ramp.

Slope Angle Recommendations - Summary



Slope Angles - Tezate



RQD logging undertaken by Geologix was conservative in areas. Re-logging by SRK showed an improvement in the ground conditions in a number of areas.

A general assumption is made that the upper 50 m in all areas is affected by the oxidation process and is weaker ground. In the east area this is up to ~75 m deep

The SE sector shows better ground conditions than the other sectors, with estimated RMR of 50 – 60 and a MRMR of 42 – 50.

Other sectors have an estimated RMR of 40 to 52 and a MRMR of 34 - 42

Slope Recommendation
Upper 50 m in all areas max of 40˚deg
Lower SE Sector IRA = 52˚
Remaining sectors IRA = 48˚
Maximum IRA height is 100 m

For a 200 m high slope angle (including ramp)
SE sector: Maximum overall is 46˚
Other sectors: Maximum overall is 43˚

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