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LA VERDE COPPER PROJECT
MICHOACÁN STATE, MEXICO
TECHNICAL REPORT
for
CATALYST COPPER CORP.

Prepared by AMC Mining Consultants (Canada) Ltd.
In accordance with the requirements of National Instrument 43-101, "Standards of Disclosure for Mineral Project", of the Canadian Securities Administrators

AMC 712013

Effective Date 30 September 2012

Patrick R. Stephenson
Alan Riles
Mo Molavi
Margaret Harder
Michael F. O'Brien

1 SUMMARY

Introduction, Location and Ownership

This Technical Report on the La Verde Copper Property (the Property) in Michoacan, Mexico, includes the current La Verde Project (the Project) and the immediate surrounding land. It has been prepared by AMC Mining Consultants (Canada) Ltd (AMC) of Vancouver, Canada, on behalf of Catalyst Copper Corporation (Catalyst) of Vancouver, Canada.

This report comprises a Preliminary Economic Assessment (PEA) of the Project and is based in part on a preceding document titled "Technical Report on the La Verde Property, Michoacan, Mexico", by Margaret Harder, P.Geo and Qualified Person, and Michael F, O'Brien, P.Geo and Qualified Person, both of Tetra Tech WEI Inc (Tetra Tech), dated 19 September 2012 (the Tetra Tech September 2012 Report), and filed by Catalyst in November 2012.The present report has been prepared in accordance with the requirements of National Instrument 43-101 (NI 43-101), "Standards of Disclosure for Mineral Projects", of the Canadian Securities Administrators (CSA) for lodgment on CSA's "System for Electronic Document Analysis and Retrieval" (SEDAR).

The preliminary economic assessment is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the preliminary economic assessment will be realized.

The La Verde Project is located in the state of Michoacán, approximately 320 km west of Mexico City. The known copper (Cu) deposits on the Property lie 8 km north-east of the town of Nueva Italia de Ruiz and 8 km south-west of the town of Lombardia. Uruapan is the largest city near the Property and is 60 km to the north.

The Project consists of two concessions totalling 16,919 ha situated along the southern margin of the Mexican Neovolcanic Axis and within the Sierra Madre del Sur.

Geology and Mineralization

Most of the Project area is underlain by the northwestern margin of the mid-tertiary Huacana granodiorite/quartz monzonite batholith. This same batholith is host to the San Isidro and Inguaran copper breccia pipes located roughly 20 km and 50 km, respectively, to the southeast of La Verde.

The intrusive complex at La Verde is dominated by quartz-diorite and forms an east- west trending arcuate range roughly 5.5 km long, 1 km wide, and up to 290 m above the surrounding blanket of Quaternary sediments. The range is known as the Sierra del Marqués and is divided into the Cerro La Laguna (West Hill) and Cerro Mina La Verde (East Hill) by a topographic low known as La Puerta located roughly in the centre of the arc.

The East Hill contains four main copper mineralized zones. Three of these zones (#1, #3 and #4) are hosted within altered and brecciated quartz-diorite in close proximity to dykes and stocks of quartz-feldspar porphyry, while the fourth zone (#2) is hosted largely within unbrecciated quartz-feldspar porphyry. All four mineralized zones form a roughly circular pattern on the western half of the East Hill, in a plan view. Alteration tends to be fairly tightly restricted to zones of brecciation, except in the case of large-scale calcium-sodium metasomatism within quartz-diorite adjacent to quartz-feldspar porphyry.

The West Hill is characterized by approximately east-west trending bands of phyllic/propylitic alteration with associated pyrite-chalcopyrite ± arsenopyrite- pyrrhotite veining at the contact between quartz-diorite porphyry and quartz-diorite. The lateral extent of these mineralized east-west striking veins forms a north-northeast south-southwest trending roughly elliptical shaped deposit, in a plan view. The western half of the West Hill consists of “red diorite” stained red by inclusions of hematite. A major north-northwest trending magnetic lineament occupies this region of the West Hill and may reflect a structural break separating red diorite to the west from quartz-diorite to the east. Apart from a few small vein showings, no significant copper-mineralization has been intersected to date within the red diorite.

Exploration

The Project has been explored since 1906. Most of the drilling has been focused on the West Hill and East Hill deposits. Minera Hill 29 S.A. de C.V. (MinHill) conducted an induced polarization (IP) survey and completed drilling on 20 drillholes in 2010. In 2011 an additional 24 drillholes were completed, while the latest 2012 campaign added 12 new holes. Combined with the prior exploration work, the database contains 641 drillholes totalling 114,824 m.

Mineral Resources

Table ES.1 summarizes the La Verde resource as of September 2012 at cut-off grades of 0.10%, 0.20% and 0.30% Cu.

Table ES.1 Mineral Resource Statement

Resource Class	Cut-off (Cu%)	Zone	Tonnes (000s)	Cu (%)	Ag (g/t)	Au (g/t)	As (%)
Measured	0.1	East Hill	57,963	0.37	2.75	0.05	0.02
		West Hill	20,995	0.37	1.95	0.01	0.05
		Total	78,958	0.37	2.54	0.04	0.03
	0.2	East Hill	41,262	0.46	3.30	0.06	0.02
		West Hill	16,265	0.43	2.03	0.01	0.05
		Total	**57,527**	**0.45**	**2.94**	**0.05**	**0.03**
	0.3	East Hill	29,896	0.54	3.68	0.07	0.03
		West Hill	11,194	0.52	2.15	0.01	0.06
		Total	41,090	0.53	3.26	0.05	0.03
Indicated	0.1	East Hill	322,936	0.26	2.32	0.05	0.02
		West Hill	262,669	0.33	1.39	<0.01	0.04
		Total	585,605	0.30	1.91	0.03	0.03
	0.2	East Hill	163,604	0.39	3.31	0.06	0.04
		West Hill	186,838	0.41	1.47	0.01	0.03
		Total	**350,442**	**0.40**	**2.33**	**0.03**	**0.04**
	0.3	East Hill	98,784	0.49	4.04	0.07	0.05
		West Hill	126,505	0.49	1.55	<0.01	0.03
		Total	225,289	0.49	2.65	0.04	0.04
Measured + Indicated, 0.1% Cu			664,563	0.30	1.98	0.03	0.03
Measured + Indicated, 0.2% Cu			**407,969**	**0.41**	**2.42**	**0.03**	**0.04**
Measured + Indicated, 0.3% Cu			266,379	0.50	2.74	0.04	0.04
Inferred	0.1	East Hill	434,614	0.20	1.51	0.03	0.02
		West Hill	330,426	0.29	1.17	<0.01	0.03
		Total	765,040	0.24	1.36	0.02	0.02
	0.2	East Hill	140,566	0.35	2.79	0.05	0.04
		West Hill	197,272	0.38	1.33	<0.01	0.02
		Total	**337,838**	**0.37**	**1.94**	**0.02**	**0.03**
	0.3	East Hill	61,761	0.49	4.20	0.07	0.07
		West Hill	126,222	0.46	1.45	<0.01	0.02
		Total	187,983	0.47	2.36	0.03	0.04

Mining

The La Verde project lends itself to a conventional truck and shovel open pit mining method as the mineralized rock outcrops at surface at both the East Hill and West Hill. A full time mine and mill operation is proposed for La Verde. The proposed mining schedule calls for ex-pit mining rates of 181 Ktonne/day during Year 1, 264 Ktonne/day from Year 2 to 4, and an average of 296 Ktonne/day from Year 5 to 18.

For this size of mining operation and the selectivity required in this type of deposit, large size mining equipment is proposed. Both productivities and costs have been estimated on the following operational scheme:

- Drilling and Blasting on 15 m benches
- Primary loading with rope and hydraulic shovels
- Use of 225 tonnes haul trucks

Use of electric equipment (primary drills and rope shovels) is considered.

Net smelter return (NSR) calculations were undertaken on all mining blocks by taking into consideration metal prices for copper, silver and gold, smelter costs, transport costs and refining costs as well as arsenic-related penalties. The NSR values were used in the life of mine schedule to optimize the value of the project. The life of mine plan presented in this report is based on optimizing the overall value for the Project by using a variable NSR cut-off grade by period and selectively stockpiling lower grade material after meeting processing plant requirements of 30 Mtpa. Stockpiled lower grade material is reclaimed at the most opportune time throughout the mine life to complement mill feed sourced directly from the open pits. Stockpiled materials are fully reclaimed by the end of the mine life.

The milling life is expected to span 19 years at a target plant throughput rate of 30 Mtpa with a total of 588 Mt of mineralized rock delivered to the plant. Life-of-mine copper concentrate production is estimated to be around 7.17 million tonnes averaging 26.7 % Cu. Mining and mill feed schedules are presented on a yearly basis in Table ES.2.

Table ES.2 Life of Mine Schedule

	YEAR	Y(-1)	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10	Y 11	Y 12	Y 13	Y 14	Y 15	Y 16	Y 17	Y 18	Y 19	Y 20	LOM
	Total Ex-Pit, Mt	**65.0**	**95.0**	**95.0**	**95.0**	**95.0**	**120.0**	**120.0**	**120.0**	**120.0**	**120.0**	**120.0**	**115.0**	**115.0**	**115.0**	**115.0**	**115.0**	**80.0**	**80.0**	**38.1**			**1,938**
	Avg. Ex-Pit Rate, ktpd	181	264	264	264	264	333	333	333	333	333	333	319	319	319	319	319	222	222	106			283
Ex-Pit Strip Ratio	By Year	na	3.0	2.2	2.2	2.4	4.1	3.0	3.0	3.0	3.0	4.0	2.8	5.4	9.4	3.9	3.8	1.9	1.7	0.3			
	Cumulative	na	5.7	3.7	3.2	3.0	3.2	3.2	3.1	3.1	3.1	3.2	3.2	3.3	3.5	3.5	3.5	3.4	3.3	3.1			3.1
	Stock to Mill, Mt		6.2	0.0	0.0	2.4	6.6	0.0	0.0	0.0	0.0	5.9	0.0	12.3	18.9	6.9	6.2	3.1	0.0	0.0	29.7	17.5	116
	Total Mill Feed, Mt	**0.0**	**30.0**	**30.0**	**30.0**	**30.0**	**29.9**	**30.0**	**29.9**	**29.8**	**30.0**	**30.0**	**30.0**	**30.3**	**30.0**	**30.2**	**30.1**	**30.3**	**30.0**	**30.0**	**29.7**	**17.5**	**588**
	Overall Strip Ratio - Cumulative	na	4.3	3.3	2.9	2.7	2.8	2.8	2.8	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.8	2.7	2.6	2.4	2.3	2.3
	Stock Size EOY, Mt	16.5	27.8	50.5	75.5	89.8	94.2	99.9	99.9	100.3	100.3	94.4	94.6	82.4	63.4	56.6	50.4	47.3	47.3	47.3	17.5	0.0	

Note that the mill feed schedule is provided in accordance with the requirements of 22 (b) of Form 43-101F1. It does not represent an estimate of Mineral Reserves.

Processing and Engineering

The major component of potentially economic material derives from the copper content of the sulphides with a minor component from the East Hill oxides. West Hill oxide material is essentially waste due to the copper being in chlorite and showing very low recoveries.

Although high priority work is required in the next phase of study on investigations into geo-metallurgical variability, AMC is satisfied that, based on the initial metallurgical testwork, 90% copper recovery to a 26-27% Cu concentrate is achievable on the sulphide material with a conventional crushing, grinding (SAG-Ball mill- pebble Crusher (SABC)) flotation circuit. The mineralized material is moderately hard but with good liberation characteristics at a primary grind size of approximately 200 µm.

The circuit would be configured in two trains of crushing, grinding and rougher flotation with a common regrind and three stage cleaning circuit. A tailings thickener has been included in the flowsheet, mainly to maximize tailings storage efficiencies and also to minimize water consumption.

The main risk to marketability of concentrate is its projected arsenic content, especially in East Hill sulphides where the arsenic is present as the copper-arsenic sulphosalt, tennantite. Although there are investigations currently underway into downstream concentrate treatment options to remove arsenic, the focus of the PEA has been on proven roaster technology to treat the La Verde concentrates and recover arsenic into a stable and environmentally inert compound.

Services such as power and supply roads essential to the La Verde project are in close proximity to the site. Two main highways are located to the east and the west of the property. The railway proposed to be used for concentrate shipment to the port of Lazaro Cardenas is sited a few kilometres north of the processing plant.

Although water is abundant in the region, including an irrigation canal that transects the property, significant community and government consultation will be required in the next project phase to determine the actual availability of water for he proposed project. During the next phase of this project, capacities, availability and permission to use these services should be investigated further.

The tailings management facility (TMF) storage area required for this Project is significant, with an estimated capacity of 367 million cubic meters. Several natural valleys located between 5 to 12 km of the Project have been identified as suitable, from an engineering perspective, for tailings storage for the first half of the mine life. Additional storage can be found further north-west or west of the Property. Alternative mining scenarios should be evaluated to mine the two open pits sequentially, thereby presenting an opportunity for backfilling the first pit with tailings or waste material. The land required for the proposed TMF locations is not currently owned by Catalyst and remains a project risk.

Capital and Operating Cost – Project Metrics

Life of mine mining costs have been estimated to average 1.66 US$/t of material moved. Processing costs have been estimated at 5.84US$/t milled. Roasting costs have been estimated at 30US$/t of concentrate treated.

Yearly revenue has been calculated applying assumed international metal prices, as follows:

- Copper: 2.7 US$/lb (5,950 US$/tonne)
- Gold: 1,200 US$/troy ounce (38.58 US$/g)
- Silver: 25 US$/troy ounce (0.80 US$/g)

These prices have been applied evenly throughout the mine life. First revenue is obtained in year Y1.

Pre-production capital has been estimated as 1,160MUS$, to be spent in three years from Year (-3) to Year (-1).

Based on these estimates and parameters the La Verde Net Present Value is estimated at 617MUS$ using a discount rate of 8%, before depreciation and taxes.

Table ES.3 Key Project Metrics

Item	Assumption and Metrics
Mill feed	587 Million tonnes at 0.37% Cu, 0.03 g/t Au and 2.3 g/t Ag
Mining rate	105 Mtpa (avg. LOM)
Total material moved	2,054 Mt
Mining method	Rope shovels and haul trucks
Ownership	Owner operated
Mining life	19 years
Processing life	20 years
Processing method	Crush, grind, flotation, concentrate roasting
Processing rate	30 Mtpa
Concentrate produced (LOM)	7.17 Million tonnes at 26.7% Cu
Processing recoveries	Cu, 90% - Au, 77.6% - Ag, 79%
Pre-Production Capital	1,160MUS$
Indicative NPV at 8% (pre-tax)	617MUS$

The preliminary economic assessment is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the preliminary economic assessment will be realized.

The mill feed figures shown in Table ES.3 do not represent an estimate of Mineral Reserves.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Conclusions and Recommendations

The present study indicates that La Verde is a potentially attractive mining opportunity. The key financial indicators, based on reasonable future copper prices and capital and operational cost estimates, justify advancing the Project to a Pre-Feasibility study stage.

There are key areas of risk or uncertainty that need to be addressed in moving the Project forward:

- Additional drilling is required to (a) upgrade the substantial component of Inferred Resources to Indicated or Measured Resources, (b) provide samples for metallurgical testwork, and c) provide additional geologic and alteration information.
- Management of high arsenic grades and arsenic byproducts, both from an economic and an environmental point of view.
- Availability of reliable and convenient power and water sources.
- Identification of TMF locations for the substantial projected LOM shortfall in tailings dam capacity.
- Acquisition of the land required for TMF locations.
- Social license. The initiation of systematic social and environmental assessment of the water access, infrastructure and tailings disposal assumptions contained in the study is required. Continued development of good relationships with the local stakeholders, particularly with the nearby communities.

AMC makes the following recommendations:

1. Advance the Project to a Pre-Feasibility level.
2. Initiate formal discussions with the relevant Mexican government agencies to investigate the feasibility of the proposed TMF solution outlined in this study.
3. Initiate formal discussions with the relevant Mexican government agencies to investigate a reliable power supply for the power consumption levels outlined in this study.

4. Strengthen Company involvement with local communities and initiate conversation with the relevant stakeholders regarding the extension and considerations of the project. Particularly relevant is the community engagement respect the project's water and land uses.
5. Initiate environmental and social baseline studies. On the environmental side, include the potential for contamination by arsenic from waste dumps and stockpiles from the earliest evaluation stages. See Table ES.4 for cost estimate.
6. In the next mine planning exercise, investigate options to deplete West Hill pit earlier to provide an opportunity for backfilling the pit with tailings or waste material. For costing purposes, it is included in the PFS.
7. Initiate a program investigating the geo-metallurgical variability across the deposits to develop a geo-metallurgical map and appropriate geo-metallurgical domains. See Table ES.4 for cost estimate.
8. Investigate options to improve precious metals recovery, especially gold in West Hill sulphides. See Table ES.4 for cost estimate.
9. Undertake a trade-off study on the use of high pressure grinding rolls (HPGR). See Table ES.4 for cost estimate.
10. Carry out concentrate roaster testwork and associated preliminary design studies. See Table ES.4 for cost estimate
11. With respect to pit wall planning:
    a. Undertake a geotechnical drilling program using orientated, NQ3 or HQ3 diamond drill core.
    b. Develop a geotechnical and structural model at pit scale to help with the modelling of the pit wall stability.
    c. Conduct a hydrogeological study to assess the presence, nature, and depth of the water table, and how this may be affected by mining.
    d. Undertake an assessment of the rock mass strength and carry out a slope stability analysis at bench, inter-ramp and overall scale to assess the slope parameters for each pit.

Table ES.4 Estimated Cost of Recommendations

Activity	Cost Estimate (C$m)
Pre-Feasibility study	1.0
Additional drilling to upgrade Resources + first Geomechanic studies	2.9
Environmental and social baseline studies	1.1
Geo-metallurgical investigations	0.8
HPGR Testing	0.1
Concentrate Roasting testwork	0.2

QUALITY CONTROL

The signing of this statement confirms this report has been prepared and checked in accordance with the AMC Peer Review Process. AMC's Peer Review Policy can be viewed at www.amcconsultants.com.

Project Manager		
Braulio Lanas	Signed	17 January 2013 Date
Peer Reviewer		
Pat Stephenson	Signed	17 January 2013 Date

IMPORTANT INFORMATION ABOUT THIS REPORT

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TABLES

FIGURES

Distribution list

Digital copies to:
Terry Hodson, Catalyst Copper Corp
Margaret Harder, Tetra Tech WEI Inc

Hard copy to:
Terry Hodson, Catalyst Copper Corp

2 INTRODUCTION

2.1 General and Terms of Reference

This Technical Report on the La Verde Copper Property (the Property) in Michoacan, Mexico, includes the current La Verde Project (the Project) and the immediate surrounding land. It has been prepared by AMC Mining Consultants (Canada) Ltd (AMC) of Vancouver, Canada, on behalf of Catalyst Copper Corporation (Catalyst) of Vancouver, Canada.

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2.2 The Issuer

Catalyst is a mining company based in Vancouver, BC, and is currently trading on the Toronto Stock Exchange (TSX) Venture Exchange under the symbol CCY and on the OTCQX market under the symbol CATXF. Catalyst is a copper and base metal developer with a focus on copper mining projects in Mexico. Catalyst’s current focus is the La Verde copper property.

2.3 Report Authors

A listing of the authors of this Report, together with the sections for which they are responsible, is given in Table 2.1.

Table 2.1 Persons who Prepared or Contributed to this Technical Report

Qualified Persons responsible for the preparation of this Technical Report						
Qualified Person	**Position**	**Employer**	**Independent of Catalyst?**	**Date of Last Site Visit**	**Professional Designation**	**Sections of Report**
Mr P R Stephenson	Director, General Manager, Principal Geologist	AMC Mining Consultants (Canada) Ltd	Yes	No visit	P.Geo., BSc (Hons) MCIM FAIG FAusIMM (CP)	1 to 5, 15, 16, 19 to 26
Mr A Riles	Principal Metallurgical Consultant	Riles Integrated Resource Management Ltd	Yes	11-12 April 2012	BSc (Hons) Grad Dipl Business Management MAIG, MAusIMM (CP)	13, 17, and 19. Part 18 – 21 and contributed to 25 and 26
Ms Margaret Harder	Geologist	Tetra Tech WEI Inc.	Yes	August 21, 2012	M.Sc.,P.Geo.	6 to 12, and 14.
Mr Michael F. O'Brien	Chief Geologist	Tetra Tech WEI Inc.	Yes	No visit	M.Sc., Pr.Sci.Nat., FGSSA, FAusIMM, FSAIMM	14
Mr Mo Molavi	Principal Mining Engineer and Vancouver Manager of Mining Services	AMC Mining Consultants (Canada) Ltd	Yes	No visit	P.Eng., M Eng, B Eng	18
Other Experts Who Assisted the Qualified Persons						
Expert	**Position**	**Employer**	**Independent of Catalyst**	**Visited Site**	**Sections of Report**	
Mr Braulio Lanas	Senior Mining Consultant	AMC Mining Consultants (Canada) Ltd	Yes	11-12 April 2012	16, 21 and 22. Overall compilation of report	
Mr. James Stoddart	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd	Yes	No visit	Peer Review of 16, 21 and 22	
Ms Adrienne Ross	Senior Geologist	AMC Mining Consultants (Canada) Ltd	Yes	No visit	General assistance	

Personal inspections by QPs and others of the Property took place as follows:

- The Tetra Tech site visit was conducted by QP Margaret Harder on 21 August 2012.
- The AMC site visit was conducted by QP Alan Riles and Senior Mining Consultant Braulio Lanas, from 11 to 12 April 2012. During the visit, the core shack and core storage facility, the Project site and surroundings, and the Project office in Uruapan were visited. The visit covered relevant geo-metallurgy aspects, including inspections of core boxes at the core shed, and surrounding potential locations for relevant infrastructure components such as tailings disposal facilities, concentrator and powerlines. The visit also included discussions with Catalyst staff regarding drill core care, sampling and assaying procedures.

This report is effective 30 September 2012.

2.4 Units of Measure, Currency and Acronyms

Throughout this report, measurements are in metric units and currency in US dollars unless otherwise stated. Tables 2.2 and 2.3 show key terms, acronyms, units and other abbreviations used.

Table 2.2 List of Institutions and Corresponding Acronyms

Institution	Acronym used	First introduced in Section (other than Summary)
Aur	Minera Aur Mexico S.A. de C.V.	
Catalyst	Catalyst Copper Corp.	
CIM	Canadian Institute of Mining, Metallurgy and Petroleum	
MinHill	Minera Hill 29 S.A. de C.V	
MTO	Minera Torre de Oro, S.A. de C.V	
Noranda	Noranda Mining and Exploration Inc	
the Project	the La Verde Project	
the Property	the La Verde Property	
Tetra Tech	Tetra Tech WEI Inc.	

Table 2.3 Units, Terms and Abbreviations

Unit	Abbreviation	Term	Abbreviation
Percent	%	Above mean sea level	AMSL
Per pound (avdp)	/lb	Bond work index	BWI
Per ounce (avdp)	/oz	Cyanide	CN
Per kilowatt hour	/kW.hr	Diameter	dia
Per cubic metre	/m3	Earnings before interest, tax, depreciation, and amortization	EBITDA
Metre(s)	m	Engineering, procurement, and contract management	EPCM
Per tonne kilometre	/t.km	General and administration	G&A
One millionth of a meter	µm	Hydraulic Radius	HR
Square meters	m^2	Internal rate of return	IRR
Cubic meters	m^3	Modified Stability Number	N’
Cubic meters per hour	m^3/hr	Preliminary Economic Assessment	PEA
Millimeters	mm	Pre-feasibility study	PFS
Million ounces (avdp)	Moz	Net present value	NPV
dry metric tonnes	dmt	Acidity or basicity	pH
Megapascals	MPa	Pyrite	Py
Megatonnes	Mt	Rock Mass Rating	RMR

Unit	Abbreviation	Term	Abbreviation
Megatonnes per annum	Mtpa	Rock quality designation	RQD
Grams /t	g/t	Rock work index	RWI
Grams /t of gold	g/t Au	Semi-autogenous grinding	SAG
Megawatts	MW	Tailings management facility	TMF
Per annum	p.a	Universal transverse mercator	UTM
Per tonne	/t	Fire Assay	FA
Kilogram(s)	kg	Atomic Absorption	AA
Kilograms per cubic metre	kg/m^3	Atomic Absorption Spectrometry	AAS
Kilometre(s)	km	Gold	Au
Square kilometers	km^2	Inductively-Coupled Plasma	ICP
Cubic kilometers per annum	km^3/a	Tonne(s)	t
Kilopascal	kpa	Tonnes per hour	tph
Kilotonne per annum	kt/a	Volt(s)	V
Kilowatt	kW	Weight for weight	w/w
Kilowatt-hours	kWh	Wet metric tonnes	wmt
Litre	l	Tonnes per cubic metre	t/m^3
Pound (avdp)	lb	Tonnes per day	tpd
ID	inversed distance	OK	ordinary kriging
IDW	inversed distance weighted	SG	specific gravity
NN	nearest neighbor	SRM	Standard Reference Material
Kilopascal	kpa	Tonnes per hour	tph
Kilotonne per annum	kt/a	Volt(s)	V
Kilowatt	kW	Weight for weight	w/w

3 RELIANCE ON OTHER EXPERTS

AMC has not included any other expert opinion or report for this study.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Property is located in the state of Michoacán, approximately 320 km west of Mexico City. The known copper deposits on the Property area lie 8 km northeast of the town of Nueva Italia de Ruiz (population ~50,000) and 8 km southwest of the town of Lombardia (population ~25,000). Uruapan (population ~316,000) is the largest city near the Property and is 60 km to the north.

The center of the Property is 19°05'14"N latitude and 102°00'41"W longitude, equivalent to coordinates 814302 mE and 2113035 mN in Universal Transverse Mercator (UTM), Zone 13Q.

Figure 4.1 La Verde Regional Location Map

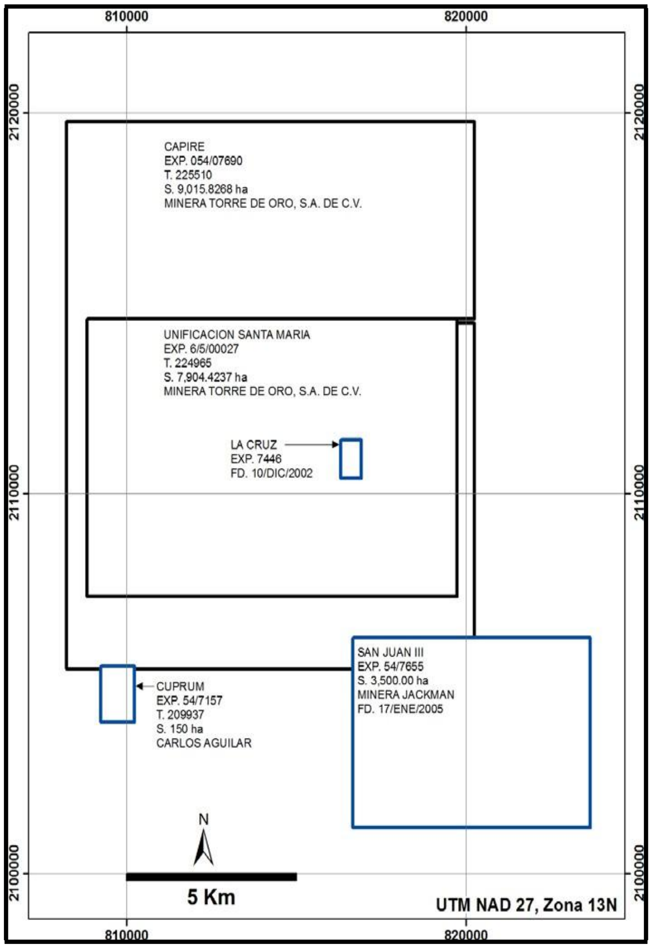
4.2 Concessions

The Property consists of two concessions as outlined in Table 4.1 and illustrated in Figure 4.2.

Table 4.1 La Verde Concessions

Claim	Title No.	Area (ha)
Unificacion Santa Maria	219919	7,903.5246
Capire		9,015.6268
Total		**16,919.1514**

Figure 4.2 La Verde Claims Map



All of the original claims which constitute the Unificacion Santa Maria were acquired by Mr. Jorge Ordoñez, the principal owner of Minera Cima S.A. de C.V. (Cima), in a Mexican claim lottery. In February 2005, the original seven claims were unified and deemed the Unificacion Santa Maria claim. Unification of contiguous claims allowed for distribution of exploration expenditures throughout the land position.

Minera Aur Mexico S.A. de C.V. (Aur) executed a purchase agreement with Cima to acquire a 100% interest in the original seven mining claims upon completion of the payments listed in Table 4.2.

Table 4.2 Cima Agreement

Anniversary	Date	Payment ($)
0	on-signing	50,000
1	5 November 2005	100,000
2	5 November 2006	150,000
3	5 November 2007	250,000
4	5 November 2008	250,000
5	5 November 2009	6,000,000
	Total	**6,800,000**

In addition, Cima retained a 0.50% net smelter royalty (NSR). In May 2005, the Capire protection claim was located around the core of La Verde claims and became part of the Cima agreement. Teck Resources Limited (Teck) acquired all of the outstanding common shares of Aur's parent company in September 2007. Aur changed its name to Minera Torre de Oro, S.A. de C.V. (MTO), and has completed all option payments and now owns a 100% interest in the Property, subject only to the underlying NSR royalty.

According to Mexican mining law, claims are located by establishing a concrete claim monument (Punto de Partida), which is positioned within the Property or less than 3 km from any of the boundaries, if it is outside. A registered surveyor must survey the Punto de Partida and link it to the national survey network. The surveyor files a report with the Mexican mining agency. The report includes the location of the Punto de Partida and the development of the perimeter of the claim. AMC has been advised that this was the procedure used to locate the Property boundaries.

Svit Gold Corp. (Svit) initiated talks with MTO in 2009 to acquire the Property. A Property of Merit report was prepared by Micon International Limited (Micon), dated 19 January 2010 as a requirement for Svit to re-list on the TSXV. Svit changed its name to Catalyst Copper Corp. at the time of re-listing. The option agreement between MTO and Minera Hill 29 S.A. de C.V (MinHill), Catalyst's Mexican subsidiary is as follows:

- Catalyst must incorporate MinHill, a wholly owned subsidiary of Catalyst, under Mexican law (completed).
- MTO and MinHill will incorporate a separate joint venture company "Holdco" under Mexican law (completed).
- Within ten days of signing, MinHill will advance US$6,000,000 to MTO. MinHill will also fund US$10,000,000 in expenditures (work commitments) before 31 December 2012. This will earn MinHill its initial 60% interest in the Property through Holdco (S-1 Option) (completed).
- The work commitment includes a minimum of 200 km of IP survey, a minimum of 30,000 m of drilling and a NI 43-101 compliant mineral resource estimate of the Property (nearing completion).

- MinHill is responsible for entering into access agreements with the local ejidos (completed).

Upon earning a 60% interest in the Project, Catalyst is required to deliver MTO “Notice”; that includes a statement in reasonable detail evidencing it’s expenditures on the Project (including required work commitments) and a technical report on the results obtained from such expenditures. The date of delivery of such Notice is referred to herein as the “Catalyst Earn-In Date”.

MTO may elect, at any time up to 60 days after the Catalyst Earn-In Date, to:

(a) Invoke its option (the “T-1Option”) to earn back a 20% interest in the Project, to hold a 60% interest; or

(b) Form a joint venture, whereupon the interests would be Catalyst 60%, MTO 40%.

If MTO elects to exercise the T-1 Option, it is required to incur and sole fund aggregate additional expenditures of twice the expenditures incurred by Catalyst to a maximum of $20 million, within three years of such election (MTO has the right to fund any shortfall in funding by paying such shortfall in cash to Catalyst). If MTO fails either to complete such expenditures or pay the shortfall in cash, then the interests in the Project would be Catalyst 60%, MTO 40%.

If MTO elects (or is deemed to have elected) not to invoke the T-1 Option, Catalyst has the right, for a period of 60 days to purchase MTO’s remaining 40% interest by paying MTO $20 million.

The option agreement warrants that the concessions and other rights are in good standing and that MTO has paid the semi-annual taxes and completed all annual work commitments under Mexican law. It also warrants that the concessions are free and clear of all liens, charges and encumbrances arising from operations on or related to the Property.

AMC is not aware of any environmental or social issues that may affect access, title, or the right or ability to perform work on the property. All exploration activities conducted on the property have been in compliance with relevant environmental permitting requirements. To AMC’s knowledge, MinHill has obtained the appropriate permits covering surface rights.

Regarding the mineral claims standing, AMC has been shown the updated payments done by MinHill to MTO for the concept of updating the “Unificación Santa María” and “Capire” mine claims. This has been deemed by AMC as sufficient proof of the Catalyst’s rights on the La Verde property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

Two paved highways provide access to the Property including the new Morelia – Lazaro Cárdenas highway (toll road) and the original Lombardia-Nueva Italia highway (Mexico Highway 37) (Figure 4.1). Within the Property there are a series of all-weather dirt roads and old drill roads.

The closest international airport is at Morelia (2010 population 730,000). The General Francisco J. Mujica International Airport is approximately 170 km to the northeast of the Property via the new Morelia-Lazaro Cárdenas Autopista. A second international airport (Ixtapa Zihuatanejo International Airport) is located approximately 200 km south of the Property. Uruapan is served by the Uruapan International Airport.

5.2 Climate

The climate in the Project area is classified as semi-dry temperate with rain in the summer, an average annual temperature of 27.3°C, and a mean temperature of 25.3°C in the coldest month of the year. Average annual rainfall is 707.5 mm with the bulk of the rainfall occurring in June, July, August, and September (588 mm). Exploration and mining can be carried out all year long.

5.3 Local Resources

There are several villages within the concession boundary (see Figure 4.1). The two closest villages are El Huaco, immediately south-west of the known deposits, and La Laguna, immediately to the north-west. Both villages have a couple of hundred residents. These villages could provide some of the workforce for the Project but could not provide the necessary equipment.

The larger town of Nueva Italia de Ruiz (2010 population 32,467) is immediately southwest of the concession boundary. The Aur / MTO core and rejects that were stored in a warehouse in La Laguna have been transported to a much larger warehouse facility 4 km north of Nueva Italia de Ruiz. The front of the warehouse opens onto the main Lombardia-Nueva Italia de Ruiz highway, while the main railway to the coast is adjacent to the rear of the warehouse (Figure 5.1).

Figure 5.1 Catalyst Core Storage and Sampling Facility

Lombardia (population ~25,000) is immediately north of the Property boundary. Most of the labourers and equipment required for the Project could be sourced from either of these larger towns. Uruapan (2010 population 315,379) is approximately 60 km north of Nueva Italia de Ruiz.

More specialized equipment and skilled labourers may be sourced from Uruapan, Morelia (population ~608,000) and Lazaro Cardenas (population ~75,000). Difficult items may have to be sourced from Mexico City or Guadalajara. It should not be necessary to source many items from outside Mexico. Drill (core and reverse circulation (RC)) equipment and personnel can be sourced from other regions within Mexico.

The known copper deposits are on the common land owned by the El Huaco Ejido (communal land cooperative). Aur had difficulty in negotiating access rights with the El Huaco Ejido. MTO has since improved community relations between itself and the ejido, and has continued to improve relations by assisting in constructing several projects for the cooperative as well as hiring some of its members during the exploration seasons. This has resulted in the negotiation of a four year access agreement with El Huaco which allows MinHill access to El Huaco common ground for exploration purposes over four years.

5.4 Infrastructure

There is a high volume irrigation canal transecting the concession which is used by the local mango and cucumber farmers. The canal empties into a steep canyon (El Marqués River gorge) near El Huaco. To date, the availability and access to water for the exploration programs has been favourably negotiated with local communities.

The road system in this part of Mexico is excellent. The rail link to the deep sea port of Lazaro Cardenas is in excellent condition. A siding already exists at the new core storage warehouse facility north of Nueva Italia de Ruiz which could be used to handle bulk imports and copper concentrates.

There are several high-tension electrical lines (66 kVA) within 5 km of the Project. Telephone / cell coverage in the area is excellent.

5.5 Physiography

The topography of the area is dominated by a ridge of prominent hills surrounded by low relief farm land and occasional deeply incised canyon drainages. The main area of interest is characterized by a series of hills approximately 5.5 km long (east-west) by 1 km wide (north-south) referred to as the Sierra del Marqués. The maximum elevation on the Property is approximately 750 masl and occurs on Cerro Mina La Verde (East Hill). The valley floor is at approximately 425 masl to the south of Sierra del Marqués and 500 masl to the north.

Vegetation in the district is classified as Selva Baja Caducifolia and most hills are covered with secondary vegetation consisting of small trees, bushes and vines. Large trees include tamarindo, pinzon and cuatro hojas, however much of the primary vegetation has been stripped and secondary vegetation dominates the non-irrigated land.

Irrigation of the valley floor has generated a strong agricultural industry. The main crops in this area include mango, rice, cucumber, tomato, corn, lime and grapefruit. Agricultural fields are active year round with some areas generating up to four harvests per year.

Figure 5.2 Irrigation Canal at El Marqués River Gorge

The US Geological Survey has recorded three earthquakes in proximity to Property. On 29 October 1990 there was a 5.1 magnitude earthquake at 98.6 km depth 16.5 km to the southeast of the Property. On 3 July 1973 there was a 5.6 magnitude earthquake at 125 km depth 22 km to the east of the Property. Then on 28 May 1985 there was a 5.2 magnitude earthquake at 105 km depth 27.6 km to the southeast of the Property.

6 HISTORY

This section has been extracted and adapted from the Tetra Tech September 2012 Report.

6.1 Regional Mining History

The Property is part of the Michoacán Copper Belt. This metallogenic area has been explored and mined on a small scale since the early 1900s, focussing on high-grade showings. Since the early 1960s, the world mining industry has focused on large tonnage, low-grade porphyry copper exploration and mining. This new direction has brought significant activity to this area. Copper-bearing breccia pipes in the Huacana pluton, including La Verde, San Isidro and Inguaran, underwent extensive exploration by mining companies and by various academic organizations including the US Geological Survey and the Servicio Geologico Mexicano.

The largest copper mine in the region is located approximately 50 km east-southeast of La Verde at the Inguaran mine. American Smelting and Refining Company (ASARCO) (now Grupo Mexico or GMEXICO) mined several breccia bodies at Inguaran from 1971 to 1982 and extracted some 7,000,000 t of ore grading 1.2% Cu (Osoria et al. 1991).

Copper mining in other parts of the region has been restricted to small operations centred on high-grade structures in andesites to the south of La Verde and in the Huacana pluton. Mining operations such as Las Mexicanas produced small amounts of ore which were treated at a small mill in Oropeda. A small underground mine and mill was developed at Brazil (also known as Calzontzin) where limited tonnage was mined from a copper-rich narrow structure (~2 m wide) in the Huacana pluton.

6.2 La Verde Mining History

Small scale copper mining began at Cerro Mina La Verde (East Hill) in 1906 and continued intermittently until the early 1960s. Approximately 45,000 t of ore was mined from high-grade sections using underground mining methods. Mined ore was hand-sorted, and material grading greater than 10% Cu was shipped to local smelters (Coochey and Eckman 1978). Evidence of previous mining is scattered throughout the Property, including underground workings (as shown in Figure 6.1) and large open stopes on the south slope of East Hill, as well as open cuts and smaller stopes on Cerro La Laguna (West Hill). The stopes on the West Hill continue to a depth of approximately 35 m but have a short, generally less than 25 m, strike-length.

Figure 6.1 La Verde Historic Mining

6.3 La Verde Exploration History

The Property was virtually unexplored by modern methods until Consejo de Recursos No Renovables (CRNR) conducted rock chip sampling, geophysics and diamond drilling in the late 1950s. Since this initial phase of exploration, three major exploration efforts have been completed on the Property.

A summary of exploration activity at the Property is presented in Table 6.1.

Table 6.1 Summary of Exploration Activity

Year	Activity
1906	Onset of mining high-grade copper zones.
1957	Mine operated and owned by Sr. Jose Maria Flores Barron. CRNR completed mapping, geophysics (ground MAG) and underground sampling of stopes and adits, 370 samples taken with average grade of 1.4% Cu.
1958	CRNR completed 11 diamond drillholes on the southern flank of the East Hill, totaling 741 m of BQ core.
1967 to 1972	Lytton enters into agreement with Flores Barron and completes a comprehensive program of geological mapping, soil geochemistry, IP geophysics and approximately 50,201 m of diamond drilling and 12,584 m of percussion drilling. First feasibility study completed.
1974	Hudson Bay Mining and Smelting (HBM&S) acquires control of Lytton. An additional 6,659 m of diamond drilling is completed, as well as a second feasibility study.
1976	D. Coochey completes resources estimate of West Hill.
1984	D. Coochey completes updated estimate of West Hill resource.
1995 to 1996	Mexican government mining agency, El Fideicomiso de Fomento Minero (FIFOMI) promotes the Property. Noranda completes due diligence including underground mapping and sampling, IP geophysics and 300 m of RC drilling in the plains to the south of the Sierra del Marqués.
2003	Claims enter lottery and are acquired by Cima.
2004 to 2005	Aur enters into an option to purchase agreement with Cima. Aur completed an airborne magnetometer, electromagnetic and radiometric survey in January 2005.
Late 2005	Aur acquires old data including the complete 1972, 1974 feasibility reports, archived drill logs, surface geologic maps and various other data from HBM&S data warehouse in Flin Flon, Manitoba.
2005 to 2007	Aur completed 24 diamond core drillholes for a total of 9,600 m and stores core in La Laguna warehouse. Aur completed a series of short lined IP surveys over the Property in 2005. The resulting cut lines are still extremely visible from surface and satellite. Aur had progressively poorer community relations with the El Huaco Ejido until the company was barred from the Property.
2007 to 2009	Teck acquires all of the outstanding common shares of Aur’s parent company in September 2007 and MTO completes four diamond core drillholes totaling 1,562 m in 2008. The core is also stored in the La Laguna warehouse. Community relations with the El Huaco Ejido have improved.
2009 to 2012	MinHill options the Property from MTO and completes additional diamond drilling and geophysical surveys (see details in Sections 9 and 10).

6.3.1 Lytton / HBM&S program

Lytton completed an extensive exploration program at La Verde from 1967 to 1972 including geological mapping and sampling, soil geochemistry, IP geophysics, underground development and sampling, bulk sampling and drilling. This work resulted in the delineation of a large mineralized vein complex, known as the West Hill Zone, and four distinct mineralized breccia zones within the East Hill area. Exploration activity is summarized in Table 6.2.

Table 6.2 Lytton / HBM&S Exploration Program

Company	Year	Activity	Description
Lytton	1966-1967	IP Survey	47.5 km
	1966-1967	Soil Geochemistry	-
	-	**Drilling – core**	50,201 m
	1967-1972	Surface	152 holes, V-1 to V-142
	1969-1972	Underground	97 holes, UV-1 to UV-90
	-	**Drilling – percussion**	12,584 m
	1967-1972	Surface	198 holes, S-1 to S-213, and R-2 to R-24
	1969-1972	Underground	91 holes, US-1 to US-204
	1970	IP Survey	179.5 km
	1968-1971	Underground Development	2.2 m wide crosscuts and drifts totaling 3,386 m
	1971	Bulk Sampling	245 m of crosscut sampling
	1967-1972	Geologic Mapping	1:4000 scale map of the Sierra del Marqués
	1968-1971	Underground Mapping	1:200 scale adit map
HBM&S	1974-1976	**Drilling – core**	6,659 m
	-	Surface	27 holes, V-143 to V-169

All IP surveys done for Lytton were undertaken by Seigel Asociados. Numerous anomalies were identified on both West and East Hill that correspond well to the known mineralized zones. Lytton and HBM&S used much of this IP data to target additional drillholes. Detailed descriptions of individual anomalies and methods used are located in Seigel 1970.

Extensive drilling was completed on the Project, mostly by Lytton (62,785 m) and later by HBM&S (6,659 m). A database for these drillholes includes short form logs that record rock types and assays. Generally both the East and West Hills were drilled along 50 m sections and at 50 m intervals.

6.3.2 Noranda Program

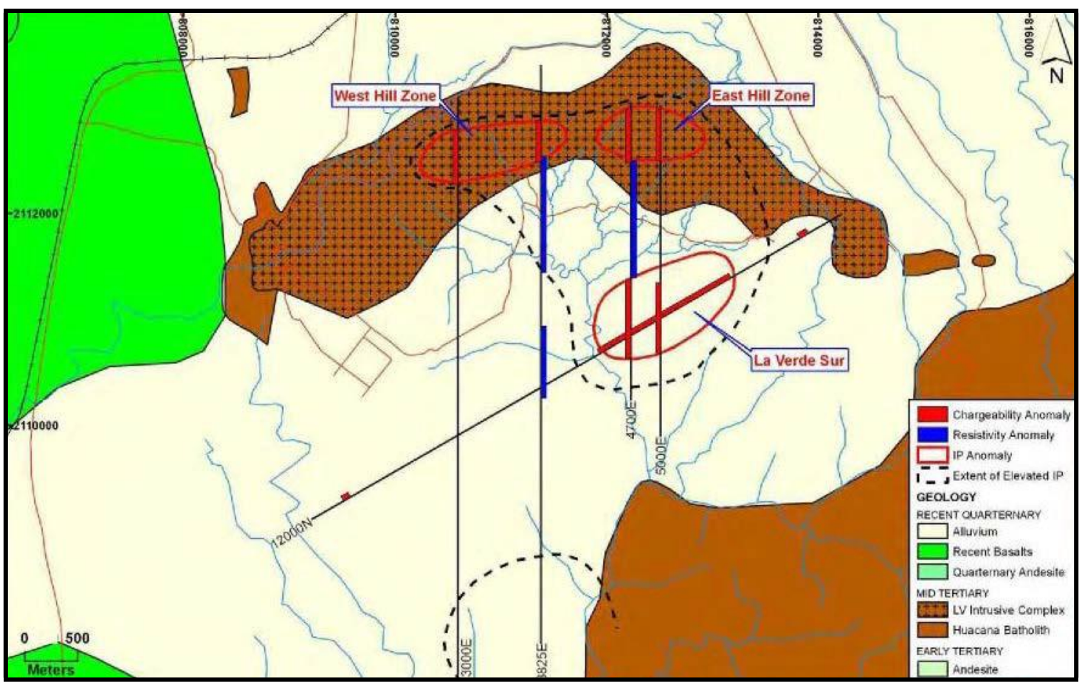
In early 1995 Noranda Mining and Exploration Inc. (Noranda) completed a significant amount of field work on the Property which is summarized in Table 6.3.

Table 6.3 Noranda Exploration Program

Year	Activity	Description
1995	Geological mapping – Underground – Surface - regional	East Hill and selective West Hill adits
1995	Sampling – Underground – Silt - regional IP survey	East Hill and selective West Hill adits 27 km

The underground work by Noranda corroborated the mineralization in drillholes and Noranda concluded that the historic grades could be considered valid, with the exception of gold which it believed was lower than that reported in the 1972 feasibility study. The most significant finding of the Noranda program, however, was the identification of a 1 by 2 km IP anomaly at 100 m depth known as "La Verde Sur", located roughly 1 km south of the East Hill in an area of alluvial cover as illustrated in Figure 6.2.

Figure 6.2 Noranda IP Results



6.3.3 Aur Program

Aur conducted an aggressive exploration program that is summarized in Table 6.4.

Table 6.4 Aur Exploration Program

Year	Activity	Description
2004 to 2006	Line cutting	65 km, N-S lines 100 to 200 m apart
	Geophysical surveys	
	– Airborne	593 line km – mag, electromagnetic (EM) and radiometric IP (60 km) and magnetometer
	– Ground	487 samples
	Soil Geochemistry	
	Geological mapping	1:20,000
	– Surface - regional	
	Drilling - core	
	– Surface	24 holes, 9,600 m, confirmation drilling and IP anomaly testing

The airborne, helicopter-based geophysics was completed by McPhar Geosurveys of Canada in January 2005. Zang Geophysical Consultants interpreted the results in April 2005. The airborne magnetics identified the eastern two-thirds of the La Verde intrusive as a large prominent magnetic low especially around the known mineralized zones. Red diorite in the western half of West Hill is thought to result in the magnetic high in this part of the intrusive. The magnetic low anomaly continues to the northwest of La Verde but shifts westerly at the edge of the East Hill known mineralized zones and continues under the alluvium in the La Laguna area. The electromagnetic (EM) survey identified several resistivity anomalies that correlate to mineralized zones. The radiometric survey outlines the location of exposed Huacana batholith, including the La Verde intrusive complex.

The soil geochemical surveys focused on the western end of the Sierra del Marqués and Las Minitas. The most significant copper soil anomaly was in the Las Minitas prospect area.

Ground geophysics was run over the cut lines and undertaken by Quantec Geophysics in February / March 2005. IP spacing was 100 m over the East and West Hill deposits and 50 m on the perimeter of the deposits and Las Minitas prospect. The results outlined a large chargeability anomaly at a depth of 200 m below the surface of Sierra del Marqués. Several anomalies within the area have not been drill tested. The ground magnetometer survey showed good correlation to the airborne magnetometer results.

Geological reconnaissance mapping was completed to understand the property-scale geology, and to investigate mineral occurrences as well as airborne geophysical anomalies outside of Sierra del Marqués. Much of the Property has yet to be mapped as the program was cut short due to the onset of the rainy season; however, a 1:20,000 scale preliminary geological map was developed.

A total of 9,600 m of surface diamond drilling (7,085 m NQ size, 2,515 m HQ size) in 24 drillholes was completed on the Property by Aur between November 2004 and August 2005. Seventeen of the holes confirmed each of the mineralized zones in both West (two holes) and East Hill (15 holes).

Aur’s sample intervals within drill core were primarily selected based on the degree of visible copper mineralization within a given zone (i.e. zones of high grade material were separated out from zones of lower grade material). When present, major lithological breaks were used to begin / end an individual sample interval.

While blanket sampling was not applied in the Aur drill program at La Verde, barren intersections up to 20 m in width within otherwise mineralized zones were generally included in sampling for continuity purposes. Overall approximately 64% of all core drilled at La Verde was sampled for assaying.

6.3.4 MTO program

Teck acquired all outstanding common shares of Aur's parent company in September 2007. The following work was completed by MTO (Table 6.5).

Table 6.5 MTO Exploration Program (Years 2007-2009)

Activity	Description
Geological	Reinterpretation of older data
Geophysical	Reinterpretation of older data
Drilling – core	
– Surface	Four holes, 1,562 m, perimeter holes

From 2008 to 2009 MTO completed a re-evaluation of the historic geology and drill database. A new deposit model was developed by MTO geologists (Chamberlain, 2009). Through this work, modifications to logging and mapping and recommendations for future exploration were put forth. Historic geophysical data was re-interpreted by MTO geophysics personnel (V. Sterritt 2008 and B. Lum 2009). Conclusions and recommendations from this work were as follows:

- Using the airborne data, several regional structures (i.e. faults and contacts) and large zones of potential porphyry-style alteration were inferred.
- There is a decent correlation between structures identified from the aeromagnetic data and the regional geology and there is potential for developing a map of the bedrock geology under alluvium using the magnetics.
- The mineralized zones occur in a strong magnetic low, which corresponds well with the mineralized lithologies.
- The magnetic low is much larger than the exposed Sierra del Marqués, indicating that there is potential for mineralization under cover to the north and south of the known deposits.
- The IP / resistivity data indicates potential for untested sulfide mineralization between East and West Hill, south of East and West Hill beyond the depth of investigation of the survey and drilling and to the northwest beyond the survey extents.

Four diamond drillholes totalling 1,562 m were completed on the Property. The four holes explored IP anomalies outside of the Sierra del Marqués ridge as MTO was still in the process of negotiating an access agreement with the El Huaco Ejido. Anomalous intersections were cut in one of these holes. This data, coupled with the geophysical data indicate that there may be more mineralization at La Verde than has been identified at the East and West Hill deposits.

MTO geologists used the same sample methodology as Aur. Sample intervals were generally 1.5 m. Blanks were obtained from the same cinder cone quarry near Nueva Italia as used by Aur.

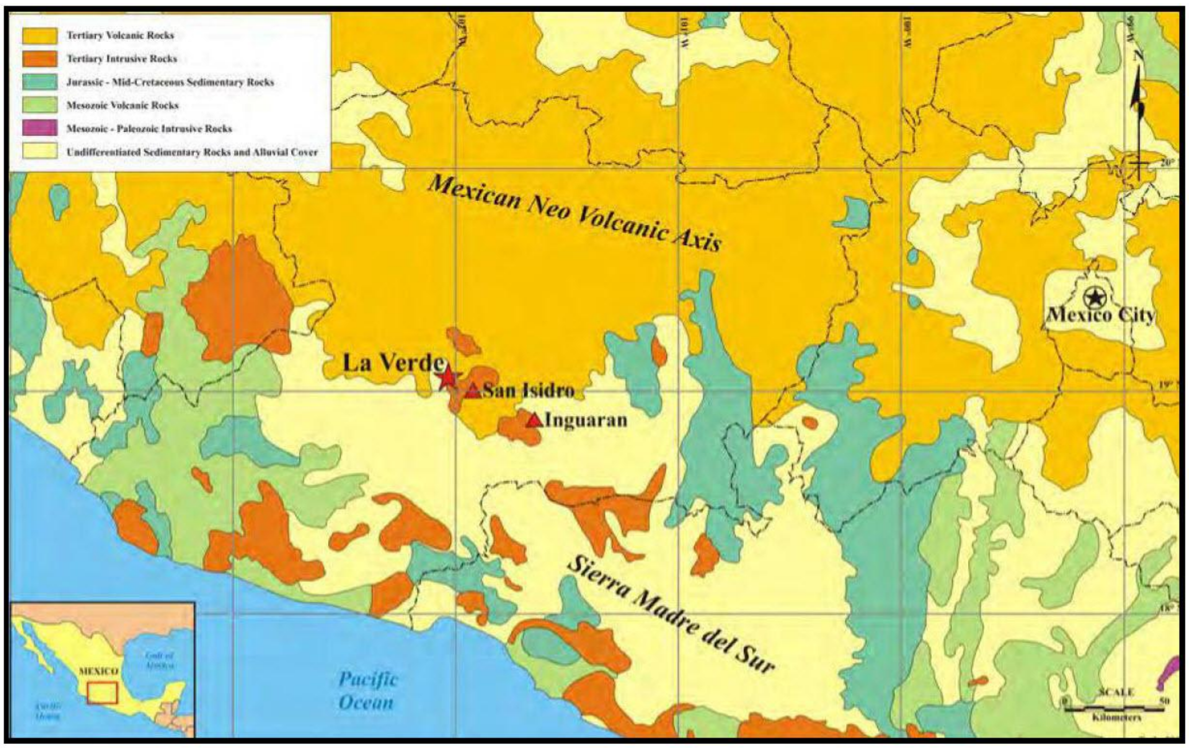
7 GEOLOGICAL SETTING AND MINERALIZATION

This section has been extracted and adapted from the Tetra Tech September 2012 Report.

7.1 Regional Geology

The Property is situated along the southern margin of the Mexican Neovolcanic Axis and within the Sierra Madre del Sur (Figure 7.1). Most of the Project area is underlain by the north-western margin of the mid-Tertiary Huacana granodiorite / quartz monzonite batholith. This same batholith is host to the San Isidro and Inguaran copper breccia pipes located roughly 20 km and 50 km, respectively, to the southeast of La Verde (Figure 7.1).

Figure 7.1 Regional Geology



7.2 Local Geology

The surficial geology in this region is characterized by a blanket of Quaternary sediments dominated by thick sequences of bedded conglomerate, lesser sandstone, mudstone and recent volcanic rocks (Coochey and Eckman 1978). The La Verde intrusive complex represents a window of exposed intrusive basement and forms an east-west trending arcuate range.

Early Tertiary supracrustal basement rocks are present in the El Marqués River gorge south of the Property where interlayered porphyritic andesite and lesser basalt flows are exposed beneath Quaternary andesite flows.

7.3 Property Geology

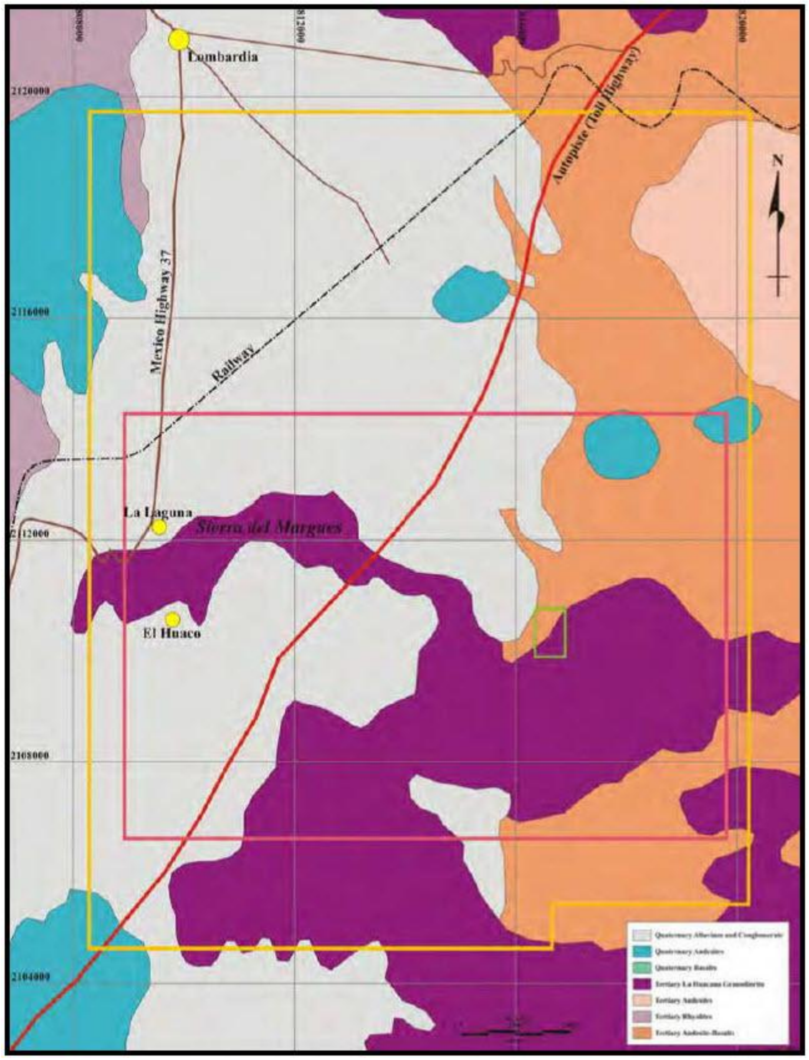
7.3.1 Summary

The intrusive complex at La Verde is dominated by quartz-diorite, which is exposed along an east-west-trending arcuate mountain range roughly 5.5 km long, 1 km wide and up to 290 m above the surrounding blanket of Quaternary sediments (Figure 7.2 and Figure 7.3). The range is known as the Sierra del Marqués and is divided into the Cerro La Laguna (West Hill) and Cerro Mina La Verde (East Hill) by a topographic low known as La Puerta located roughly in the centre of the arc.

The East Hill contains four main copper mineralized zones. Three of these zones (#1, #3 and #4) are hosted within altered and brecciated quartz-diorite in close proximity to dykes and stocks of quartz-feldspar porphyry, while the fourth zone (#2) is hosted largely within unbrecciated quartz-feldspar porphyry. All four mineralized zones form a roughly circular pattern on the western half of the East Hill, in a plan view. Brecciation mechanisms on the East Hill vary from mechanical milling to hydrothermal cracking. Alteration tends to be fairly tightly restricted to zones of brecciation, except in the case of large-scale calcium-sodium metasomatism within quartz-diorite adjacent to quartz-feldspar porphyry. Potassic feldspar alteration was noted in drillholes located on the north side of the East Hill.

The West Hill is characterized by approximately east-west trending bands of phyllic / propylitic alteration with associated pyrite-chalcopyrite ±arsenopyrite pyrrhotite veining at the contact between quartz-diorite porphyry and equigranular quartz-diorite. The lateral extent of these mineralized east-west striking veins forms a north-northeast south-southwest trending roughly elliptical shaped mineralized body, in a plan view. The western half of the West Hill consists of equigranular “red diorite” stained red by inclusions of hematite. A major north-northwest-trending magnetic lineament occupies this region of the West Hill and may reflect a structural break separating red diorite to the west from quartz-diorite to the east. Apart from a few small vein showings, no significant copper mineralization has been intersected to date within the red diorite.

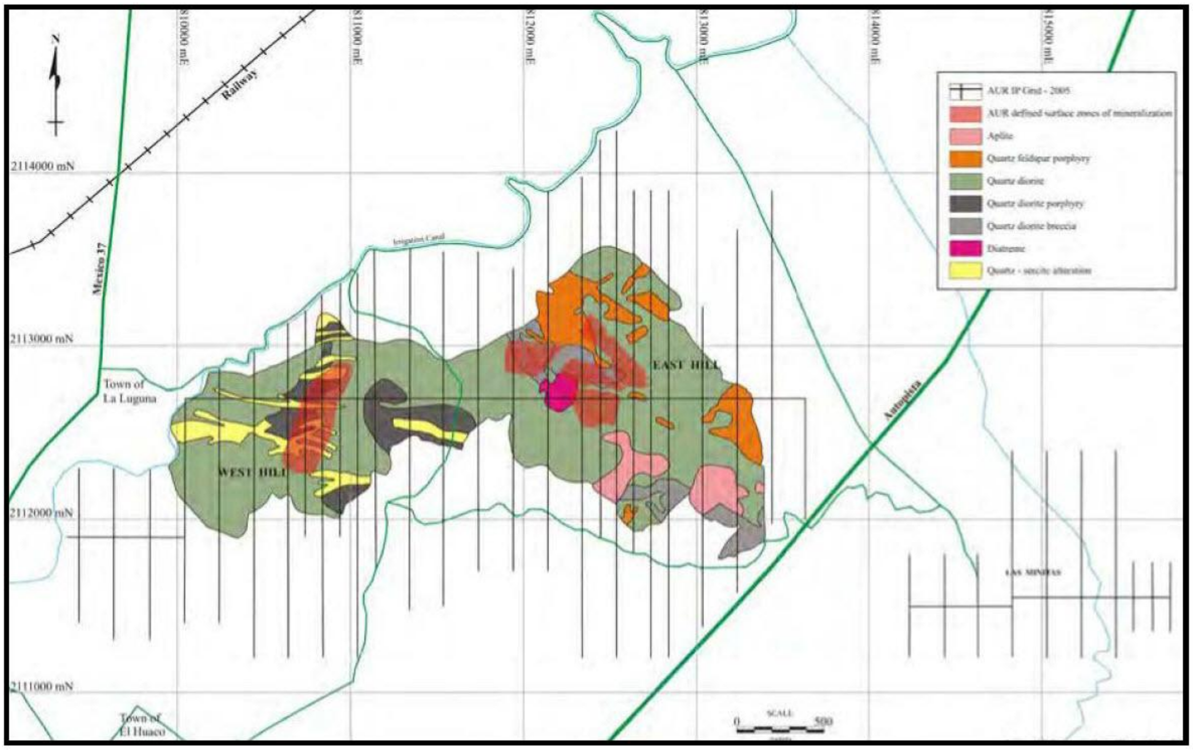
Figure 7.2 Local Geology Map



The Las Minitas prospect (south-east from La Verde in Figure 7.3) is located roughly 2 km east-southeast of the East Hill. It consists of a narrow, east-west-trending zone of propylitic / silica alteration with associated weak chalcopyrite-pyrite mineralization.

Descriptions of the various rock units at the Property are summarized below and are extracted from Wilson 2005a, b, Weston et. al. 2006, and Chamberlain 2009.

Figure 7.3 Geology of the La Verde Intrusive Complex



Intrusive rocks at La Verde are sub-alkaline and range in composition from diorite to granodiorite.

Various types of brecciated intrusive rocks are common, specifically at the East Hill, and host much of the copper mineralization discovered there to date.

7.3.2 Quartz-Diorite Group

Quartz diorite forms the dominant rock type on the Property. It is the host rock in the West Hill and both the host rock and the most significant component of clasts in the breccias.

The mineralogy of quartz-dioritic rocks at La Verde consists dominantly of plagioclase, quartz, biotite, actinolite with varying amounts of hornblende, magnetite and a variety of alteration products including chlorite-epidote-carbonate-sericite / clays-iron (Fe)-titanium (Ti) oxides-Fe/Cu sulphides. In hand sample, fresh quartz-diorite is dark grey, coarse-grained, weak to strongly feldspar porphyritic and moderate to strongly magnetic. All dioritic rocks are affected by late stage magmatic deuteric alteration of pyroxene to a combination of hornblende-actinolite-magnetite such that very little if any primary pyroxene is preserved. Subsequent chlorite alteration of amphibole is fairly ubiquitous but generally quite weak. Fresh potassium feldspar is rare and is most often turbid in thin section, variably altered to sericite and clays.

7.3.3 Feldspar +/- Quartz Porphyritic Felsic Intrusive Group

Felsic intrusive rocks have been restricted to the East Hill. Based on available core and limited outcrops there are at least four main types of felsic intrusive rocks present as breccia clasts or as lithological units:

- feldspar-biotite porphyry
- hornblende-quartz-feldspar porphyry
- hornblende-feldspar-quartz-eye porphyry
- fine to medium-grained aplite

Hornblende-quartz-feldspar porphyry is the most abundant type of felsic intrusion on East Hill. Detailed mapping would be required to substantiate this idea, as feldspar-biotite porphyry and hornblende-quartz-feldspar porphyry have historically been mapped as a single unit.

7.3.4 Breccia

Brecciated quartz-diorite, feldspar±quartz porphyritic felsic intrusions host the bulk of copper mineralization. The breccias show a wide range of characteristics, commonly complicated by secondary alteration and mineralization. There are three types of breccia that have been recognized at La Verde. These three types have been subdivided according to cement versus matrix fill, lithofacies and components (Chamberlain 2009).

Two hydrothermal events have been recognized. The initial stage is associated with the copper mineralizing event. The secondary stage crosscuts the lithology. A possible third event involves tourmaline ±hematite cemented quartz and sericite altered diorite clast breccias.

7.4 Structure

No significant faults have been identified in drilling or in outcrops although there are numerous minor structures / faults trending east-west, northeast-southwest and southeast-northwest. Several regional structures (faults and / or contacts) have been identified by airborne magnetic and electromagnetic geophysics as well as ground magnetometer and IP surveys run over the Property. The interpreted aeromagnetic geophysical survey data indicates that there could be a left-handed slip fault (or fold) separating the East Hill from the West Hill, striking at approximately 045° azimuth. Very limited drilling in the projected fault area has not identified this structure.

No folding has been recognized in the drilling or in outcrops.

7.5 Alteration

In the West Hill area vein-controlled phyllic and propylitic alteration is oriented east-northeast within the quartz-gabbro porphyry stock, east-southeast within surrounding quartzdiorite, and east-west common to both. Historic mapping indicates linear east trending zones of quartz-sericite alteration that can be traced for up to 900 m on surface.

Alteration on East Hill is limited to mineralized zones and associated breccias although there are a few broad zones of calcium-sodium metasomatism found on the East Hill. Potassic alteration was observed in drill cores on the north side of the East Hill deposits. Outside of these zones, little to no pervasive wall-rock alteration is present.

7.6 Mineralization

Mineralization discovered to date at La Verde occurs predominantly within five main zones, four of which are situated within the East Hill area. Detailed descriptions of all five zones are contained in the Weston and MacLean 2006 report, but are summarized in the following sections.

7.6.1 West Hill

The West Hill deposit has a north-northeast-trending elliptical outline, approximately 300 to 500 m wide by 800 m in length. Mineralization on the West Hill deposit occurs in veins within phyllic-propylitic alteration envelopes and the mineralization outcrops in a number of locations, commonly marked by the appearance of limonite-malachite stained waste piles and / or sparse vegetation “kill” zones.

The West Hill consists of steeply-dipping to vertical bands of epidote-sericite-quartz-calcite-altered quartzdiorite and quartz-gabbro porphyry containing veinlets, veins and disseminations of pyrite-chalcopyrite ±pyrrhotite-arsenopyrite.

Phyllic-propylitic alteration veins strike east-northeast within the quartz-gabbro porphyry stock, east-southeast within surrounding quartz-diorite, and east-west common to both. Historic mapping indicates linear east-trending zones of quartz sericite alteration that can be traced for up to 900 m on surface.

In drill core, mineralized veins vary from a few centimetres to several metres in width. Zones of mineralization often consist of groups or swarms of mineralized veins with increased alteration intensity correlating with increased vein density.

A low vein density usually corresponds with weakly mineralized zones, with sulphides generally restricted to veins. Wall-rock alteration occurs as discrete envelopes around the veins. Surrounding host rocks are generally unaltered.

Where there is high vein density, alteration in vein selvages coalesces and sulphides commonly occur as disseminations in the vein selvages. The altered and / or mineralized host rock is characteristically lighter in colour, non-magnetic and visually distinct from unaltered quartz-diorite / quartz-gabbro host rock.

Localized high grade mineralization has been observed in drill core. As an example, drillhole C11-04-02 intersected massive chalcopyrite veins of greater than 2 m core-width. The non-true-width sample assayed 13.88% Cu over 3.99 m. Underground mapping indicates that these wider zones of high grade mineralization pinch and swell over the width of the drift and plunge either steeply east or steeply west.

It is difficult to correlate the mineralized structures, due to the variability of copper grades and widths between drillholes. As a result, previous resource estimates have used phyllic-propylitic alteration zones as a guide for connecting the mineralized intervals. These have demonstrated the vertical continuity of the alteration system. Indeed, past and present drilling has intersected zones of alteration / mineralization to a depth of 0 masl or 600 m below the top of West Hill.

Aur tried to model the West Hill vein structures by vertical and horizontal sections spaced 50 m apart using various grade and waste parameters in an attempt to identify the orientation of broad corridors of lower grade mineralization, and the orientation of narrow higher grade vein structures. Unfortunately, due to the complexity of the mineralizing system, a representative geological solid model which confidently reflected the size and shape of the West Hill Zone has not yet been successfully produced. From this data, there appears to be a northeast-southwest trend to the mineralization.

7.6.2 East Hill

7.6.2.1 #1 Zone (Breccia)

The #1 Zone is exposed at surface between sections 812300 to 812600 m E as a roughly east-trending series of malachite-stained quartz-diorite breccia outcrops. Weathering and copper-oxide mineralization is generally restricted to within 20 m of surface.

Mineralization occurs principally as disseminations and blebs of bornite-chalcopyrite± arsenopyrite-molybdenite throughout the matrix and within fragments of brecciated quartz-diorite. Some mineralization also occurs as planar fracture veinlets and joint coatings within large quartz-diorite blocks and unbrecciated quartz-diorite wall-rock. The breccias are generally weakly to moderately altered to an assemblage of chlorite-sericite-calcite±quartz. Zones of patchy pink alteration have been interpreted to be potassically altered (potassium feldspar-chloritesericite± quartz-epidote-calcite) fragments and are almost always associated with higher grade bornite±molybdenite-arsenopyrite mineralization within the #1 Zone.

The #1 Zone forms a well-defined, fairly consistently mineralized body at a cut-off grade of 0.3% Cu. It has rough dimensions of 150 to 220 m wide, 100 to 150 m deep and 200 m long. The western boundary of the zone is quite sharp as it passes into sporadically mineralized breccia. The zone thins into two separate mineralized bodies to the east in section 812450 m E, before it apparently pinches out in section 812500 m E. This area is poorly delineated, but the IP chargeability inversion data indicates that there is potential for undiscovered mineralization at depth or along strike.

The lower contact of the #1 Zone dips gently to the north between 20° to 30° and roughly parallels the breccia / quartz-diorite footwall contact, but extends into unbrecciated footwall up to 90 m in section 812350 m E. Mineralization within the footwall differs in that it occurs principally as veinlets and disseminations associated with a mix of alteration assemblages including quartz-potassium feldspar-sericite-chlorite. Previous work suggests this footwall style mineralization may represent part of a feeder system to the overlying mineralized breccias.

7.6.2.2 #2 Zone

Copper mineralization within the #2 Zone generally does not begin until 50 to 150 m below surface. However, the western extension between lines 812300 m E to 812350 m E comes within 5 to 10 m of surface and is clearly visible as a chargeability inversion anomaly. The upper 100 to 200 m of this zone is hosted within a rootless, unbrecciated, quartz-feldspar porphyry which shows little alteration outside of the mineralized veinlets. Below the porphyry, mineralization continues down into quartz-diorite and quartz-diorite breccia.

Within quartz-feldspar porphyry, the mineralization is characterized by 1 to 2 mm wide sub-vertical sheeted veins of chalcopyrite-bornite±molybdenite-arsenopyrite within discrete alteration envelopes of quartz-chlorite-sericite. The veins are generally sub-parallel but conjugate sets have been observed. Mineralization can also occur as finely disseminated chalcopyrite-bornite associated with sericite-chlorite altered amphibole crystals within quartz-feldspar porphyry. At depth, the host rock changes to quartz-diorite and quartz-diorite breccia with local zones of mixed breccia containing quartz-feldspar porphyry fragments intermingled with quartz-diorite breccia. The mineralization style largely remains as fracture veinlets and fine disseminations, often associated with sericite-chlorite-quartz± potassium feldspar-calcite alteration envelopes. However, with depth a strong and pervasive calcium-sodium-silicon (Ca-Na-Si) metasomatism pervades both quartz diorite and quartz diorite breccia often masking igneous textures. This widespread alteration type may indicate the presence of a large felsic intrusion at depth as there appears to be a close spatial relationship at surface between calcium-sodium ±silicon metasomatism and quartz-feldspar porphyry.

The #2 Zone trends east-southeast and at a cut-off grade of 0.3% Cu forms a well-defined, consistently mineralized, vertically dipping tabular body between 50 to 150 m wide, up to 250 to 300 m in strike length, and with a dip extent of 350 m that is open at depth. The bottom of the mineralizing system has yet to be defined for most of its strike length and there is potential to add significant tonnage to this zone at depth.

Previous work indicates a lateral zonation in sulphide mineralogy as the zone tends to be chalcopyrite dominant in the south and bornite dominant in the north. As well, in sections with deep drillhole penetration the grade appears to decrease gradually with depth.

The western boundary of the #2 Zone is poorly defined and currently extends into the wide and deeply mineralized area of the #4 Zone. Mineralization within this region contains some characteristics of the #2 Zone, with fracture veins of bornite±chalcopyrite within quartz-feldspar porphyry recorded in previous drill logs as far west as section 812250 m E.

To the east, the down dip extent of the #2 Zone appears to merge with the #1 Zone as the mineralization shifts from #1-style blebs to #2-style sub-parallel fracture veins in a continuously mineralized section intersected in hole C11-05-06. This area is also characterized by a lateral displacement of the #2 Zone to the south before it gradually pinches to less than 20 m width at its eastern boundary. The cause of this south, right-lateral shift is unclear as the majority of late faults in the East Hill have been previously mapped as trending northeast, east and southeast, and there does not appear to be a southerly shift in the outline of the #1 Zone in this area.

7.6.2.3 #3 Zone

The #3 Zone is intermittently exposed on surface for roughly 400 m between sections 812325 to 812725 m E as a southeast trending zone of malachite-stained breccias of quartz-feldspar porphyry in the northwest and quartz-diorite breccia extending from the centre to the southeast. Mixed breccias of quartz-feldspar porphyry and quartz-diorite occur within the contact areas between the two main rock types. Weathering and copper oxide mineralization is generally restricted to within 20 m of surface except in zones of faulting where surface waters have percolated to greater depths.

Mineralization within the #3 Zone occurs as blebs, disseminations and veins of chalcopyrite-bornite within sericite/kaolinite-chlorite-quartz±calcite-potassium feldspar altered framework supported breccias of quartz-feldspar porphyry, quartz-diorite and mixed varieties of the two. Quartz-feldspar porphyry breccias tend to be more tightly packed with less rotational movement than brecciated quartz-diorite, and in several cases brecciated dykes of quartz-feldspar porphyry are observable in drillcore.

At a cut-off grade of 0.3% Cu, the #3 Zone forms a fairly continuously mineralized southeast-trending elongate corridor that extends for roughly 500 m. In cross section, however, this corridor consists of one to three mineralized sections which merge and bifurcate along strike. In the northwest the zone consists of two mineralized panels dipping to the northeast at 50 to 60° with long axis 120 to 170 m and width 30 to 80 m. At roughly 812550 m E these two panels merge to form a single intermittently mineralized body roughly 150 m wide and 100 to 220 m long. By 812650 m E, the zone splits again to form two mineralized zones, one of which exhibits a strong vertical aspect.

7.6.2.4 #4 Zone

Copper mineralization within the #4 Zone does not outcrop and generally does not begin until 100 to 300 m below surface. Historic mapping indicates that it is manifested at surface as a 60 to 90 m wide northwest trending band of weakly to non-mineralized green sericitized crackle breccia between sections 811925 to 812250 m E. Near its eastern boundary in section 812225 m E, the #4 Zone approaches within 20 to 30 m of surface where it appears to merge with the #2 Zone.

Historically the #4 Zone was the last to be discovered at La Verde. Due to economic restrictions inherent with the depth of mineralization, its dimensions were never properly delineated, and as such there is good potential to add significant tonnage to this zone in the future.

Mineralization within the zone is characterized by blebby to disseminated chalcopyrite-pyrite±bornite-arsenopyrite occurring principally within the matrix to a hydrothermal “crackle” breccia cemented by a combination of quartz-calcite-tourmaline-chlorite gangue. Brecciated host rocks are dominantly quartz-diorite, but several intervals of brecciated quartz-feldspar porphyry indicate the area was intruded by several felsic dykes prior to, and possibly coincidental with brecciation. Indeed, a large feldspar±quartz porphyritic stock occurs immediately north of the #4 Zone, and a mineralized feldspar-quartz-biotite bearing granodiorite has been intersected at depth in drilling to the north in hole C11-05-15.

An early episode of silicification associated with the #4 Zone is evident as numerous cross-cutting quartz veinlets at surface, and coarse, drusy quartz±tourmaline veins up to 4 m in core-width observed at depth. Subsequent brecciation of the host rock ensued, as evidenced by the presence of quartz-diorite fragments with quartz veinlets near surface, and brecciated quartz±tourmaline vein fragments at depth. Pervasive wall-rock alteration followed (with zoned breccia fragments containing light grey sericite/kaolinite-calcite altered cores and dark green chlorite-tourmaline altered margins) along with cementation of the breccia matrix by a combination of quartz-calcite-tourmaline-chlorite-chalcopyrite.

At a cut-off grade of 0.3% Cu (which includes 9 m internal waste), the zone is a large, roughly east-west, irregularly shaped body with strike length up to 300 m, width varying from 100 to 350 m, and an open down-dip extent up to 400 m. The western boundary of the zone is quite sharp, terminating abruptly in section 811936 m E. To the east the zone pinches from 200 m width in section 812085 m E to less than 100 m width in section 812130 m E before it appears to merge with the #2 Zone. A large chargeability inversion anomaly to the south suggests sulphide mineralization may continue for several hundred metres beyond the presently assumed southern limit of mineralization. The northern boundary appears to dip steeply to the north between 55 to 75° until section 812225 m E where it dips vertically, similar to the #2 Zone. The upper surface of the #4 Zone is highly irregular in form and may in fact be sub-horizontal with sub-vertical mineralized shoots emanating upward from the main mineralized zone.

7.6.3 Las Minitas

The Las Minitas prospect occurs 2 km east-southeast of the East Hill, just east of the town of Las Minitas. The prospect is characterized by an area, roughly 100 by 50 m, of exposed malachite chips and a few shallow historic trenches. Recent geophysical and geochemical surveys revealed an east-west trending IP inversion anomaly with coincident copper-in-soil anomaly over a strike length of roughly 700 m. Subsequent drilling intersected significant copper mineralization in only one of five holes drilled to test the area.

Hole C11-05-20 confirmed the existence of shallow copper mineralization of limited width and strike extent immediately east of the historic showing. Mineralization here occurs as a 15 m wide discrete band of chalcopyrite-pyrite disseminations and veins within pervasively quartz-chlorite-epidote altered massive medium-grain quartz-monzodiorite. The mineralized band is interpreted from geophysics to dip moderately (~45°) to the south and was not intersected in any other holes located 200 m, 400 m and 640 m to the east.

8 DEPOSIT TYPES

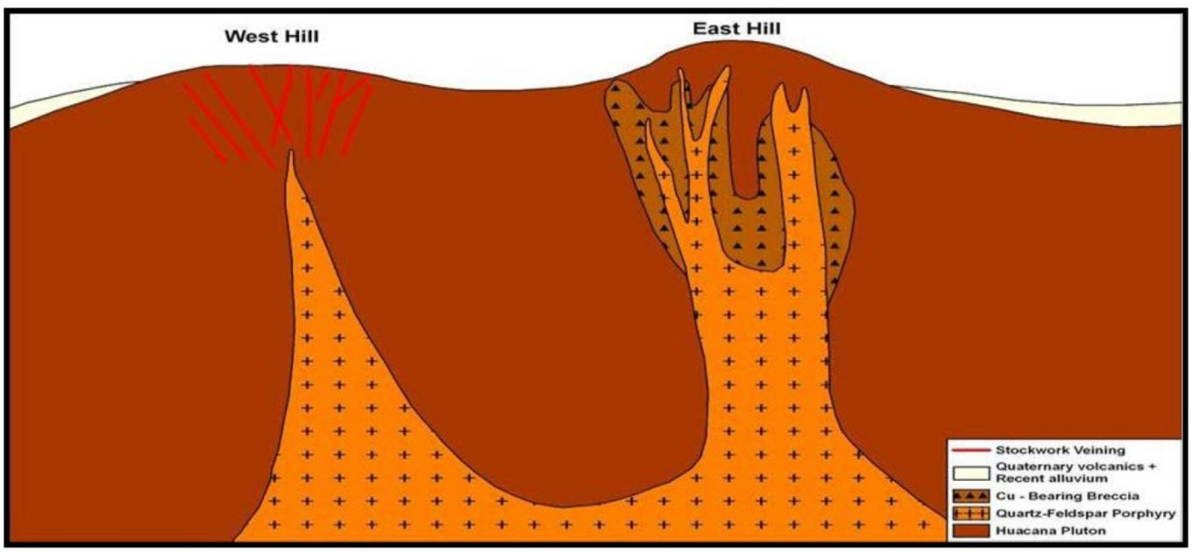
This section has been extracted and adapted from the Tetra Tech September 2012 Report.

The La Verde deposit represents the upper portions of a copper porphyry deposit.

Several models have been developed for the La Verde deposit. One of these is illustrated in Figure 8.1, put forth by Weston and MacLean (2006). The figure illustrates the relationship between intrusive breccias (East Hill) and buried porphyry (West Hill) copper systems at depth.

Examples of similar mineralized intrusive breccias are the Willa breccia pipe in southeastern BC (Wong and Spence, 1995), the mineralized breccia pipes of Copper Basin, Arizona (Johnston and Lowell, 1961), and numerous copper bearing tourmaline breccia pipes in Chile as described by Sillitoe and Sawkins (1971).

Figure 8.1 Idealized Porphyry Copper Model for La Verde

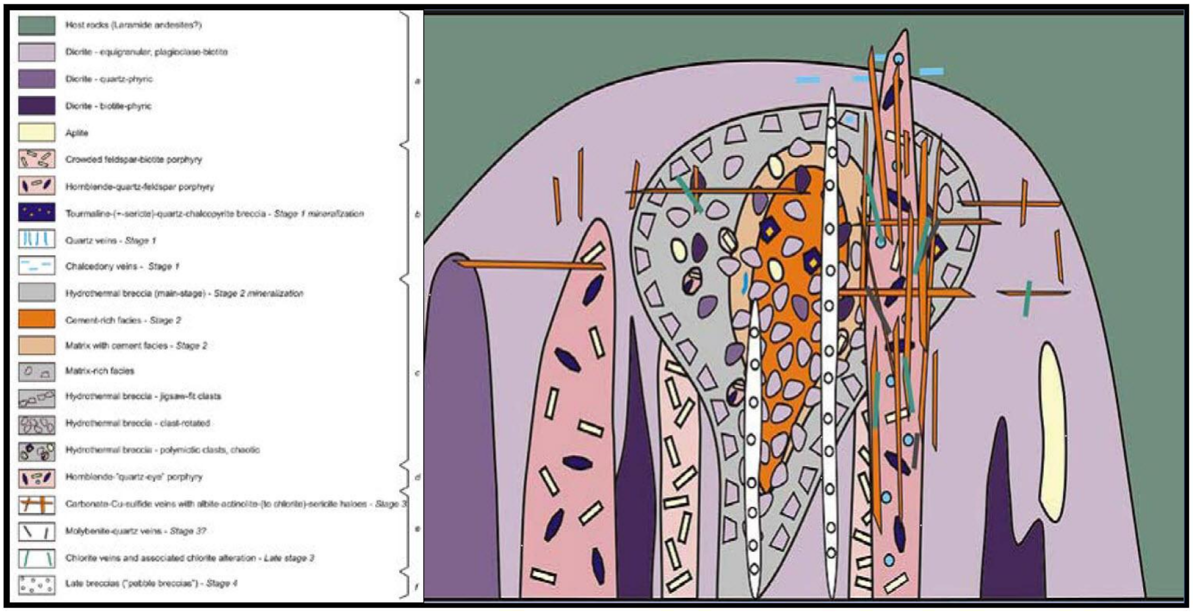


Source: Weston and MacLean (2006)

The existence of a separate intrusion of quartz-gabbro porphyry is debatable, as past geologists and recent observations suggest that porphyritic quartz-gabbro may simply be a contiguous phase of the quartz-diorite intrusion forming the West Hill.

An alternative model for the East Hill deposit has been developed by Chamberlain (2009) and is illustrated in Figure 8.2.

Figure 8.2 Idealized Porphyry Copper Model for East Hill



Source: Chamberlain (2009)

This model includes the initial diorite intrusion which is overprinted by a series of pulsed breccia facies with and without copper mineralization. A series of carbonate copper sulphide-sheeted veins cross cut the lithology followed by small-scale bornite±chalcopyrite veinlets that cross cut the porphyry and diorite. This is followed by chlorite veins and pervasive chlorite alteration which occurs and replaces mafic mineral phases. The final recognized event is the emplacement of a hydrothermal breccia with calcite-quartz-tourmaline-chalcopyrite cement.

9 EXPLORATION

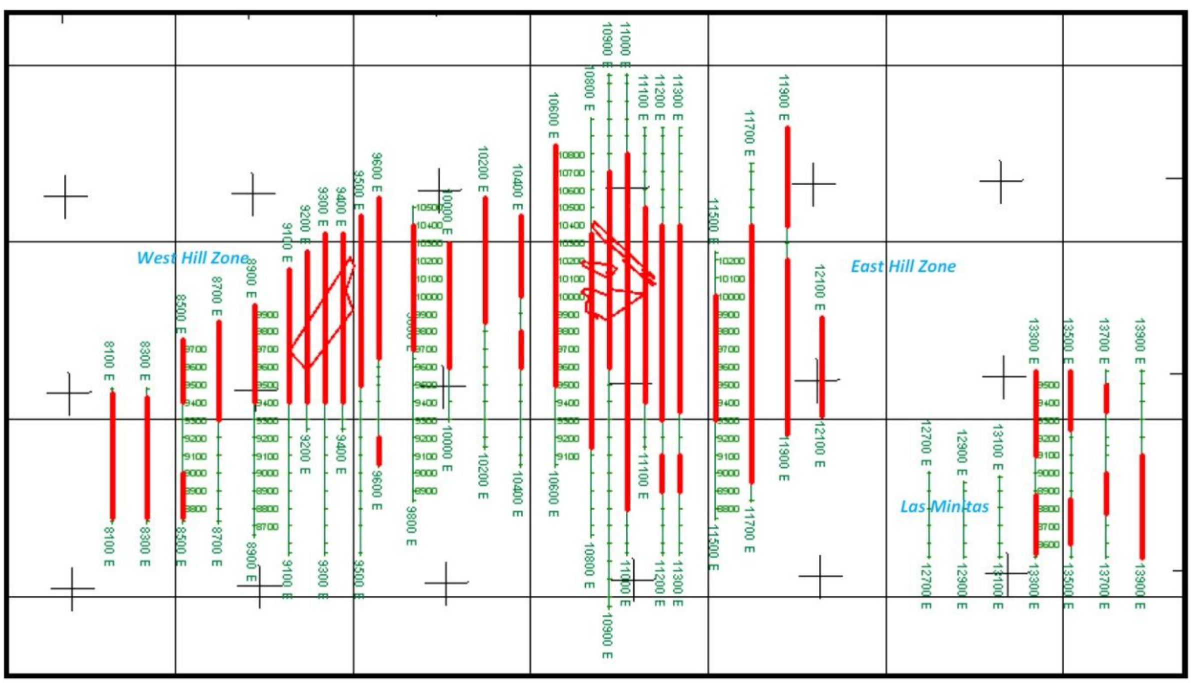
This section has been extracted from the Tetra Tech September 2012 Report.

9.1 2010 Exploration

Based on the “Property of Merit” report (Makepeace, 2010) recommendations, MHill carried out an exploration program in 2010. A 150-line km pole-dipole IP program was completed in October 2010 (Lajoie 2010), and was conducted by Pacific Geophysical Ltd (Pacific Geophysical) of Vancouver, BC. The geophysical survey covered the East and West Hill as well as to the northern portion of the Huacana Batholith area to the south end of the concessions. A standard pole dipole array with a=100 m and the majority of the separation was n =1 to 12 which improved depth detection and target delineation capability.

The results indicated that the southern portion of the concession hosted several IP anomalies that can be attributed to hematite in quartz monzonite (Figure 9.1). The hematite mineralization appears to be highly chargeable at times which can create shallow IP anomalies. Several drill target areas were discovered from the survey including Huaco, Huaco South, Tziritzicuaro North. A review of the previous data confirmed that the strongest chargeabilities in the vicinity of West Hill mineralization were toward the north. This suggests that the area between West Hill deposit and the main irrigation canal has mineral potential.

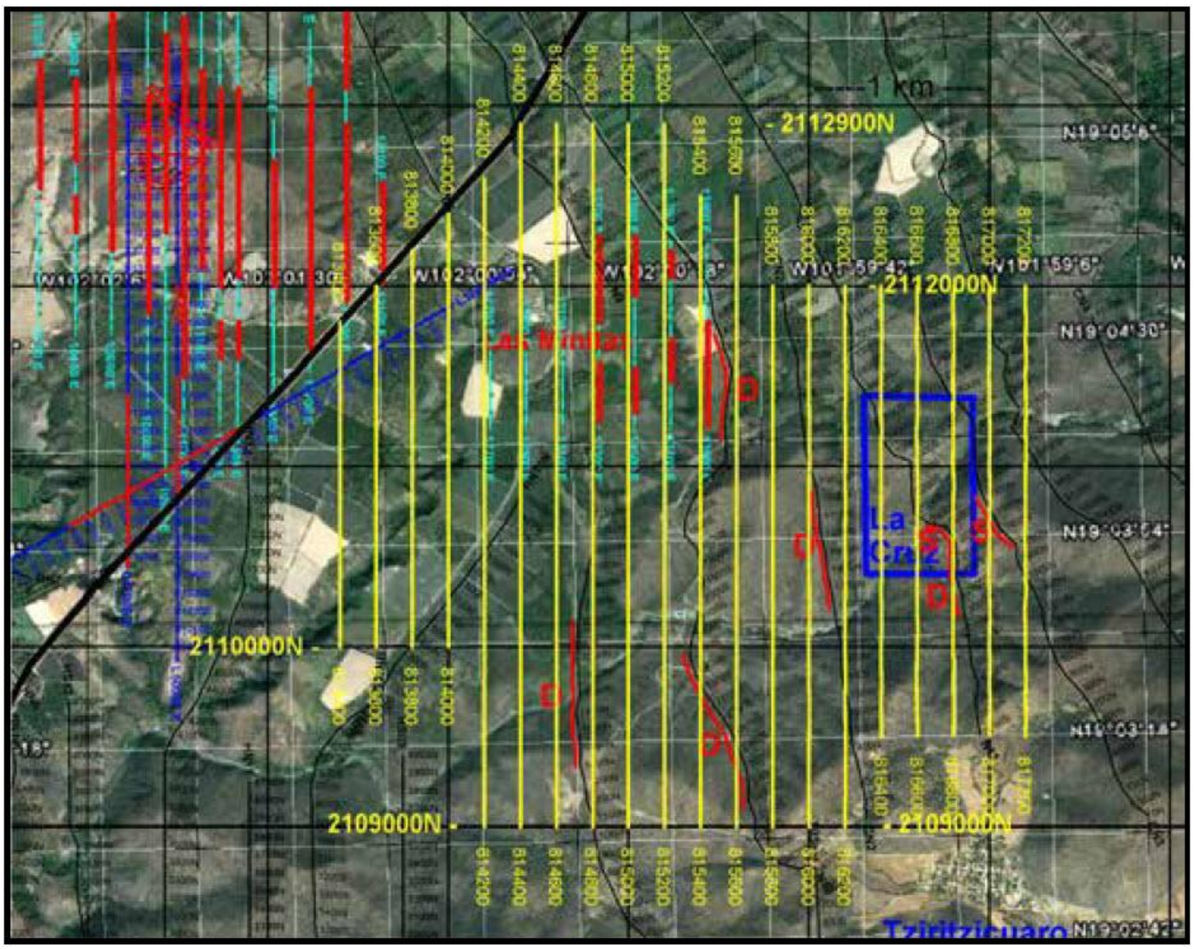
Figure 9.1 IP survey. Anomalies shown in red.



9.2 2012 Exploration

In 2012, a 59 line-km pole-dipole IP survey was completed in the Las Minitas area, southeast of East Hill. The 2012 survey was also conducted by Pacific Geophysical. The survey was conducted on a UTM grid with 200 m line spacing. Figure 9.2 shows the location of the 2012 lines relative to previously completed survey lines in the Las Minitas region; the La Cruz concession is indicated by the blue rectangle.

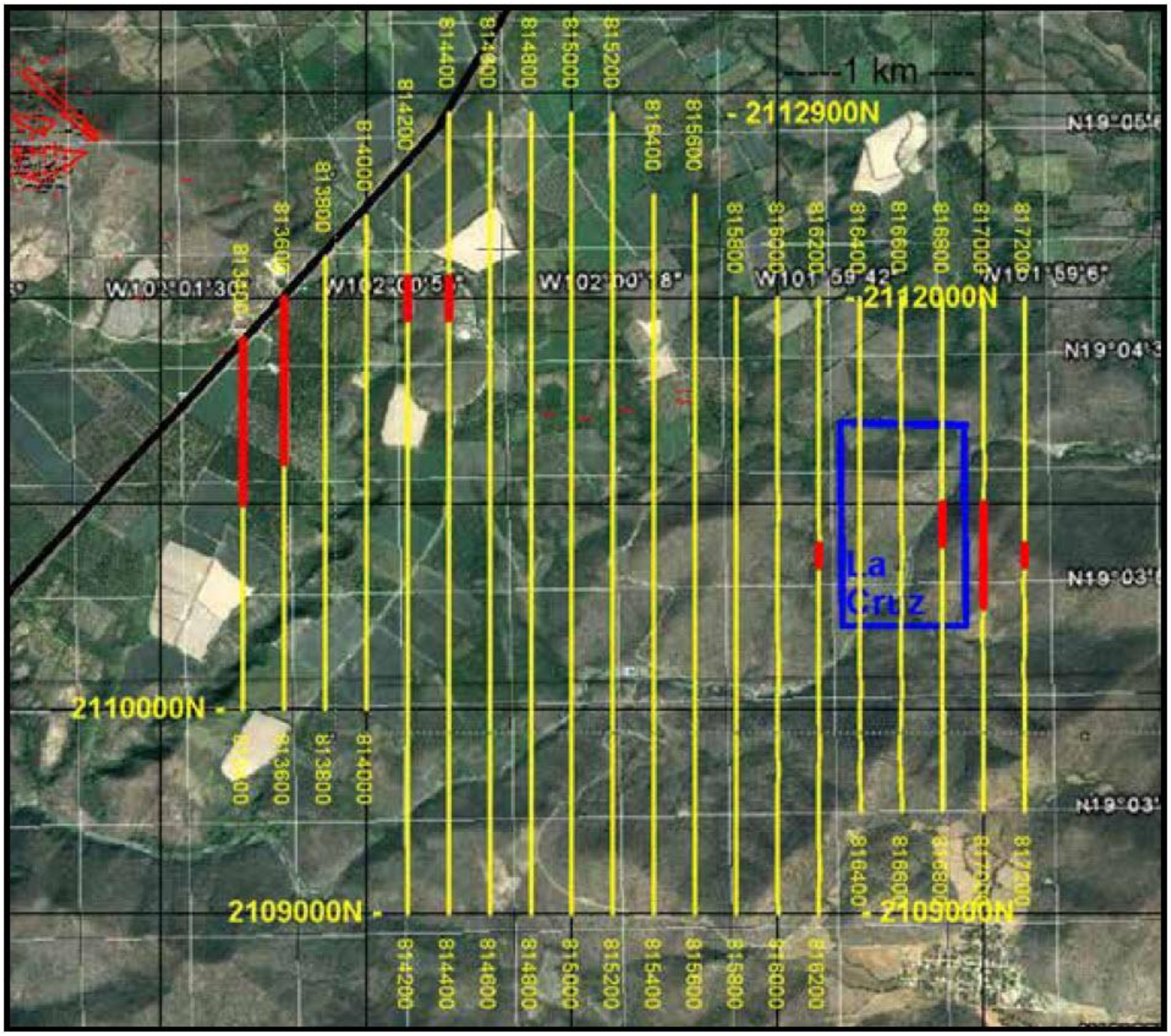
Figure 9.2 2012 IP Survey Relative to Previous Surveys



Yellow Lines = 2012 IP Survey. Blue/Green/Red Lines=Previous Survey

The results show several areas with anomalous IP responses that are prospective for further exploration. These anomalous areas are indicated by the red portions of the survey lines on Figure 9.3. These results identify three areas of potential interest with anomalous IP responses, however the IP responses are lower than those of the West Hill and East Hill mineralized zones. Previous drill testing of other IP anomalies in this area of the Property have intersected hematite, and not copper, mineralization.

Figure 9.3 2012 IP Survey Anomalous IP responses



Red Lines = anomalous IP chargeability responses

10 DRILLING

This section has been extracted from the Tetra Tech September 2012 Report.

10.1 2010 and 2011 Drilling

The previous resource estimate (Maunula, 2012) was based on drilling completed up to the end of 2011. MHill completed drilling on 20 drillholes in 2010 and an additional 24 drillholes in 2011. Combined with the prior exploration work, the database contains 629 drillholes totalling 107,768.89 m. Figure 10.1 illustrates the distribution of 2010 and 2011 drillholes (blue squares) relative to the 2012 drillholes as well as holes completed by previous operators. The MHill drill program used two skid-mounted drill rigs from Falcon Drilling. The holes were collared using HQ to a depth of approximately 400 to 500 m and then reduced to NQ.

10.1.1 2010 Drill Program

Twenty diamond drillholes were completed on the Property in 2010 for a total of 12,279.6 m. Twelve holes for a total of 6,717.45 m were completed to verify historical drillhole mineralization. Seven holes (4,871 m) were completed to expand the known mineralization, especially at depth. A 687.25 m diamond drillhole was completed to identify any mineralization associated with an IP anomaly that had been discovered from the 2010 geophysical survey.

10.1.2 2011 Drill Program

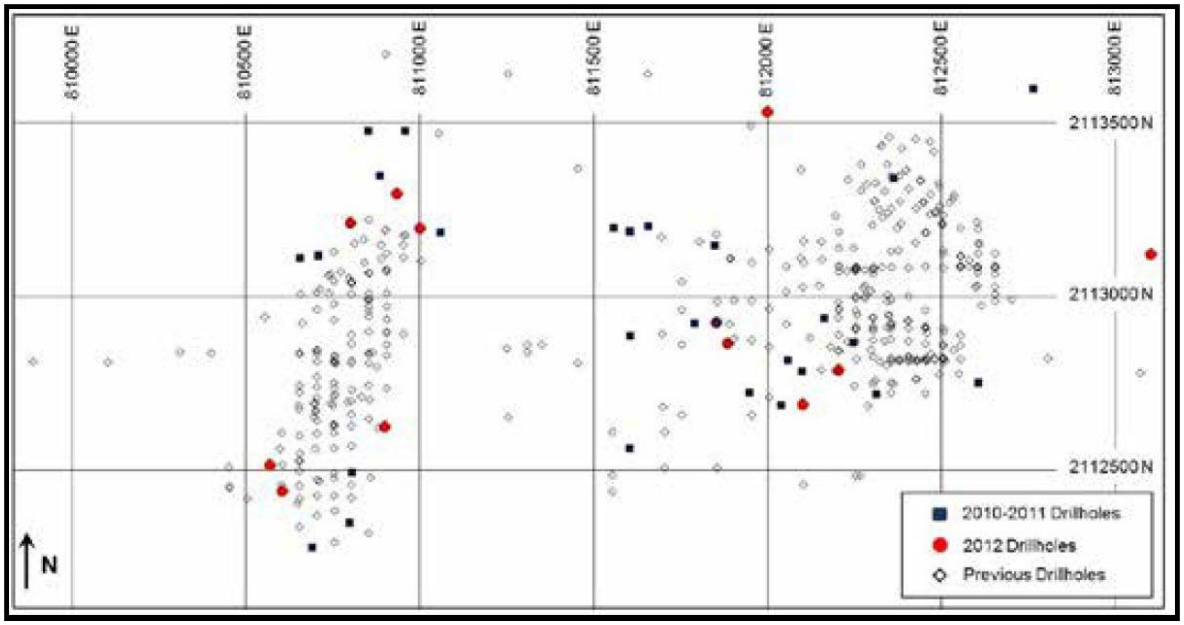
Twenty-four drillholes were completed in 2011 for a total of 15,918.0 m. Two drillholes targeted the Noranda IP target La Verde Sur. The remaining 22 drillholes focused on infill drilling of East and West Hill.

10.2 2012 Drilling

Twelve drillholes were completed in 2012 for a total of 6770.61 m. All but one drillhole was more than 400 m in length. The 2012 drilling focused on expanding the known mineralization of both East and West Hill mineralization, including some infill and some delineation drilling. As in the 2010 and 2011 drilling programs, the 2012 program used two skid-mounted drill rigs from Falcon Drilling. The holes were collared using HQ to a depth of approximately 400 to 500 m and then reduced to NQ.

Figure 10.1 shows the location of 2012 drillholes (red closed circles) relative to previous MHill drilling. Table 10.1 summarizes significant results intersected in 2012 drilling.

Figure 10.1 La Verde Drillhole Collar Map



Note: Grid display easting (E) and northing (N) values (units in metres).

Table 10.1 2012 Drilling – Summary of Significant Results

Hill	Drillhole	From (m)	To (m)	Length (m)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)
East	**LV12-037**	118.70	125.00	6.30	0.423	0.160	1	0.001
		129.00	135.00	6.00	0.119	0.042	1	0.001
		140.00	148.00	8.00	0.250	0.196	2	0.001
		221.80	229.50	7.70	0.116	0.043	1	0.009
		261.50	265.50	4.00	0.102	0.006	1	0.006
		277.30	280.90	3.60	0.145	0.078	1	0.011
		315.40	319.75	4.35	0.116	0.007	1	0.001
		329.20	332.45	3.25	0.123	0.033	1	0.001
East	**LV12-038**	35.80	39.80	4.00	0.266	0.015	1	0.001
		112.00	114.85	2.85	0.118	0.007	2	0.001
		216.80	224.80	8.00	0.105	0.004	1	0.001
		230.80	234.80	4.00	0.105	0.014	1	0.001
		248.50	265.70	17.20	0.108	0.003	1	0.001
		271.00	276.50	5.50	0.154	0.011	1	0.001
		276.50	303.70	27.20	0.576	0.275	1	0.001
		303.70	324.25	20.55	0.121	0.041	1	0.001
		333.15	336.65	3.50	0.599	0.127	11	0.001
		336.65	367.20	30.55	0.132	0.036	1	0.001

Hill	Drillhole	From (m)	To (m)	Length (m)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)
		370.40	380.10	9.70	0.159	0.063	1	0.001
		381.70	384.70	3.00	0.230	0.031	1	0.001
		392.70	396.70	4.00	0.117	0.032	1	0.001
		396.70	461.25	64.55	0.371	0.121	4	0.004
		464.60	468.60	4.00	0.102	0.026	1	0.002
		483.25	498.10	14.85	0.096	0.042	1	0.001
		498.10	675.30	177.20	0.536	0.041	5	0.006
		675.30	682.80	7.50	0.179	0.002	2	0.002
East	**LV12-039**	89.05	92.20	3.15	0.184	0.038	2	0.001
		236.80	250.80	14.00	0.349	0.028	2	0.001
		332.00	346.00	14.00	0.234	0.008	3	0.001
		354.00	358.15	4.15	0.249	0.023	4	0.001
		360.30	368.00	7.70	0.106	0.023	2	0.001
		376.00	603.05	226.95	0.673	0.076	6	0.003
		617.00	621.00	4.00	0.131	0.008	2	0.004
		627.00	631.00	4.00	0.435	0.005	5	0.014
		646.00	651.50	5.50	0.221	0.004	2	0.012
		661.60	673.35	11.75	0.932	0.009	6	0.009
West	**LV12-040**	59.00	69.25	10.25	0.121	0.002	1	0.001
		69.25	74.90	5.65	1.659	0.009	4	0.001
		143.00	147.00	4.00	0.265	0.001	1	0.001
		161.70	165.10	3.40	0.120	0.001	1	0.001
		181.90	216.00	34.10	0.855	0.003	2	0.001
		226.00	229.60	3.60	0.100	0.001	1	0.001
		231.20	236.70	5.50	0.828	0.001	3	0.001
		236.70	244.70	8.00	0.109	0.003	1	0.001
		256.60	259.30	2.70	0.282	0.002	2	0.001
		288.30	291.70	3.40	0.259	0.002	1	0.001
		320.20	323.10	2.90	0.250	0.001	1	0.001
		327.30	331.50	4.20	0.222	0.003	1	0.001
		364.60	372.35	7.75	1.477	0.006	2	0.001
		383.05	387.35	4.30	0.263	0.002	1	0.001
West	**LV12-041**	69.00	76.45	7.45	0.423	0.011	3	0.007
		93.00	106.00	13.00	0.601	0.011	2	0.003
		185.00	193.00	8.00	1.324	0.005	7	0.002

Hill	Drillhole	From (m)	To (m)	Length (m)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)
		223.50	227.30	3.80	0.581	0.005	4	0.001
		249.55	259.00	9.45	0.269	0.005	1	0.003
		284.50	311.40	26.90	0.286	0.012	1	0.002
		337.30	408.00	70.70	0.857	0.003	3	0.001
		419.00	438.00	19.00	1.946	0.003	5	0.001
		461.60	477.50	15.90	0.614	0.003	2	0.001
		487.00	496.00	9.00	1.898	0.180	5	0.001
		533.60	539.00	5.40	0.641	0.001	2	0.001
		549.00	553.70	4.70	0.608	0.001	1	0.001
		581.20	585.00	3.80	1.397	0.003	3	0.001
		594.25	597.20	2.95	0.612	0.001	1	0.001
		605.20	640.10	34.90	0.603	0.002	1	0.001
East	**LV12-042**	47.30	50.70	3.40	0.266	0.019	1	0.001
		100.75	243.00	142.25	0.287	0.007	3	0.001
		289.60	293.70	4.10	0.229	0.008	2	0.001
		325.60	330.70	5.10	0.239	0.001	1	0.001
		359.15	557.95	198.80	0.375	0.078	5	0.004
West	**LV12-043**	30.80	34.15	3.35	0.237	0.004	2	0.001
		115.60	119.80	4.20	1.008	0.046	5	0.001
		176.00	203.00	27.00	0.687	0.006	2	0.001
		211.20	221.30	10.10	0.689	0.036	2	0.001
		283.50	287.50	4.00	0.238	0.005	1	0.009
		311.20	316.25	5.05	0.295	0.017	1	0.002
		336.50	345.80	9.30	0.682	0.021	3	0.005
East	**LV12-044**	197.10	201.15	4.05	0.237	0.018	2	0.002
		238.00	246.00	8.00	0.295	0.016	5	0.001
		278.55	283.30	4.75	0.245	0.022	4	0.003
		303.50	307.50	4.00	0.231	0.021	3	0.001
		366.20	369.65	3.45	0.390	0.050	3	0.001
		400.50	406.50	6.00	0.200	0.140	2	0.002
		432.85	436.70	3.85	0.213	0.113	2	0.002
		483.00	487.00	4.00	0.563	0.095	5	0.005
		506.40	510.40	4.00	0.647	0.504	7	0.003
		533.70	575.35	41.65	0.555	0.014	5	0.003
		657.00	746.50	89.50	0.526	0.014	7	0.007

Hill	Drillhole	From (m)	To (m)	Length (m)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)
West	**LV12-045**	115.00	118.50	3.50	0.250	0.002	2	0.003
		133.15	150.90	17.75	1.246	0.019	5	0.003
		233.00	255.90	22.90	0.972	0.015	4	0.001
		295.60	299.00	3.40	0.269	0.002	2	0.001
		326.70	352.70	26.00	1.683	0.007	4	0.001
		381.00	395.25	14.25	2.038	0.010	6	0.001
		403.15	410.50	7.35	0.389	0.003	2	0.001
East	**LV12-046**	187.80	227.10	39.30	0.227	0.038	2	0.001
		256.30	259.55	3.25	0.223	0.041	3	0.001
		267.50	274.90	7.40	0.217	0.062	2	0.001
		279.60	283.60	4.00	0.291	0.045	4	0.001
		314.50	319.70	5.20	0.206	0.055	3	0.003
		414.00	418.00	4.00	0.299	0.003	8	0.001
		445.40	452.50	7.10	0.238	0.092	3	0.002
		466.30	472.00	5.70	0.380	0.094	4	0.002
		493.00	528.95	35.95	0.587	0.114	6	0.002
		562.35	611.40	49.05	1.188	0.039	10	0.003
		623.85	630.10	6.25	0.276	0.005	3	0.012
		635.00	641.00	6.00	0.214	0.046	3	0.010
West	**LV12-047**	81.70	86.30	4.60	0.294	0.003	2	0.001
		114.60	123.35	8.75	0.263	0.004	2	0.001
		132.80	140.85	8.05	0.874	0.009	4	0.001
		185.70	188.70	3.00	0.461	0.003	3	0.001
		196.30	205.30	9.00	0.627	0.042	4	0.004
		209.30	213.30	4.00	0.251	0.005	2	0.001
		259.00	263.00	4.00	0.229	0.002	2	0.001
		298.00	330.70	32.70	0.387	0.003	2	0.001
		378.00	381.95	3.95	1.639	0.014	5	0.001
		413.45	422.75	9.30	0.668	0.018	2	0.001
		433.40	437.60	4.20	0.220	0.002	2	0.001
		452.00	458.85	6.85	0.405	0.002	2	0.001
		478.30	480.25	1.95	0.353	0.003	2	0.001
		493.60	584.90	91.30	0.530	0.003	2	0.001
West	**LV12-048**	92.00	94.80	2.80	0.345	0.002	2	0.001
		98.60	102.55	3.95	0.818	0.002	2	0.001

Hill	Drillhole	From (m)	To (m)	Length (m)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)
		115.00	124.25	9.25	0.882	0.002	2	0.002
		146.40	153.85	7.45	0.225	0.002	2	0.001
		179.35	188.70	9.35	0.469	0.004	2	0.001
		208.50	216.00	7.50	0.264	0.002	2	0.001
		222.00	226.00	4.00	0.645	0.004	2	0.001
		244.00	252.00	8.00	0.634	0.005	2	0.001
		261.60	269.25	7.65	1.010	0.006	2	0.001
		279.00	283.00	4.00	0.202	0.002	2	0.001
		298.50	308.60	10.10	2.020	0.003	3	0.002
		320.20	324.20	4.00	0.296	0.002	2	0.001
		352.20	356.20	4.00	0.267	0.007	2	0.001
		373.20	377.10	3.90	2.148	0.007	4	0.001
		382.40	385.75	3.35	0.380	0.004	2	0.001
		411.80	428.05	16.25	0.474	0.002	2	0.001
		456.95	465.35	8.40	0.299	0.002	2	0.001

10.3 Verification Drillholes

The verification, or twinned, drillholes (T suffix in Table 10.2) were drilled as close as possible to the historic holes, based on the known coordinates of those holes. In some cases the exact collar location of the historic drillhole could not be determined. Half of the historic holes had no down-the-hole surveys so their path was assumed to be linear while the verification holes were all surveyed at regular intervals throughout the holes. The verification holes were drilled deeper than the historic holes because mineralization continued at depth. Deeper mineralized intervals were intersected in the majority of the verification holes.

Table 10.2 compares the weighted average of the initial continuous mineralized intervals within each of the historic holes as compared to the verification holes. This analysis illustrates that although the continuous interval is large, the majority of the verification holes compare favourably with the historical data. There are a few holes that do not compare well but are still anomalous. These differences could be explained by:

- The historic and verification holes may be further apart from one another, possibly due to survey accuracy of the collar location of the historic hole with respect to the verification hole collar
- The historic holes that have no down-hole surveys could have deviated independently to the verification holes creating an increasing distance between the holes as the hole depths increased
- Small-scale variations or high-grade veinlets may have been intersected in one hole and not the other due to the nature of the geological formations (e.g. breccias) being intersected.

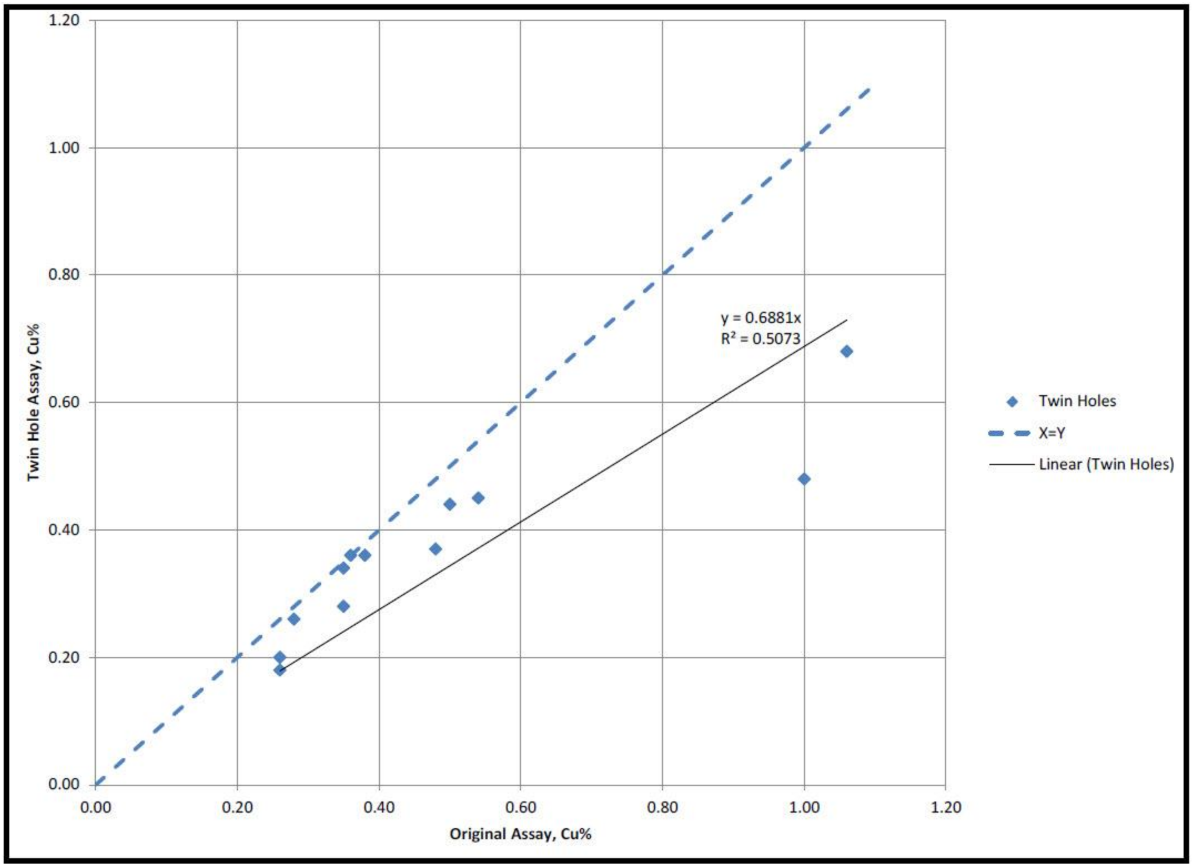
Table 10.2 Twinned Drillholes

Holes	From (m)	To (m)	Interval (m)	Grade (Cu %)
V-126 V-126T	228.05 227.53	360.25 360.85	132.20 133.32	0.48 0.48
V-12	96.80	240.70	143.90	1.06
V-12T	96.00	240.00	144.00	0.68
V-13	27.80	276.50	248.70	0.26
V-13T	28.71	276.70	247.99	0.20
V-14	16.30	167.00	150.70	0.38
V-14T	16.10	165.40	149.30	0.36
V-15	13.40	281.20	267.80	0.54
V-15T	13.00	282.00	269.00	0.45
V-17	80.80	351.00	270.20	0.26
V-17T	79.50	352.00	272.50	0.44
V-19	37.00	261.25	224.25	0.50
V-19T	37.50	260.86	223.36	0.34
V-21	20.15	310.50	290.35	0.28
V-21T	20.20	311.00	290.80	0.26
V-38	70.35	243.90	173.55	0.35
V-38T	70.10	243.00	172.90	0.18
V-47	108.30	390.15	281.85	0.36
V-47T	108.60	391.00	282.40	0.37
V-75	74.70	473.00	398.30	0.35
V-75T	74.50	473.00	398.50	0.28
V-76	0.00	90.85	90.85	1.00
V-76T	0.00	91.00	91.00	0.36

The quantile-quantile (QQ) plot is illustrated in Figure 10.2 and demonstrates that the twin hole assays were generally lower in grade, but match reasonably well up to 0.5% copper. High-grade chalcopyrite veinlets results in a greater assay difference at higher grades, which is to be expected given the nature of high grade mineralization.

Tetra Tech feels that the historical holes can be used as part of the drill database.

Figure 10.2 QQ Plot, Twin Hole Assays



10.4 Drillhole Survey

The Catalyst diamond drillhole collars were initially surveyed with a Garmin GPS 60CSx which has an accuracy of ±3 m. All 2010 and 2011 drillholes were later resurveyed by an external surveyor. A rented FlexIT Smart Tool Survey System (No. 54611) was used for all down-the-hole surveys.

Prior to the Maunula (2012) resource estimate, all drillhole collars were translated to WGS84 datum to conform with the new topography survey by Aero Geometrics Ltd.

10.5 Mineral Sample Length Versus True Thickness

The relationship between sample length and true thickness cannot always be established for porphyry style mineralization. At the Property, there is disseminated and stockwork mineralization. Both the East and West Hill mineralized zones are irregular, coupled with multiple phases of intrusion. This makes it difficult to know the orientation and dimensions of mineralization relative to an individual drillhole. Therefore, although mineralized intervals may be selected on the basis of grade, the true thickness of the mineralized zone is not readily apparent.

However, the West Hill mineralized vein structures typically are oriented with an east-west strike (090° azimuth) and steeply dip to the south. Therefore, in this area the drillholes have a bearing of 000° azimuth, which is approximately perpendicular to the strike of the veins. The holes are inclined and depending on the dip inclination of the individual holes, the true thickness would be approximately 60 to 90% of the drill core intervals.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following Sections 11.1 through 11.5 are a summarized version of the sample preparation methodology used for the La Verde Mineral Resource and Mineral Reserve. The summary is derived from the Tetra Tech September 2012 Report to which the reader is referred for further details.

11.1 Sample Preparation

For each drillhole, one geologist was assigned responsibility for all aspects of the core sampling method.

Core boxes were transported twice daily from the drill station to the core facility. Rock quality designation (RQD) and geology were logged on paper. Sample intervals were marked on the core box and sample tags were also stapled to it. A digital photo was taken of each core box prior to cutting.

Core was cut in half with a diamond core saw. Sample intervals were generally 2 m in length. The samples were placed in heavy mil plastic bags with a sample tag inside the bag and the sample number written on the outside. Samples were then placed in nylon sacks. Sample numbers were written on each sack. A lab sample sheet documents each sack and each sample within the sack.

Quality assurance / quality control samples (QA / QC) were inserted into the sequence of sample numbers for each hole. A standard was inserted every 20 samples and a blank was inserted every 30 samples. Duplicate samples, in the form of a quarter of the remaining core interval, were inserted every 40 samples.

Geological logs, photos and samples sheets were input into a laptop on a daily basis. This data was backed up onto a second computer and an external drive each day.

11.1.1 Sampling of Previously Drilled Core

In 2010 and 2011, MHill sampled core from drillholes completed by Aur in 2005 to 2007. Aur sampled only 64% of the core, leaving many gaps including apparently barren intervals within mineralized areas. MHill sampled these gaps to provide continuous sampling in mineralized areas. A total of 524 samples were collected from 14 Aur drillholes. Sample preparation, storage, analysis, and QA / QC for these samples is the same as for other samples collected by MHill, described elsewhere in this section.

11.2 Core Storage Facility

MinHill established a new secure core storage facility at a large warehouse in Estación Nueva Italia which is approximately 4 km north of Nueva Italia de Ruiz along Mexico Highway 37. The warehouse backs onto a siding which is part of the main line railway to the ocean port facilities at Lazaro Cardenas.

The facility has ample room for core logging, core sawing / sampling and core / reject / pulp storage. All Aur and MTO core, rejects and pulps have been moved from the original warehouse at La Laguna to this new facility.

11.3 Sample Analysis

Samples were taken to Acme Analytical Laboratories Mexico S.A. de C.V., Guadalajara Mexico, for sample preparation. An employee from Acme signed for the samples and drove them to their laboratory. Each sample was pulverized and sieved generating a pulp and rejects. The sample was crushed to 80% passing 10 mesh, split 500 g and pulverized to 85% passing 200 mesh (R200-500). The rejects were returned to MHill in Estación Nueva Itália, by Acme drivers.

The pulps were sent by air express cargo to Acme Analytical Laboratories in Vancouver, BC, Canada for analysis. The analysis was initially a 36 element inductively coupled plasma (ICP) analysis (Acme Group 1D). A 0.5 g sample split is leached in hot (95°C) aqua regia with the resulting aliquot being analysed by an inductively coupled plasma emission spectrometry (ICP-ES). This initial analysis was almost immediately changed to achieve optimum precision especially in copper (Acme Group 7AR). The aqua regia aliquot is analyzed by ICP-ES emission spectrometry giving "%" concentrations for base and precious metals. If the concentration of copper or molybdenum were over 10,000 ppm (1%) then the sample had an atomic absorption spectroscopy (AAS) finish (Acme Group 8AR). If the concentration of gold was above 0.5 g/t, that sample was fire assayed (Acme Group G01).

The results of the analysis were digitally sent to MHill and followed up by hardcopy assay certificates.

Acme Analytical Laboratories Ltd. (AcmeLabs) is an independent, internationally recognized, ISO 9001:2008 certified facility (Quality Management System Certificate FM 63007). They are working toward ISO / IES 17025:2005 (General Requirements for the Competence of Testing and Calibration Laboratories) certification.

Catalyst has also sent a random series of 100 pulps to ALS Minerals' laboratory in North Vancouver, BC, Canada as a check on AcmeLab's performance. ALS Minerals is also an ISO 9001:2008 certified facility and operates in compliance with ISO / IES 17025.

11.4 Review of Quality Assurance / Quality Control

MHill used a series of standard reference materials (SRMs), blank reference materials (Blanks) and duplicates as part of their QA / QC program.

11.4.1 Standards

Four SRMs from CDN Resource Laboratories Ltd. (CDN Resource) in Langley, BC, Canada were used during the exploration program: CDN-CM-5, CDN-CM-6, CDNCM- 8, and CDN-CM-12.

The SRMs analyzed by ACME report close to the recommended values.

Table 11.1 is a summary of the text from the Tetra Tech September 2012 Report. The table lists the number of SRMs analyzed by ACME, the results of those assays, the number of times an insertion error occurred and the number of times the standard was outside acceptable limits.

Table 11.1 Summary of Standard Reference Material Results

SRM	No. of SRM Analyzed	Average Cu (%)	Two Standard Deviations	Insertion Errors*	Fails
ACME CDN-CM5	180	0.316	0.026	9	2
Expected	n/a	0.319	0.020	n/a	n/a
ACME CDN-CM6	338	0.737	0.042	9	2
Expected	n/a	0.737	0.039	n/a	n/a
ACME CDN-CM8	196	0.367	0.012	3	0
Expected	n/a	0.364	0.024	n/a	n/a
ACME CDN-CM12	16	0.914	0.024	0	1
Expected	n/a	0.917	0.044	n/a	n/a

*An insertion error is where the SRM was identified as the wrong standard. Insertion errors are not considered fails.

In addition, for CDN-CM-5, ten samples had insufficient material for gold analysis. For CDN-CM-6, twelve samples had insufficient material for gold analysis and one sample had insufficient material for copper analysis. For CDN-CM-8, fifteen samples had insufficient material for gold analysis and two samples for copper.

The following charts, Figures 11.1 to 11.4, illustrate the standard assay results for standards CDN-CM-5, CDN-CM-6, CDN-CM-8 and CDN-CM-12.

Figure 11.1 SRM CDN-CM-5 Copper Assay Results

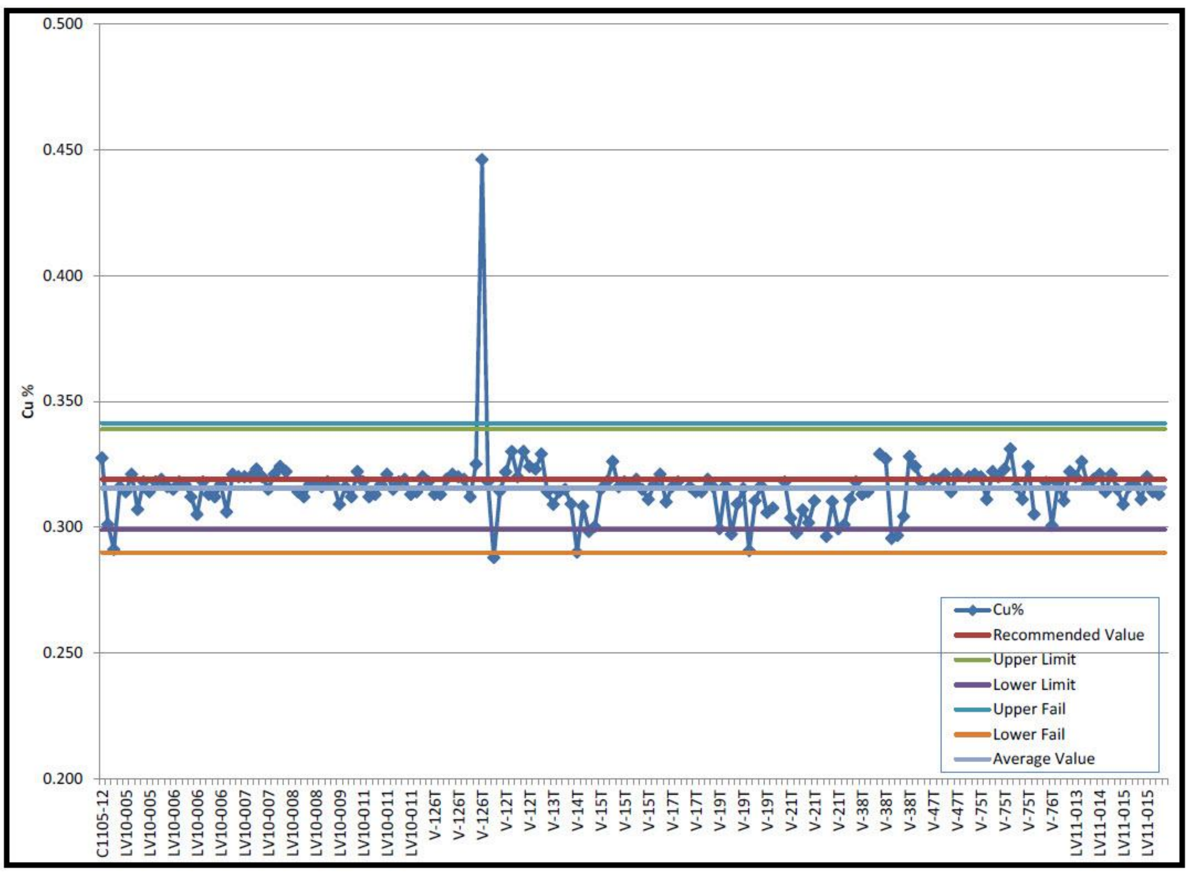


Figure 11.2 SRM CDN-CM-6 Copper Assay Results

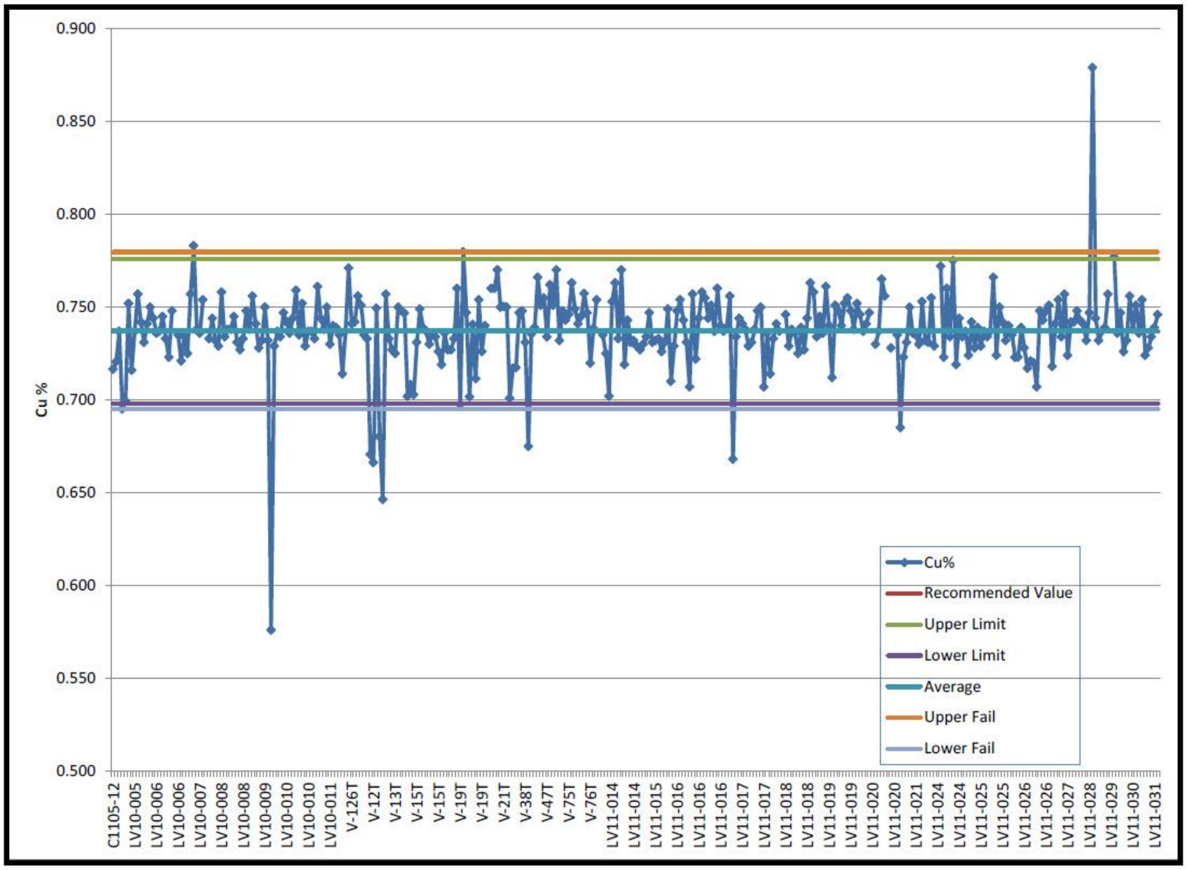


Figure 11.3 SRM CDN-CM-8 Copper Assay Results

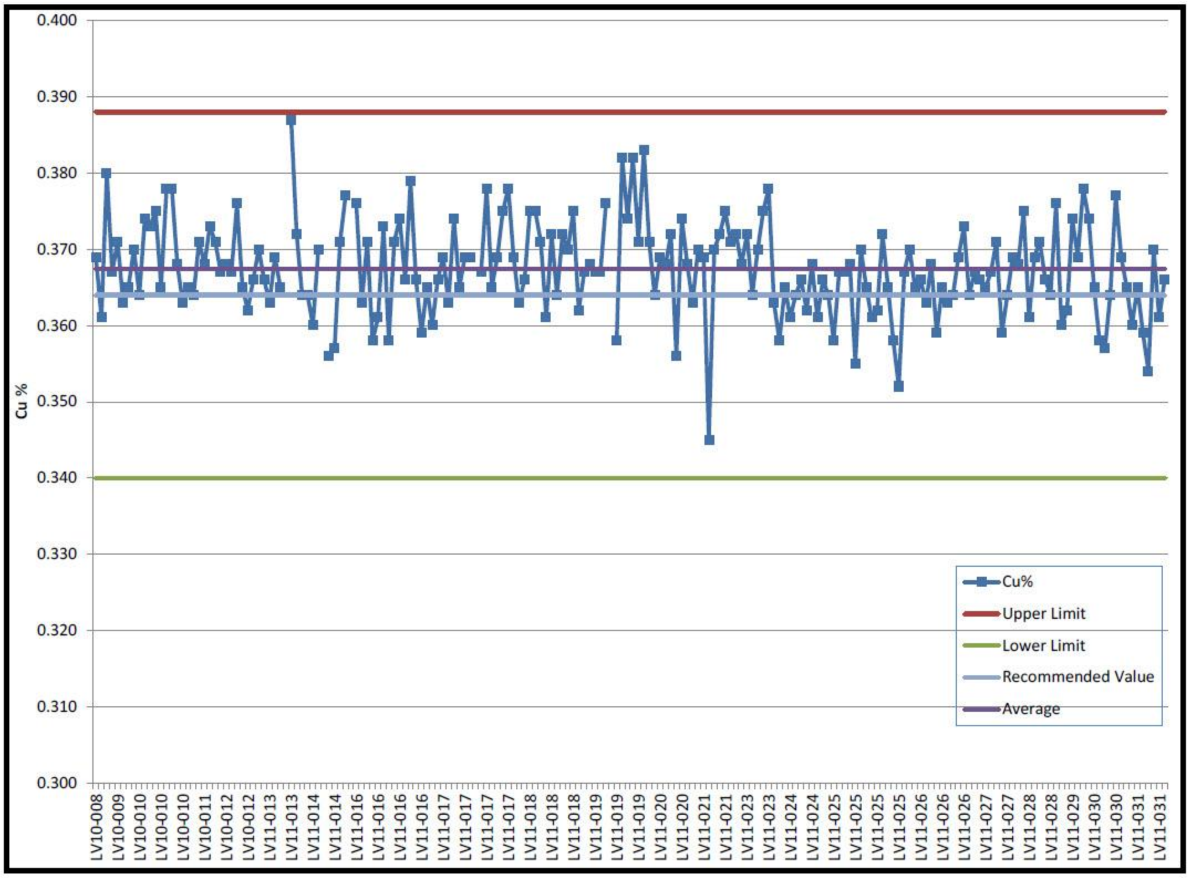
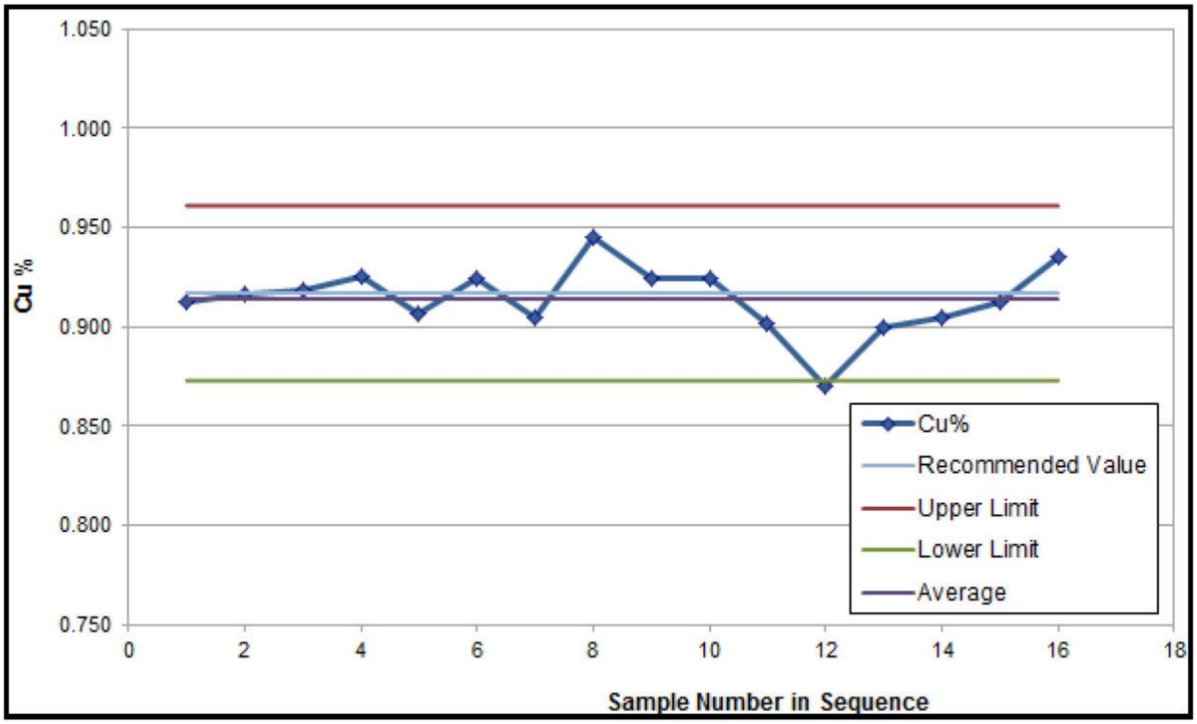


Figure 11.4 SRM CDN-CM-12 Copper Assay Results

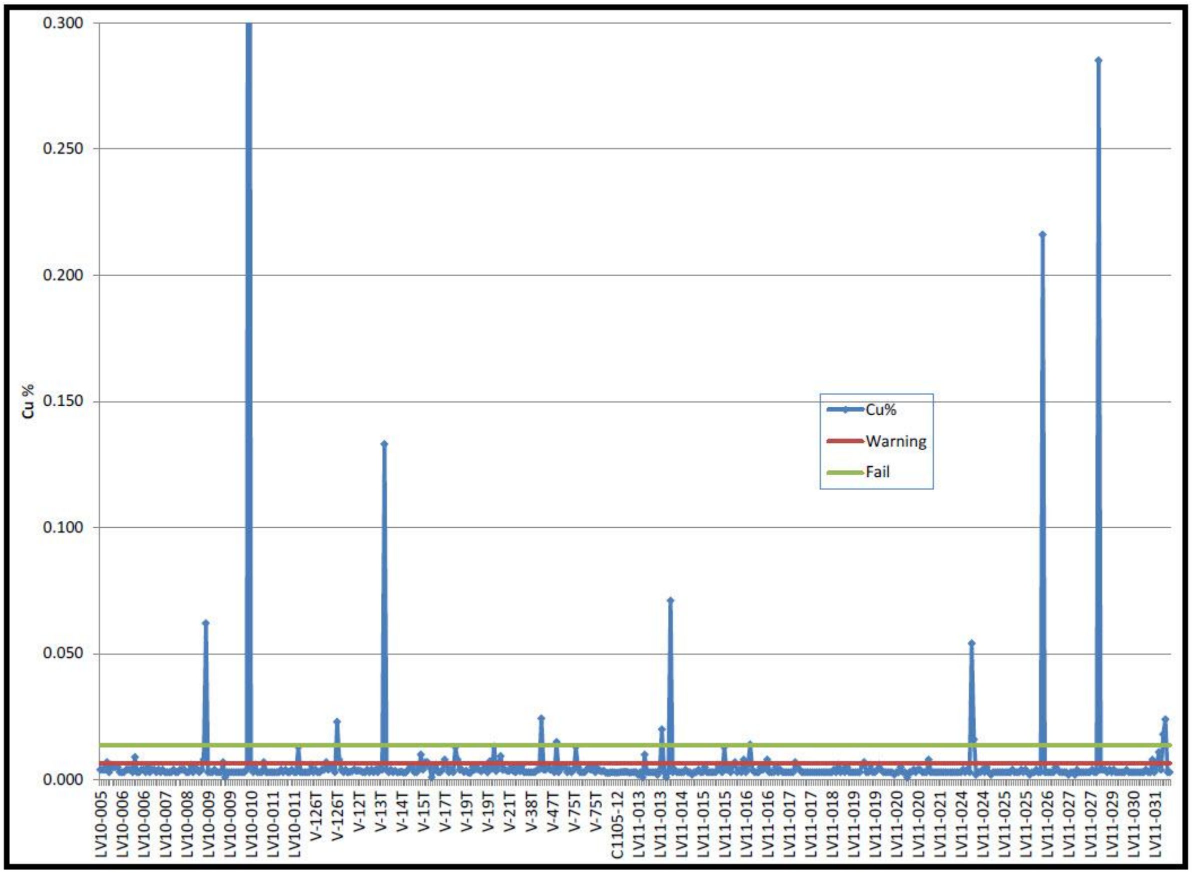


11.4.2 Blanks

There were 498 blanks inserted in the program. The blanks were material from a local gravel pit (volcanic cinder cone). The copper assays from the blank samples were quoted in parts per million rather than copper percent, due to the very low concentration of copper in the blanks. Ten independent samples averaging 3.1 kg were sent to AcmeLabs for analysis. The average of these samples was 28.9 ppm copper ± 2.2 ppm copper. Figure 11.5 illustrates the blank sample assay results.

There are many sample results that plot above the second standard deviation level. This can partially be attributed to the lower detection limit (1 ppm copper) of the equipment used and possibly the potential inhomogeneity of the blank material. Although the copper levels in the blank material are extremely low, Micon (Makepeace, 2011) had recommended that a standard blank sample be used in further drilling programs as part of the QA / QC process. This was not implemented by MHill in the 2011 drill program. Using three times the “accepted average”, there were 15 blank samples that failed. One may be an insertion error, as it appears to be a SRM sample for CM-6.

Figure 11.5 Blank Sample Copper Assay Results



11.4.3 Duplicates

There were 382 duplicate samples analysed during the program. Figure 11.6 illustrates the comparison between the two assay results.

The duplicates report 70% within 20% relative difference, the target is generally set for 90%. These results somewhat reflect the influence of the high-grade copper veinlets within the core.

The QQ plot in Figure 11.7 shows that there is no bias present in the samples. Generally, the distribution tracks closely to the X=Y line up to 3% copper. Above the 3% copper point, the duplicate assay is higher than the original assay.

Figure 11.6 Coarse Reject Duplicate Relative Difference Plot

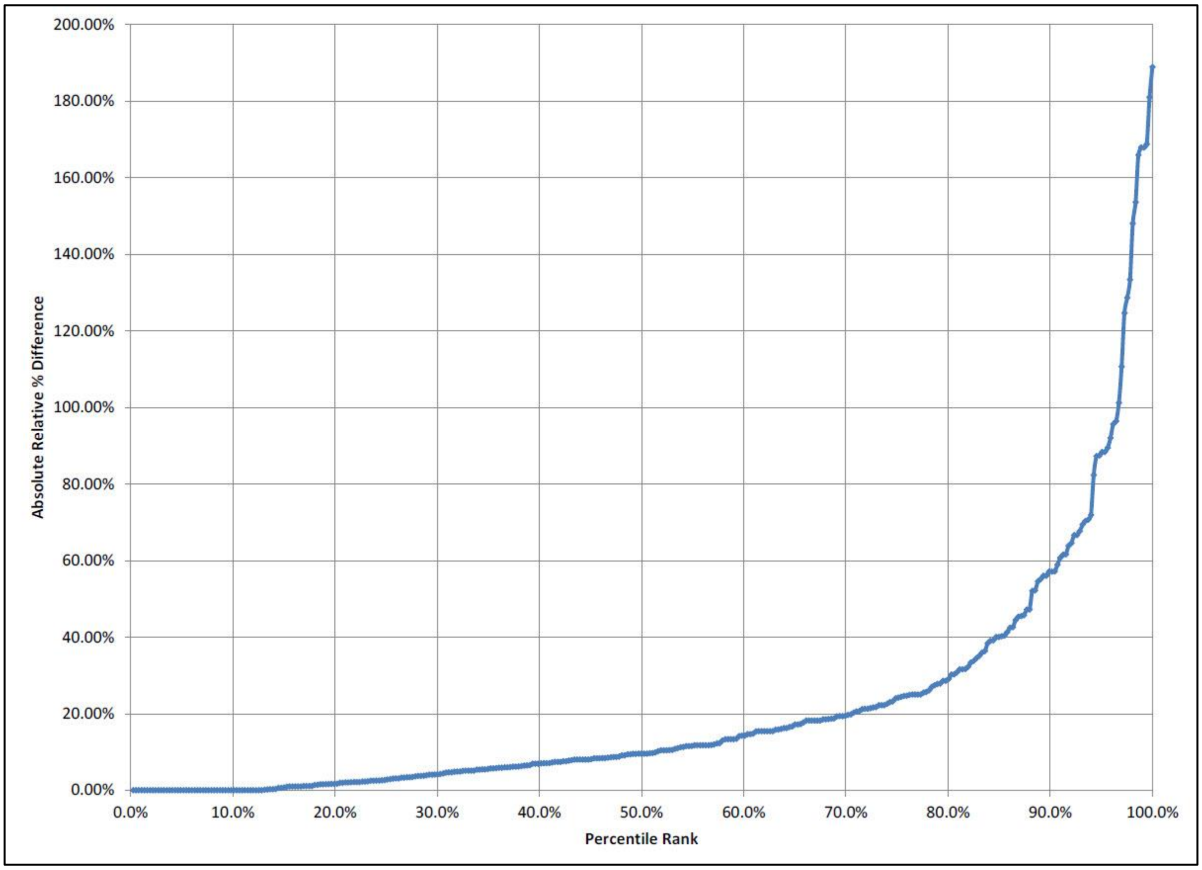
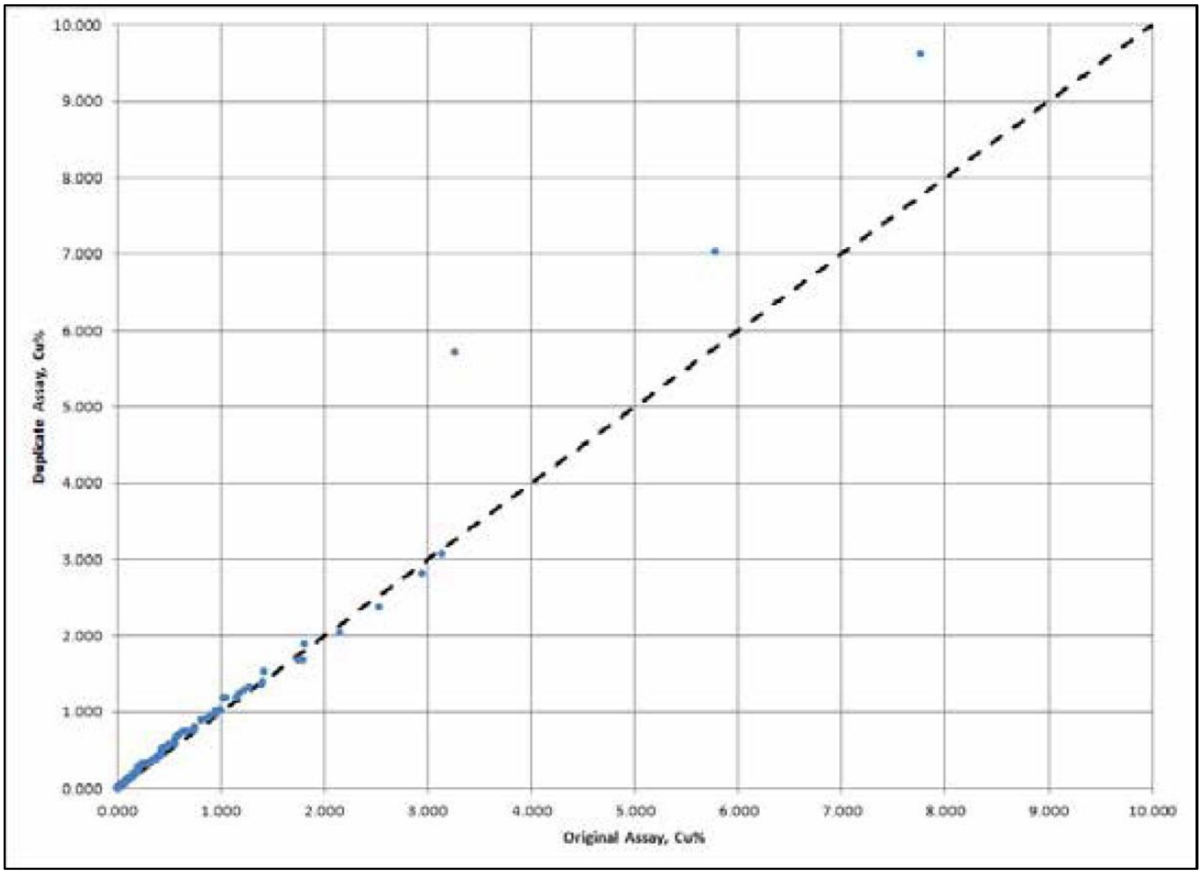


Figure 11.7 Coarse Reject Duplicate QQ Plot



11.4.4 Second Laboratory

The 100 samples that were re-assayed at ALS Minerals have been compared to the AcmeLabs analysis. Based on this small number of samples, the AcmeLabs analysis reports 0.004% copper below the ALS Minerals analysis with a standard deviation of 0.035% copper. An analysis of the QA / QC standards, used in the MHill drill program, was also compared. The results showed similar variations to the 100 samples. This nominal difference between laboratories is considered acceptable.

11.5 Discussion

Tetra Tech believes that the QA / QC program that was initiated by MHill during the 2010 drilling program and continued through the 2011 and 2012 programs follows industry accepted guidelines and the results confirm the validity of the assays obtained from that program.

Tetra Tech recommends timely monitoring of QA / QC samples through control charts to further improve the QA / QC program.

12 DATA VERIFICATION

Sections 12.2 and 12.4 are a summarized version of the logging and check assay methodology. The summary is derived from the Tetra Tech September 2012 Report of Tetra Tech to which the reader is referred for further details. The remaining sections are an extraction from Tetra Tech September 2012 Report.

The Tetra Tech site visit was conducted by Margaret Harder, P.Geo., 21 on August 2012. During the site visit to the Property, the core shack and core storage facility, which are located at the Project office 8 km northeast of the town of Nueva Italia de Ruiz, were visited. The Project office in Uruapan was also visited following the Project site visit.

Tetra Tech conducted verification of two drillholes LV12-039 and LV12-045 which is 16% of the 12 drillholes completed in 2012. Tetra Tech found that MHill followed industry accepted guidelines for drillhole data management.

12.1 Drillhole Collar

Two collars were checked in the field during the site visit. Table 12.1 compares the collar locations. Tetra Tech used a Garmin Oregon 550 for the field readings. The accuracy for this global positioning system (GPS) instrument is less than 10 m. The collar coordinates were generally confirmed within that level of accuracy, although the easting reading for LV11-039 was slightly outside of the error limit. The elevations for both holes were also slightly outside these error limits, although elevations values are often somewhat less accurate than easting and northing readings.

Table 12.1 Field Check – Collar Survey

Drillhole	Coordinate	MHill	Tetra Tech
LV11-039	Easting	812,100	812,088
	Northing	2,112,690	2,112,691
	Elevation	483	495
LV11-045	Easting	810,935	810,937
	Northing	2,113,295	2,113,297
	Elevation	523.5	534

Figure 12.1 illustrates the method employed By MHill to mark the collars in the field.

Figure 12.1 Collar Marker, LV12-045

12.2 Logging

The core boxes of the two drillholes were laid out in MHill's warehouse. Core box labelling and footage markers and assay tags were reviewed. No errors were noted.

The logged lithology was compared with the database record to confirm data entry along with consistency and accuracy in logging. The contacts were found, in some cases, to be transitional so the position of the contact was subject to interpretation. As the grade is transitional across most contacts, subjectivity about the contact location will not impact the grade estimation.

Core logging by MHill geologists appears to have been completed to industry standards. Although there is some inconsistency in identification of alteration types, this is not unusual in porphyry deposits as the nature of alteration is typically gradational and distinctions between different alteration styles can be subtle.

12.3 Data Verification

The original assay certificates were compared with the database records to confirm data loading. A total of 679 assay records for drillholes LV12-039 and LV12-045 were verified out of a total of 3,938 records, or 17% of all assays collected in 2012. No errors were found.

12.4 Check Assays

Tetra Tech collected four pulp samples from drillholes LV11-039 and LV11-045. Tetra Tech maintained custody of the samples from the Project site until shipped to the check laboratory, ALS Minerals in North Vancouver, BC.

The check assay results compared reasonably well with the original assays. The four Tetra Tech check assays returned results somewhat lower than those of MHill; however, given the small number of samples collected this is not considered statistically meaningful. The Tetra Tech check samples confirmed anomalous grade results within the expected grade range for the rock types and areas sampled.

12.5 Discussion

Tetra Tech believes that the dataset is sufficiently robust to support resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The word "ore" is used in this section in a generic sense and does not imply that Mineral Reserves have been estimated.

Historic testwork programs and their results have already been reported in the Tetra Tech September 2012 Report. They are summarized below.

The 1972 metallurgical studies by Lytton (raw data lost) showed 55% recovery from oxides, >90% recovery from sulphides, both to superior grade (>35% Cu) concentrates, especially early in the mine life. Additional work in 1976 by Hazen focused on grind size and reagent optimization and also attempted to address the issues around arsenic levels in the concentrates. The results as of 1976 are shown in Table 13.1.

Table 13.1 1976 Hazen Metallurgical Study Results

Mineralization Type	Feed Cu %	Recovery % Cu	Conc Grade Cu %
Mixed ox/sulphide	0.76	54.8	37.4
East Hill zones 1, 3	0.7	93.7	36.6
East Hill zones 2	0.68	90.9	26.6
West Hill	0.68	91.9	25.5
Stockpile	0.27	87.6	27.6
Average	0.65	90.5	32.1

The average expectation was of >90% copper recovery to a copper concentrate of 32% Cu, with the higher grades coming from the mixed oxide-sulphide material and from the secondary sulphides in East Hill zones 1 and 3.

The most recent metallurgical testwork was initiated by Catalyst as part of this current PEA. A preliminary phase was carried out at G&T Metallurgical Services Ltd (G&T) in Kamloops in late 2011.

The key objectives were to prepare four global composites based on oxide and sulphide material for each of West Hill and East Hill and, on these, carry out a standard set of comminution tests followed by flotation tests to maximize copper recoveries into high grade copper concentrates. In addition, a detailed assessment of concentrate quality was carried out.

Subsequently, and pursuant to the high levels of arsenic found to report to the flotation concentrates in 2011, a further round of testwork was conducted in April 2012 to confirm previous results, and generate larger amounts of rougher and final concentrates for testing at other laboratories of downstream treatment options.

These two programs (named KM2900 and KM3306, respectively) are reported separately in Section 13.3.

13.2 Metallurgical Samples

The metallurgical samples were taken from a selection of holes from West Hill and East Hill whose collars covered the grid areas 2112300-2113200 N by 810700-811100 E and 212800-2113300 N by 812200-812500 E, respectively.

AMC considers that the samples provide adequate spatial representation but at a lower density in West Hill.

For each of West Hill and East Hill, the samples were composited by oxide or sulphide characteristics, thus providing four composites. The West Hill oxide composite covered depths to 40 m and the sulphide 50–140 m, although heavily weighted to deeper intersections. The East Hill oxide composite covered depths to 30 m and the sulphide 30–130 m, more evenly spread.

AMC considers that, to date, the oxide-sulphide transition zone in West Hill has been inadequately represented and this may constitute a geometallurgical risk. AMC understands that this will be addressed in the next phase of study with acid soluble copper assays being included in the in-fill drilling assays.

AMC also considers that inadequate account has been taken of the alteration chemistry although this is to a certain extent a function of the incorporation of historical drilling data where core logs were not as comprehensive as the more recent drilling. As is discussed in Section 13.3, the mineralogical characteristics of the two oxide samples are very different, with East Hill oxide exhibiting a “classic” supergene” weathering profile, whereas West Hill oxide appears to be a result of a different (hypogene?) mechanism.

The East Hill sulphide composite does not take account of the zoning described in Section 7.6.2, as indeed had been done by Hazen; therefore, AMC believes that the composite selection does not address any potential mineralogical and resulting geometallurgical differences arising from the alteration geochemistry.

AMC’s experience on the smaller porphyritic systems and associated vein-fracture hosted deposits in Mexico (as opposed to the giant Chilean porphyries where sheer scale imposes some homogeneity) is that there is often significant geometallurgical variability, especially associated with alteration geochemistry.

In summary, AMC believes that the metallurgical samples are spatially representative but that there should have been more consideration of the alteration geochemistry and resulting geometallurgical implications. Although the samples are adequate for this preliminary assessment, AMC considers it essential that high priority be given early in subsequent studies to investigations of the geometallurgical variability. AMC understands that this is well understood by Catalyst and that the in-fill drilling program of the next phase of study will include variability testing to address some of these issues around alteration geochemistry.

13.3 Metallurgical Testwork

13.3.1 KM2900 – August-November 2011

13.3.1.1 Chemical and Mineralogical Characteristics

89 individual intersections of half drill core were received, coarse-crushed, and 200g assayed for Cu, Cu_{OX} and Cu_{CN}. The last two represent acid soluble oxide copper and cyanide-soluble secondary copper, respectively.

For both East and West Hill material, high levels of oxide copper were most evident at depths shallower than 30 m. Below that, oxide copper levels were generally around 5% of the total copper, except for occasional higher values (10–20%) in East Hill at around 80 m depth.

Cyanide soluble copper levels showed that a general rising trend with depth to levels containing > 50% of total copper in East Hill and in West Hill were generally much lower (10%) apart from slightly elevated (20%) levels at around 20 m depth.

The composites were WH-1 and WH-2, being oxide and sulphide material respectively from West Hill and EH-3 and EH-4, being similarly oxide and sulphide material from East Hill.

The chemical content of the four composites is shown in Table 13.1.

Table 13.2 Chemical Analysis of Composites

Composite	Cu	CU (ox)	Cu (CN)	Mo	Mo(ox)	Fe	S	Ag	Au	As	Co	C
	%	%	%	%	%	%	%	g/t	g/t	%	g/t	%
WH-1	0.36	0.17	0.03	0.004	0.004	6.35	0.06	1	<0.01	0.058	25	0.02
WH-2	0.63	0.01	0.01	0.003	0.002	4.65	1.57	2	0.04	0.17	188	0.55
EH-3	0.45	0.33	0.05	0.021	0.021	3.43	0.07	1	0.13	0.009	19	0.39
EH-4	0.72	0.03	0.34	0.002	0.001	2.47	0.58	6.5	0.09	0.058	100	0.66

AMC notes that both oxide composites confirmed the high levels of acid soluble copper expected from the individual intersections. This is generally difficult to recover via flotation.

The East Hill sulphide material had just under half the copper in cyanide soluble form suggesting the presence of secondary copper minerals such as chalcocite and covellite. These secondary copper sulphides, although often slower floating than chalcopyrite, should result in higher grade copper concentrates, as indicated by the earlier work at Hazen.

Molybdenum was predominantly in the oxide form and, therefore, not expected to be readily recoverable by flotation

The copper deportment of the four composites in mineralogical terms is summarized in Table 13.2.

Table 13.3 Copper Deportment

Composite	% of Copper Occurring in Copper-Bearing Minerals								
	Cp	**Bn**	**Ch/Cv**	**Mal**	**Chrys**	**in Feox**	**in Chlor**	**in Mnox**	**Tenn**
WH-1	3.5	0.3		2.5	4.4	21.4	66.2	1.7	
WH-2	93.5	2.5	3.1	1.0					
EH-3	4.6	1.6	4.2	51.0	6.1	8.4	24.2		
EH-4	44.1	32.5	20.4						3.0

From Table 13.3, it is evident that the two oxide samples WH-1 and EH-3 are very different, as mentioned in Section 13.2.

While the East Hill oxide material has the acid soluble copper content largely as malachite (difficult to float but possible with sulphidation by sodium hydrosulphide NaHS), the non-sulphide copper in West Hill is predominantly contained in chlorite and in iron and manganese oxides. Flotation recovery of these constituents would be expected to be close to zero and, in any case, even if recoverable to some extent, would result in concentrates of too low a grade to be sold. This West Hill oxide is, effectively, waste.

AMC believes that whereas the East Hill weathering profile is a typical supergene phenomenon with typical malachite, the weathering mechanism at West Hill appears to be different with much higher proportions of copper-bearing chlorite, possibly related to the alteration geochemistry and hypogene effects.

Gangue mineralogy was also reported in the G&T report and shows the predominantly quartz-feldspar mineralogy of East Hill as opposed to the quartz-epidote veining of the West Hill. West Hill generally shows more mineralogical evidence of the propylitic/phyllic alteration.

With respect to the sulphide mineralogy, the East Hill copper deportment studies confirm the high proportion (almost 50%) of secondary sulphides. It is also significant that, although arsenopyrite was reported in the West Hill sulphide composite, the arsenic in East Hill occurred as tennantite-tetrahedrite, so any arsenic depression in the latter case would adversely impact on copper recovery.

Liberation studies reported by G&T indicated that with the nominal primary grind of 150-200 microns an acceptable liberation would be achieved for sulphides, whereas the East Hill oxide material appeared to be more sensitive to grind size and a finer grind may be required.

13.3.1.2 Comminution Tests

The comminution test data, including Bond and SMC tests is shown in Table 13.4.

Table 13.4 Comminution Results

Composite	Bond Indices		SMC Test Results						Calculated	
	Abrasion	BWI	DWi kWh/m3	Mia kWh/t	A	b	S.G.	t_a	A*b	Mib kWh/t
WH-1	0.023	15.6	3.8	11.9	60.9	1.21	2.81	0.68	73.7	21.4
WH-2	0.178	16.8	8.1	21.6	65.9	0.53	2.83	0.32	34.9	22.8
EH-3	0.091	15.6	6.8	20.3	71.3	0.54	2.62	0.38	38.5	22.0
EH-4	0.132	16.1	7.7	22.5	78.6	0.43	2.62	0.34	33.8	23.9
Average	0.106	16.0	6.6	19.1			2.72	0.4	45.2	22.5
Average (excl WH-1)	0.134	16.2	7.5	21.5			2.7	0.3	35.7	22.9

Note that in addition to the results reported by G&T, AMC has also added the simple calculation of A*b and the SMC M_{ib} parameter calculated from the original Bond ball mill work index data.

With the exception of the much softer WH-1 oxide material, different for the reasons already discussed and which will effectively be classified as waste for the purposes of this study, the La Verde material showed limited variability across the three composites tested and would be regarded as moderately hard.

AMC notes that the Bond ball mill work index tests were carried out at a closing size of 106 µm, coarser than the expected grind size at least for the sulphides, and this should be revised in the next phase of study to be closer to the expected grind size.

The material tested as being of relatively low abrasivity.

This data was sufficient for the preliminary grinding equipment sizing and power demand estimations carried out in Section 17.

13.3.1.3 Flotation Tests

Rougher flotation tests on the WH-1 oxide sample confirmed the poor recoveries expected from the mineralogy and no further work on this composite was carried out.

The rougher flotation results for the other three composites, based on a potassium amyl xanthate (PAX) collector with sodium hydrosulphide (NaSH) as a sulphidizing agent for East Hill oxide sample, are summarized in Table 13.5 and Figure 13.1.

Table 13.1 Rougher Flotation Results – Copper Recovery vs Mass Pull wt%

	wt%	Cu rec %	wt%	Cu rec %	wt%	Cu rec %
Grind Size	147 (WH)		217 (WH)		305 (WH)	
WH-2	5.1	94.9	4.7	93.6	4.4	91.6
	6.0	96.2	5.9	95.8	5.7	94.4
	7.0	96.9	6.9	96.7	7.1	95.6
Grind Size	145 (EH)		223 EH)		300 (EH)	
EH-4	4.5	95.3	4.1	94.8	4.2	92.7
	5.5	96.4	4.8	95.9	5.3	94.5
	7.1	97.3	5.5	96.6	6.4	95.6

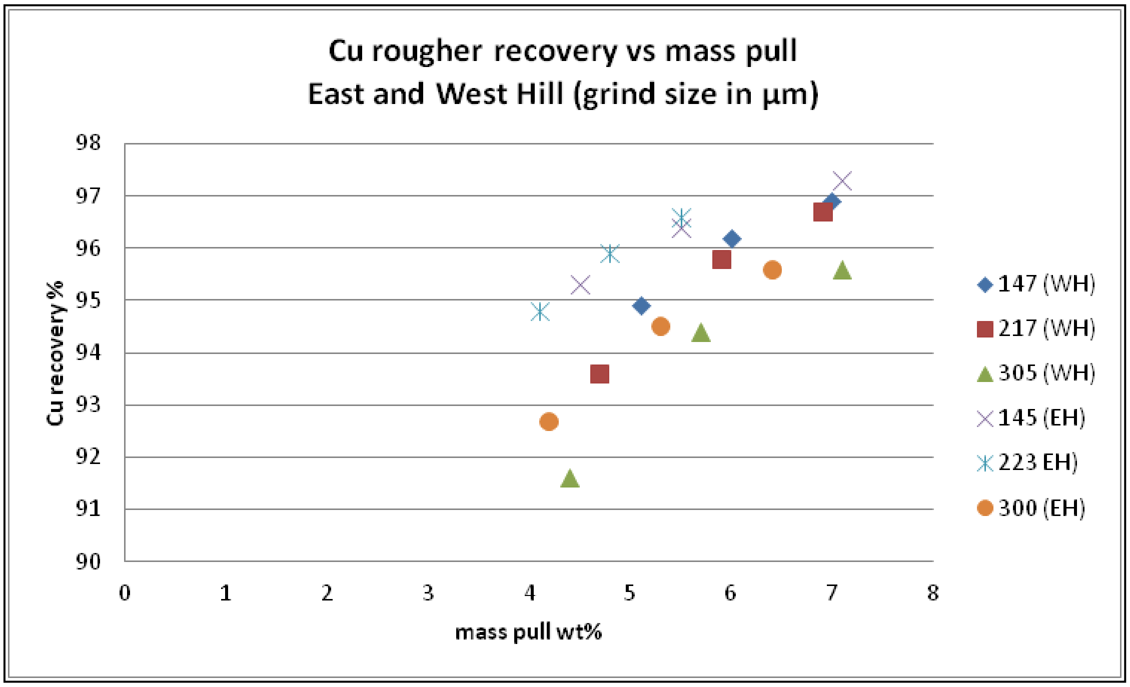
Two things are evident from Figure 13.1:

i. West Hill and East Hill sit on two slightly different grade recovery curves, with East Hill being slightly superior perhaps due to the secondary copper sulphides enhancing concentrate grades.

ii. For both East and West, the 145–147 µm and 217–223 µm curves are very similar but the 300–305 µm curves are inferior. So although the G&T report states that a primary grind of 300 µm would be adequate, AMC considers that a slightly finer grind, say 220 µm, would provide an extra two points recovery. At this stage AMC would recommend this finer grind subject to more a detailed trade-off and optimization study to balance additional recovery against additional grinding costs at the next phase of study.

Molybdenum recovery attempts were unsuccessful, due to it being in the oxide form.

Figure 13.1 Sulphide Cu Rougher Performance vs Grind Size



In the cleaner tests EH-3 oxide sample appeared insensitive to cleaner / regrind conditions and overall recoveries (batch basis) of 40% to a concentrate assaying at least 30% Cu appeared achievable.

In the case of the sulphide composites, West Hill appeared to benefit from floating at a higher ph (9.5) in terms of improving concentrate grade with no impact on recovery, whereas with East Hill a higher pH did improve concentrate grade but at the expense of copper recovery.

Regrinding improved concentrate grade for both West and East and for East appeared to be the key to realizing the concentrate grade of the secondary copper sulphides.

The other key result from the cleaner work was that arsenic will be an issue with concentrate quality. Although some success was achieved with rejecting arsenopyrite from the West Hill sulphides, especially with elevated pH (11) where less than 10% arsenic recovery was achieved whilst still maintaining copper recoveries in the 80–90% range.

However, with East Hill the arsenic being as tennantite-tetrahedrite meant that similar differentiation was not possible.

Flotation kinetics was not specifically addressed but typical flotation times were 6–10 minutes for roughers and up to five minutes for cleaners. The higher end of the range of times was used in the design criteria presented in Section 17.

The above discussions have all been related to batch tests, essential for establishing, and then optimizing flotation conditions. However, definitive results for plant recovery predictions can best be derived from locked cycle tests in which intermediate streams are recycled, and these results are tabulated in Tables 13.6 and 13.7.

Table 13.2 Locked Cycle Test Results – Sulphides

Product	Assay						Distribution %				
	wt%	Cu %	S %	As %	Ag g/t	Au g/t	Cu	S	As	Ag	Au
WH-2											
Feed	100	0.62	1.6	0.17	3	0.02	100	100	100	100	100
Rghr Conc	7.6	7.75	16.7	1.49	27	0.21	94.2	79.3	66.8	68.7	62.8
Final Conc	2.2	26	38.1	0.94	81	0.12	91.8	52.4	12.2	60.5	10.6
Clnr 1 tail	5.4	0.28	7.96	1.71	4	0.24	2.5	26.8	54.6	8.2	52.2
Rghr Tail	92.4	0.04	0.36	0.06	1	0.01	5.8	20.7	33.2	31.3	37.2
EH-4											
Feed	100	0.74	0.59	0.05	8	0.06	100	100	100	100	100
Rghr Conc	5.8	12.1	8.6	0.68	116	0.84	93.8	83.9	74.2	87.6	83.7
Final Conc	2.5	27.4	19.3	1.50	260	1.87	92.2	81.6	70.9	85.5	80.9
Clnr 1 tail	3.3	0.36	0.42	0.05	5	0.05	1.6	2.3	3.4	2.1	2.8
Rghr Tail	94.2	0.05	0.1	0.01	1	0.01	6.2	16.1	25.8	12.4	16.3

Note that for these sulphide locked cycle tests, a primary grind of 300 µm was employed, coarser than ideal in AMC's opinion, and for WH-2 there were three stages of cleaning at pH 9.5. For EH-4 there were also three stages of cleaning, but at natural pH and with no regrind. AMC believes that the higher grade potential of the secondary copper sulphides was not realized under these conditions and there remains upside in the concentrate copper grade.

As expected from the cleaner tests, arsenic levels in final concentrate are high. In East Hill, despite the lower feed As%, unacceptably high (>1% As) from a concentrate marketing perspective, due to the co-recovery of tennanite an copper arsenic sulphosalt.

The very low levels of gold recovery to final concentrate for WH-2, despite reasonable recovery to rougher concentrate, appear anomalous, but AMC has referred back to the original results and can find no obvious error. This should be investigated further.

Table 13.3 Locked Cycle Test Results – Oxides

Product	Assay						Distribution %				
	wt%	Cu %	S %	As %	Ag g/t	Au g/t	Cu	S	As	Ag	Au
EH-3											
Feed	100	0.47	0.08	0.02	2	0.06	100	100	100	100	100
Rghr Conc	1.9	14.2	1.59	0.23	63	3.43	57.6	39	19.1	58.6	101.4
Final Conc	0.8	28.8	3.07	0.42	124	5.91	51.5	33.2	14.9	51.3	77
Rghr Tail	99.2	0.23	0.05	0.02	1	0.02	48.5	66.8	85.1	48.7	23

For the oxide locked cycle test a finer primary grind of 138 µm was employed and a negative electrochemical potential (relative to Ag/AgCl-Pt electrode) was maintained using NaHS to promote the flotation of malachite and azurite. The recovery relative to the batch tests was significantly increased due to the recycling of first cleaner tail. Only one stage of cleaning was necessary.

13.3.2 KM3306 – April 2012

13.3.2.1 Chemical and Mineralogical Characteristics

This second phase was conducted principally to generate additional concentrates for downstream treatment option studies, targeted at reducing arsenic levels in the concentrates. Material remaining from the KM 2900 initial phase was used and additional composites were prepared from similar intersections.

Table 13.8 shows the chemical analysis of the new composites (WH-5 and EH-6), both sulphides, compared to the equivalent composites from KM 2900. The general match is good except that arsenic levels are about half the earlier values. Mineral deportment was also similar, although again there were some differences with respect to arsenic with arsenopyrite dominant and both new composites and less tennantite-tetrahedrite in EH-6 than in EH-4. EH-6, however, also contained cobaltite which is assumed will float similarly to the copper arsenic sulphosalts.

Liberation characteristics were reported as being similar.

Table 13.4 KM 3306 Chemical Analysis of Composites

Composite	Cu %	CU (ox) %	Cu (CN) %	Fe %	S %	As %	C %
KM2900 WH-2	0.63	0.009	0.01	4.65	1.57	0.17	0.55
KM3306 WH-5	0.68	0.007	0.013	5.21	1.79	0.078	0.38
KM2900 EH-4	0.72	0.029	0.34	2.47	0.58	0.058	0.66
KM3306 EH-6	0.77	0.03	0.37	2.36	0.49	0.029	0.6

13.3.2.2 Concentrate Generation

Following rougher tests at a primary grind of around 300 µm which produced similar results as previously although with slightly lower recovery, probably due to the coarser grind, locked cycle tests were again carried out.

Table 13.9 shows the results of these locked cycle tests.

WH-2 concentrate grade was significantly lower, thought to be due to reagent overdosing, but recovery was lower too. Interestingly the concentrate grade for EH-6 was significantly higher despite neither regrinding nor a higher pH being employed.

Arsenic levels were still in excess of 1% in final concentrate despite the lower head grade and arsenic recovery was significantly higher for WH-5, presumably related to the reagent overdosing already referred to.

Table 13.5 Locked Cycle Test Results – Sulphides

Product	Assay					Distribution %			
	wt%	Cu %	Fe %	S %	As %	Cu	Fe	S	As
WH-5									
Feed	100	0.66	4.92	1.68	0.08	100	100	100	100
Rghr Conc	9.4	6.2	15.7	12.8	0.50	87.6	30	71.5	62.3
Final Conc	3.0	19.0	33.4	37.9	1.15	86.8	20.5	68.1	41.9
Clnr 1 tail	6.0	0.08	7.3	0.9	0.28	0.7	8.9	3.2	20.1
Rghr Tail	90.6	0.09	3.8	0.53	0.03	12.4	70	28.5	37.7
EH-6									
Feed	100	0.69	2.49	0.51	0.03	100	100	100	100
Rghr Conc	7.0	9.0	7.0	6.2	0.30	91.4	19.8	84.5	77.5
Final Conc	1.8	34.4	16.7	23.3	1.16	88.3	11.9	81.1	71.3
Clnr 1 tail	3.6	0.39	3.61	0.31	0.04	2.0	5.2	2.2	4.4
Rghr Tail	93	0.06	2.15	0.09	0.01	8.6	80.2	15.5	22.5

13.4 Metallurgical Grade-Recovery Predictions

Based predominantly on the KM2900 testwork, Table 13.10 summarizes the grade-recovery predictions on which the pit optimization and flowsheet development are based.

AMC notes the following with respect to the figures highlighted in red:

- Gold recovery in West Hill sulphides at 10% appears anomalously low, as mentioned previously, and merits further investigation.
- Arsenic recovery in West Hill sulphides has been predicted at 20%, higher than the 12% reported in the testwork, and conservatively reflecting the possibility of a much higher figure (42%) as found in the KM3306 work.
- In the absence of data, arsenic recovery for East Hill oxides is assumed to be the same as for the sulphides, but the proportion of oxide ore is low, so this is hardly material.
- East Hill sulphides concentrate grade is reported as 27.5% per KM2900, although the KM3306 work indicates that higher grades can be achieved. However KM3306 Cu recovery was lower so this potential for a higher grade, although quite likely based on the mineralogy, is not certain and should be subject to confirmation in the next phase of study.

Table 13.6 Metallurgical Recovery and Concentrate Grade parameters

	Metallurgical Parameters											
	Cu			**Au**			**Ag**			**As**		
	W Su	E Su	E ox	W Su	E Su	E ox	W Su	E Su	E ox	W Su	E Su	E ox
Recovery	90%	90%	50%	**10%**	80%	75%	60%	85%	50%	**20%**	70%	**70%**
Conc. grade	26%	**27.5%**	29%									

13.5 Concentrate Quality

The testwork has shown that the La Verde sulphide mineralization, which constitutes the major part of the deposit, is metallurgically amenable to a conventional flowsheet utilizing a relatively coarse grind and a straightforward flotation circuit.

However, the presence of arsenic (and antimony) as deleterious elements does present an adverse impact on the economic extraction of the mineral. Table 13.11 lists the important elemental analyses for the three composites of economic value.

Table 13.7 Copper Concentrates Analysis

Element	Symbol	Units	WH-2	EH-4	EH-3
Aluminium	Al	%	0.24	2.59	2.35
Antimony	Sb	%	0.01	**0.56**	0.05
Arsenic	As	%	**0.94**	**1.5**	0.16
Bismuth	Bi	g/t	<20	62	<20
Cadmium	Cd	g/t	44	20	2
Calcium	Ca	%	0.21	1.37	1.15
Carbon	C	%	0.05	0.31	2.62
Cobalt	Co	g/t	1162	994	188
Copper	Cu	%	**26**	**27.4**	**28.8**
Fluorine	F	g/t	31	133	163
Gold	Au	g/t	**0.12**	**1.87**	**5.88**
Iron	Fe	%	32	14.5	10.6
Lead	Pb	%	0.04	0.02	0.04
Magnesium	Mg	%	0.054	0.29	0.5
Manganese	Mn	%	0.005	0.039	0.15
Mercury	Hg	g/t	<1	1	<1
Molybdenum	Mo	%	0.017	0.029	0.42
Nickel	Ni	g/t	246	92	160
Palladium	Pd	g/t	0.015	0.005	0.008
Phosphorus	P	g/t	19	234	656
Platinum	Pt	g/t	<0.001	<0.001	0.22
Selenium	Se	g/t	47	38	19
Silicon	Si	%	0.45	8.03	7.53
Sulphur	S	%	38.1	19.3	3.07
Silver	Ag	g/t	**76**	**258**	**124**
Zinc	Zn	%	0.54	0.11	0.03

Copper concentrates are regarded as “clean” with arsenic levels less than 0.1% As. Penalties generally commence at 0.3% As and arsenic levels greater than 1% As pose significant marketing issues, especially as China, an important destination for copper concentrates in the global market, imposes a 0.5% As limit on imported material. This is discussed further in Section 19.

Preliminary results have been received from the hydrometallurgical and roasting testwork carried out on the concentrates generated in the KM 3306 program of April 2012. AMC has summarized the key outcomes:

- Galvanox™ leach technology did not appear suitable. This process relies on the chalcopyrite-pyrite galvanic couple to dissolve chalcopyrite under atmospheric leach conditions, but the La Verde concentrates are low in pyrite; therefore, a pyrite addition was required. Even with that, 80% extraction was the maximum achieved and extractions > 90% were only possible with the addition of petroleum coke. AMC considers that the core feature of the galvanic couple that makes this process attractive is not applicable to La Verde and therefore does not believe this process merits further testwork.

- Teck's CESL process was also tested. This process consists of pressure oxidation and an "enhanced atmospheric leach" stage in the presence of chloride ions, followed by pressure cyanidation for precious metals recovery. Testwork objectives were achieved, namely:
  - >97. 5% copper extraction
  - >85% gold and silver extraction
  - Arsenic was precipitated and As in solution was < 20ppm

However, AMC notes the following:

- The process is relatively complex and chloride solutions will incur materials of construction issues.
- Despite several years of operation of the CESL demonstration plant, no commercial scale plant is in operation.

Nevertheless AMC believes that the excellent results achieved with respect to metal extractions and arsenic precipitation merit further investigations.

- Partial roasting for As and Sb removal was tested at Outotec's laboratory in Sweden. Outotec concluded that both the East Hill and West Hill concentrates were suitable for fluidized bed roasting with calcine arsenic levels expected to be <0.3% As, and possibly as low as 0.1-0.2% As in full scale operation. However, the East Hill material contained coarse lime or dolomite which formed calcium arsenates and limited arsenic removal to 0.35% As in calcine. The coarseness of these carbonates indicated possible contamination and Outotec recommended that further investigations be carried out to verify if they were intrinsic to the concentrates or indeed extraneous contamination.
- AMC believes that these results indicate that roasting can achieve saleable concentrates for La Verde, notwithstanding the carbonate issue. Also, in view of the well-established commercialization of roasting and current advanced investigations into roasting at other operations, including Aranzazu in Mexico and Codelco's Mina Ministro Hales operation in Chile, AMC believes that roasting is the most viable concentrate treatment option for consideration in this PEA.

14 MINERAL RESOURCE ESTIMATES

Sections 14.1 to 14.6 and 14.10 have been extracted from the Tetra Tech September 2012 Report. Sections 14.7, 14.11 and 14.12 have been modified slightly from the Tetra Tech September 2012 Report

Tetra Tech completed the Mineral Resource estimation using MineSight™ MS3D v.7.0-6 software. The Tetra Tech team comprises Margaret Harder, M.Sc., P.Geo., who completed the Resource estimate and Michael O'Brien, M.Sc., Pr.Sci.Nat., FGSSA, FAusIMM, FSAIMM, who reviewed the Resource estimate. The effective date of this Resource estimate is 19 September 2012.

14.1 Drillhole Database

The drillhole sample database was compiled by MHill and reviewed by Tetra Tech, as described in Section 12.0. Based on this review, Tetra Tech determined that the database is acceptable for use in Resource estimation.

The database contains collar locations, drillhole orientations with downhole deviation surveys, assay intervals with results, and intervals with geology and alteration logging data.

The Resource database contained information from 641 complete drillholes totalling 114,823.93 m of drilling. The average drillhole length is 179.12 m and the maximum length is 924.05 m. A total of 49,597 copper assays averaging 0.33% copper were contained in the assay table and 34,993 copper composites (3 m in length) averaging 0.28% copper in the composite table.

14.2 Project Parameters

The deposit was modelled for copper, gold, silver, molybdenum, and arsenic content. The block model was developed based on the selected block size for a parallel preliminary economic assessment for an open pit operation, using a block size 20 m wide by 20 m long and a bench height of 15 m. The block model origin, in UTM coordinates, was 809,000 m east, 2,110,000 m north and -250 m elevation. The block model limits are tabulated in Table 14.1.

Table 14.1 La Verde Block Model Limits

Minimum	Maximum	Size (m)	Number
809,000	814,000	20	250
2,110,000	2,114,000	20	200
-250	800	15	70

14.3 Geological Model

Three-dimensional geology solids were initially developed by MHill on north-south sections at 50 m spacing for the quartz-feldspar porphyry, quartz diorite, breccia, and overburden lithologies. These solids were used by Tetra Tech as a guide for further geological modelling using Leapfrog v.2.5.1.13 software. Eight geology domains were created, as well as an overburden surface which modelled the overburden depths as logged in drillholes (where overburden was absent, the overburden surface was merged with topography).

The modelling in Leapfrog was conducted by selecting the lithology of drillhole composites using a window filter of 3 m and with a minimum percentage of 50% for any lithology type to be incorporated into the wireframe mesh. The wireframe meshes were constructed using a 30 m triangulation resolution and an off-surface point clipping of 100 m. Contact points identified from drillhole logs were honoured in the boundary meshes. The lithology meshes, overburden, and topography were assigned unique codes which were used to code both the block model and the drillhole composites. During interpolation, the software would select data based on matching block and composite lithology codes. Tetra Tech regards these parameters as appropriate for geological modelling of a porphyry-type deposit.

Similar modelling of alteration domains was conducted, also using Leapfrog software. The alteration wireframes did show reasonable correlation with mineralization in some areas; however the identification of alteration types was too inconsistent to apply for modelling on a deposit-wide scale. As such, the alteration wireframes could not be used for selection of data during resource estimation.

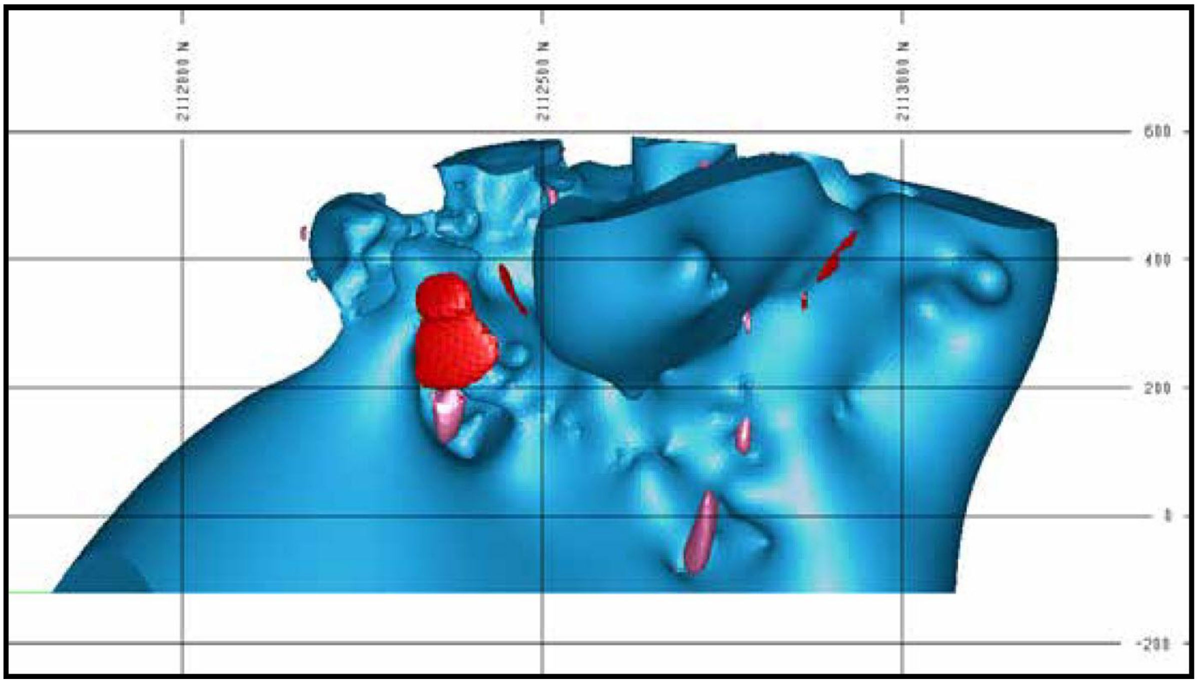
The geological domains and their associated block model codes are listed in Table 14.2.

Table 14.2 Geological Domains

Code	Description
1	East Hill Breccias
2	East Hill Quartz Diorite Porphyry
3	East Hill Quartz-Feldspar Porphyry
4	West Hill Breccias
5	West Hill Quartz Diorite Porphyry
6	West Hill Quartz-Feldspar Porphyry
7	East Hill Quartz Diorite
8	West Hill Quartz Diorite

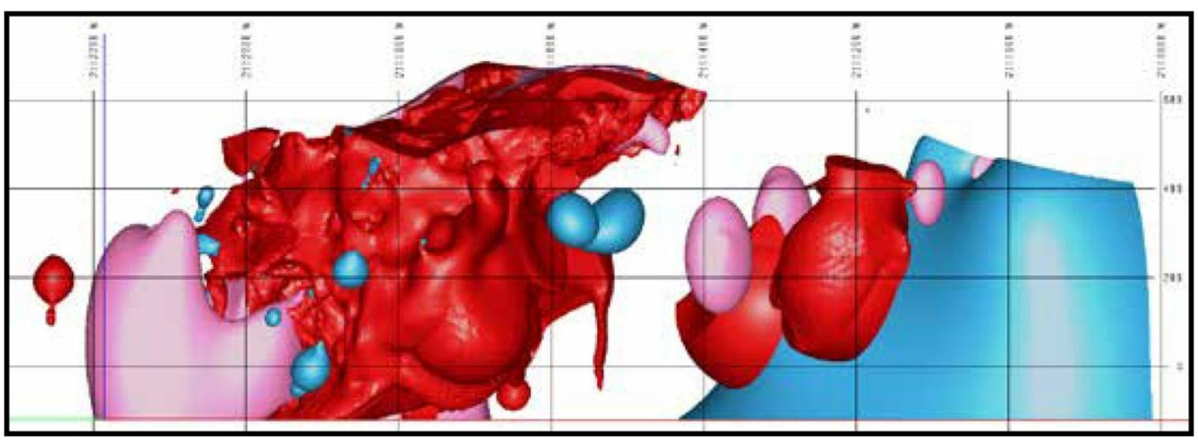
Figure 14.1 and Figure 14.2 show geology wireframes for breccias (red), quartz diorite porphyry (blue), and quartz-feldspar porphyry (pink) for the West and East Hill, respectively. In both the West and East Hill zones, these rock types are surrounded by the quartz diorite wireframe for that zone, which is not shown in Figure 14.1 or Figure 14.2 for illustration purposes. Figure 14.3 shows the domain models for all rock types for both the West and East Hills, with the respective quartz diorite wireframes transparent to show the other enclosed rock types.

Figure 14.1 West Hill Geology Domains



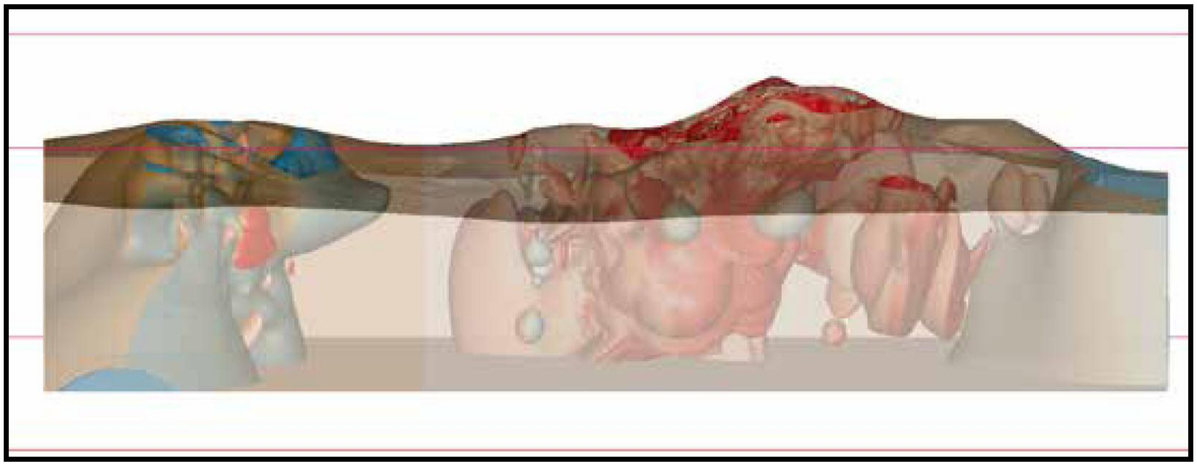
Note: View looking west-northwest. Grid displays northing and elevation values (units in metres). Red=Breccia, Blue=Quartz Diorite Porphyry and Pink = Quartz-Feldspar Porphyry

Figure 14.2 East Hill Geology Domains



Note: View looking northeast. Grid displays northing and elevation values (units in metres). Red= Breccia, Blue=Quartz Diorite Porphyry and Pink = Quartz-Feldspar Porphyry

Figure 14.3 West and East Hill Geology Domains



Note: View looking north. Total extent is approximately 870 m in elevation, 2,700 m in width. Red= Breccia, Blue=Quartz Diorite Porphyry and Pink = Quartz-Feldspar Porphyry, Dark beige (left side of image)= West Hill Quartz Diorite, Light beige (Right side of image)= East Hill Quartz Diorite

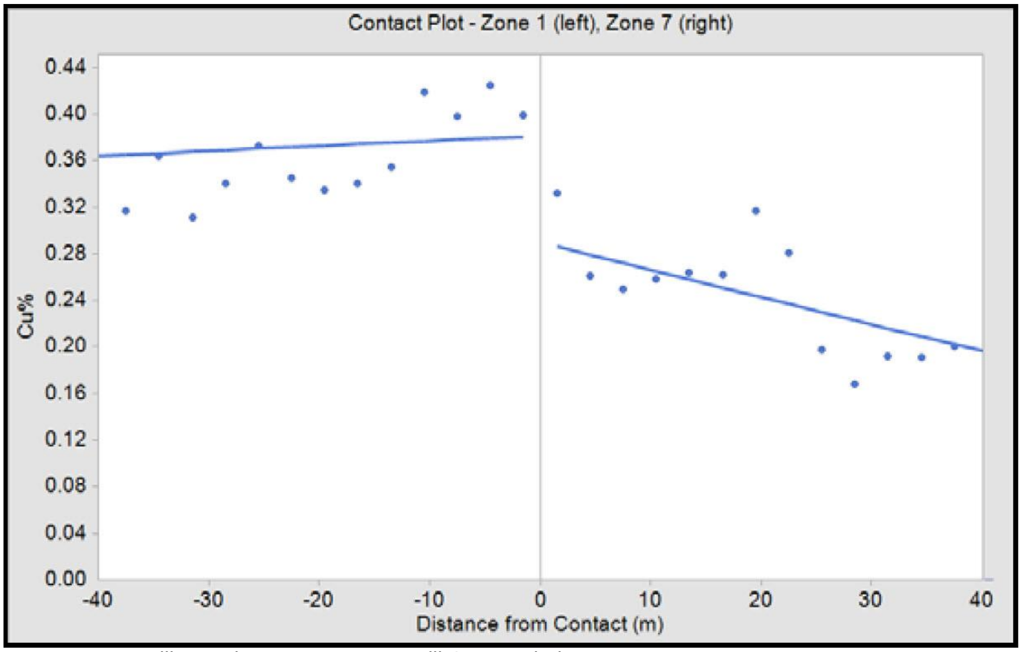
14.3.1 Contact Plots

The validity of the geology domain wireframes was evaluated using contact plots, plotting the copper grade in a set distance on either side of the contact of interest.

14.3.1.1 East Hill

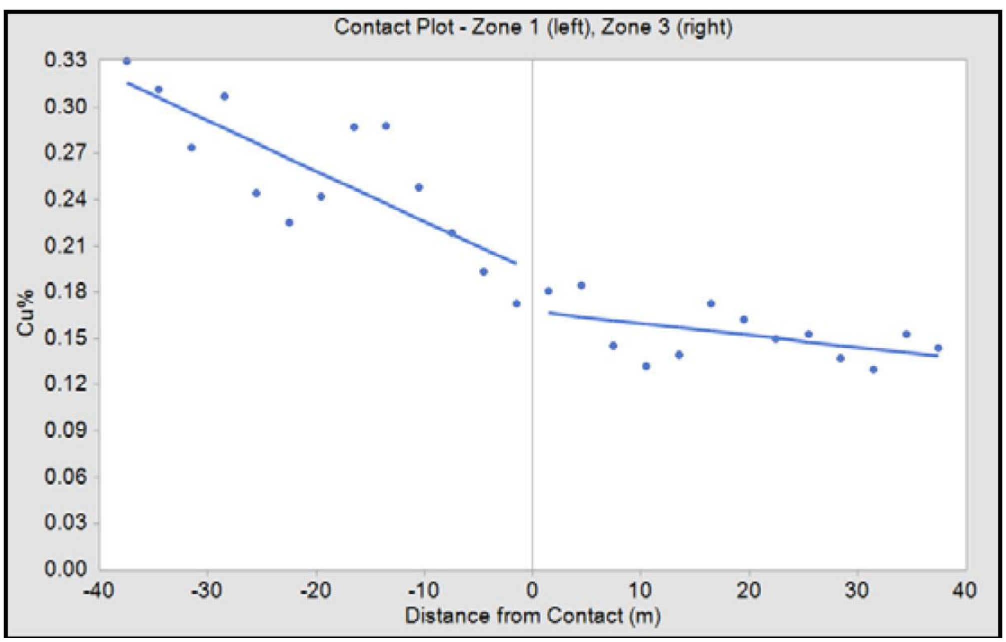
Contact plots for the East Hill geology solids are shown in Figure 14.4 to Figure 14.6. These plots highlight that much of the mineralization in the East Hill is hosted by breccias, with a gradual increase in the grade of adjacent quartz diorite towards the brecciated zones (Figure 14.4). The grade is fairly flat across the contact between quartz diorite and quartz diorite porphyry and the grade is lower in the porphyry adjacent to both the breccia zone and the quartz diorite (Figure 14.5 and Figure 14.6). The quartz diorite porphyry rock type is a minor host of mineralization on the East Hill and, therefore, is not shown in contact plots.

Figure 14.4 Contact Plot for Geology Wireframes Zone 1 and Zone 7



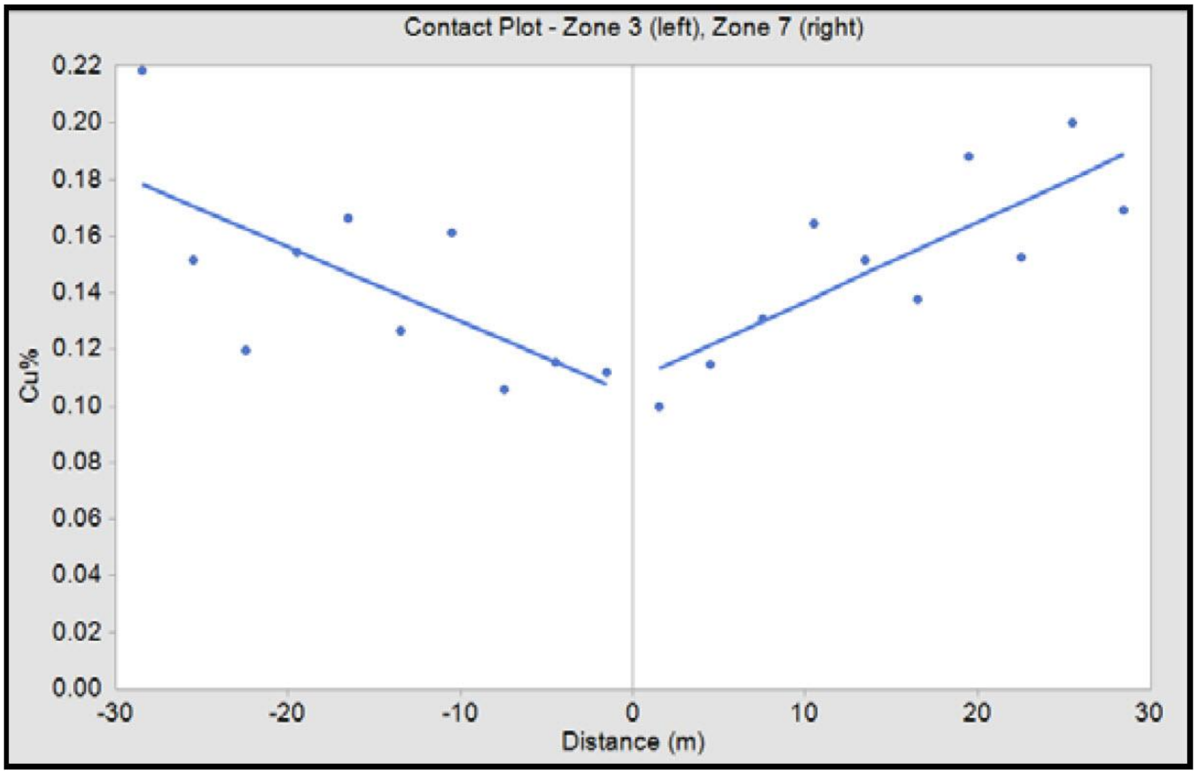
Zone1 = East Hill Breccias, Zone 7 = East Hill Quartz Diorite

Figure 14.5 Contact Plot for Geology Wireframes Zone 1 and Zone 3



Zone1 = East Hill Breccias, Zone 3 = East Hill Quartz-Feldspar Porphyry

Figure 14.6 Contact Plot for Geology Wireframes Zone 2 and Zone 7

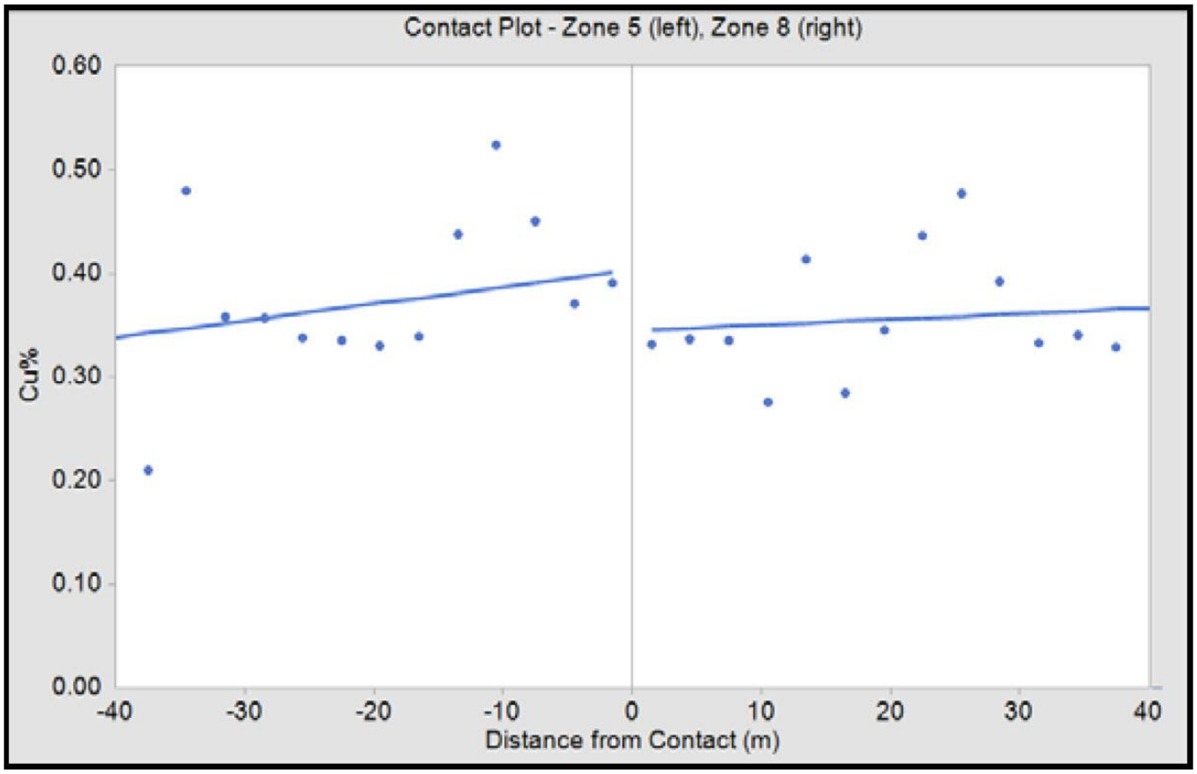


Zone 2 = East Hill Quartz-Diorite Porphyry, Zone 7 = East Hill Quartz Diorite

14.3.1.2 West Hill

Mineralization in the West Hill does not appear to have a strong preference for either of the dominant West Hill rock types of quartz diorite and quartz diorite porphyry. Overall the contact plot shows a fairly flat copper profile across the contact, with a slight decrease in grade in the quartz diorite porphyry with increasing distance away from the quartz diorite (Figure 14.7). Unlike the East Hill, breccia and quartz-feldspar porphyry rock types are volumetrically minor in the West Hill and therefore contact plots for these wireframes are not shown.

Figure 14.7 Contact Plot for Geology Wireframes Zone 5 and Zone 8



Zone 5 = West Hill Quartz Diorite Porphyry, Zone 8 = West Hill Quartz Diorite

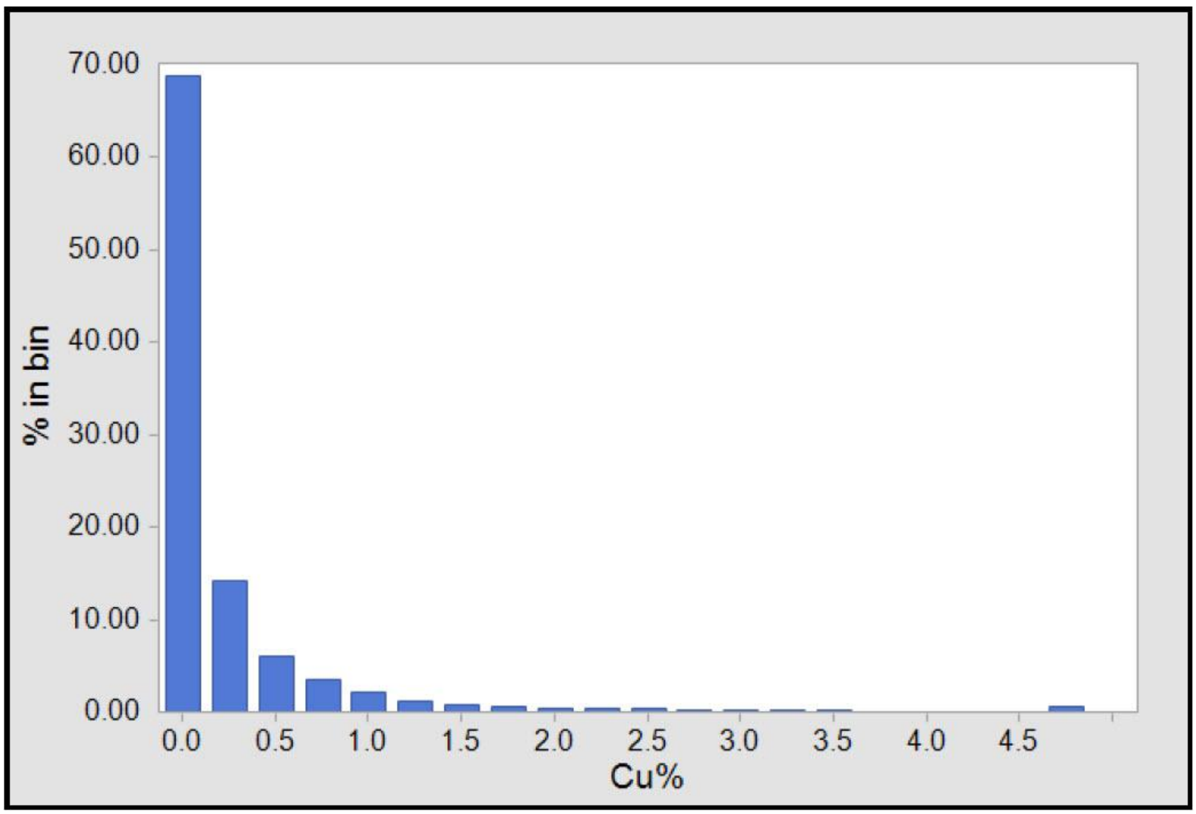
14.4 Assay Statistics

The assay data was evaluated based on the geological domains (as defined in Table 14.3) using standard descriptive statistics and graphical techniques including boxplots, scatterplots, and histograms. Table 14.3 summarizes the descriptive statistics for all copper assays in the database used for resource estimation. Figure 14.8 shows a histogram of copper assay data for all drillholes, and across all rock types. The histogram shows strong positive skewness, typical of this type of deposit.

Table 14.3 La Verde Assay Descriptive Statistics for Copper, all Drillholes and Rock Types

Statistic	**Raw Assay Samples**
Number of Samples	49,597
Minimum (Cu%)	0.0
Maximum (Cu%)	33.19
Mean (Cu%)	0.328
Median (Cu%)	0.104
Standard Deviation	0.902
Sample Variance	0.813
Coefficient of Variance	2.753
Kurtosis	306.799
Skewness	13.806

Figure 14.8 Histogram of Copper Assays, all Drillhole and Rock Types



14.5 Compositing

The raw drillhole assay data was composited into 3.0 m intervals starting at the collar and continuing to the bottom of the hole. More than 95% of assay samples are 3 m in length or less, and therefore this was selected as the most appropriate composite length. Composite lengths less than 1.5 m at the end of the hole were combined with the previous interval. The composites were then coded to the geology wireframes (zone item) based on the majority code for each composite. Composite summary statistics are reported in Table 14.4 for all drillholes and rock types. Composite summary statistics by geology domain are reported in Table 14.5. Figure 14.9 shows boxplots of the copper grade of 3 m composites by geology domain.

Table 14.4 La Verde 3 m Composite Descriptive Statistics for Copper, all Drillholes and Rock Types

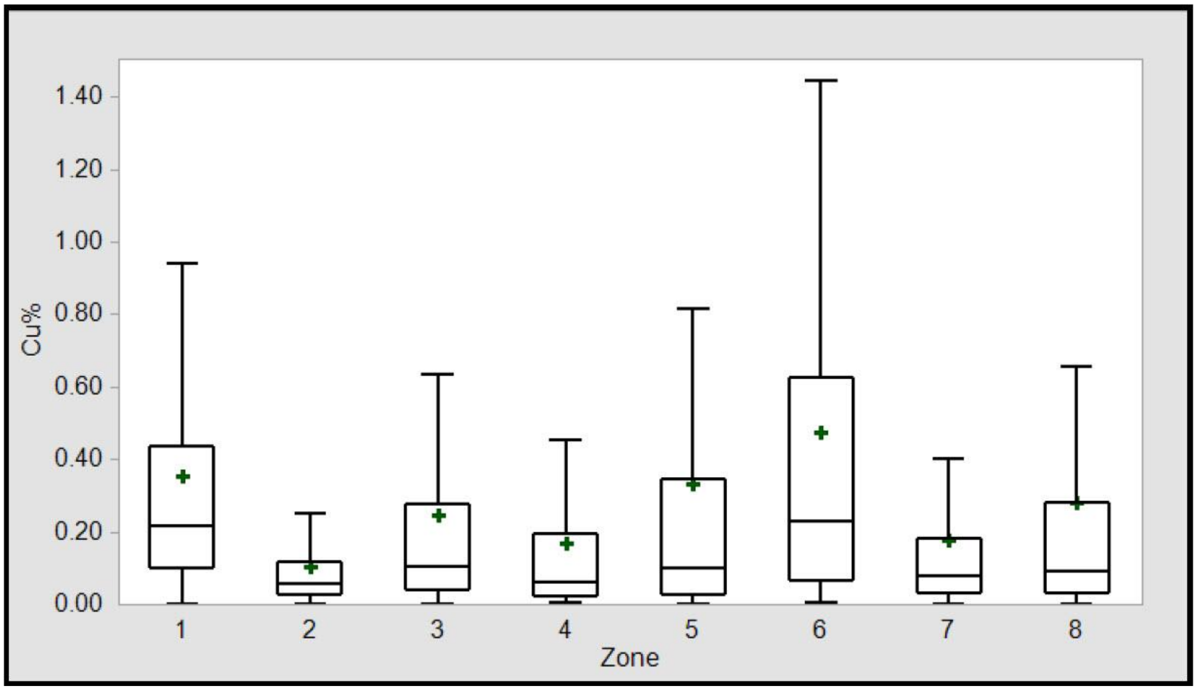
Statistics	3 m Composites
Number of Samples	34993
Minimum (Cu%)	0
Maximum (Cu%)	17.59
Mean (Cu%)	0.28
Median (Cu%)	0.117
Standard Deviation	0.511
Sample Variance	0.261
Coefficient of Variance	1.826
Kurtosis	127.714
Skewness	7.794

Table 14.5 La Verde 3 m Composite Descriptive Statistics for Copper by Geology Domain

Statistic	EH Breccia	EH Quartz Diorite Porphyry	EH Quartz-Feldspar Porphyry	WH Breccia	WH Quartz Diorite Porphyry	WH Quartz-Feldspar Porphyry	EH Quartz Diorite	WH Quartz-Dior
Zone	1	2	3	4	5	6	7	8
Number of Samples	10,343	436	2462	85	6,605	158	6,791	5,594
Minimum (Cu%)	0	0	0	0	0	0	0	0
Maximum (Cu%)	9.87	1.63	5.43	1.29	17.59	6.27	8.51	9.38
Mean (Cu%)	0.356	0.104	0.244	0.169	0.332	0.472	0.177	0.28
Median (Cu%)	0.215	0.056	0.104	0.061	0.099	0.23	0.078	0.09
Standard Deviation	0.474	0.161	0.365	0.254	0.719	0.694	0.354	0.548
Sample Variance	0.225	0.026	0.133	0.065	0.516	0.482	0.125	0.3
Coefficient of Variantion	1.334	1.55	1.499	1.502	2.163	1.471	2	1.956
Kurtosis	67.392	36.025	25.032	6.177	126.192	30.338	107.244	51.974
Skewness	5.768	5.087	3.671	2.438	8.452	4.48	7.941	5.647

Note: EH=East Hill, WH=West Hill

Figure 14.9 Boxplot of 3 m Composite by Geology Domain Showing Copper Grade



Note: See Table 14.3 for zones

14.6 Capping

Tetra Tech applied the same capping criteria for the current Resource estimate that was used for the Maunula (2012) estimate. Tetra Tech considers the capping strategy applied by Maunula (2012) to be reasonable for porphyry-style deposits, and retained the same strategy for consistency.

No cap was applied to copper values. Gold was capped at 3 g/t and silver was capped at 100 g/t. These caps affected ten gold values and six silver values, in both cases much less than 1% of the total number of gold or silver assays.

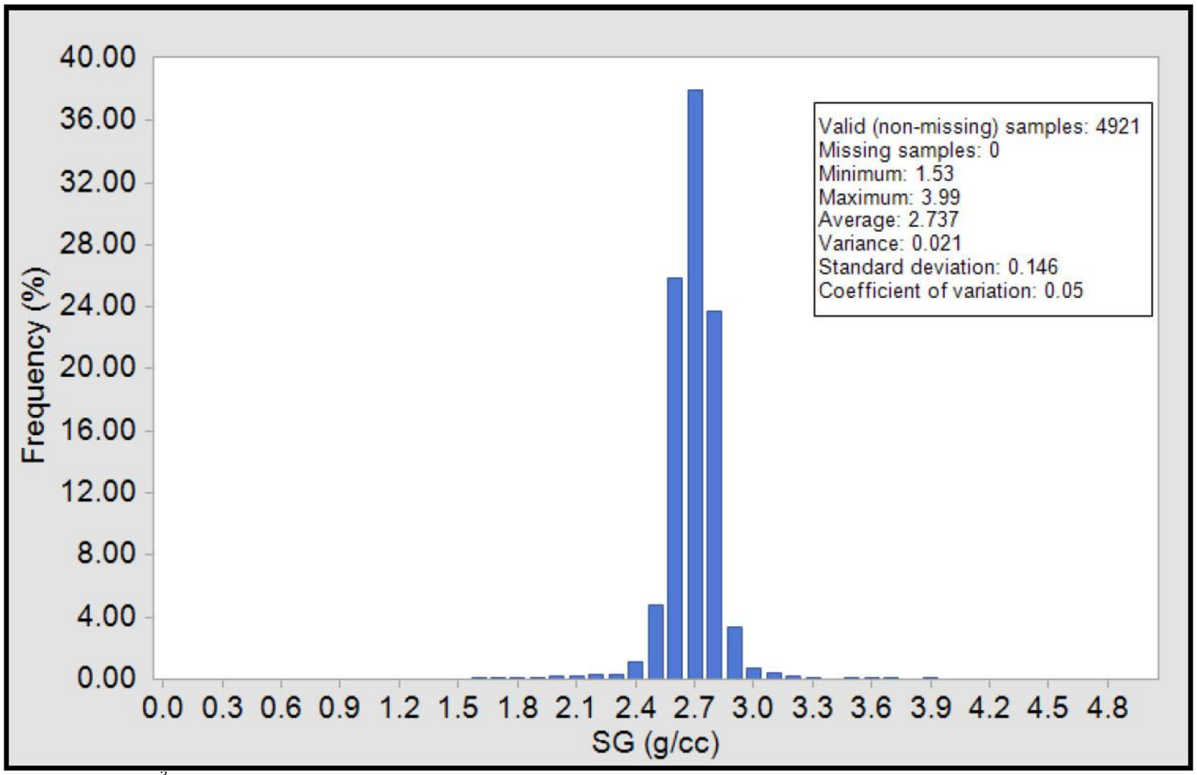
14.7 Specific Gravity

MHill determined specific gravity (SG) through immersion of drill core in water methodology. A total of 4,921 determinations were completed in 2011. Figure 14.10 is a histogram showing the distribution of measurements for SG.

The value of 2.7 g/cm^3 was selected for the Maunula (2012) La Verde model and this value is also used for the current Resource estimate. Tetra Tech recommends further analysis by rock types and mineralization styles to confirm whether one global value for the entire deposit is appropriate. Table 14.6 summarizes the existing SG data by main rock types, highlighting that 2.7 g/cm^3 is an appropriate global estimate, however more detailed SG data by rock type should be used for future estimates.

As the rocks are competent intrusive rocks, permeability and porosity are not considered material. Tetra Tech considers the SG to be representative of the bulk density.

Figure 14.10 Specific Gravity Determinations



Note: g/cc= g/cm^3

Table 14.6 Summary of La Verde SG Values by Rock Type

Rock Type	SG (g/cm3)
Quartz feldspar Porphyry	2.67
Quartz diorite porphyry	2.79
Quartz diorite	2.75
Breccias	2.69
Veins	3.4
Faults	3.63
Overburden	3.96

Note: EH=East Hill, WH=West Hill

14.8 Spatial Analysis

The spatial continuity of mineralization in a particular deposit is commonly assessed using variography. The variography of composited assay values for the deposit were assessed using MineSight™ MSDA v.2.80-02 software. Both downhole and directional variograms were initially constructed with 3 m lags, matching the composite lengths. Additional variograms were also constructed at several multiples of 3 m, up to 30 m, to test the parameters identified using the short range lags. Experimental variograms were constructed at 45° directional increments and 45° vertical increments, for a total of 10 variograms. The MineSight™ MSDA software then applied a best-fit algorithm for these 10 variograms producing the variogram model, which was exported from the MineSight™ MSDA software and imported into the MineSight™ MS3D software for interpolation. Variography was conducted separately for the East and West Hills, with the exception of arsenic, as the style of mineralization and dominant host rocks are different for each part of the deposit. Variography was also conducted separately for copper, gold, silver, and arsenic in order to best reflect the spatial distribution of these main metals of interest. The model variogram parameters for these metals are summarized in Table 14.7 for the East Hill and Table 14.8 for the West Hill.

Table 14.7 East Hill Variogram Model Parameters

Metal	Nugget	Total Sill	Range X	Range Y	Range Z	Rotation X	Rotation Y	Rotation Z
Cu	0.3421	1.4769	29.9	23.7	79.4	5.4	16.5	-2.1
Au	0.4756	1.2742	85.6	78.9	125.5	20	1.8	-5.2
Ag	0.469	1.3057	102	81	211.5	30.3	18.9	-2.5
As	0.7741	1.3407	91.6	271.5	271.5	280.7	-0.4	31.5

Table 14.8 West Hill Variogram Model Parameters

Metal	Nugget	Total Sill	Range X	Range Y	Range Z	Rotation 1	Rotation 2	Rotation 3
Cu	0.5406	1.0858	151.5	18.3	10.2	90.4	-6.3	68.8
Au	0.4777	0.8584	113.3	122.5	0.3	70.9	42	18.5
Ag	0.7148	1.2838	33.9	38.4	211.5	29	6	-3.8
As	0.7741	1.3407	91.6	271.5	271.5	280.7	-0.4	31.5

14.9 Grade Interpolation

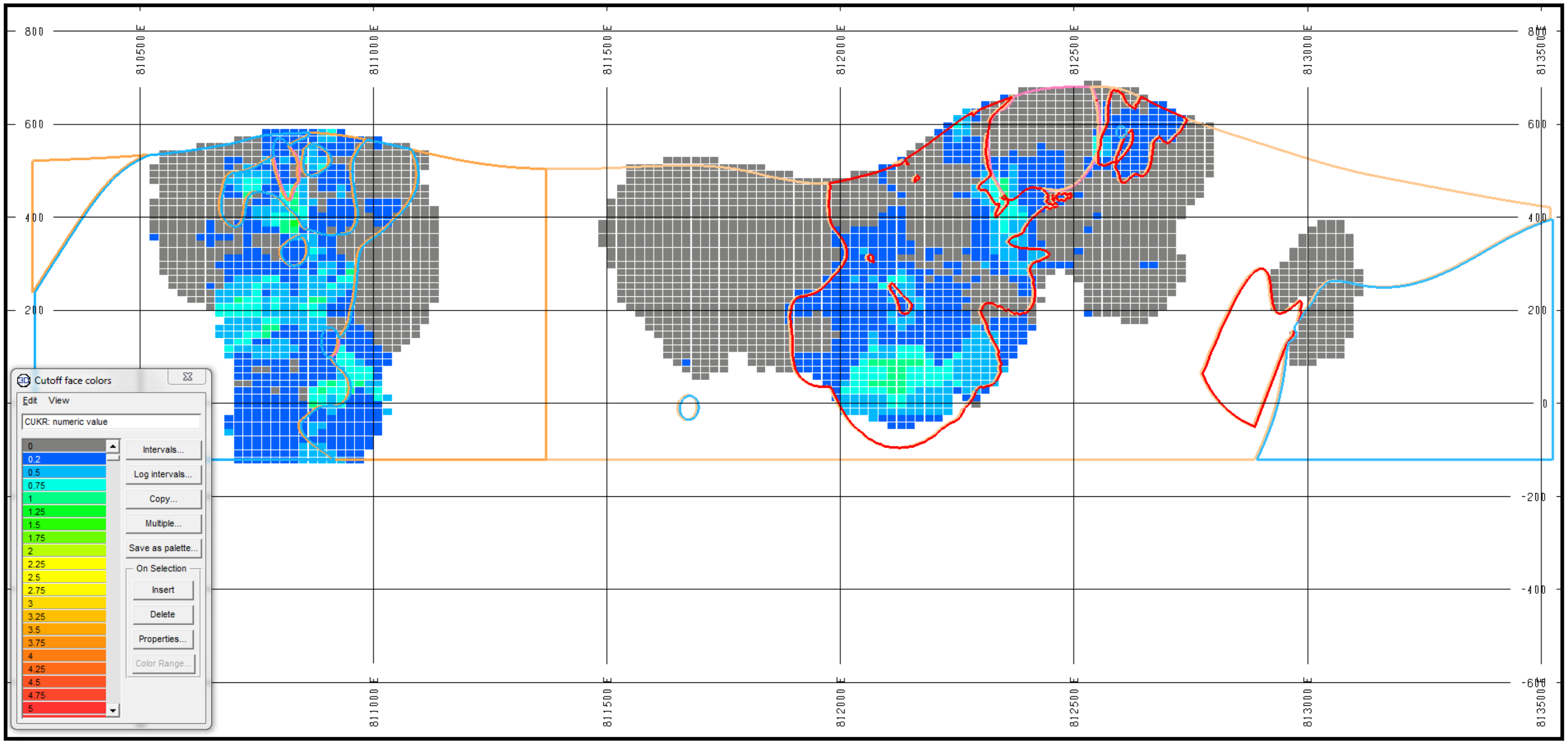
The East and West Hills were interpolated separately as each hill had different variogram models. Modelling consisted of grade interpolation by ordinary kriging (OK). An additional interpolation using inversed distance weighting (IDW) was also completed for validation purposes. The grade interpolation used search ellipses as defined in Table 14.9. These parameters were based on the geological interpretation and variogram analysis for each hill and for each metal. The search ellipses were chosen based on the ranges indicated by the variogram models, such that the ellipses were larger in all dimensions than the variogram model. In many cases, no rotation of the search ellipse was considered necessary.

Table 14.9 Search Ellipses

Deposit Area	Metal	Search Distance			MS3D Rotation		
		X	Y	Z	Rotation 1	Rotation 2	Rotation 3
East Hill	Cu	125	125	125	0	0	0
	Au	150	150	180	0	0	0
	Ag	150	150	250	30.3	18.9	-2.5
	As	275	275	275	0	0	0
West Hill	Cu	125	125	160	0	0	0
	Au	175	180	100	70.3	42	18.5
	Ag	100	100	250	29	6	-3.8
	As	275	275	275	0	0	0

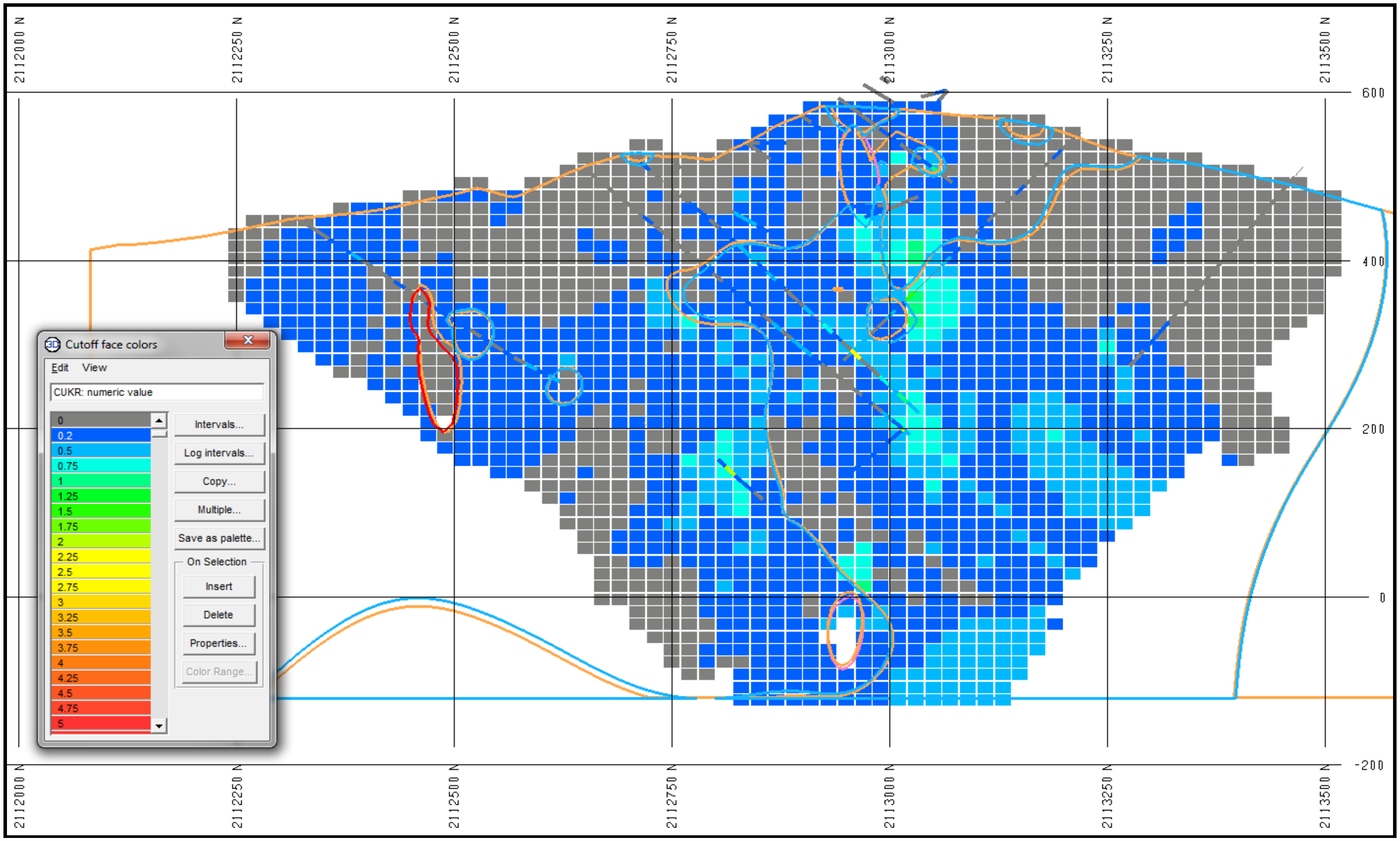
Figure 14.11 to Figure 14.13 shows the block model OK copper grade in a typical east-west section (Figure 14.11), north-south section (Figure 14.12), and plan view (Figure 14.13). Figure 14.12 also shows drillhole traces within the volume clipping of that section; the drillhole display shows 15 m composites for illustration purposes. As most drilling was completed with roughly north or south azimuths, they do not display well within the volume clipping of east-west or plan view sections and therefore are not shown on Figure 14.11 or Figure 14.13.

Figure 14.11 Copper OK Block Model, East-West Section at 2113000 North (Looking North)



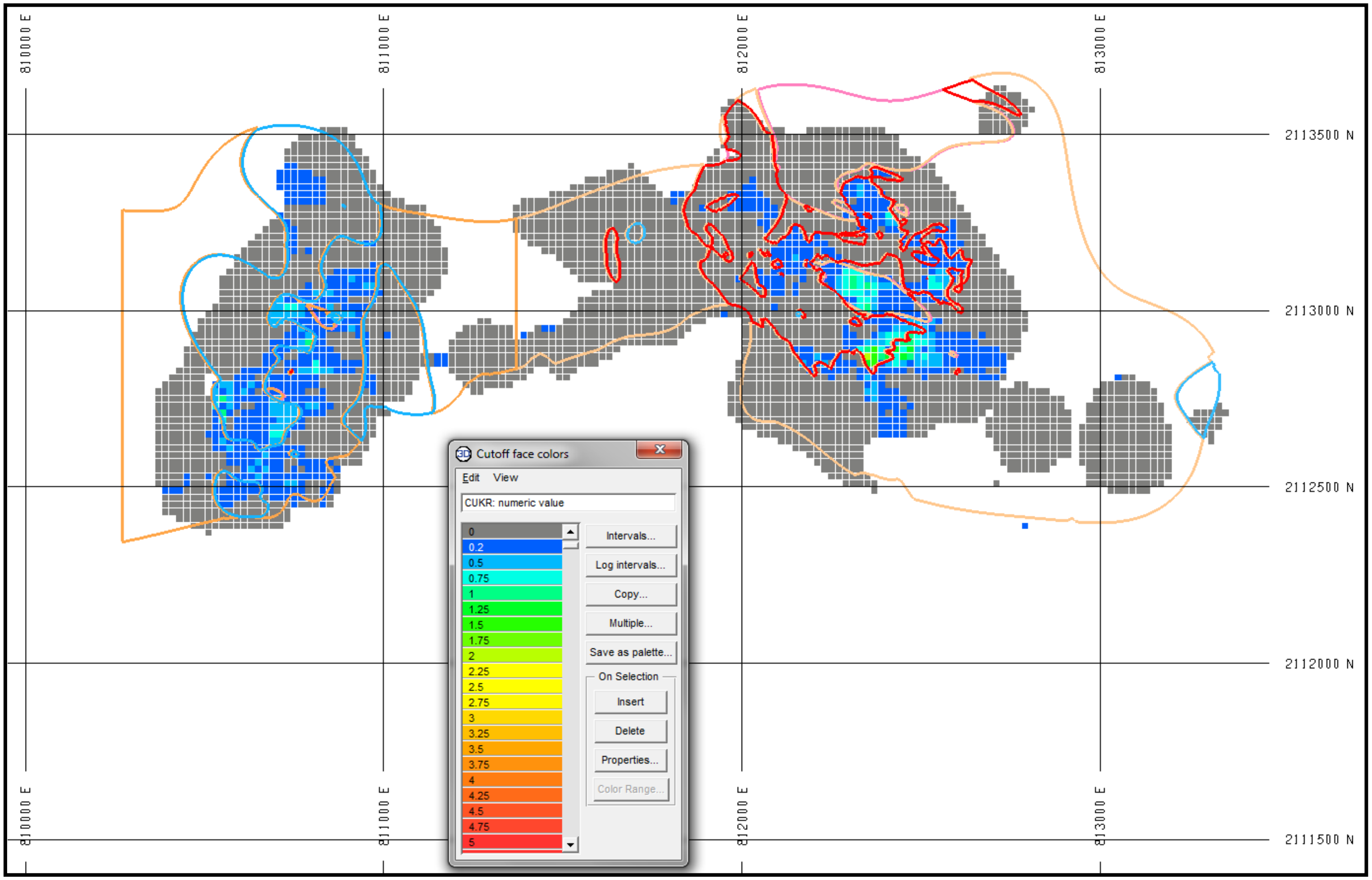
Note: Geology domain colours as described in Section 14.3. Easting and elevation in metres. Only blocks classified as Inferred-Indicated-Measured are shown.

Figure 14.12 Copper OK Block Model, North-South Section at 810850 East (Looking West)



Note: Geology domain colours as described in Section 14.3. Northing and elevation in metres. Only blocks classified as Inferred-Indicated-Measured are shown. Drillhole traces show 15 m composites.

Figure 14.13 Copper OK Block Model, Planview 480 m Elevation (Looking Down)



Note: Geology domain colours as described in Section 14.3. Easting and northing in metres. Only blocks classified as Inferred-Indicated-Measured are shown.

14.10 Block Model Validation

Tetra Tech distinguishes between verification from validation as follows:

- Verification is a manual (e.g. visual inspection) or quasi-manual (e.g. spreadsheet) check of the actual procedure used.
- Validation is a test for reasonableness using a parallel procedure, which may be either manual or (but is usually) a computer-based procedure.

14.10.1 Visual Checks

Interpolated block grades, Resource classification, geological interpretation outlines and drillhole composite intersections were verified on screen for plan and section. Based on visual inspection by Tetra Tech, the block model grades appeared to honour the data well (for example Figure 14.11). The estimated grades exhibit a satisfactory consistency with the drillhole composites.

14.10.2 Global Comparison

Tetra Tech verified the block model estimates for global bias by comparing the average copper grades from the model (OK) using IDW estimates. The results (Table 14.10) show that the IDW results compare well with the OK results. A higher degree of smoothing is evident in the IDW model.

Table 14.10 Global Comparison, 0.1% Copper Cut-off

Block Selection	CUKR (%)	CUID (%)
Inferred + Measured + Indicated	0.268	0.241
Measured + Indicated	0.303	0.280

Note: CUKR = OK results; CUID = IDW results

Histogram and boxplots were created of the block model data and reviewed with respect to the corresponding composite plots. No apparent bias was noted. Table 14.11 summarizes the global statistics of composites versus blocks.

Table 14.11 Composite versus Block Model Statistics for all Blocks

Source	Cu% (Mean)	Standard Deviation	Coefficient of Variation
Drillhole Composites	0.280	0.511	1.83
Block Model	0.168	0.170	1.011

14.10.3 Adequacy of Resource Estimation Methods

The deposit has been estimated using modern block modelling techniques in industry-standard software. This included proper geologic input, appropriate block model cell sizes, review of grade capping, assay compositing and reasonable interpolation parameters. The results have been verified by peer review, by visual review, and by statistical comparisons between the estimated block grades and the composites used to assign them. The OK has also been validated with an alternative estimation method, IDW. No biases have been identified in the model.

14.11 Mineral Resource Classification and Resource Statement

14.11.1 Resource Classification

The Mineral Resources are classified under the categories of Measured, Indicated and Inferred Mineral Resources, in accordance with CIM Definition Standards. The classification reflects confidence of grade continuity as a function of: QA / QC procedures, quality of assay and density data, and sample spacing relative to geological and geostatistical observations regarding continuity of mineralization.

Tetra Tech is not aware of any environmental, permitting, legal, title, taxation, socioeconomic, marketing or other relevant issues that may affect the estimate of Mineral Resources at this time. The volume of material removed by historical mining is insignificant compared with the total volume of mineralized material, and therefore this material was not subtracted from the tonnages below.

The Resource model blocks were classified into Measured, Indicated and Inferred Resource categories based on the level of confidence in the grade estimate for each block. The classification was based on a combination of kriging variance and distance from the nearest composite. The classification boundaries used in the Maunula (2012) estimate (nearest composite less than 15 m for Measured, less than 50 m for Indicated, less 160 m for Inferred) were used as a basis to evaluate the kriging variance around these distances in both the East and West Hills. The kriging variance was then used to generate meshes containing blocks of similar estimation confidence, using kriging variance as an inverse proxy measure of confidence. Kriging variance increases with increasing distance from the data used for the kriging estimation, at a rate dependent on the variography for the variable. It is, therefore, a useful measure of uncertainty (or inverse measure of confidence) in the estimation process. The meshes of these kriging confidence solids were constructed using a 100 m triangulation resolution. The Resource classification then assigned a confidence level to blocks within each of the Measured, Indicated, and Inferred kriging variance solids.

Tetra Tech regards this process as appropriate to guide the definition of boundaries between the Mineral Resource categories in this deposit.

14.11.2 Mineral Resource Statement

The Mineral Resource, as of 19 September 2012 comprises Measured, Indicated and Inferred Resources. The classified Mineral Resources are shown in Table 14.12. At a base-case cut-off grade of 0.2% copper, the total Measured Resource is 57,527,000 t at 0.45% copper, the Indicated Resource is 350,442,000 t at 0.40% copper, and the Measured + Indicated Resource is 407,969,000 t at 0.41% copper. The total Inferred Resource at a 0.2% copper cut-off is 337,838,000 t at 0.37% copper.

Table 14.12 Mineral Resource Statement, La Verde Project

Resource Class	Cut-off (Cu%)	Zone	Tonnes (000s)	Cu (%)	Ag (g/t)	Au (g/t)	As (%)
Measured	0.1	East Hill	57,963	0.37	2.75	0.05	0.02
		West Hill	20,995	0.37	1.95	0.01	0.05
		Total	78,958	0.37	2.54	0.04	0.03
	0.2	East Hill	41,262	0.46	3.30	0.06	0.02
		West Hill	16,265	0.43	2.03	0.01	0.05
		Total	**57,527**	**0.45**	**2.94**	**0.05**	**0.03**
	0.3	East Hill	29,896	0.54	3.68	0.07	0.03
		West Hill	11,194	0.52	2.15	0.01	0.06
		Total	41,090	0.53	3.26	0.05	0.03
Indicated	0.1	East Hill	322,936	0.26	2.32	0.05	0.02
		West Hill	262,669	0.33	1.39	<0.01	0.04
		Total	585,605	0.30	1.91	0.03	0.03
	0.2	East Hill	163,604	0.39	3.31	0.06	0.04
		West Hill	186,838	0.41	1.47	0.01	0.03
		Total	**350,442**	**0.40**	**2.33**	**0.03**	**0.04**
	0.3	East Hill	98,784	0.49	4.04	0.07	0.05
		West Hill	126,505	0.49	1.55	<0.01	0.03
		Total	225,289	0.49	2.65	0.04	0.04
Measured + Indicated, 0.1% Cu			664,563	0.30	1.98	0.03	0.03
Measured + Indicated, 0.2% Cu			**407,969**	**0.41**	**2.42**	**0.03**	**0.04**
Measured + Indicated, 0.3% Cu			266,379	0.50	2.74	0.04	0.04
Inferred	0.1	East Hill	434,614	0.20	1.51	0.03	0.02
		West Hill	330,426	0.29	1.17	<0.01	0.03
		Total	765,040	0.24	1.36	0.02	0.02
	0.2	East Hill	140,566	0.35	2.79	0.05	0.04
		West Hill	197,272	0.38	1.33	<0.01	0.02
		Total	**337,838**	**0.37**	**1.94**	**0.02**	**0.03**
	0.3	East Hill	61,761	0.49	4.20	0.07	0.07
		West Hill	126,222	0.46	1.45	<0.01	0.02
		Total	187,983	0.47	2.36	0.03	0.04

This Resource estimate is not constrained by a pit shell. The cut-off grade of 0.2% copper is based on experience for similar open pit projects and a mining conceptual study which used a metal price of $2.50/lb and copper metal recovery of 92% (Maunula, 2012).

14.12 Previous Mineral Resource Estimate

Prior to the Tetra Tech September 2012 Report a NI 43-101 compliant Resource was reported by T. Maunula of Tetra Tech in January 2012. Table 14.13 summarizes the differences between the January 2012 Resource Report and the September 2012 Resource Estimate.

The difference between the January 2012 estimate and the current estimates is in part due to additional drilling completed in 2012. The 12 drillholes completed in 2012 targeted areas of the deposit for both delineation of additional mineralization and infill drilling in select areas to allow for upgrading of the Mineral Resource classification. As a result, the Resource tonnage is larger in the current Resource estimate, and includes a 15% increase in the Indicated + Measured categories and an 85% increase in the Inferred category. As expected, the grades remain similar between the two models, and, therefore, the percent difference in metal content is similar to the percent difference in tonnage.

The increase in the Inferred category is also due to a change in the parameters used for classification. Classification was based on a combination of the distance from the nearest composite and kriging variance. It was also guided by the most recent 3D geology model that was not available when the January 2012 Resource Report was written.

Table 14.13 Comparison with Previous Mineral Resource Estimate

Description	Tetra Tech January 2012		Current Estimate		Difference in Tonnes (%)	Difference in Metal (%)
	Tonnes (000s)	Cu (%)	Tonnes (000s)	Cu (%)		
Measured	29,342	0.50	57,527	0.45	96	76
Indicated	324,735	0.40	350,442	0.40	8	8
Measured + Indicated	354,077	0.41	407,969	0.41	15	15
Inferred	181,739	0.40	337,838	0.37	86	72

15 MINERAL RESERVE ESTIMATES

Under NI43-101 guidelines, due to the Preliminary Economic Assessment level of the present study, no mineral reserves are reported.

16 MINING METHODS

The word "ore" is used in this section in a generic sense and does not imply that Mineral Reserves have been estimated.

The La Verde main mineralized regions, East Hill and West Hill, lend themselves to open pit mining as they are large bulk Resources close to surface. Additionally, the local topography is amenable to hold waste dumps and stockpiles to contain all the materials scheduled. The climate is favorable to open pit mining and there are no surface water bodies that would be impacted by the mining activity.

The proposed open pit recovers the upper portion of the two main mineralized regions leaving some deeper Resource un-mined.

16.1 Resource Model for Mining – Mining Block Model

A Resource block model was supplied by Catalyst. The Resource model received was the direct output of Tetra Tech's work on geological modelling and grades interpolation as described in the Tetra Tech September 2012 Report. This model was modified to account for practicalities of mining and used in the mining, metallurgical and economic analysis.

The significant modifications made to the model were:

- Rock type grouping – As indicated in Section 14, all grade interpolations were allocated to eight lithology classes. In the mining block model, zones were grouped into bigger categories of the West or East Hill.
- New Oxide Zone created – In the Resource model all mineralization is classified as sulphide. However oxidation is evident in the upper meters of some drillholes, as well as in the field. As copper recoveries by flotation are often lower in oxide mineralization than in sulphide mineralization, it was agreed with Catalyst that a simplified 20 m thick oxide blanket would be added to the model in order to account for this effect. All original grade estimates were retained within the oxide blanket.
- Resource classification – Mineralized material was classified in the Resource model as Measured, Indicated, Inferred and Not-Classified. As this is a PEA, all classified mineralization has been included in the analysis so in the mining block model Measured, Indicated and Inferred categories were grouped as one. The Not-Classified category was treated as waste,
- Grades: Kriged estimates for copper (CUKR), gold (AUKR), silver (AGKR) and arsenic (ASKR) were selected for the development of the study. In agreement with Catalyst, molybdenum grades were deemed too low to consider this metal as a recoverable product. Hence, while molybdenum is included in the Resource statement, it was excluded of any further analysis.

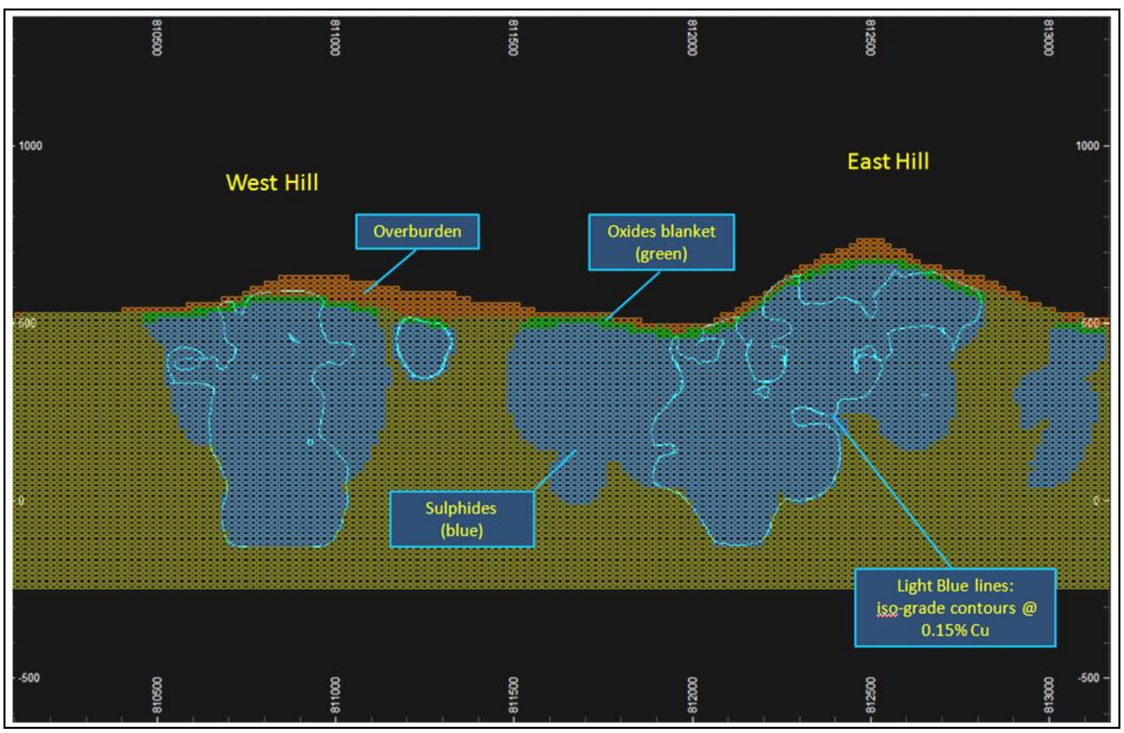
Table 16.1 maps the categories used in the Resource model to those in the mining model.

Table 16.1 Rock Types in the Mining Block Model

Resource Model Zone and Category		Mining Block Model Rocktypes	
Zone	**Classification**	**Sulphides**	**Oxides**
1 – **East Hill** Breccia	Measured + Indicated + Inferred	ESU	EOX
2 – **East Hill** Quartz Diorite Porphyry			
3 – **East Hill** Quartz Feldspar Porphyry			
7 **–** **East Hill** Quartz Diorite			
4 – **West Hill** Breccia	Measured + Indicated + Inferred	WSU	WOX
5 – **West Hill** Quartz Diorite Porphyry			
6 – **West Hill** Quartz Feldspar Porphyry			
8 – **West Hill** Quartz Diorite			
All Zones	Not Classified	Added to background waste	

Figure 16.1 shows the rock types in the mining block model and their spatial distribution.

Figure 16.1 Rock Types in a Typical Section through West and East Hill, Facing North



16.1.1 Processing Eligibility and NSR Calculations

As described in Sections 13.4 and 13.5, metallurgy testing to date suggests that arsenic in the La Verde final concentrates can reach levels that, if not reduced before selling, would incur severe smelter penalties, and even potentially make the concentrates unmarketable. Penalties ultimately increase the copper selling cost and they can be as high as to reduce the concentrate value close to zero.

On the other hand, high-arsenic concentrates can be treated on site, potentially by roasting, to remove a significant proportion of the contained arsenic before releasing the final product to copper smelters. In this case, the value of the concentrate is also reduced by the operating costs of the arsenic reduction stages.

In order to properly reflect the economic impact of the arsenic values reporting to the final concentrates, an NSR equivalent grade was calculated for all potential ore blocks. This equivalent grade was used in the Lerchs-Grossman analysis to identify the ultimate pit used for mine scheduling. In all cases the NSR value was estimated to the concentrate filtering stage (i.e. mill product). Table 16.2 summarizes the parameters used in the NSR calculations. The sources for these parameters can be found in Sections17 and 19. Note that for smelter charges and refining charges, two set of parameters, denoted by ‘(I)’ and ‘(II)’, will be found. These were used to calculate alternative NSR values depending on the metallurgical course of action selected.

Table 16.2 NSR Calculation Parameters and Assumptions

Parameter		Description	Value	Comment
Primary Recovery Method		Single concentrate flotation		
Metals Adding Value		Cu, Ag, Au		Au, Ag paid as by-products. Arsenic penalties at smelter and refinery
Mill Recoveries	Cu	Sulphides	East Hill : 90% West Hill : 90%	See Table 17.1
		Oxides	East Hill: 50% West Hill: 0%	
	Au	Sulphides	East Hill: 80% West Hill: 0%	
		Oxides	East Hill: 75% West Hill: 0%	
	Ag	Sulphides	East Hill: 85% West Hill: 60%	
		Oxides	East Hill: 50% West Hill: 50%	
	As	Sulphides	East Hill: 70% West Hill: 20%	
		Oxides	East Hill: 70% West Hill: 20%	
Concentrate Cu grade		Sulphides	East Hill: 27.5% West Hill: 26%	
		Oxides	East Hill: 29% West Hill: 0%	
Cost of Roasting		On site	30 US$/tonne	Per tonne of concentrate
Cost of Transport to Smelter			70 US$/tonne	Per tonne of concentrate
% Payable	Cu		96.5%	
	Au		95%	Minimum payable content of 1 g per concentrate tonne
	Ag		90%	Minimum payable content of 30 g per concentrate tonne
Smelter Charges (I) (when sold in restricted market for “dirty” concentrates)		As < 0.5%	108 US$/ dry tonne	
		0.5% < As < 1.0%	128 US$/dry tonne	
		As > 1.0%	190 US$/ dry tonne	Plus an additional 5 US$/ dry tonne per each 0.01% above 1%
Refining Charges (I) (when sold in restricted market for “dirty” concentrates)		As < 0.5%	0.1075 US$/Cu lb	
		0.5% < As < 1.0%	0.1275 US$/Cu lb	

Parameter		Description	Value	Comment
		As > 1.0%	0.19 US$/Cu lb	
Smelter Charges (II) (when sold in regular market as "clean" concentrate)		As < 0.2%	70 US$/ dry tonne	
		0.2% < As < 1.0%	70 US$/dry tonne	Plus an additional 3 US$/ dry tonne per each 0.1% above 0.2%
Refining Charges (II) (when sold in regular market as "clean" concentrate)		As < 0.2%	0.07 US$/Cu lb	
		0.2% < As < 1.0%	0.07 US$/Cu lb	
Metal Prices	Cu		2.7 US$/lb	
	Au		1,200 US$/tOz	
	Ag		25 US$/tOz	

16.2 Strategic Mine Planning

Since this is the first economic analysis of the La Verde deposit, some strategic technical-economic assumptions had to be made. These included: processing and mining capacities, cut-off grade strategy, options for arsenic reduction, and concentrate marketing. These assumptions are not final and even though specific values have been used, significant further analysis is required to advance the project to pre-feasibility level.

16.2.1 Selection of the Size of the Operation

Prior to the present study, preliminary pit optimization runs showed potential mill feed tonnages in the 350 Mt – 550 Mt range. Some benchmarking in similar deposits (porphyries) led Catalyst to consider a likely range of milling capacity for La Verde in the 40 ktpd – 90 ktpd range.

As part of the present study, a high-level analysis was undertaken in order to estimate a preliminary capacity where a maximum value could be found. For this comparative purpose, the NPV was calculated as the difference between the sum of the discounted operating surplus associated with a given mining schedule, and a corresponding estimate of the initial capital. Plant throughputs from 22 Mtpa (60.3 ktpd) to 55 Mtpa (151 ktpd), with their corresponding mine capacities and capital costs were analyzed. The Lerchs-Grossman (LG) algorithm for optimum pit geometry search was used in the exercise.

This high-level analysis showed a relatively wide and stable maximum value around a processing capacity of 30 mtpa. Hence, this throughput was selected for this PEA.

AMC emphasizes the utmost importance of this strategic analysis in the proper development of any mining project. Should the La Verde project be advanced to a pre-feasibility stage, a detailed study to estimate the optimal mix of mining rate, processing rate and cut off grade should be performed.

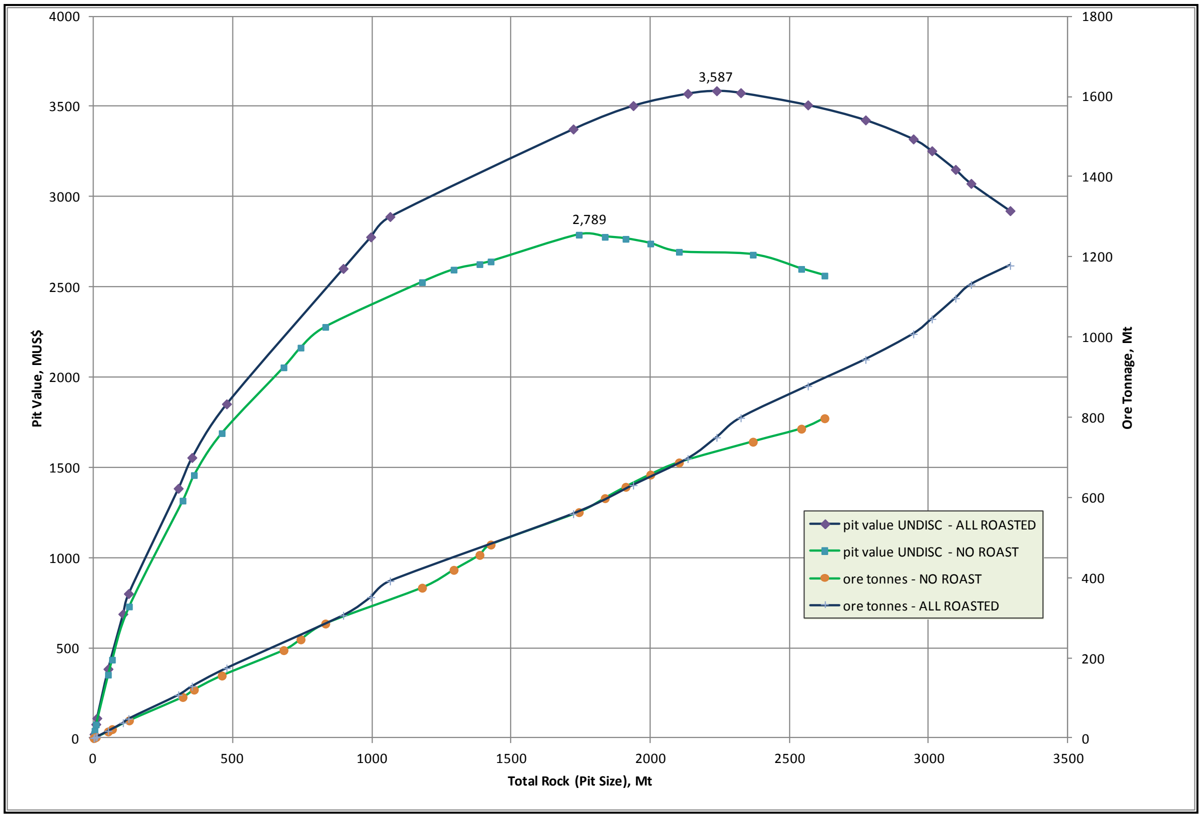
16.2.2 Inclusion of a Roasting Stage

As discussed, metallurgy testing to date suggests that arsenic in the La Verde final concentrates can reach levels that will adversely impact the economic value of the project due to additional cost being incurred after the concentrate is realized at the mill. Additional costs incurred are dependent on which of the two potential courses of action are chosen, namely:

- Producing "dirty" concentrates. In this case the selling cost will be higher due to the high smelter penalties imposed by the very restricted market which can accept high arsenic grades. Ultimately, the increase in the selling cost can be as high as to reduce the value of the concentrate to close to zero.
- Producing "clean" concentrates and selling them in the regular smelting market. For the anticipated arsenic grades in the La Verde concentrate, this option requires treating the concentrate on site by partial roasting in order to bring the arsenic levels to the "clean" ranges. In this option, the value of the project is reduced by the capital and operating cost of the roasting stage.

Both options were analyzed for La Verde. For each case, the corresponding NSR calculation accounts for the change in value of any eligible block according to the post-mill path option selected. By the use of LG algorithm for optimum pit geometry search, the series of pits were obtained for each NSR version (roast, no-roast). Figure 16.2 shows the total operating surplus and the total mill feed tonnage for different optimal pit sizes (expressed as tonnage of total rock on the X-axis). It can be seen that the non-roasting option curve lies beneath the roasting option curve along the whole range. This indicates that, for the same set of parameters, the non-roasting option always yields lower value pits. For the non-roast option the maximum undiscounted pit value is 2,789 MUS$ at a pit size of around 1,700 Mt containing around 560 Mt of mill feed. For the roasting option maximum undiscounted pit value is 3,587 MUS$ at a pit size of around 2,200 Mt containing around 750 Mt of mill feed. The same behavior and relative position of the curves is observed when discounted pit values are used.

Figure 16.2 Undiscounted Pit Value and Mill Feed Yield versus Pit Size – Roast / No-Roast Options



The preceding results support adopting the roasting option as the base case for the current PEA.

Figures 16.3 and 16.4 illustrate the change in pit limit between the roast and no-roast options.

Figure 16.3 No-Roasting NSR Values and Optimal Pit – Typical Section through West and East Hill, facing North

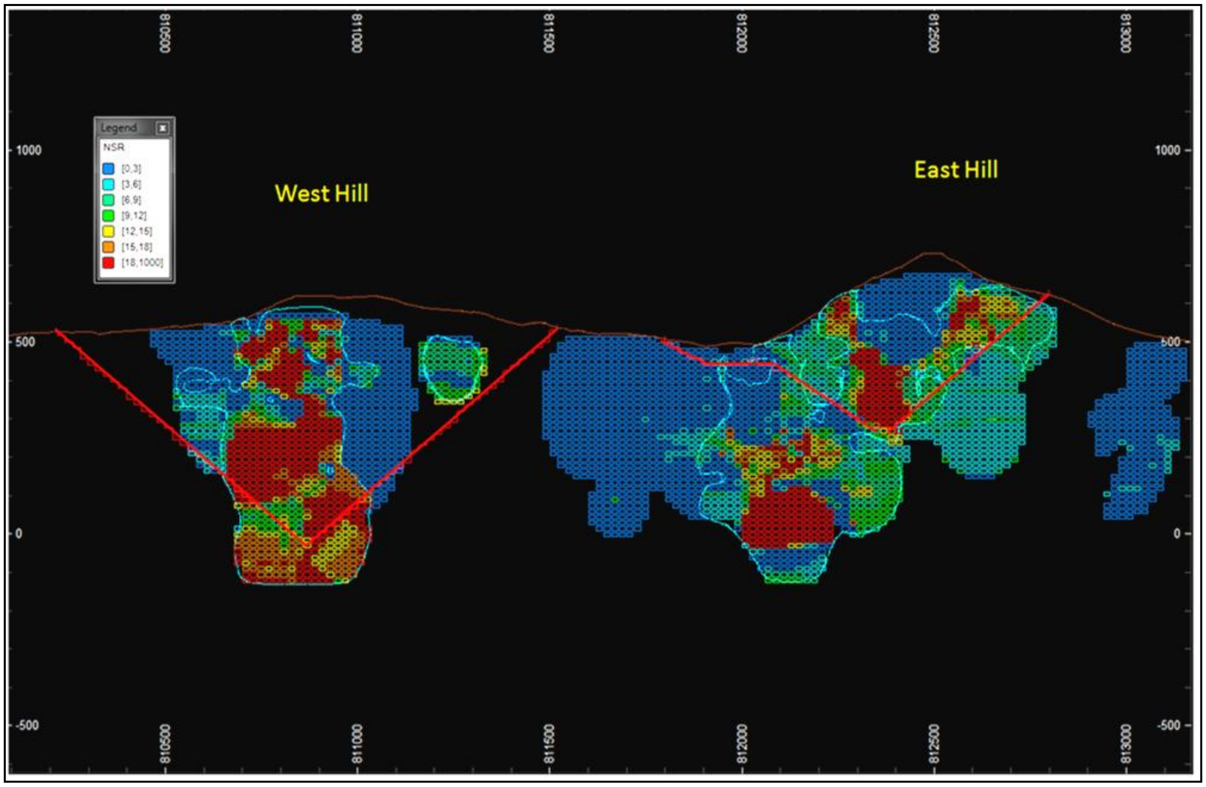


Figure 16.3 shows a typical cross section showing block NSR values for the no-roast option. The thick red lines depict the optimal pit for a given set of parameters. The pit presented corresponds to the one giving the maximum undiscounted value on green curve in Figure 16.2.

Figure 16.4 Roasting NSR Values and Optimal Pit – Typical Section through West and East Hill, facing North

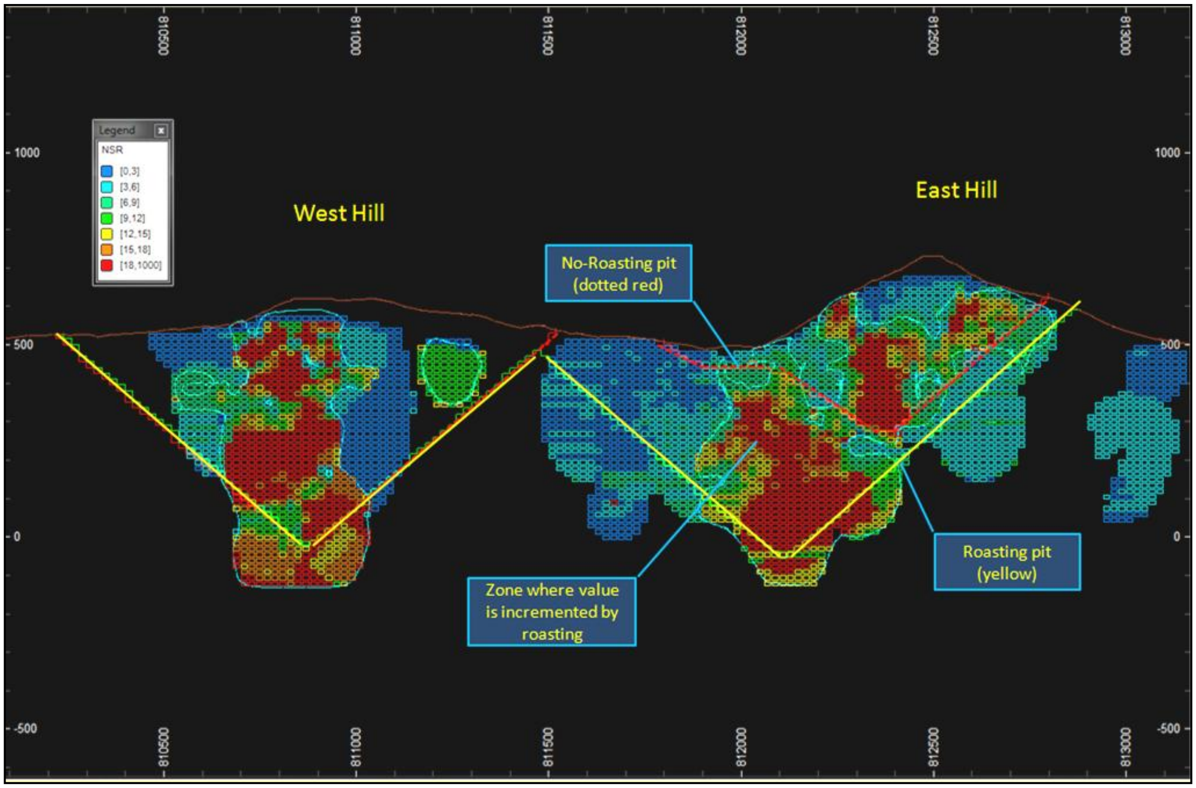


Figure 16.4 shows a typical cross section showing block NSR values for the roast option. The thick yellow lines depict the optimal pit for the same set of parameters used for the no-roast option. While the West Hill pit remains the same, the East Hill pits grows significantly down and westwards due to the gain in value of the corresponding regions in the deposit.

These results identified the roasting option as preferable.

The potential to add value by deferring the roaster construction (and postponing capital expenses), and by using an elevated cut-off grade (COG) in the early years was tested. The answer to this question was sought by a high-level strategic LG-based analysis varying the number of time periods (years) of no-roasting operation and the COG policy applied in the early years. Each COG sequence was called a Series. The Series defined for the study are shown in Table 16.3.

Table 16.3 Cut-off Grade Series Tested

COG series - NSR, US$/t														
	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
Series 0 (Base Case)	6	… >	Flat Marginal COG											
Series 1	12	12	10	10	8	8	6	… >						
Series 2	14	14	12	12	10	10	8	8	6	… >				
Series 3	16	16	14	14	12	12	10	10	8	8	6	… >		
Series 4	18	18	16	16	14	14	12	12	10	10	8	8	6	… >

The analysis allowed exploring for the best timing for the roaster construction as well as the COG policy which gave the highest value. Each case had two stages of operation: a Stage 1 with no-roasting, followed by Stage 2 with roasting for the remainder of the mine life. Depending on the COG Series applied some mineralized stockpile was produced and the assumption was that all the stockpiled material was processed at the end of the mine life.

A high-level analysis was undertaken in order to delineate where the maximum value could be found. In this case, “value” was calculated as the difference between the sum of the discounted operating surplus associated with a given mining schedule – a sequence of Stage 1 followed by a Stage 2 – and the corresponding discounted roaster capital which was inserted in the year before the commencement of Stage 2 (i.e. last year of Stage 1). The results of the analysis are shown in Table 16.5.

Table 16.4 Normalized results for the search of best roasting inception

	Series 0 (Base Case)	Series 1	Series 2	Series 3	Series 4
At Startup	**100%**	106%	**111%**	106%	95%
After 3 yrs	**--**	101%	110%	109%	--
After 5 yrs	**--**	99%	110%	99%	--
After 6 yrs	**--**	--	101%	--	--

Note: The notation “--“ stands for “not done”.

In Table 16.5, columns represent the COG policy applied and rows represent the roasting inception time during mine life: row “At Startup” indicates that roasting is applied from day 1; row “After 3 yrs” indicates that roasting starts in Year 4; row “After 5 yrs” indicates that roasting starts in Year 6, and row “After 6 yrs” indicates that roasting starts in Year 7. For clarity, as the focus of the analysis is on the relative economic value of options. The “Series 0”/”At Startup” pair was selected as the 100%.

The results show how the maximum value is associated with the Series 2 COG policy. All row maxima are located in this column. The highest value obtained, 111%, is given by the combination of roasting from the mill startup and applying the Series 2 COG policy over the first nine years of mining.

This is a high level analysis and a more detailed analysis of the mining rate, treatment rate and COG policy should be performed as part of the Pre-feasibility study.

The combination of options described above was accepted as a robust course of action for advancing the PEA.

16.3 Ultimate Pit Selection

Ultimate pit estimation and intermediate pit stages for the selected pits were obtained using the Lerschs & Grossman pit optimizing algorithm and the Milawa production scheduling algorithm. The Whittle Four-X package was used to accomplish the optimization and scheduling. The ultimate pit selection followed the following steps:

After a series of trial runs, including the strategic analysis, the ultimate pit was selected from a family of nested pits obtained using the parameters listed in Table 16.5.

- The NSR value corresponding to the all-roasting option drove the blocks valuation and the pit optimization.
- The ultimate pit was selected by considering the highest discounted operating surplus obtained from feasible mining and milling schedules. The schedules are based on the "Series 2" declining COG policy outlined in the preceding section. An 8% discount rate was used.
- The selected pit corresponds to the Revenue Factor of 90% (RF=0.9). Figure 16.5 below shows the region of discounted pit value curve where the selected pit lies.

Figure 16.5 Undiscounted and Discounted Pit Value Curves

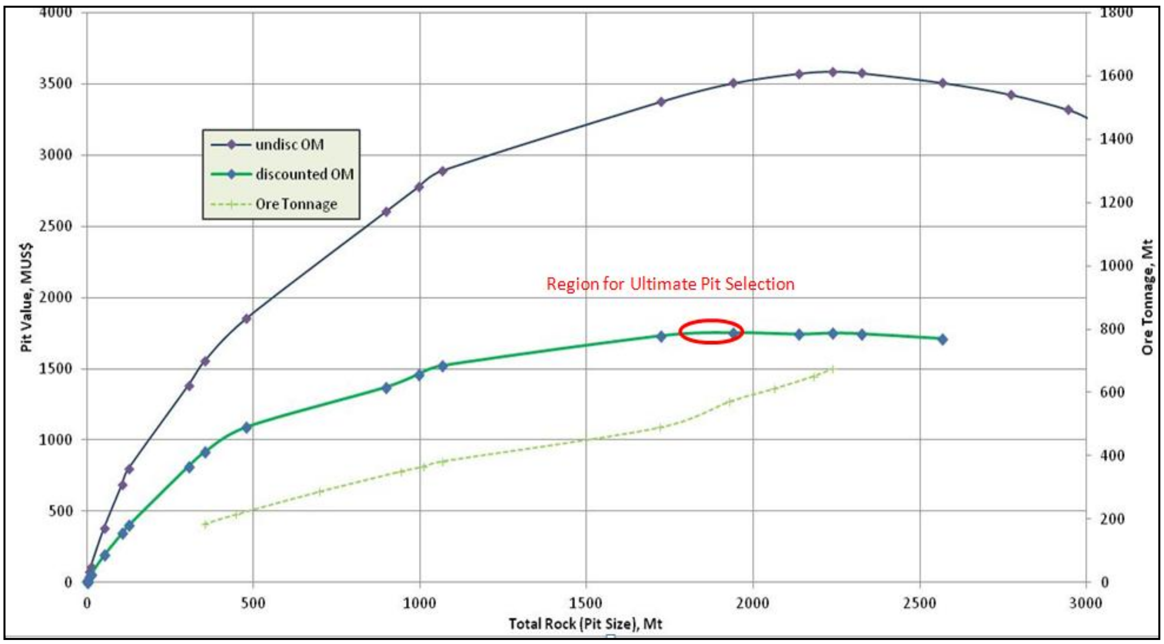


Table 16.5 Pit Optimization Input

Parameter	Value	Comment
Block Size	20 m x 20 m x 15 m	No re-blocking used
Overall Slope Angle	39 degrees	Limited geotechnical data was available. The overall slope angle used is based on experience in similar rock types and the expected total wall height.
Unit Cost of Mining	1.70 US$ per tonne mined	Benchmark data based on a mining rate of 200ktpd to 300ktpd
Unit Cost of Processing	6.00 US$ per Mill Feed tonne	Refer to Section 17
Unit G&A Costs	0.35 US$ per Mill Feed tonne	10 – 11 MUS$ per year was estimated for the expected operation size.
Eligible Rock Types for Processing	Four rock types as described in Table 16.1. These include Measured, Indicated and Inferred Resources	
Metal Prices	Included in the NSR calculation (see Table 16.2)	
Processing Recoveries		
Selling Prices		
Mine Dilution Factor	1	The porphyry nature of the mineralization means that , for the given block size, that dilution and mining losses are likely to be similar in magnitude and hence, maintaining these two factor at the unity value is appropriate for this level of study
Mine Recovery Factor	1	

The selected ultimate pit-shell features are summarized in Table 16.6.

Table 16.6 Features of the Selected Ultimate Pit

Item	**Value**	**Units**
Total rock	1,938	million tonnes
Waste	1,350	million tonnes
Total Mill Feed	588	million tonnes[1]
Overall strip ratio	2.3	
Mill Feed mean grades		
NSR	17.3	US$/tonne
Copper	0.37	%
Gold	0.03	g/tonne
Silver	2.3	g/tonne
Arsenic	0.033	%
Lowest bench elevation (toe)	-55	m

Figures 16.5 through 16.9 illustrate the ultimate pit selected and its relationship to the Mineral Resources. In some figures the ultimate pit trace has been highlighted to improve visualization.

[1] Please note this tonnage cannot be directly correlated to the Resource statement given in Table 14.11 as it is based on a NSR marginal cut off grade of 6.0 US$/tonne. By visual examination of the tonnage-grade graphs it is inferred that a comparable cut-off grade is in the vicinity of 0.15% Cu.

Figure 16.6 Selected Ultimate Pit and NSR Values – Typical Section through West and East Hill, Facing North

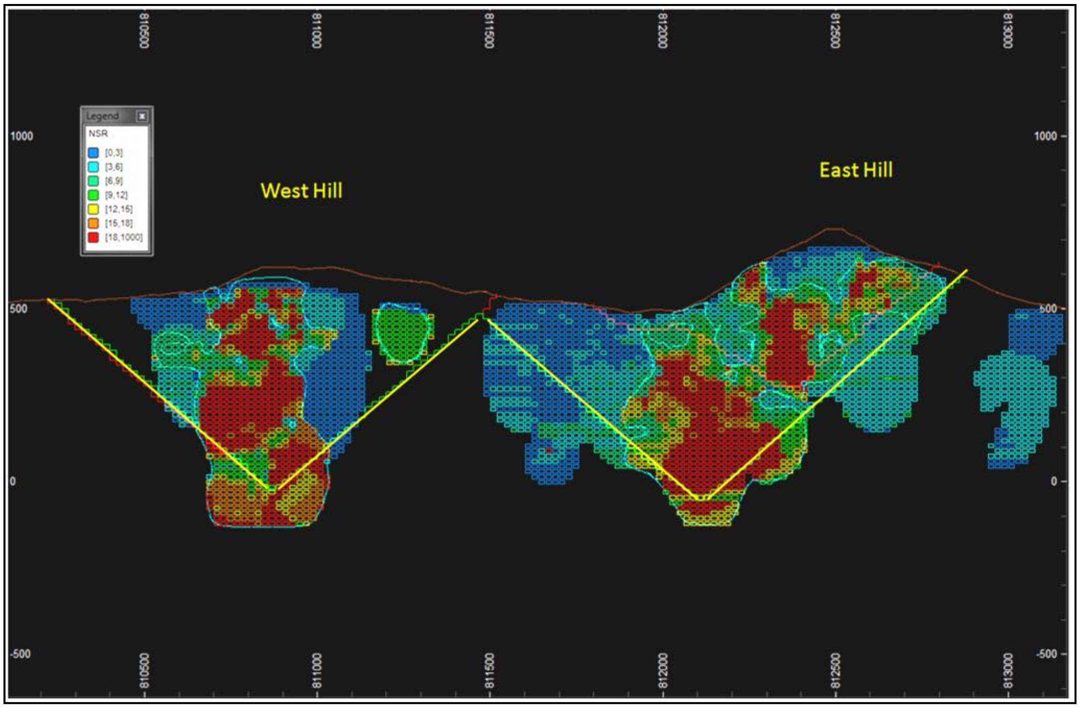


Figure 16.7 Selected Ultimate Pit and NSR Values – Plan View through West and East Hill at Elevation 390

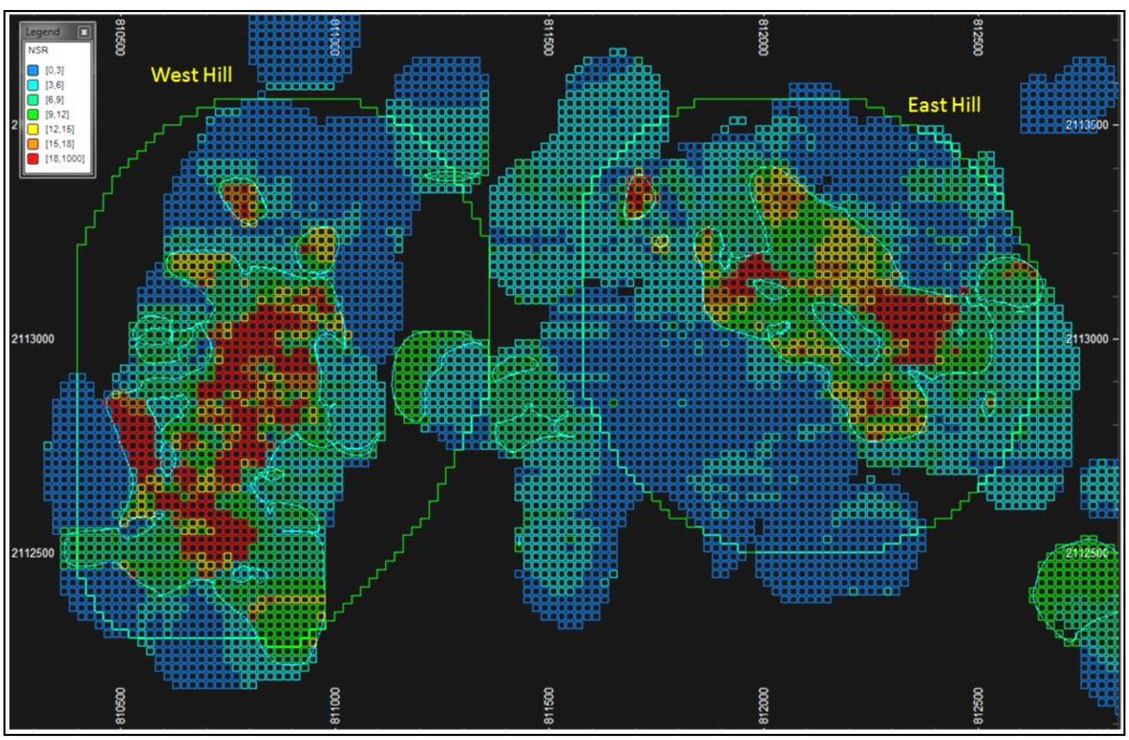


Figure 16.8 Selected Ultimate Pit and NSR Values – Typical Section through West Hill, Facing East

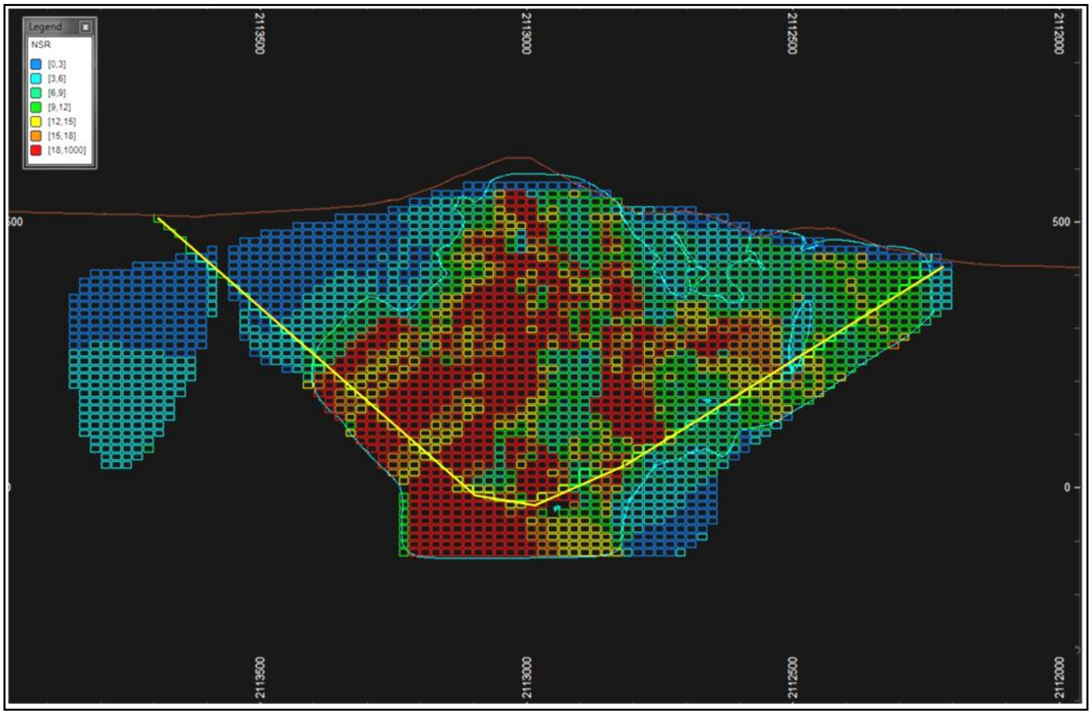
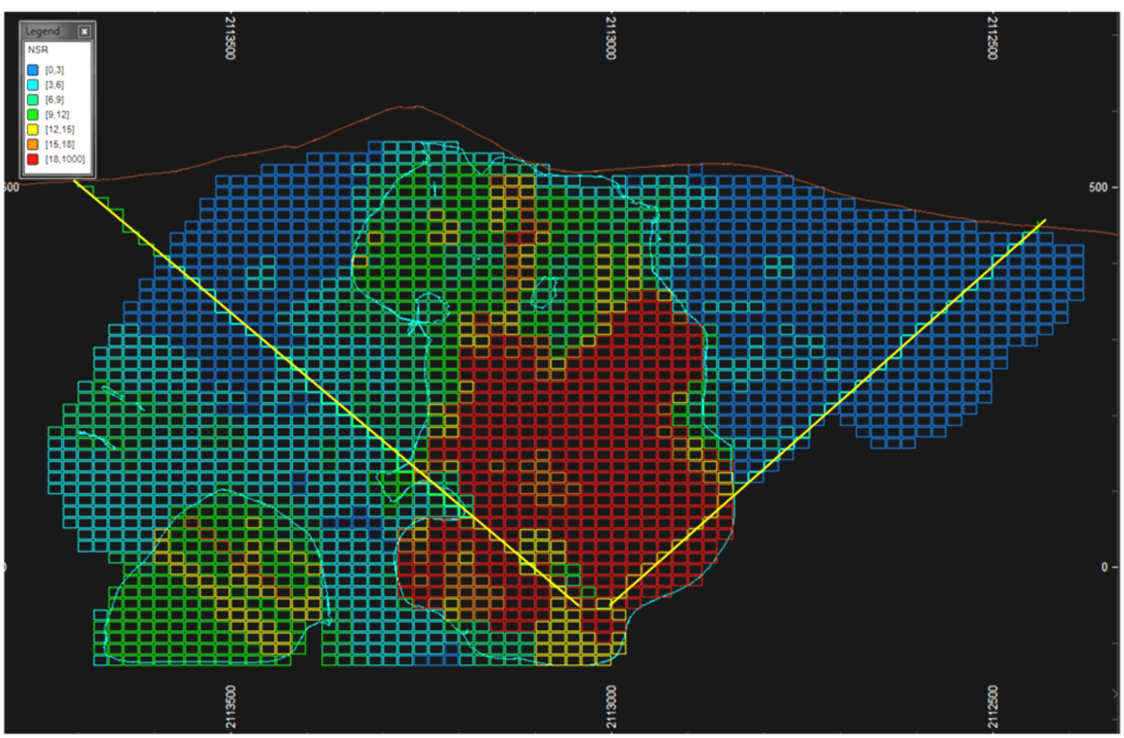


Figure 16.9 Selected Ultimate Pit and NSR Values. Typical Section through East Hill, Facing East



Figures 16.10 and 16.11 show the ultimate pit selected and its relationship to the fraction of the Mineral Resources with NSR greater than 6US$/t, represented by the red bodies. In Figure 16.11 a view "from underground" shows the fraction not included in the selected pit. An increase of the grade of these fractions (when more drilling is available), or higher metals price rise might include part of these areas in future pit optimization runs.

Figure 16.10 Selected Ultimate Pit and the Recovery of NSR Values – View Facing North

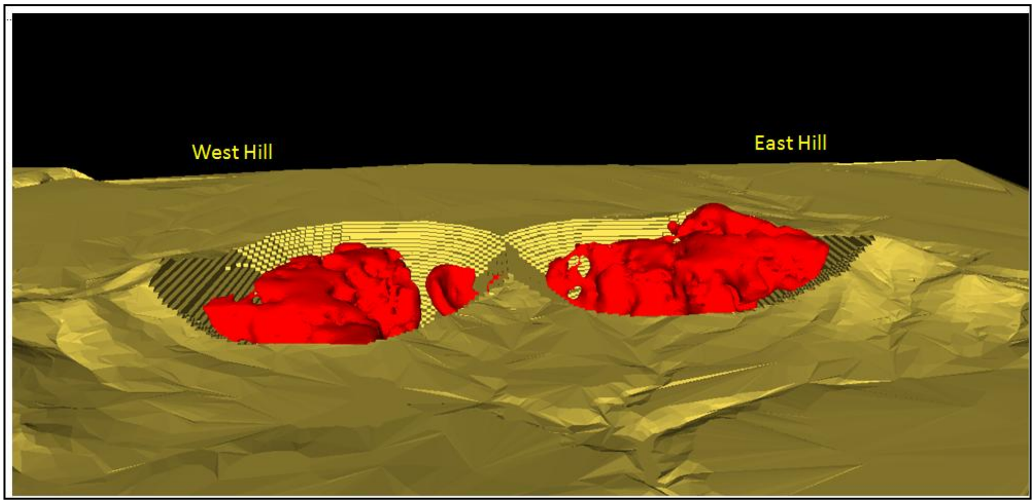
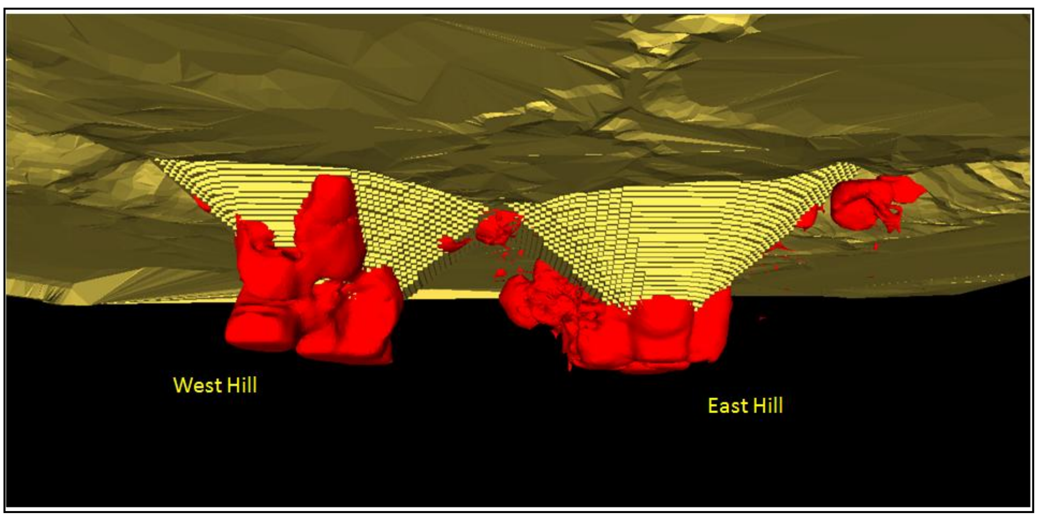


Figure 16.11 NSR Values not Mined by the Ultimate Pit – View from Beneath Facing North



16.4 Ultimate Pit Sensitivity

Ultimate pit sensitivity to some relevant parameters was tested. Results are shown in Table 16.7.

Table 16.7 Ultimate Pit Sensitivity to Selected Operational Parameters

Variation in Unit Mining Cost. Current Value: 1.7 US$/tonne					
		1.9 US$/tonne (+12%)		2.1 US$/tonne (+24%)	
Feature	**Current Value (a)**	**Sensitivity Value (b)**	**Variation, % (b/a)**	**Sensitivity Value (b)**	**Variation, % (b/a)**
Total Rock (pit size), Mtonne	1,938	1,723	89%	1,082	56%
Total to Processing, Mtonne	588	488	83%	388	66%
Mean NSR Grade, US$/tonne	17.3	17.3	100%	18.2	105%
Mean Copper Grade, %	0.37	0.37	100%	0.39	105%
Variation in Overall Slope Angle. Current value: 39 degrees					
		37 degrees (- 2 flatter)		42 degrees (+3 steeper)	
Feature	**Current Value (a)**	**Sensitivity Value (b)**	**Variation, % (b/a)**	**Sensitivity Value (b)**	**Variation, % (b/a)**
Total Rock (pit size), Mtonne	1,938	2,031	105%	2,004	103%
Total to Processing, Mtonne	588	565	96%	591	101%
Mean NSR Grade, US$/tonne	17.3	16.9	98%	17.7	102%
Mean Copper Grade, %	0.37	0.36	97%	0.38	103%

16.5 Geotechnical Data – Rock Mass Classification

16.5.1 Data

The rock mass characteristics at La Verde have been assessed using the limited data provided to AMC, which includes:

- Resource borehole collar locations and downhole survey
- Local topography
- Drill hole logging data (RQD data only)

The geotechnical data base used in this study is comprised of 51 diamond drill holes. All drill holes were logged by Catalyst personnel. The data base was reviewed using core photographs from four drill holes outlined in Table 16.5.

It should be noted that AMC did not have an opportunity to review the drill core on site from a geotechnical point of view.

Based on the limited information available, logging of geotechnical parameters (i.e. RQD) appears to be logged to industry standard.

Table 16.8 Drill Hole Core Photos Used to Review Catalyst Logging of RQD Values

Bore Hole ID	Easting (mE)	Northing (mN)	Elevation (mRL)	Depth Length (m)
LV12-041	810800.0	2113210.0	565.0	641.9
LV12-042	812204.0	2112788.0	534.0	565.1
LV12-043	811000.0	2113195.0	540.0	389.0
LV12-044	811852.0	2112925.0	450.0	682.3

16.5.2 Rock Quality Designation (RQD)

RQD is a fundamental input into several rock mass characterization schemes, and is generally regarded as a reliable, basic indicator of ground conditions. However, it should be noted that RQD is not a substitute for a more representative rock mass classification scheme such as RMR_{1989} (Bieniawski, 1989) or NGI Q (Barton et. al., 1974).

The RQD is calculated as the ratio of the sum of the lengths of all core sticks greater than 10 cm in length, to the total length of the drill core run, expressed as a percentage. Values and descriptions of RQD are presented in Table 16.6.

Table 16.9 RQD Values and Descriptions (Deere, 1964)

Rock Quality Designation (Description)	RQD Value (%)
Very Poor	0 to 25
Poor	25 to 50
Fair	50 to 75
Good	75 to 90
Excellent	90 to 100

A preliminary review of geotechnical logging suggests that the uppermost section of the drill holes typically display low RQD recorded values (after Deere, 1964). This zone is thought to be associated with a depth of weathering, which is estimated to be around 25 m.

Figure 16.12 indicates that the weathered zone in the West Hill deposit (WWP Domain) is characterized by RQD values of 0 to 10%, while the unweathered rock (UWP Domain) below features values of 70% to 90%.

It can be seen from Figure 16.13 that the weathered zone in the East Hill deposit (WEP Domain) typically features RQD values of 0 to 10% and 60 to 70%. The unweathered drill core (UEP Domain) is characterized by RQD values of 70 to 90%.

Figure 16.12 RQD Values by Geotechnical Domain – Logged From Drill Holes Located in the proposed West Hill Pit

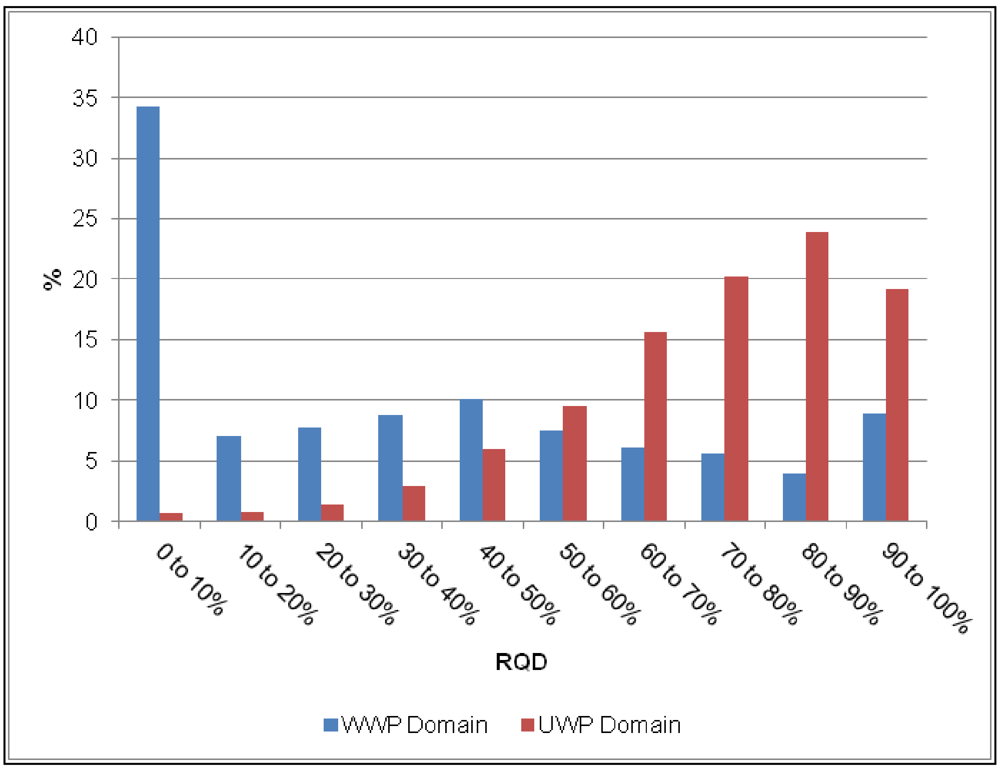
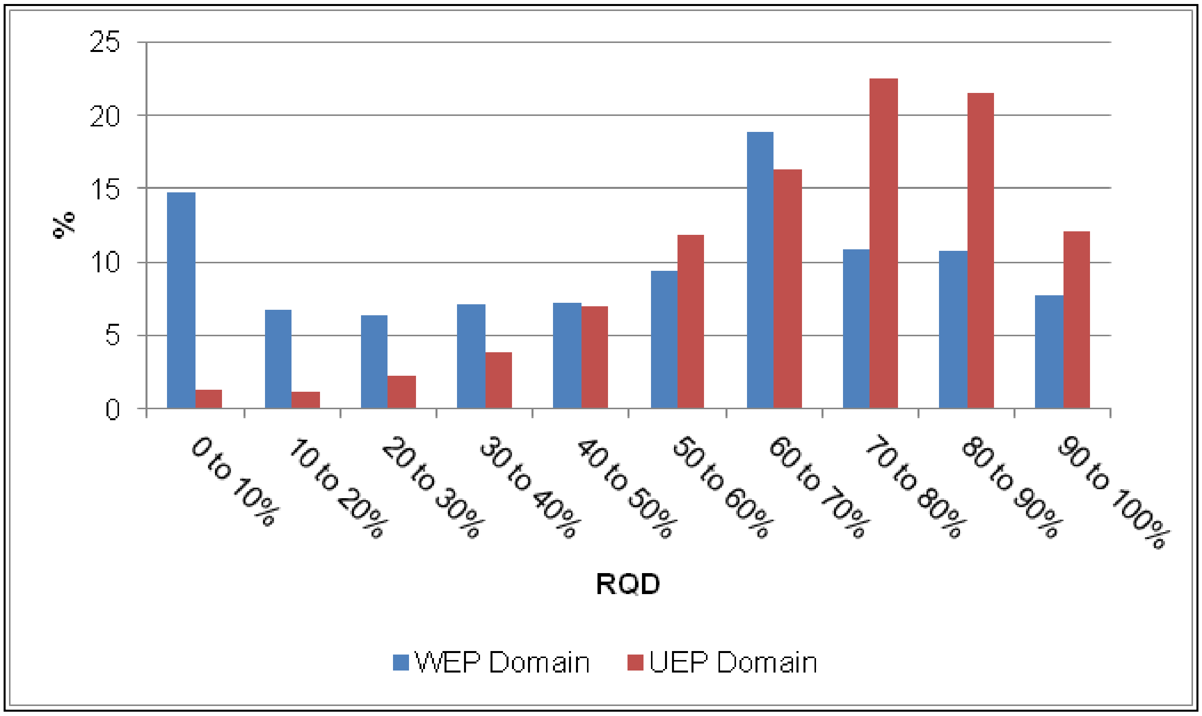


Figure 16.13 RQD Values by Geotechnical Domain; Logged From Drill Holes Located in Proposed East Pit



16.5.3 Assumed Design Parameters

An Overall Slope Angle (OSA) of 39° has been assumed for this study, which is reasonable for a PEA level of analysis.

Geotechnical information that is typically required to undertake reliable slope stability analysis includes, but is not limited to:

- Lithology types and locations
- Estimated intact rock strengths
- The orientation and condition of geological structures (such as joints, shears, and deposit-scale faults)
- Rock property data derived from laboratory testing
- Hydrological information (such as location and extent of ground water, along with the associated pore water pressures)

None of the information outlined above has been supplied to AMC for this study.

While RQD values have been provided, it should be noted that this form of assessment does not take into consideration the effects of hydrological conditions, nor deposit-scale geological structures, which may adversely affect slope stability. Furthermore, the RQD estimates are sourced from drill holes that are positioned within the deposit, and not from locations where pit walls are likely to be located.

16.6 Layout of Other Mining Facilities around the Pits

As shown in Table 16.3, 1,350 million tonnes of waste need to extracted from the pits and stored on surface. Additionally, the extraction schedule calls for the production of up to 110 million tonnes of stockpile materials.

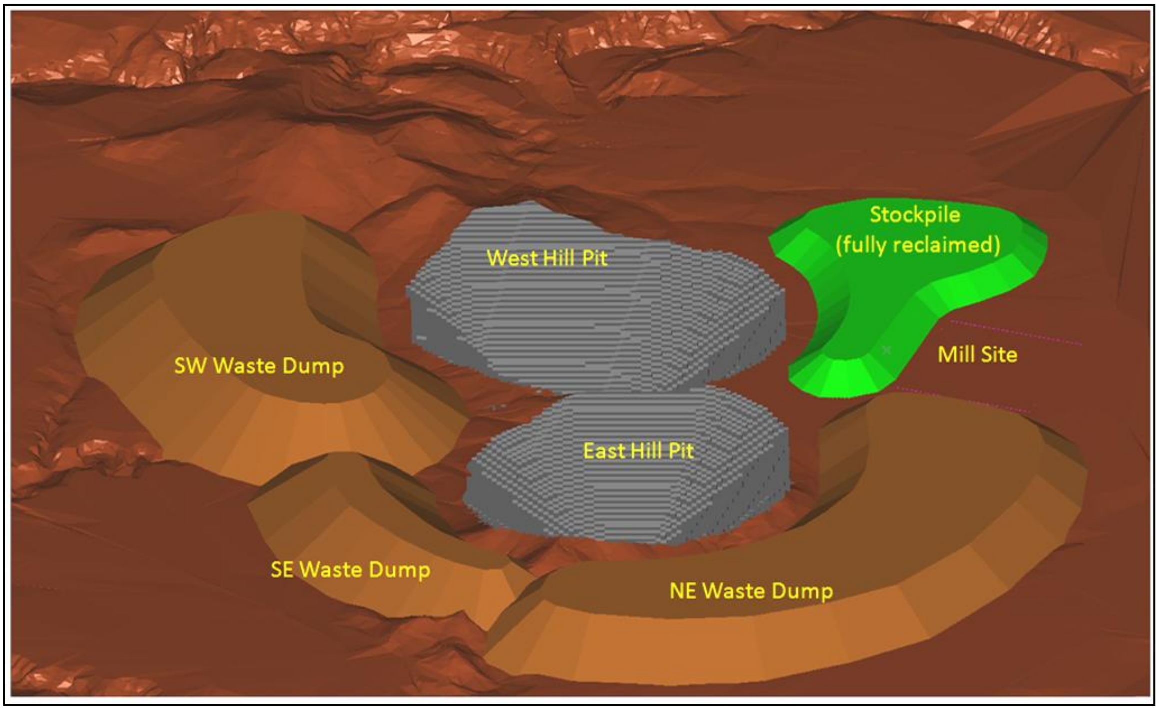
Currently, Catalyst does not own the surface land required to hold the pit and the surrounding mining facilities outlined in this section. Notwithstanding, under the assumption that any land required for surface dumps and storage can be acquired, sufficient space for the scheduled dumps and stockpiles is available in the immediate surroundings of the pit.

After defining the location of the mill at the north side of the mine, most of the areas surrounding the pit have been used for surface dumps and stockpiles. The basic criteria for selecting the locations have been as follows:

- Minimize haulage. Dump as close as possible to the pit rims and at elevations as low as possible.
- Offset from communities. A minimum 500 m between current community perimeters and the closest dump / stockpile toe has been used. Viability of this offset has not been explored with the local communities.
- Offset from pit. Toes of waste dumps and stockpiles no closer than 200 m to pit rims (allowance for potential pit expansion).
- All waste dumps and stockpiles were designed at an average slope angle of 28 degrees. This angle includes a 37 degree angle of repose and allowance for haul roads (not shown at this stage).

The resulting layout is shown in Figure 16.14. The stockpile would be fully reclaimed by the end of the mine life.

Figure 16.14 General Layout including Mill Site, Stockpile and Waste Dumps Facing West



16.7 Mining Schedule

The open pits were scheduled using the Milawa algorithm which is part of the Whittle Four-X package. Highlights of the selected mining schedule are:

- The selected schedule calls for 21 years of mining activity, including Year -1 devoted to mining 65 million tonnes of pre-stripping, and Years 20 and 21 only reclaiming stockpile material to mill.
- Metal prices and mill recoveries are the same as those used for NSR calculation and pit optimization.

Figures 16.15 to 16.17 show some selected end of period mine status to illustrate the mining development. Blocks legend is based in NSR values, the warmer colours corresponding to the higher NSR values. Year -1 is the first mining year. Milling starts in Year 1.

Figure 16.15 Mining Schedule, End of Year 1 (end of pre-stripping) – Facing North View

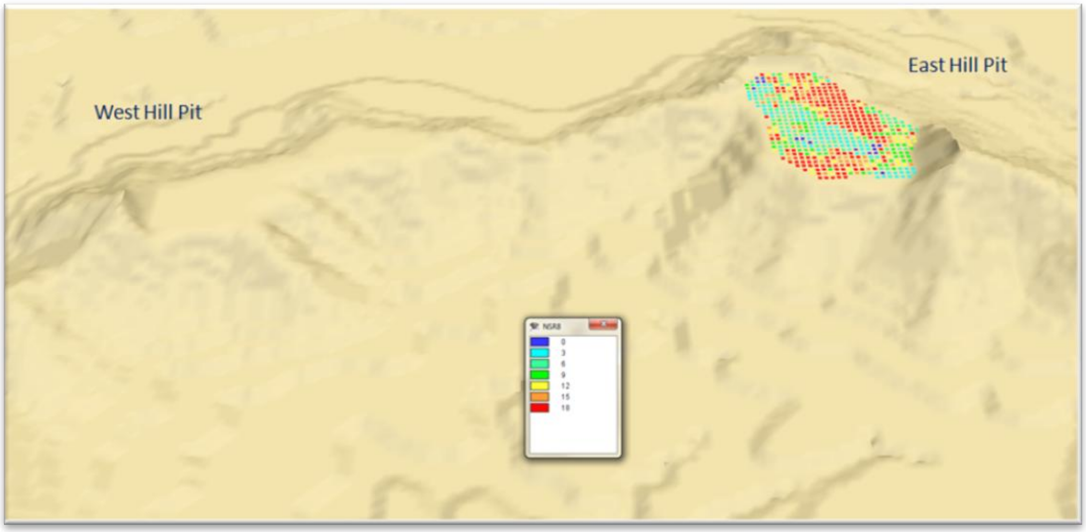


Figure 16.16 Mining Schedule, End of Year 5 – Facing North View

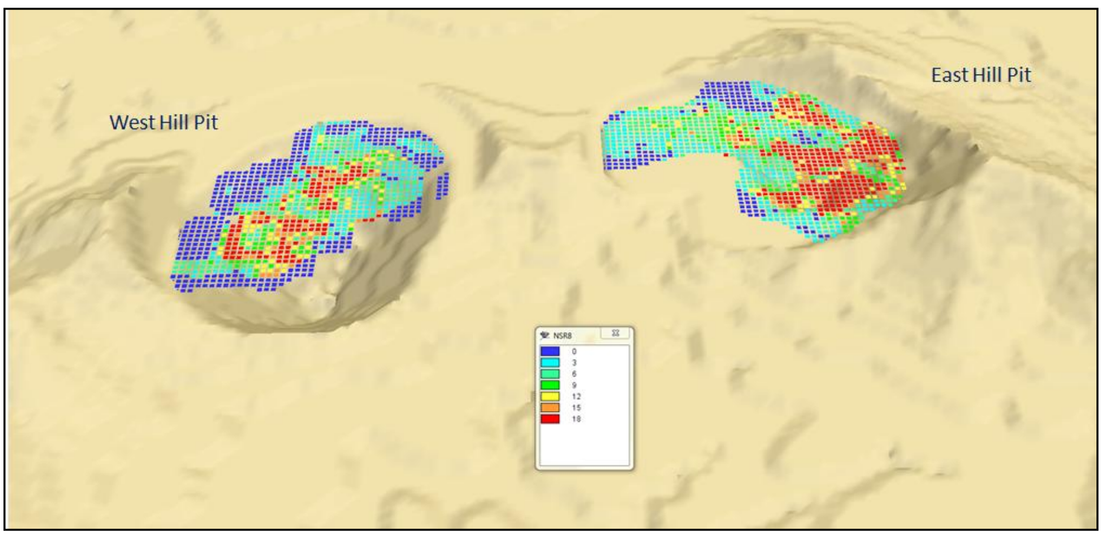


Figure 16.17 Mining Schedule, End of Year 10 – Facing North View

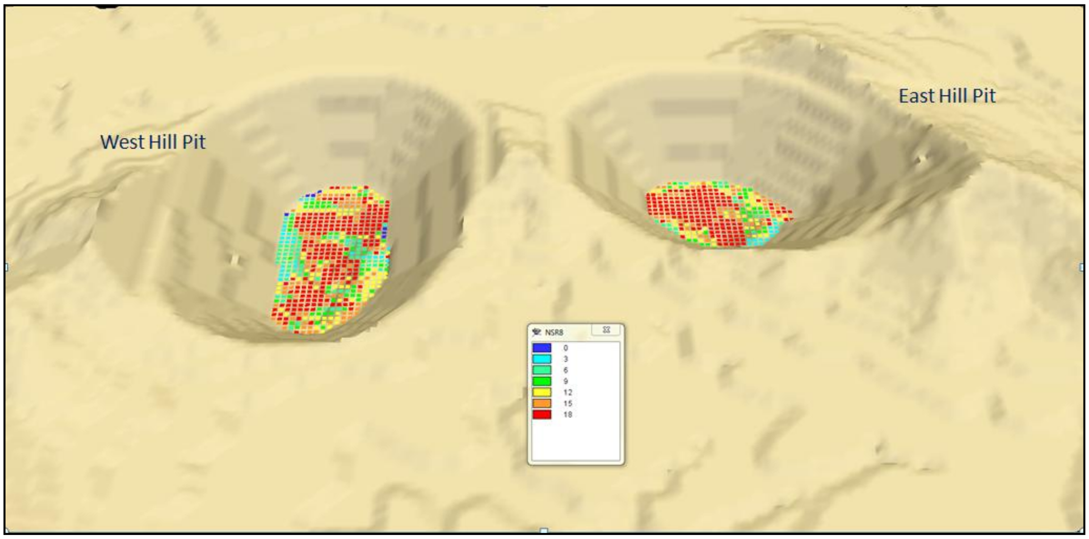
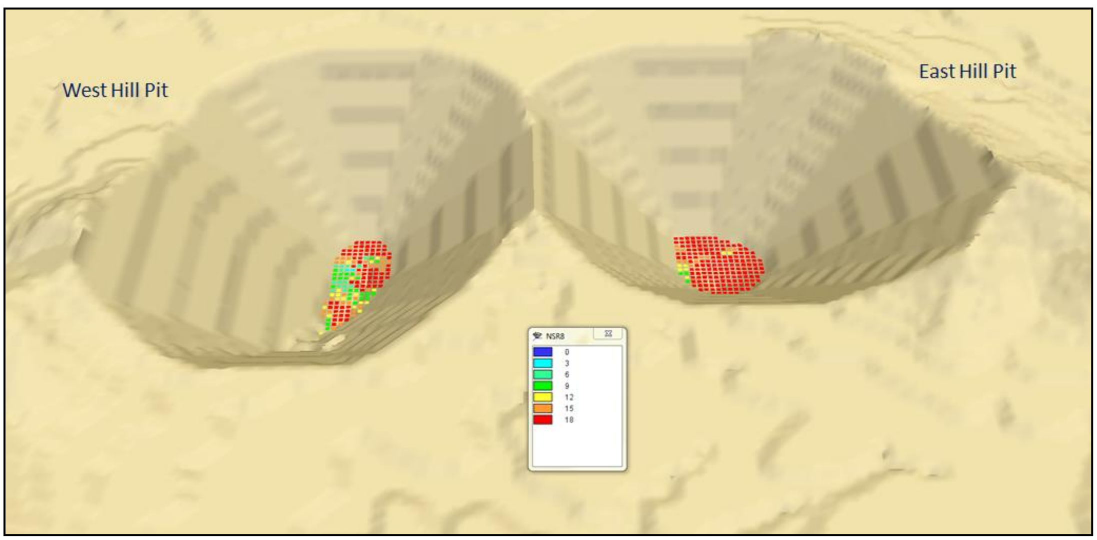


Figure 16.18 Mining Schedule, End of Pit Life – Facing North View



The detail of the schedule is presented in Table 16.7. Note that only a portion of the mill feed goes directly to the mill with the remaining material extracted going to stockpile for later treatment. This is a result of the higher cut-off grades applied in early years in order to increase the NPV. The detailed mill feed composition is given in Table 16.8.

Table 16.10 Pit Extraction Schedule. Detail by Mine Phases

	YEAR	Y(-1)	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10	Y 11	Y 12	Y 13	Y 14	Y 15	Y 16	Y 17	Y 18	Y 19	Y 20	LOM
Phase 1	to Mill, Mt	-	23.8	29.9	28.1	13.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-			
	to Stocks, Mt	16.5	17.5	22.5	20.9	2.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-			
	Waste, Mt	48.5	53.7	37.0	14.3	1.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-			
Phase 2	to Mill, Mt	-	-	0.1	1.9	13.7	21.4	-	-	-	-	-	-	-	-	-	-	-	-	-			
	to Stocks, Mt	-	-	0.3	4.0	14.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-			
	Waste, Mt	-	-	5.3	25.8	48.9	13.1	-	-	-	-	-	-	-	-	-	-	-	-	-			
Phase 3	to Mill, Mt	-	-	-	-	-	1.8	30.0	22.9	-	-	-	-	-	-	-	-	-	-	-			
	to Stocks, Mt	-	-	-	-	-	10.3	5.7	-	-	-	-	-	-	-	-	-	-	-	-			
	Waste, Mt	-	-	-	-	-	53.9	84.3	8.3	-	-	-	-	-	-	-	-	-	-	-			
Phase 4	to Mill, Mt	-	-	-	-	-	0.2	-	7.0	27.8	12.9	-	-	-	-	-	-	-	-	-			
	to Stocks, Mt	-	-	-	-	-	0.7	-	-	-	-	-	-	-	-	-	-	-	-	-			
	Waste, Mt	-	-	-	-	-	18.7	-	81.8	49.2	1.4	-	-	-	-	-	-	-	-	-			
Phase 5	to Mill, Mt	-	-	-	-	-	-	-	-	1.7	17.1	19.0	-	-	-	-	-	-	-	-			
	to Stocks, Mt	-	-	-	-	-	-	-	-	0.2	-	-	-	-	-	-	-	-	-	-			
	Waste, Mt	-	-	-	-	-	-	-	-	27.2	88.6	3.6	-	-	-	-	-	-	-	-			
Phase 6	to Mill, Mt	-	-	-	-	-	-	-	-	0.4	-	5.1	26.9	10.6	-	-	-	-	-	-			
	to Stocks, Mt	-	-	-	-	-	-	-	-	0.2	-	-	-	-	-	-	-	-	-	-			
	Waste, Mt	-	-	-	-	-	-	-	-	13.5	-	92.3	36.7	0.5	-	-	-	-	-	-			
Phase 7	to Mill, Mt	-	-	-	-	-	-	-	-	-	-	-	3.1	7.3	11.1	23.3	20.0	21.8	14.8	0.7			
	to Stocks, Mt	-	-	-	-	-	-	-	-	-	-	-	0.3	0.0	-	-	-	-	-	-			
	Waste, Mt	-	-	-	-	-	-	-	-	-	-	-	48.1	96.5	99.6	91.7	18.5	6.5	15.9	0.0			
Phase 8	to Mill, Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3.9	5.4	15.2	29.3			
	to Stocks, Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-			
	Waste, Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	4.3	-	72.5	46.3	34.1	8.1			
Totals	to Mill, Mt	-	23.8	30.0	30.0	27.6	23.3	30.0	29.9	29.8	30.0	24.1	30.0	18.0	11.1	23.3	23.9	27.2	30.0	30.0			472
	to Stocks, Mt	16.5	17.5	22.7	24.9	16.7	11.0	5.7	-	0.4	-	-	0.3	0.0	-	-	-	-	-	-			116
	Waste, Mt	48.5	53.7	42.3	40.1	50.7	85.7	84.3	90.1	89.8	90.0	95.9	84.8	97.0	103.9	91.7	91.1	52.8	50.0	8.1			1,350
	Total Ex-Pit, Mt	**65.0**	**95.0**	**95.0**	**95.0**	**95.0**	**120.0**	**120.0**	**120.0**	**120.0**	**120.0**	**120.0**	**115.0**	**115.0**	**115.0**	**115.0**	**115.0**	**80.0**	**80.0**	**38.1**			**1,938**
	Avg. Ex-Pit Rate, ktpd	181	264	264	264	264	333	333	333	333	333	333	319	319	319	319	319	222	222	106			283
Ex-Pit Strip Ratio	By Year	na	3.0	2.2	2.2	2.4	4.1	3.0	3.0	3.0	3.0	4.0	2.8	5.4	9.4	3.9	3.8	1.9	1.7	0.3			
	Cumulative	na	5.7	3.7	3.2	3.0	3.2	3.2	3.1	3.1	3.1	3.2	3.2	3.3	3.5	3.5	3.5	3.4	3.3	3.1			3.1
	Stock to Mill, Mt		6.2	0.0	0.0	2.4	6.6	0.0	0.0	0.0	0.0	5.9	0.0	12.3	18.9	6.9	6.2	3.1	0.0	0.0	29.7	17.5	116
	Total Mill Feed, Mt	**0.0**	**30.0**	**30.0**	**30.0**	**30.0**	**29.9**	**30.0**	**29.9**	**29.8**	**30.0**	**30.0**	**30.0**	**30.3**	**30.0**	**30.2**	**30.1**	**30.3**	**30.0**	**30.0**	**29.7**	**17.5**	**588**
	Overall Strip Ratio - Cumulative	na	4.3	3.3	2.9	2.7	2.8	2.8	2.8	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.8	2.7	2.6	2.4	2.3	2.3
	Stock Size EOY, Mt	16.5	27.8	50.5	75.5	89.8	94.2	99.9	99.9	100.3	100.3	94.4	94.6	82.4	63.4	56.6	50.4	47.3	47.3	47.3	17.5	0.0	

Table 16.11 Mill Feed Schedule

YEAR	Y(-1)	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10	Y 11	Y 12	Y 13	Y 14	Y 15	Y 16	Y 17	Y 18	Y 19	Y 20	**LOM**
Total Mill Feed, Mt	-	30.0	30.0	30.0	30.0	29.9	30.0	29.9	29.8	30.0	30.0	30.0	30.0	30.0	30.1	30.1	30.2	30.0	30.0	30.0	17.7	**588**
Cut Off Grade - NSR, US$/t		14.0	13.0	13.0	13.0	12.0	12.0	11.0	11.0	10.0	10.0	9.0	9.0	7.0	6.0	6.0	6.0	6.0	6.0	6.0	6.0	
Mean Grades																						
NSR, US$/t		25.0	25.3	25.1	22.9	19.0	16.5	22.4	18.1	19.7	17.3	16.5	12.7	9.2	10.6	12.3	16.2	16.5	19.3	8.6	8.2	**17.3**
Cu, %		0.53	0.54	0.56	0.51	0.43	0.33	0.43	0.40	0.43	0.36	0.35	0.27	0.19	0.22	0.26	0.33	0.35	0.41	0.18	0.17	**0.37**
Au, g/t		0.09	0.03	0.02	0.02	0.02	0.04	0.07	0.02	0.03	0.04	0.03	0.03	0.02	0.03	0.02	0.05	0.04	0.03	0.01	0.03	**0.03**
Ag, g/t		4.0	3.4	2.4	2.4	2.1	2.4	4.2	2.1	2.1	2.2	2.0	1.7	1.4	1.6	1.8	2.3	2.1	2.4	1.5	1.3	**2.3**
As, %		0.020	0.036	0.041	0.032	0.031	0.022	0.076	0.034	0.029	0.029	0.030	0.029	0.028	0.023	0.030	0.038	0.039	0.035	0.039	0.009	**0.033**

Note that the mill feed schedule is provided in accordance with the requirements of 22 (b) of Form 43-101F1. It does not represent an estimate of Mineral Reserves

16.8 Mine Equipment and Personnel

A full time mine and mill operation is proposed for La Verde, i.e. a continuous operation of 360 days per year and 24 hours per day. No source of non-usable time other than significant holidays is anticipated.

The selected schedule calls for ex-pit mining rates of 181 Ktonne / day during Year 1, 264 Ktonne / day from Year 2 to 4, and an average of 296 Ktonne / day from Year 5 to 18.

For this size of mining operation and the selectivity required in this type of deposit, large size mining equipment is proposed. Both productivities and costs have been estimated on the following operational scheme:

- Drilling and Blasting on 15 m benches
- Primary loading with rope and hydraulic shovels
- Use of 225 tonnes haul trucks
- Use of electric equipment (primary drills and rope shovels)

In Table 16.12, the mine operation units and associated equipment are described.

Table 16.12 Mine Unit Operations Descriptors

Unit Operation	Equipment	Normal Use	Key Indicators
Drilling	Electric rotary drills. 90 klb pulldown class	Vertical drilling. 10 5/8” – 12 ¼” hole diam. range	29 drilled meters per hour (LOM avg)
Blasting	Factory trucks. Owner operated	Patterns for ore and waste. Avg. Yield, 2500 tonne per hole	LOM powder factor, 280 g/t
Loading	Four shovels in the 37 - 42 cu. meter bucket range. Two electric rope shovels. Two hydraulic (diesel). Two complementary loaders	LOM average, 75 ktonne per day per unit	5500 tph avg. capacity with truck (not idle included)
Haulage	225-tonne-class payload trucks	Shovel and FEL loading	830 tph avg LOM productivity
Support Equipment	Conventional equipment: water trucks, wheel dozers, bulldozers and motor graders		84,000 operating hours per year (LOM avg)

The number of equipment units required per year and the purchasing schedule (new and replacement units) are provided in Table 16.13.

Table 16.13 Mine Mobile Equipment. Active Units Per Year and Acquisition Schedule

		Mine Operations and Maintenance Mobile Equipment - UNITS in USE																						
	Unit	Y(-3)	Y(-2)	Y(-1)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20
Primary Drill	u			3	5	5	5	5	6	6	7	7	7	7	7	7	6	6	6	5	5	3	3	0
Secondary Drill	u			1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Explosive Factory Truck	u			2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	3	3	0	0
Rope Shovel	u				1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	0
Hydraulic Shovel	u			2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Front-End Loader	u			1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1
Haultruck	u			9	11	11	11	11	15	15	16	18	18	22	22	24	24	24	24	24	24	15	5	3
Wheel Dozer	u			3	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	4	4	3	2	2
Bulldozer	u			3	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	4	4	4	2	2
Water Truck	u			2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	3	2	2	2	2
Grader	u			2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Cable Handler	u			1	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3	2	2	2	1	1
Crane	u			2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3
Tool Truck (Maintenance)	u			8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	9
		Aquisition Schedule - Includes additions and replacements - NUMBER of UNITS																						
	Unit	Y(-3)	Y(-2)	Y(-1)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20
Primary Drill	u		2	1	2				1		1				2									
Secondary Drill	u			1						1							1							
Explosive Factory Truck	u		2	2							2	2						2	2					
Rope Shovel	u			1				1																
Hydraulicl Shovel	u		1	1											1									
Front-End Loader	u		1						1					1										
Haultruck	u		4	5	2				4	1	2		4	9	4				4					
Wheel Dozer	u		2	2	1							2	2	1										
Bulldozer	u		2	2	1							2	2	1										
Water Truck	u		1	1	2						2	2							2	2				
Grader	u			1	1									1	1									
Cable Handler	u			1	1			1								1	1							
Crane	u			1	1									1	1									
Tool Truck (Maintenance)	u			4	4						7							7						

Mine personnel includes staff and shift workers from Mine Operations, Mine Maintenance, Mine Engineering and Production Geology. For shift work, four crews have been assumed with number one working on day shift, number two working on night shift, and numbers three and four on off day.

Personnel are predominantly composed of Mexican nationals; expatriates have been allocated to some key positions only. In Table 16.14, a summary of mine personnel is given.

Table 16.14 Number of Employees and Labor Expenses

	Count	Yearly Avg per Person '000 US$
Expats	3	300
Professionals & Supervisors	89	78
Tradesmen	35	25
Shift Workers	377	16
Total	504	n.a

17 RECOVERY METHODS

17.1 Key Metallurgical Parameters

The key metallurgical parameters from Section 13 are shown again in Table 17.1 for ease of reference.

These have been used in conjunction with the mine schedule to develop the production schedule shown in Table 17.3 and to input into the key throughput parameters for equipment sizing.

Table 17.1 Metallurgical Recovery and Concentrate Grade parameters

Metallurgical Parameters												
	Cu			Au			Ag			As		
	W Su	E Su	E ox	W Su	E Su	E ox	W Su	E Su	E ox	W Su	E Su	E ox
Recovery	90%	90%	50%	10%	80%	75%	60%	85%	50%	20%	70%	70%
Conc. grade	26%	27.5%	29%									

Summary design criteria are shown in Table 17.2.

These are based on some generic industry standards, e.g. for availabilities, the key testwork outcomes described in Section 13, and some factors based on AMC's experience of similar projects. As mentioned in Section 13, note that flotation kinetics were not specifically reported in the metallurgical testwork studies but the residence time assumptions cited in the Table 17.2 are in accord with the range recorded in the raw test data and, in AMC's opinion, in line with expected conditions for such a copper flotation circuit.

These parameters were used along with the recoveries and others from Table 17.1 to develop a high-level mass balance. They also provided inputs for the power and water consumption calculations presented in Section 17.5.

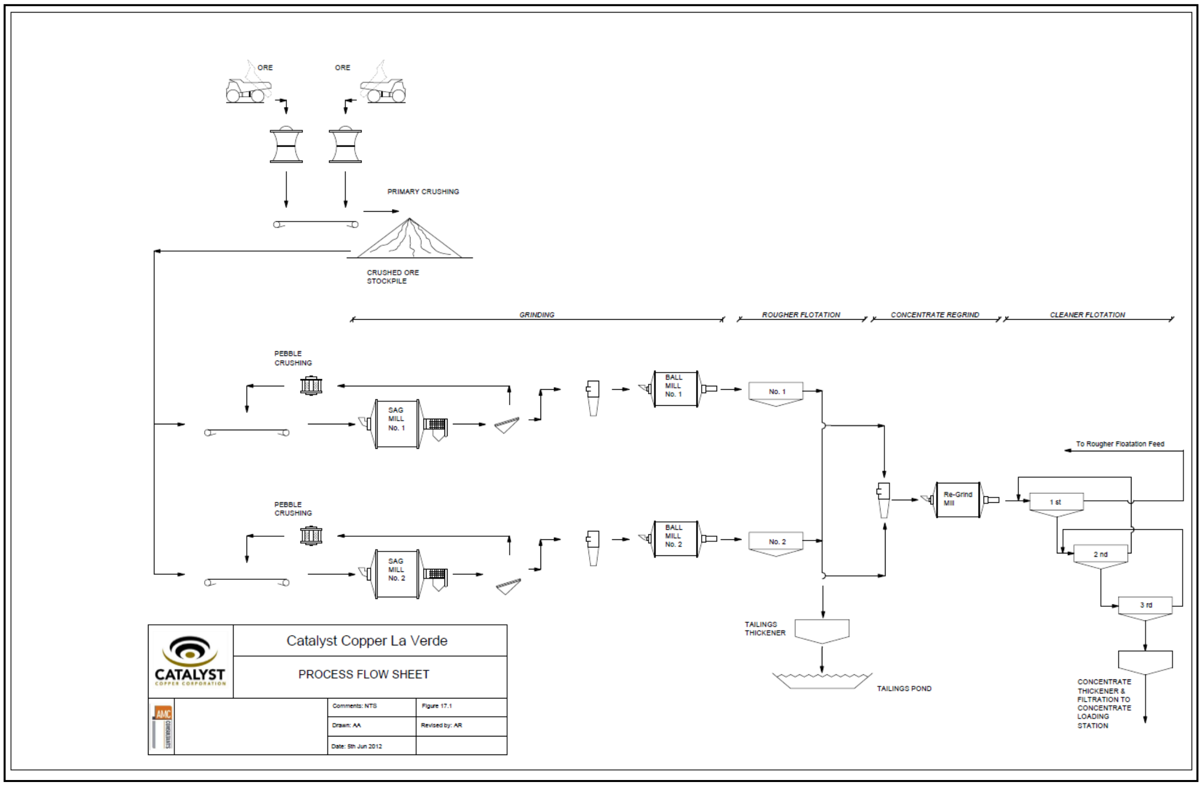
Table 17.2 Overall Design Criteria

Parameter	Value
General:	
Operating days p.a.	360
Availabilities:	
crusher	70%
grinding / flotation	95%
Grinding:	
Abrasion Index	0.134
SAG parameters:-	
A*b (JK)	35.7
Mia (SMC)	21.5
Feed size F80 mm	125
Product Size P80 µ	1000
Ball mill parameters:-	
BWI	16.2
Mib (SMC)	22.9
Feed size F80 µ	1000.0
Product Size P80 µ	220.0
Regrind parameters:-	
Feed size F80 µ	100.0
Product Size P80 µ	40.0
Flotation:	
% solids:-	
roughers	33
cleaners	15
residence times (lab) mins	
roughers	10
cleaners	5
scale up factor	3 x
Tailings Thickener:	
% solids in underflow	60

17.2 Flowsheet

The proposed flowsheet is shown in schematic form in Figure 17.1. It is a conventional copper concentrator flowsheet comprising crushing, grinding (SAG, Ball mill pebble crusher – SABC), and flotation with concentrate regrind and three stages of cleaning. There will be two trains of grinding and two rougher flotation banks followed by a common regrind and cleaner circuit.

Figure 17.1 Process Flowsheet



For reasons of promoting maximum settled density in the tailings storage facility, which maximizes the storage efficiency in the face of significant challenges in identifying suitable sites and adequate storage volume, a tailings thickener has been included in the flowsheet. This will also minimize net water usage by reducing evaporation losses and the amount of water lock-up in the tailings solids.

In addition, following the concentrate treatment testwork reported in Section 13, partial roasting of the concentrates for arsenic removal has been included in the flowsheet. Its general principles are described below.

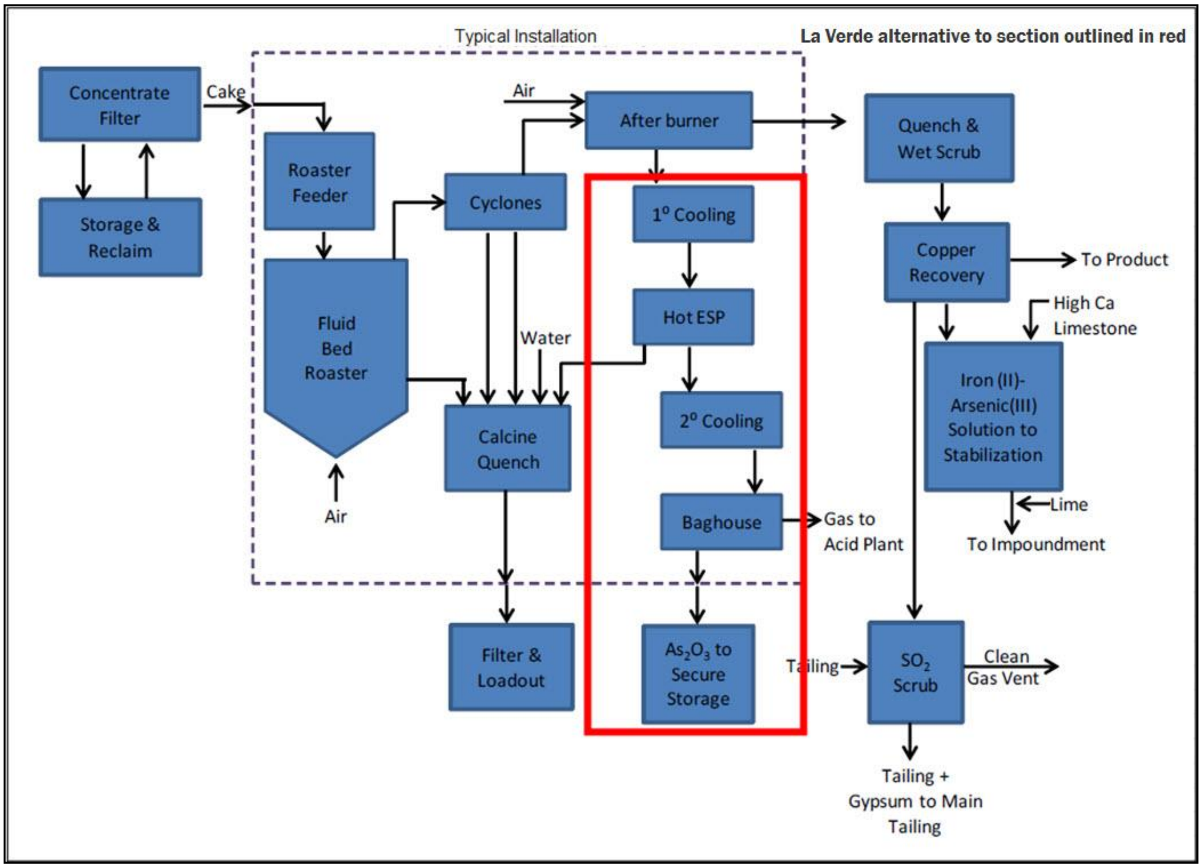
Partial roasting in fluidized bed roaster is well-established technology. The roasting principle in general prepares concentrates for subsequent pyrometallurgical or hydrometallurgical operations by decreasing sulphur levels, removing arsenic and antimony or rendering the concentrate leachable. Partial roasting in particular is accomplished by controlling the access of the air to the concentrate; a predetermined amount of sulphur is removed and only part of the iron sulphide is oxidized although most of the volatile elements such as arsenic and antimony are removed, leaving the copper sulphide otherwise relatively unchanged. There are operating examples in China, Philippines and Europe and an advanced project underway with one of the major South American copper producers.

“Fixing” the arsenic as a stable waste product is also a vital part of the process and this will be achieved at La Verde by wet scrubbing of the roaster exhaust gases to capture the arsenic (as water soluble arsenic trioxide) and the oxidized particulate (mainly iron oxide). The iron oxide will dissolve in the acid scrub liquor to provide the necessary iron (minimum 4:1 Fe:As stoichiometric ratio), along with limestone to neutralize the liquor and air oxidation, to stabilize the arsenic as hydrated ferric arsenate ($FeAsO4.2H2O$; artificial scorodite). This is environmentally stable over a much wider range of pH and oxidation potential than a calcium arsenate produced by simple neutralization with lime. Modern flue gas treatment technology will also capture $SO2$ and meet any environmental concerns regarding roasters that may have applied in the past.

The suggested partial roasting flowsheet for La Verde is shown in Figure 17.2. Typically a partial roaster would utilize the installation shown in the dashed-line box in order to produce arsenic trioxide for storage. However the electrostatic precipitator and baghouse are both capital intensive and high maintenance unit operations and for La Verde iron stabilisation of arsenic as described above is the preferred option and the red box would be replaced by the alternative wet scrubbing flowsheet shown.

For the purposes of this PEA, it is assumed that on-site disposal of the stable ferric arsenate is viable although sensitivity to a potential off-site registered land-fill disposal scenario has been considered, based on another project in Mexico known to AMC where such a scenario has been costed.

Figure 17.2 Partial Roasting Flowsheet



17.3 Equipment Sizing and Description

Equipment sizing has been carried out from a combination of first principles calculations for critical equipment such as grinding mills and flotation cells that are mineralization-specific, and benchmark comparables for other, less critical more generic equipment. Details by section are provided below:

Crushing equipment is based on the AMC database for similar size base metal projects. Two gyratory crushers of 60” x 89” are proposed.

Grinding is based on the SMC testwork results tabulated in Section 13 and the SMC methodology for determining mill power requirements. A 20% allowance has been added for losses and design margin.

From this AMC has determined that two 22 MW SAG mills (12.2 m x 6.7 m with wraparound gearless motors) and two 15 MW ball mills are required. A single regrind mill of 2.15 MW capacity is also indicated.

The ball mills are in closed circuit with cyclones.

Flotation circuit design and sizing is based on the maximum annual feed grade in the production schedule (Table 17.3) of 0.59% Cu, and with a 20% margin for grade fluctuations (relevant to cleaner circuit). Residence time assumptions have been shown in Table 17.2 and a minimum of five cells per bank has been observed to minimize short-circuiting.

300m3 cells have been selected for rougher-scavenger flotation (two trains of 10 cells) and 40m3 cells for cleaning (9, 7, 5 cells for first, second, third, cleaners respectively). Note that an opportunity exists to improve capital and power efficiency by selecting the 500m3 cells, which have recently been released by one of the equipment manufacturers.

No thickening or filtration testwork has been performed at this stage (typically done by vendors) and AMC has not attempted any sizing of concentrating dewatering equipment. However the regrind size would indicate that a pressure filter would be required for concentrate filtration and for the purposes of the capital cost estimate presented in chapter 21, AMC has relied on benchmark comparables from a similar size project.

Tailings pumping is a significant item, particularly if suitable tailings storage facilities are only available at a considerable distance. High-level pump calculations were carried out on a thickened tails product and it was determined that a power load of approximately 7 MW would be required to pump just over 3500 tph a distance of up to 10 km, assuming minimal elevation change to any of a potential range of sites in the flat non-farmed land to the north-west of the project. Several parallel pumping stations and lines (probably four) would be required.

For the purposes of this PEA the roaster was sized at 400,000 tpa concentrate feed. This was based on two criteria, referring to the mill feed production schedule shown in Table 17.3:

- Being able to treat 80% of the average 500,000 tpa concentrate for first five years at 0.61% As average arsenic content, with the balance being for direct sales at approximately 0.5% As
- Not being over-sized for the balance of the mine life where all concentrates need to be roasted (average 1.4% As) and tonnage only rarely exceeds 400,000 tpa by a significant amount.

Clearly this is an item for further optimization at the next stage of study when more detailed production schedules are available, based on the additional geometallurgical variability information that will be generated from the testwork already recommended.

17.4 Production Schedule

The mill feed schedule is presented in Table 17.3. The mine schedule described in Section 16 was derived not only with an elevated cut-off grade strategy to maximize early cash flow and project NPV but also with the arsenic recovery data, smelter penalties (these are especially penurious above 1% As in the concentrate) and the cost of roasting built into the NSR values in the block model.

The recoveries and grades for the main mineralization types were applied to the mine schedule by individual mineralization type and the schedule represents the blended feed.

The pit optimization exercise was a good example of how combining the usual grade parameters with downstream metallurgical parameters facilitated selective mining of lower arsenic content material to meet desired concentrate specifications.

Table 17.3 Mill Feed Schedule

		Feed Grades				Concentrate	Conc Grades				Recoveries			
Period	Mill Feed	Cu	Au	Ag	As	Produced	Cu	Au	Ag	As	Cu	Au	Ag	As
	000 tonnes	%	g/t	g/t	%	000 tonnes	%	g/t	g/t	%				
1	30.0	0.53	0.09	4.0	0.02	480	27.5	4.20	194	0.70	83.6%	78.6%	77.2%	55.4%
2	30.0	0.54	0.03	3.4	0.04	540	26.8	1.41	151	0.78	90.0%	74.0%	80.0%	38.6%
3	30.0	0.56	0.02	2.4	0.04	573	26.4	0.64	92	0.56	89.8%	62.6%	72.0%	26.2%
4	30.0	0.51	0.02	2.4	0.03	516	26.4	0.89	101	0.47	89.4%	69.1%	70.9%	25.2%
5	29.9	0.43	0.02	2.1	0.03	429	26.3	0.94	98	0.53	87.8%	67.5%	66.0%	24.5%
6	30.0	0.33	0.04	2.4	0.02	329	27.4	2.90	184	1.29	90.0%	79.6%	84.1%	64.1%
7	29.9	0.43	0.07	4.2	0.08	428	27.1	3.77	240	3.40	90.0%	79.0%	82.5%	64.3%
8	29.8	0.40	0.02	2.1	0.03	408	26.3	0.82	101	0.69	89.8%	65.9%	67.4%	27.5%
9	30.0	0.43	0.03	2.1	0.03	440	26.4	1.26	104	0.80	90.0%	72.1%	72.9%	40.0%
10	30.0	0.36	0.04	2.2	0.03	360	26.7	2.83	143	1.03	89.0%	76.1%	76.5%	42.2%
11	30.0	0.35	0.03	2.0	0.03	355	26.6	1.97	122	0.96	89.9%	74.7%	74.0%	38.5%
12	30.3	0.27	0.03	1.7	0.03	269	26.8	2.33	144	1.40	88.7%	73.4%	75.4%	43.7%
13	30.0	0.19	0.02	1.4	0.03	189	26.8	2.89	164	1.32	89.0%	73.8%	74.6%	30.1%
14	30.2	0.22	0.03	1.6	0.02	220	26.7	2.69	160	1.08	90.0%	73.8%	74.0%	34.1%
15	30.1	0.26	0.02	1.8	0.03	261	26.6	2.07	153	1.33	89.9%	73.5%	74.7%	38.8%
16	30.3	0.33	0.05	2.3	0.04	330	26.8	3.42	167	1.78	89.5%	77.6%	78.4%	51.5%
17	30.0	0.35	0.04	2.1	0.04	350	26.6	2.26	136	1.75	90.0%	74.6%	76.0%	52.4%
18	30.0	0.41	0.03	2.4	0.04	419	26.6	1.49	133	1.21	90.0%	71.4%	76.2%	48.3%
19	29.7	0.18	0.01	1.5	0.04	187	26.3	0.87	161	1.37	89.9%	53.4%	65.8%	22.1%
20	17.5	0.17	0.03	1.3	0.01	89	27.6	5.41	211	0.98	83.7%	79.3%	81.4%	55.9%
Totals	**587.6**	**0.37**	**0.03**	**2.3**	**0.03**	**7169**	**26.7**	**2.05**	**142**	**1.12**	**89.1%**	**74.6%**	**75.6%**	**41.6%**

17.5 Process Inputs

The key process inputs for a large low grade project of this nature are power and water.

Power demand was estimated from first principle calculations for grinding and flotation and using vendor data for flotation cells. Grinding and flotation typically account for over 60% of the total power usage in a concentrator. Power requirements for ancillary equipment and services were estimated by a factoring approach. Because the tailings pumping power would be a function of the tailings storage facility location, this component was separately calculated as described earlier so it could be varied if necessary.

Total power demand amounted to 120 MW drawn, of which 7 MW was for tailings, assuming a pumping distance of 10 km. This total demand translated to 33 kWh/t in unit power consumption terms.

With a tailings thickener included in the flowsheet, it was assumed that recovery of water from the thickened material in the tailings storage facility would be negligible and, therefore, assuming no other significant losses, the net water consumption would equal the water contained in the thickener underflow. This amounts to 0.65 m3/T of feed or 19.5 million m3 of water per annum.

Other significant process inputs are grinding steel and reagents / consumables. Grinding steel was estimated from the abrasion index derived in the comminution testwork to be 1.2

kg/tonne of mill feed. Overall flotation reagent consumption was estimated at 0.1 kg/tonne, typical of a simple copper flotation circuit.

These power, steel and reagent unit consumption figures provided the basis for the operating costs tabulated in Section 21.

18 PROJECT INFRASTRUCTURE

18.1 Tailings Disposal

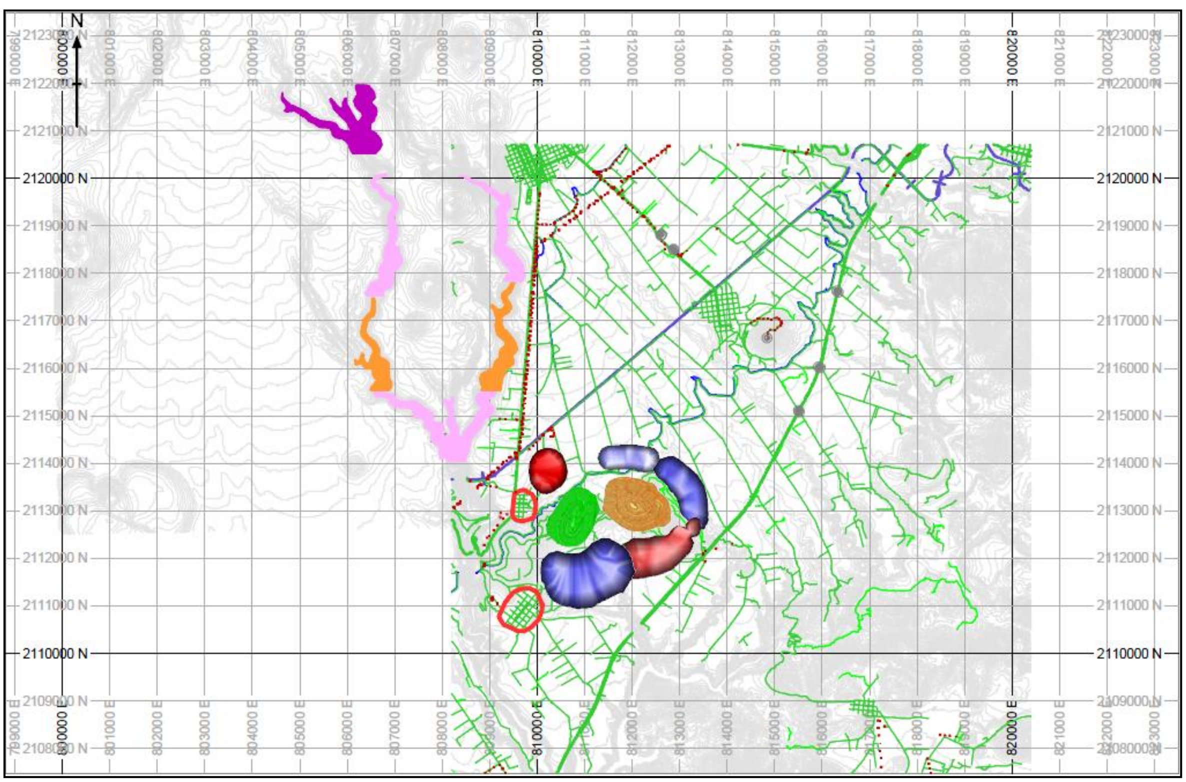
As indicated in Table 17.3, 587.6 million tonnes will be processed through the mine life. Using an estimate of 1.6 tonne / cubic meter for the settled density of tailings — and neglecting the mass recovered as concentrate — there is a requirement of 367 million cubic meters for storage of all the tailings for the LOM.

From a purely engineering and conceptual perspective a series of narrow valleys located around the property have initially been considered for this disposal. For the purpose of advancing this study, AMC and Catalyst have agreed that these narrow valleys can reasonably be considered as the closest technically feasible and economically convenient location for the TMF. AMC and Catalyst understand that a full process of consultation to local communities and authorities will be required to assess all the TMF options outlined in the study. Additionally, AMC notes that Catalyst does not own the land required for this intended use.

Preliminary estimations indicate that a total storage capacity of 150 M m^3 is achievable with construction of six dams.

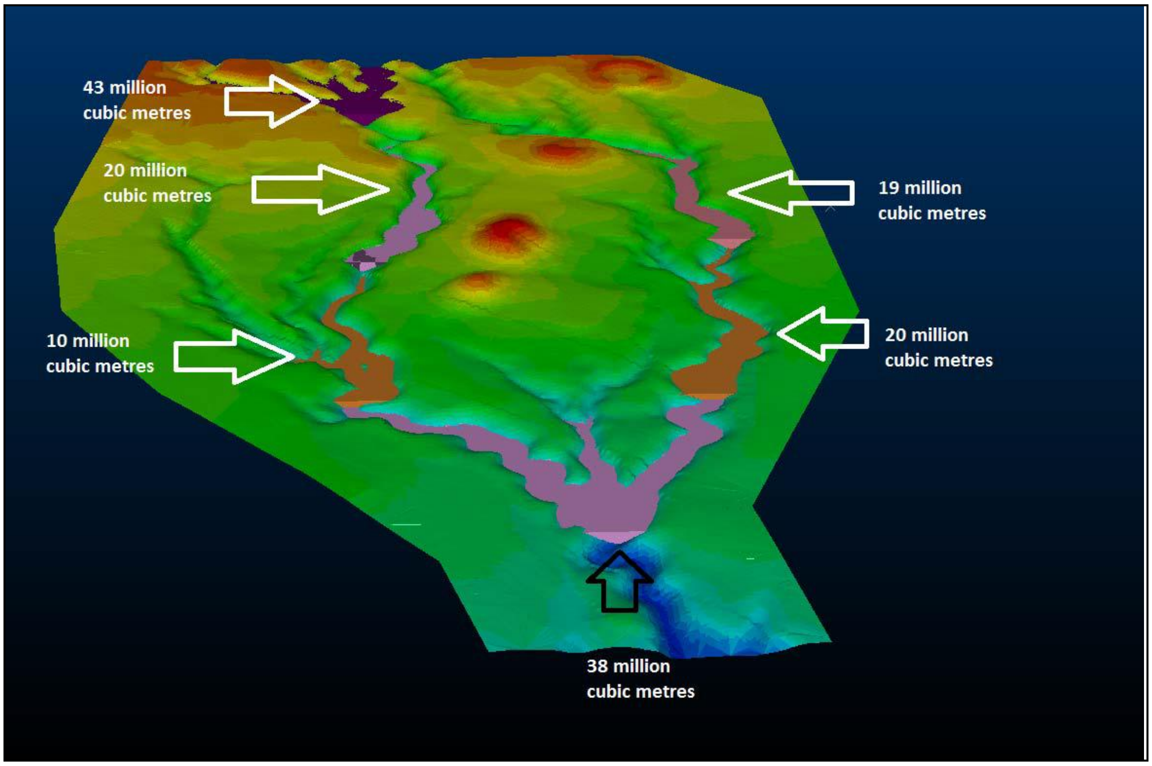
Figure 18.1 shows one of the possible locations of the proposed six dams in relation to the mine site.

Figure 18.1 Proposed TMF and Mine Site



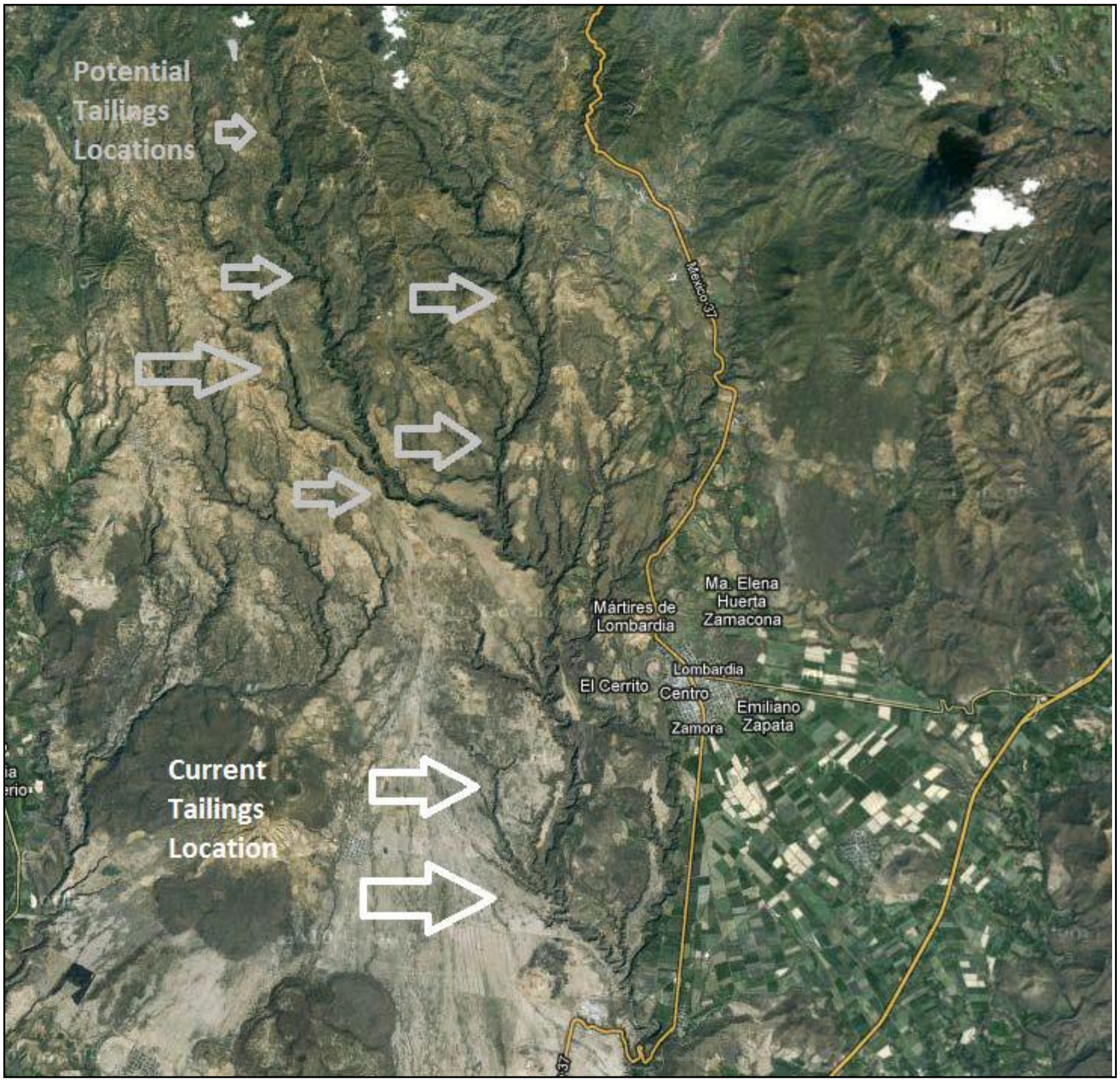
The capacities of these six dams are shown in Figure 18.2.

Figure 18.2 TMF Dam System and Capacities



This storage capacity is not sufficient for the LOM. An additional 217 M m^3 would be needed and therefore other options have to be investigated. Currently the most obvious option is the extension of valleys further to the north-west as shown in Figure 18.3.

Figure 18.3 Other Potential TMF Sites Near the Mine Site



At the conceptual engineering stage, the tailings management facility construction would commence with the closest dam to provide one and a half years storage at the production rate of 30 Mtpa. Five more dams need to be constructed to provide storage space for the following four years. The requirement is for an ultimate storage of 367 Mm^3 of tailings over the 20 year milling life.

During the early years the first dam wall would require 1.3 million cubic metres of burrowed material to be quarried from within, or in the vicinity of, the tailings dam impoundment area to form the bulk of the initial wall structure. If a contractor is brought in to excavate this rock, the cost will include mobilization, accommodation for a significant workforce, amortization of the contractor’s equipment and the contractor’s margin.

18.1.1 Design Basis and Operating Criteria

The principal objective of the design and operation of the TMF is to ensure secure containment for tailings solids and impounded process water. The TMF would serve as the primary water management facility for the project, providing a buffering volume for the mill process water demands, as well as collecting and storing the necessary quantities of precipitation and runoff.

The total tailings production is assumed to be about 82,200 t/d, or 100% of the milled mineralized rock. Tailings from the mill would be discharged to the TMF as a slurry at an average unthickened solids content of approximately 35% (by weight).

The initial six TMF dams are sized to store the estimated volume of tailings produced in the first eight years of mill operation (240 million tonnes). There will be a need for one or several more dams to store the estimated total of 367 Mm^3 of tailings over the 20 year milling life.

The topography is not favourable to yield a relatively large storage volume as compared with the quantity of material required for embankment construction. As a result TMF capital costs can be high. In the initial capital only the first six dams has been considered.

18.1.2 Tailings Management Facility Embankments

The first TMF embankment would be raised in the NW of the site, more embankments would be constructed upstream of the first with each embankment providing the required capacity for that particular period until the next embankment is completed, while always maintaining minimum storm water storage, wave run-up, and freeboard requirements. It is expected that the construction design of the embankments would be reviewed annually and refined as required, to accommodate the availability of construction materials and to incorporate experience gained with local conditions and constraints.

Small temporary coffer dams would be constructed for each embankment upstream of the main TMF embankment footprint. These dams would allow the TMF dam foundation area to be dewatered, cleared, and stripped prior to preparation for construction of the embankment.

A network of 150 mm-diameter perforated high-density polyethylene (HDPE) interceptor pipes placed in a dendritic or herringbone pattern would underlie each dam foundation. The drains would be surrounded by appropriate filter and drainage materials. The individual interceptor drains would connect to 300 mm diameter HDPE main collector pipes to transport seepage to recycle ponds located at the topographic low points below each embankment. The underdrain network would be expanded as the staged embankments are constructed. They would also provide foundation dewatering during initial construction.

Each dam has a minimum horizontal width of 15 m to allow placement of burrow material by the required fleet. Rock would be dumped, spread, and compacted to the specified density. All embankments would have a minimum final crest width of 15 m to allow for vehicle access and pipelines. The dams would be designed as water retaining structures to allow the supernatant pond to submerge the tailings beaches to the extent possible. A central low-permeability core zone would provide a positive hydraulic cut-off in the dam. Appropriate filter zones would be placed downstream of the low-permeability core. It is envisaged that these would be constructed by a specialized contractor using locally sourced borrow material. Ongoing construction of the dam shell zone would make use of suitable low-sulphur, geochemically innocuous rock from the area. The shell zones would need to be constructed prior to beginning work on the crest, core, and filter zones to allow access for equipment. Geomechanical waste rock characterizations are needed to confirm that this is technically and economically feasible.

18.1.3 Mill Tailings Transport and Deposition System

The tailings transport system would be constructed in stages throughout the life of the project. Tailings would initially be pumped from the mill directly to the main TMF embankment through a 560 mm-diameter rubber lined steel pipeline; a 560 mm-diameter HDPE pipeline would be used to distribute tailings to off takes located along the embankment.

18.1.4 Reclaim Water System

The reclaim water system would comprise both pumped and gravity flow components. Supernatant reclaim water from the TMF would be pumped over the dam to a head tank directly above the mill site at an elevation of roughly 700 m.

Reclaim pumps can be mounted on a floating barge in the TMF supernatant pond and would operate on level control from the head tank.

18.2 Waste Rock Storage (reactive and non-reactive)

Conceptually, waste dumps would be located on the south and east side of the pits as shown on the site plan, Figure 18.1.

A ditch and pumping system is required to collect and transport any run-off from the waste pile to the water treatment plant or mill.

18.3 Mill Feed Stockpiles

Mill feed stockpiles are proposed to be on the north side of the pits as shown on the site plan, Figure 18.1 with the main stockpile area on the West side of the process plant.

18.4 Irrigation Channel

An irrigation channel runs in a NE to SW direction by the proposed process plant site. It is recommended that the course of this waterway would stay as is. Capital funds have instead been allotted to channel the water in a culvert to allow for traffic movement over the channel. Catalyst envisages that any action concerning this channel is subject to engagement with the communities and government.

18.5 Access Roads

Two paved highways provide access to the Property including the new Morelia-Lazaro Cárdenas Autopista (toll road) and the original Lombardia-Nueva Italia highway (Mexico Highway 37) (Figure 18.4). Within the Property there are a series of all-weather dirt roads and old drill roads.

Figure 18.4 Property Connectivity to Nearby Locations

There is an allowance for 10 km of site access roads to connect the pits to the process plant, explosives magazine, truck shops and waste stockpiles. Some of these roads are constructed to accommodate heavy mine haul trucks, others for lighter transportation.

A 2 km road is required for transport of concentrate from the plant to the rail siding load out facility.

A 6 km road is required to place pipelines for the mill tailings to the NW area TMF. A large pipe for the tails and a smaller pipe for the reclaim water from TMF will be placed on this road. Regulations may require containment provisions in case of line breakage. Table 18.1 shows the estimated cost of site roads.

Table 18.1 La Verde PEA Estimate Road Costs

ITEM	M$
Main access	0.25
Access to TMF	2.4
To load out	0.5
Site roads	1
Divert irrigation channel	1
Total	5.15

18.6 Rail access

The rail link to the deep sea port of Lazaro Cardenas is in excellent condition (Figure 18.5). A siding already exists at the new core storage warehouse facility north of Nueva Italia de Ruiz which could be used to handle bulk imports and copper concentrates.

Figure 18.5 Active Rail Transport Close to Core Shack

18.7 Water Supply

It is intended that a large percentage of the process water with the exception of what is lost to evaporation is recycled. Potential fresh water sources include the irrigation channel mentioned in Section 18.4, the irrigation reservoir located nearby to the south east of the property, and the use of Catalyst-owned water wells drilled where permitted.

Figure 18.6 Irrigation Channel Close to the Proposed Mill Site

18.8 Power

The overall power demand for the operation has been estimated at 145 MW. The main user of power is the process plant.

There are several high-tension electrical lines (66 kVA) within 5 km of the Project. The client indicates that Mexican power authority CFE would have sufficient capacity for La Verde operations. Figure 18.7 shows a typical high tension transmission line in the vicinity of the mine site.

Figure 18.7 Two Height Tension Powerlines. La Verde Hills in the Background

Table 18.2 shows a breakdown of the initial capital cost estimate for the electrical system at site.

Table 18.2 La Verde PEA Estimate Electrical Cost

Site	MUS$
Mine site substation	30
Electrical distribution	25
communication system	1
Fences, gates, fire alarm system yard lighting	1
Pit electrical	5
Transmission lines for 5km	3
Total	65

18.9 Other Infrastructure

The project infrastructure and services have been estimated to support an operation of 30 Mtpa of copper mineralized rock to produce concentrate. Infrastructure and ancillary facilities are explained in the following section of the report

Careful attention was given to the placement of the facilities in order to minimize the overall footprint and required excavation. The layout as a whole takes advantage of the natural slope in the area and availability of services.

Table 18.3 shows the estimated cost of other infrastructure.

Table 18.3 La Verde PEA Estimate for Other Infrastructure Costs

ITEM	MUS$
Concentrate rail load-out facility	6
Construction camp	10
Offices include lockers, first aid	2
Truck shop, tools and lube oil	7
Warehouse	1
Lab and equipment	1
Fuel storage	1
Explosives magazine	1
Site prep	25
Total	54

18.9.1 Equipment Lay Down Area

No open area storage area has been specifically allocated; however, there is area within the plant site that could be fenced off to provide extra storage for equipment and materials, if required.

18.9.2 Loading and Unloading Facilities

A truck load-out facility will be required at the process plant for loading of concentrate. It is anticipated that the concentrate will be trucked to the railhead 1.5 km NW of the plant where a siding is in existence. An unloading facility for trucks and load-out onto the rail cars will be required at this site.

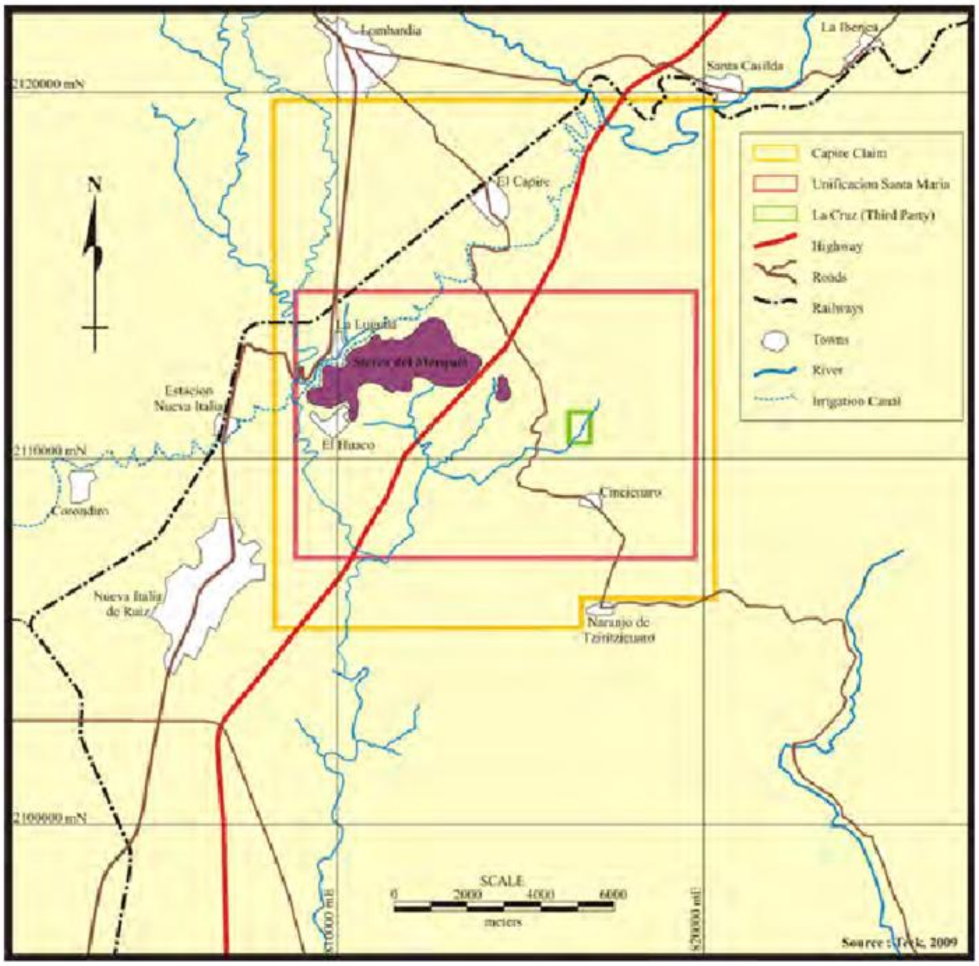
18.9.3 Water Treatment Plant

Reclaim water from the TMF and any water collected from the waste and mill feed stockpiles and effluent from the offices, truck shop and warehouse will be treated prior to use in the plant or discharge in the environment.

18.9.4 Communications

The site communications systems will be supplied as a design build package and the scope defined in future project phases.

Figure 18.8 La Verde Site, Mine Claims and Surroundings



18.9.5 Port Facilities

The port of Lazaro Cardenas is approximately 200 km south west of the mine site. This port has rail unloading and ship loading facilities.

18.9.6 Camp and Housing

A construction camp will be required during the early days of the project. Upon commissioning, it is anticipated that the workforce will be living in the many communities that are in the vicinity of the mine site.

There are provisions for a modularized structure construction camp in the capital estimate. At a later stage this structure can be converted into either offices or a temporary or permanent accommodation facility for the workforce. The construction camp is located at the plant site.

18.9.7 Administration Offices Training Room

The administration building will be a modularized structure that will provide working space for management, supervision, geology, engineering, and other operations support staff.

18.9.8 Maintenance Workshops and Warehouse Building

A stick-built maintenance / warehouse facility will be provided to service the mobile equipment and for storage of equipment spares. One repair bay will be provided for servicing light vehicles, which will also be used for tune-ups. Small tools and equipment will be provided. A waste oil system, exhaust system, lube-oil system, water system, small machine shop and equipment, and welding bay will be included.

The maintenance area will be equipped with a crane. The warehouse area will be sized to accommodate process materials and the maintenance shop supplies.

18.9.9 Mine Dry (change rooms)

A separate mine dry facility, including lockers and shower facilities, will be provided. The mine dry will be a modularized structure located at the plant site.

18.9.10 Laboratories

An assay laboratory will be located in a separate modular building as part of the mill building. The laboratory will be a single-storey structure equipped to perform daily analyses of mine and process samples.

18.9.11 Security and Guardhouse

A security / gatehouse will be located on the site access road at the plant site. The access road at Highway 37 will have a manual gate with signage indicating that they are now entering private property.

18.9.12 Medical / Ambulance / First Aid / Mine Rescue Facilities

Due to proximity of the mine site to several communities, it is envisaged that only basic medical and first aid facilities would be required to stabilize any injuries for transport to local hospitals.

18.9.13 Truck shop

The truck shop building will be a pre-engineered building, with an overhead clearance of at least 10 m. The building will be designed to provide facilities for maintenance and repair, minor office space, clean and dry areas, and general storage. It will be at the plant site near the mining haul road. The truck shop will house two maintenance bays, one light vehicle repair bay, a welding and machine shop, and an electrical and instrument shop. The truck wash and tire change building is included within the truck shop. The building will contain a wash bay, maintenance bay, tool crib, compressor room, hot water pressure system, tire change, and an oil separator. Waste oil will be disposed of in the refuse incinerator with any remaining oil removed and discarded at an approved facility.

18.9.14 Explosives Magazine

The explosives magazine is located within the mine property, south west of the West Hill pit.

18.9.15 Fuel Storage and Distribution Systems

An area has been designated near the truck shop for the storage and dispensing of fuel. The fuel storage and dispensing facility will include a lined containment area so that spills are confined and can readily be cleaned, and so that the need for extensive and costly remediation work can be avoided during site closure.

Diesel fuel will be required for the non-electrical mobile mine equipment, some small trucks, and surface vehicles. The pumping station allows for re-fuelling of both light vehicles and heavy-duty mining equipment.

19 MARKET STUDIES AND CONTRACTS

19.1 Concentrate Marketing

Although at this early stage in the project no smelter contracts have been put in place, the crucial importance of the arsenic levels in concentrates has been well recognized, as has already been discussed.

Globally there is a trend towards higher arsenic levels in copper concentrates, while on the other hand smelting capacity for arsenic-bearing materials is decreasing. Hence there is once again a strong interest in potential hydrometallurgical routes for treating high arsenic concentrates as well as traditional roaster routes, albeit with attendant environmental issues.

Specialist in high arsenic concentrate smelters like Dundee Metal's Tsumeb smelter in Namibia specialize in concentrates with levels of 5% As, higher than La Verde's under any circumstances unless a differential float to produce a high arsenic concentrate was carried out. This route is not considered any further at this stage, although it remains an option for further study.

In the "normal" copper concentrate market, "clean" concentrates are those with less than 0.1% As, penalties typically commence at 0.3% As and "dirty" concentrates used to be considered as containing greater than 1% As with 2% As as an absolute limit. However the closure of several smelters capable of treating such dirty concentrates has squeezed the market significantly and most of these dirty concentrates are treated in only one country, China.

Although Chinese smelters treat domestic concentrates with 2-3% As, there is a limit imposed on imported base metal concentrates of 0.5% As. Marketing concentrates through the large global concentrate traders does offer some blending opportunities; however 0.5% As is the new target maximum, especially for a large volume producer like La Verde where the large volumes limit blending options.

AMC has based its analysis of La Verde concentrate options on the following premises:

- Current benchmark smelter TC/RC terms for "clean" (<0.3% As) concentrates of 70/7 i.e $70/t for TC, 7c/lb Cu for RC
- Normal As penalties applying ($3/t per 0.1% As > 0.3% As) up to 1% As
- An additional $20/T TC applied for concentrates >0.5% As to allow for storage and blending costs through a concentrate trade (based on a recent example in Mexico)
- Much increased TC's for concentrates > 1% As (additional $100/t plus up to $5/t per 0.01% As, also based on a recent example in Mexico), reflecting acute difficulties in securing sales.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Due to the early stage of the La Verde copper project development Catalyst has not yet implemented any formal environmental and community impact evaluation programs. Accordingly, no specific report has been made available to AMC for review. Following are some AMC's environmental and communities-related considerations contained in this study.

20.1 Environmental and Communities Considerations Included in this Study

AMC envisages that the three most significant areas of environmental impact of this Project are: the two pits, the waste dumps and mill feed stockpiles, and the tailings management facilities. This study principally focuses on the technical and economical viability of the project, emphasizing the engineering point of view. AMC and Catalyst understand that the actual viability of the technical solutions proposed for the mine, the dumps and the TMF need to be assessed with the communities and the corresponding permits and social licences are required. In the Recommendations, AMC states the need for an early assessment of various specific environmental / social features of the Project should it be moved to more advanced stages.

20.1.1 The Mine Pits

By the natural location of the deposit, most of the land comprised within the pit rims corresponds to wild vegetation areas, currently with no agricultural or pasture use. Also, preliminary observation does not reveal the existence of any site regularly used by the communities inside the pit rims perimeters.

20.1.2 Waste Dumps and Stockpiles

A surrounding belt of massive rock storage facilities is included in the study (see Figure 16.4). Most of waste has been scheduled to be left on East and South dumps whereas two stockpiles are developed on the North side on each flank of the concentrator site. Both stockpiles will grow during the first half of the mine life to be completely reclaimed by the end of the mine life.

Both proposed waste dumps and stockpiles make extensive use of current agricultural land. This aspect will need to be consulted with the involved communities.

An opportunity of reducing the footprint of waste dumps and stockpiles may be available as most of the proposed designs have spare capacity on top. Further trade-off mining studies can examine this option.

As shown is Sections 18 and 21, waste dumps remediation is considered in the present study.

20.1.3 Tailings Management Facilities

Among other options reviewed, the valleys to the NW of project have been suggested for tailings disposal. Proximity to plant site is the main reason for this choice. One valley is dry during the non-rainy season. Provisions for the east valley that may contain water most of the year is to allow the water un-interrupted flow via culverts under the tailings.

Currently, it is observed that there is minimal vegetation or agricultural use around these valleys, as such the impact of tailings disposal would be minimized. Notwithstanding, Catalyst is committed to carry on appropriate environmental and community surveys before making any final planning for tailings disposal.

At closure, there are provisions in the budget to rehabilitate these impoundments by covering them with top soil from the vicinity.

21 CAPITAL AND OPERATING COSTS

21.1 Mining Capital and Operating Costs

21.1.1 Operating Costs

An indicative operating cost model has been used to estimate total and unit mining cost for La Verde Project. Calculated operating costs at La Verde pits are mostly driven by the haulage cost. During the first half of the mine life haulage profiles are more productive and, accordingly, haulage is comparatively cheaper. From years 11 to 19 haulage cost markedly increases due to the deeper source of ex-pit rock. Table 21.1 gives indicative estimates of the unit mining cost per year.

Table 21.1 Indicative Mining Total and Unit Costs

		Y(-3)	Y(-2)	Y(-1)	Y1	Y2	Y3	Y4	Y5	Y6 to Y10	Y11 to Y15	Y16 to Y20	LOM
	unit												
Ex-Pit Tons	Mt			65.0	95.0	95.0	95.0	95.0	120.0	600.0	575.0	198.1	1938.1
From Stocks Tons	Mt			0.0	6.2	0.0	0.0	2.4	6.6	5.9	43.9	50.7	115.8
Total Moved Tons	Mt			65.0	101.2	95.0	95.0	97.4	126.6	605.9	618.9	248.8	2053.8
Drilling	MUS$			7.4	13.0	13.2	13.3	13.1	15.7	80.2	76.9	31.7	264.6
Blasting	MUS$			12.3	20.7	22.0	22.2	21.0	24.1	120.7	113.5	44.6	401.2
Loading	MUS$			15.9	22.4	21.7	21.9	21.6	30.0	146.8	148.4	79.9	508.6
Haulage	MUS$			14.5	20.9	20.6	20.0	24.0	40.9	264.5	765.6	417.3	1588.2
Support Equipment	MUS$			10.4	16.9	16.9	16.9	16.9	16.9	84.3	83.6	47.5	310.2
Mine Eng. & Geology	MUS$			3.5	3.5	3.5	3.5	3.7	3.7	18.4	18.4	18.1	76.5
Maintenance G&A + Shops	MUS$			8.2	8.2	8.2	8.2	8.2	8.2	40.8	40.8	33.0	163.4
Mine G&A	MUS$			4.7	4.7	4.7	4.7	4.7	4.7	23.4	23.3	19.8	94.6
Total	**MUS$**			76.8	110.2	110.8	110.7	113.1	144.1	779.1	1270.6	691.8	3407.2
		Unit cost , US$ / moved tonne											
Drilling	US$/t			0.11	0.13	0.14	0.14	0.13	0.12	0.13	0.12	0.13	0.13
Blasting	US$/t			0.19	0.20	0.23	0.23	0.22	0.19	0.20	0.18	0.18	0.20
Loading	US$/t			0.24	0.22	0.23	0.23	0.22	0.24	0.24	0.24	0.32	0.25
Haulage	US$/t			0.22	0.21	0.22	0.21	0.25	0.32	0.44	1.24	1.68	0.77
Support Equipment	US$/t			0.16	0.17	0.18	0.18	0.17	0.13	0.14	0.14	0.19	0.15
Mine Eng. & Geology	US$/t			0.05	0.04	0.04	0.04	0.04	0.03	0.03	0.03	0.07	0.04
Maintenance G&A + Shops	US$/t			0.13	0.08	0.09	0.09	0.08	0.06	0.07	0.07	0.13	0.08
Mine G&A	US$/t			0.07	0.05	0.05	0.05	0.05	0.04	0.04	0.04	0.08	0.05
Total	**US$/t**			**1.18**	**1.09**	**1.17**	**1.17**	**1.16**	**1.14**	**1.29**	**2.05**	**2.78**	**1.66**

21.1.2 Capital Cost

Mine capital costs are mostly comprised of mobile equipment. Maintenance buildings and mine electric grid are included under Infrastructure. Total initial capital for the mine, including mobile equipment and first fills, amounts to 134.5 MUS$. Table 21.2 gives the amount of capital per year, including the initial capital spent in years Y(-3) to Y(-1). This includes mine operation and mobile equipment maintenance, major repairs for bigger mine equipment (shovels and drills), and ancillary.

Table 21.2 Mine Mobile Equipment Capital Per Year

		Y(-3)	Y(-2)	Y(-1)	Y1	Y2	Y3	Y4	Y5	Y6 to Y10	Y11 to Y15	Y16 to Y20	LOM
Primary Drill	MUS$	0.0	7.0	3.5	7.0	0.0	0.0	0.0	3.5	3.5	7.0	0.0	31.5
Secondary Drill	MUS$	0.0	0.0	0.6	0.0	0.0	0.0	0.0	0.0	0.6	0.6	0.0	1.8
Explosive Factory Truck	MUS$	0.0	0.8	0.8	0.0	0.0	0.0	0.0	0.0	1.6	1.6	0.0	4.8
Rope Shovel	MUS$	0.0	0.0	23.0	0.0	0.0	0.0	23.0	0.0	0.0	0.0	0.0	46.0
Hydraulic Shovel	MUS$	0.0	18.0	18.0	0.0	0.0	0.0	0.0	0.0	0.0	18.0	0.0	54.0
Front-End Loader	MUS$	0.0	4.7	0.0	0.0	0.0	0.0	0.0	4.7	4.7	0.0	0.0	14.1
Haultruck	MUS$	0.0	15.6	19.5	11.7	0.0	0.0	0.0	15.6	66.3	35.1	0.0	163.8
Wheel Dozer	MUS$	0.0	1.6	1.6	0.8	0.0	0.0	0.0	0.0	4.0	0.0	0.0	8.0
Bulldozer	MUS$	0.0	3.6	3.6	1.8	0.0	0.0	0.0	0.0	9.0	0.0	0.0	18.0
Water Truck	MUS$	0.0	1.6	1.6	3.2	0.0	0.0	0.0	0.0	6.4	3.2	3.2	19.2
Grader	MUS$	0.0	0.0	0.7	0.7	0.0	0.0	0.0	0.0	0.7	0.7	0.0	2.8
Cable Handler	MUS$	0.0	0.0	0.2	0.2	0.0	0.0	0.2	0.0	0.0	0.4	0.0	1.0
Crane	MUS$	0.0	0.0	1.1	1.1	0.0	0.0	0.0	0.0	1.1	1.1	0.0	4.4
Tool Truck (Maintenance)	MUS$	0.0	0.0	0.8	0.8	0.0	0.0	0.0	0.0	1.4	1.4	0.0	4.4
Major Repairs	MUS$	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	7.3	8.8	0.0	16.1
Other Capital Items	US$/t	0.0	2.6	4.0	0.0	3.0	0.0	1.0	0.0	1.0	1.0	0.0	12.6
Total	**US$/t**	**0.0**	**55.5**	**79.0**	**27.3**	**3.0**	**0.0**	**24.2**	**23.8**	**107.6**	**78.9**	**3.2**	**402.5**

21.2 Milling Capital and Operating Costs

For the proposed 30 mtpa capacity for La Verde concentrator following Tables 21.3, 21.4 and 21.5 give the total concentrator construction cost, mill operating costs and personnel estimates, respectively.

Table 21.3 Processing Capital Cost Estimate (Million US$)

	Mechanical Eqmt.	Installed Cost	Source / Basis
DIRECTS			
Site Preparation		10.0	AMC database (similar sized base metals project)
Crushing		58.0	
Coarse ore stockpile / reclaim		48.0	
SAG Mills (2) incl transport	62.7	125.4	Power calculations and quotes
Ball Mills (2) incl transport	28.1	56.2	
Pebble Crusher	5.0	10.0	AMC database
Flotation	17.6	35.2	mass balance calculations and quotes
Regrind	3.2	6.3	power calculations and quotes
Concentrate Dewatering	3.0	6.0	AMC database (similar sized base metals project)
Concentrate Loading		20.0	
Tailings pumping / lines	10.0	40.0	
incl thickener	10.0		
Reagents		5.0	
Services / Utilities		8.0	
Ancillary buildings (incl lab)		10.0	
TOTAL DIRECTS		**438.1**	
INDIRECTS			
EPCM etc 25%		109.5	As percentage of Directs
(incl spares / first fills)			
Temporary Facilities etc 5%		21.9	As percentage of Directs
Owners Costs 10%		43.8	As percentage of Directs
TOTAL INDIRECTS		**175.2**	
Contingency 26%		161.7	10% to equipment quotes, 30% general
MILL Grand TOTAL		**775.1**	
ROASTER		**104**	Lump sum. Scaled on benchmarking on similar application in Mexico

Table 21.4 Processing Operating Cost Estimate (US$)

Item	$ per year	Unit usage u/t	Units	Unit Cost $/Unit	Unit Cost $/t	Source
Power (milling)		31	KWh	0.1	3.1	Refer Section 17.5
Power (tailings)		2	KWh	0.1	0.20	
Grinding Media		1.2	kg	1	1.20	
Reagents (flotation)		0.1	kg	3	0.30	
Other reagents					0.30	AMC estimate
Consumables					0.20	
Labour	3.305				0.11	Refer Table 21.5
Maintenance (3% installed)	13.1				0.43	Industry factor
Other (contractors etc)	3.0				0.1	AMC estimate
(250 k/month)						
Total					**5.84**	

Note that the power cost estimate of $0.1/kWh has been confirmed as a reasonable estimate for Mexican grid power and reinforces the recommendation regarding a trade-off study for high pressure grinding rolls as an alternative to SAG milling to reduce power consumption.

Table 21.5 Processing Manning Costs (US$)

Category	Salary (all-incl) US$	Numbers	Cost $ p.a.
Operators	15,000	55	825,000
Lab	20,000	12	240,000
Tradesmen	20,000	24	480,000
Supervision	60,000	9	540,000
Professionals:-			
National	80,000	4	320,000
Ex-Pat	150,000	4	600,000
Management (ex-pat)	300,000	1	300,000
Total		**109**	**3,305,000**

As explained in Section 17.3, the roaster facility has been estimated to have a capacity to treat 400,000 tonnes per year. The estimated operation cost is US$30/tonne of concentrate fed. A high-level estimate of the total capital cost is included in Table 21.3. AMC estimates that further studies, including flotation management of the arsenic and options for mill feed scheduling, will contribute to building a more robust estimate in terms of capacity and costs.

21.3 Infrastructure Capital

The infrastructure capital includes roads, buildings, electric power and tailings management facilities. Table 21.6 shows a summary of these costs.

Table 21.6 Infrastructure Capital Costs

ITEM	COST, Million US$
Buildings	17.5
Roads	11.0
Electrical	55.0
Site preparation	15.0
TMF	164.0
Closure	92.0
Total	**356.0**

21.4 General and Administrative (G&A) Costs

G&A costs are estimated at 10MUS$ based on staffing complement of 69 and including insurance, permits, flights and miscellaneous fees.

22 ECONOMIC ANALYSIS

The preliminary economic assessment is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the preliminary economic assessment will be realized.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

22.1 Revenue Estimation

The income generated by the La Verde Project comes from selling copper concentrate to copper smelters around the world. The La Verde concentrates will additionally contain variable amounts of gold and silver in high enough concentration to be payable by smelters.

Yearly metals production in concentrate from the La Verde mill can be obtained from Table 17.3. Yearly revenue has been calculated applying assumed metal prices, as follows:

- Copper: 2.7 US$/lb (5,950 US$/tonne)
- Gold: 1,200 US$/troy ounce (38.58 US$/g)
- Silver: 25 US$/troy ounce (0.80 US$/g)

These prices have been applied evenly throughout the mine life. First revenue is obtained in year Y1.

Smelting and refining charges, maximum payable metal percentages, freight charges and arsenic-related penalties have been applied as described in Table 16.2. The resulting gross and net revenues are given in Table 22.1. All figures are million US$.

Table 22.1 Gross and Net Revenue Estimation

YEAR	Y(-1)	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10	Y 11	Y 12	Y 13	Y 14	Y 15	Y 16	Y 17	Y 18	Y 19	Y 20	**LOM**
Final Product (dry), Mt		432.3	486.1	515.3	464.1	386.0	296.2	385.1	366.8	395.6	323.9	319.2	241.9	169.7	198.2	235.3	296.6	315.4	376.7	168.0	79.7	**6,452**
Freight, US$M		30.3	34.0	36.1	32.5	27.0	20.7	27.0	25.7	27.7	22.7	22.3	16.9	11.9	13.9	16.5	20.8	22.1	26.4	11.8	5.6	**452**
Roasting, US$M																						
TC, US$M		30.3	34.0	36.1	32.5	27.0	20.7	27.0	25.7	27.7	22.7	22.3	16.9	11.9	13.9	16.5	20.8	22.1	26.4	11.8	5.6	**452**
Refining, US$M		20.4	22.4	23.3	21.0	17.4	13.9	17.9	16.5	17.9	14.8	14.6	11.1	7.8	9.1	10.7	13.6	14.4	17.2	7.6	3.8	**295**
As Penalties, US$M		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**-**
Total, US$M		80.9	90.4	95.5	86.0	71.4	55.4	71.8	67.9	73.3	60.2	59.2	45.0	31.6	36.8	43.7	55.2	58.6	69.9	31.1	14.9	**1,199**
GROSS REVENUE (BASED ON PAYABLE METALS)																						
YEAR	Y(-1)	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10	Y 11	Y 12	Y 13	Y 14	Y 15	Y 16	Y 17	Y 18	Y 19	Y 20	**LOM**
Copper, US$M		757.8	832.4	868.6	782.7	647.4	518.5	666.6	615.0	667.1	552.4	541.7	413.9	290.5	337.2	399.7	507.6	536.0	639.9	282.1	140.4	**10,998**
Gold, US$M		59.2	8.5	0.0	0.0	0.0	24.1	45.8	0.0	4.4	25.3	13.3	13.8	13.7	14.3	10.8	30.8	17.0	8.0	0.0	15.1	**304**
Silver, US$M		63.4	52.5	28.5	29.4	23.5	40.9	72.3	23.4	26.1	32.6	26.3	24.7	20.4	23.0	25.8	36.4	30.0	34.7	19.6	12.9	**646**
Total, US$M		880.5	893.4	897.1	812.1	670.9	583.5	784.7	638.4	697.7	610.3	581.3	452.3	324.6	374.5	436.3	574.7	582.9	682.5	301.8	168.4	**11,948**
NET REVENUE (NSR), US$M		**799.6**	**803.0**	**801.6**	**726.1**	**599.5**	**528.1**	**712.8**	**570.5**	**624.4**	**550.2**	**522.0**	**407.4**	**293.0**	**337.7**	**392.6**	**519.6**	**524.4**	**612.6**	**270.7**	**153.5**	**10,749**

22.2 Initial Capital Cost

Initial capital cost has been estimated as 1,160 MUS$, to be spent in three years from Year (-3) to Year (-1). It can be broken down as follows:

- Mining equipment: 134.5 MUS$
- Mill and Roaster: 879.1 MUS$
  - 775,1 MUS$ in the mill buildings and equipment
  - 104.0 MUS$ in the roasting facility
- Infrastructure: 126.5 MUS$ in TMF, buildings, electrical and other facilities

Other preproduction capital expenditures are:

  - 76.8 MUS$ in operating cost for mine pre-stripping
  - 27.6 MUS$ in mine sustaining capital (first 3 months of Year 1)
  - 47.4 MUS$ in mill sustaining capital (first 3 months of Year 1)

22.3 Cash Flow and Financial Indicators

The La Verde project cash flow combines operating and capital costs, as described in Section 21 and Section 22.2; and revenues, given in Section 22.1. The projected before-taxation cash flow is given in Table 22.2.

Table 22.2 La Verde Project Cash Flow (before taxation)

	CASH FLOW, MUS$ - BEFORE TAXATION																										
			YEARS ------->																								
			Y(-3)	Y(-2)	Y(-1)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	TOTALS
	Ex-Pit Tonnage	Mt			65.0	95.0	95.0	95.0	95.0	120.0	120.0	120.0	120.0	120.0	120.0	115.0	115.0	115.0	115.0	115.0	80.0	80.0	38.1	0.0	0.0		1,938
	Total Moved Tonnage	Mt			65.0	101.2	95.0	95.0	97.4	126.6	120.0	120.0	120.0	120.0	125.9	115.0	127.0	133.9	121.8	121.2	83.0	80.0	38.1	30.0	17.7		2,054
	Unit Cost - Per Ex-Pit Tonne	US$/t			1.18	1.16	1.17	1.17	1.19	1.20	1.20	1.27	1.27	1.38	1.38	1.37	1.90	2.33	2.57	2.87	3.15	3.11	3.18	-	-		1.76
	Unit Cost - Per Moved Tonne	US$/t			1.18	1.09	1.17	1.17	1.16	1.14	1.20	1.27	1.27	1.38	1.31	1.37	1.72	2.00	2.43	2.73	3.03	3.11	3.18	1.42	1.55		1.66
MINE	**Total Mining Op. Cost**	**US$M**			**76.8**	**110.2**	**110.8**	**110.7**	**113.1**	**144.1**	**143.8**	**152.6**	**152.3**	**165.1**	**165.2**	**157.9**	**218.6**	**267.5**	**296.0**	**330.5**	**251.6**	**249.0**	**121.1**	**42.6**	**27.5**		**3,407**
	Mine Capital - Initial	US$M		55.5	79.0																						134
	Mine Capital - Sustaining	US$M				27.3	3.0	0.0	24.2	23.8	4.5	17.9	14.2	20.8	50.2	42.4	6.7	0.8	8.4	20.6	3.2	0.0	0.0	0.0	0.0		268
	Mine Capital - Working	US$M			27.6																				-27.6		0
	Total Mine Capital	**US$M**		**55.5**	**106.6**	**27.3**	**3.0**	**0.0**	**24.2**	**23.8**	**4.5**	**17.9**	**14.2**	**20.8**	**50.2**	**42.4**	**6.7**	**0.8**	**8.4**	**20.6**	**3.2**	**0.0**	**0.0**	**0.0**	**-27.6**		**402**
	Mill Feed	Mt				30.0	30.0	30.0	30.0	29.9	30.0	29.9	29.8	30.0	30.0	30.0	30.3	30.0	30.2	30.1	30.3	30.0	30.0	29.7	17.5		588
	Unit Milling Cost	US$/t				5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84	5.84		
	Total Milling Op. Cost	**US$M**				**175.3**	**175.1**	**175.1**	**175.0**	**174.8**	**175.0**	**174.6**	**174.2**	**175.1**	**175.3**	**175.0**	**176.8**	**175.0**	**176.1**	**175.8**	**177.0**	**175.2**	**175.2**	**173.5**	**102.4**		**3,431**
	Roaster Feed	000t				400.0	400.0	400.0	400.0	400.0	329.1	400.0	400.0	400.0	359.9	354.6	268.8	188.6	220.2	261.4	329.6	350.4	418.6	186.7	88.5		6,557
MILL & ROASTER	Unit Roasting Cost	US$/t				30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0		
	Total Roasting Op. Cost	**US$M**				**12.0**	**12.0**	**12.0**	**12.0**	**12.0**	**9.9**	**12.0**	**12.0**	**12.0**	**10.8**	**10.6**	**8.1**	**5.7**	**6.6**	**7.8**	**9.9**	**10.5**	**12.6**	**5.6**	**2.7**		**197**
	Mill Capital - Initial	US$M		387.5	387.5																						775
	Roaster Capital - Initial	US$M			104.0																						104
	Mill & Roast - Sustaining Cap.	US$M				7.0	7.0	7.0	7.9	7.9	7.9	8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.8		168
	Mill & Roast - Working Cap.	US$M			46.8																				-46.8		0
	Total Mill&Roast Capital	**US$M**		**387.5**	**538.4**	**7.0**	**7.0**	**7.0**	**7.9**	**7.9**	**7.9**	**8.8**	**8.8**	**8.8**	**8.8**	**8.8**	**8.8**	**8.8**	**8.8**	**8.8**	**8.8**	**8.8**	**8.8**	**8.8**	**-38.0**		**1,047**
	Infrastructure Capital - Initial	US$M	60.0	56.5	10.0																						127
Infra structure	Infrastructure Capital - Sustaining	US$M				0.0	4.5	8.0	5.3	10.0	0.0	0.0	13.1	2.5	5.5	17.0	0.0	17.0	0.0	19.0	7.0	8.0	17.0	3.1	0.0		137
	Total Infrastructure Capital	**US$M**	**60.0**	**56.5**	**10.0**	**0.0**	**4.5**	**8.0**	**5.3**	**10.0**	**0.0**	**0.0**	**13.1**	**2.5**	**5.5**	**17.0**	**0.0**	**17.0**	**0.0**	**19.0**	**7.0**	**8.0**	**17.0**	**3.1**	**0.0**		**264**
	G& A Costs	US$M			10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	7.0	4.9	212
G&A and Others	Closure Capital	US$M																							20.0	72.0	92
	Other Liabilities	US$M				4.0	4.0	4.0	3.6	3.0	2.6	3.6	2.9	3.1	2.8	2.6	2.0	1.5	1.7	2.0	2.6	2.6	3.1	1.4	0.8		54
	Total Site Cost		**60.0**	**499.5**	**741.8**	**345.8**	**326.4**	**326.8**	**351.2**	**385.6**	**353.7**	**379.4**	**387.5**	**397.5**	**428.5**	**424.4**	**431.0**	**486.3**	**507.6**	**574.5**	**470.1**	**464.1**	**347.7**	**244.9**	**94.8**	**76.9**	**9,106**
	Gross Revenue (on % payables)	US$M				880.5	893.4	897.1	812.1	670.9	583.5	784.7	638.4	697.7	610.3	581.3	452.3	324.6	374.5	436.3	574.7	582.9	682.5	301.8	168.4		**11,948**
NSR	Selling Cost	US$M				83.3	94.6	100.7	89.5	72.3	55.4	72.7	68.1	74.5	60.2	59.2	45.0	31.6	36.8	43.7	55.2	58.6	69.9	31.1	14.9		**1,217**
	Net Revenue (NSR)	**US$M**				**797.2**	**798.8**	**796.4**	**722.7**	**598.6**	**528.1**	**712.0**	**570.3**	**623.2**	**550.2**	**522.0**	**407.4**	**293.0**	**337.7**	**392.6**	**519.6**	**524.4**	**612.6**	**270.7**	**153.5**		**10,731**
			-60.0	-499.5	-742.4	451.3	472.4	469.6	371.5	213.1	174.4	332.6	182.8	225.7	121.6	97.7	-23.7	-193.2	-169.9	-181.9	49.5	60.2	264.9	25.7	59.3	-76.9	
	Net Cash Flow		**-60.0**	**-499.5**	**-741.8**	**451.3**	**472.4**	**469.6**	**371.5**	**213.1**	**174.4**	**332.6**	**182.8**	**225.7**	**121.6**	**97.7**	**-23.7**	**-193.2**	**-169.9**	**-181.9**	**49.5**	**60.2**	**264.9**	**25.7**	**58.7**	**-76.9**	**1,625**
	Net Cash Flow (accumulated)		- 60	- 559	- 1,301	- 850	- 377	92	464	677	851	1,184	1,366	1,592	1,714	1,811	1,788	1,595	1,425	1,243	1,292	1,353	1,617	1,643	1,702	1,625	

Financial indicators derived from the cash flow are as follow:

- NPV at the beginning of year Y(-3), as a function of the discount rate is as follows:
  - If discount rate is 0%, NPV is 1,625MS$
  - If discount rate is 5%, NPV is 904MUS$
  - If discount rate is 8%, NPV is 617MUS$
  - If discount rate is 10%, NPV is 471MUS$
- Internal Rate of Return: 21.2%
- Years for recovery of the Initial Capital: Year 3
- Total undiscounted net cash flow: 1,625MUS$

AMC notes that in spite of the positive economic KPI's the cash flow given in Table 22.2 exhibits a series of successive negative years from Year 12 to Year 15. These negatives are mostly driven by a strong increase in the unit mining cost which, in turn, is the result of a corresponding increase in haulage cost in those years. Due to the PEA nature of the study, insufficient iterations of the mining schedule were done in order to specifically smooth out the haulage cost. This step should be included in any future mine schedule exercise. Nevertheless, once smoothed, the location of these negatives in the project time line is likely to have an adverse, but not severe, impact on the NPV, which could be offset by metals prices higher than the ones assumed for the study.

22.4 Cash Flow Sensitivity

The cash flow sensitivity has been tested by varying some relevant input parameters such as copper head grades, copper recovery in the mill, and initial capital cost. Table 22.3 gives the sensitivity response as percent variation of financial KPI's.

Table 22.3 Cash Flow Sensitivity

KPI --->		NPV (8%), MUS$		IRR, %		Total Undisc. CF, MUS$	
Parameter Varied		**value**	**% of Base Case**	**value**	**% of Base Case**	**value**	**% of Base Case**
	Base Case	**617**	100%	**21.2**	100%	**1,625**	100%
Copper Recovery	up to 94%	810	131%	24.0	113%	2,063	127%
	down to 85%	406	66%	17.7	83%	1,135	70%
Copper Head Grade	10% higher	1,042	169%	27.1	128%	2,591	159%
	10% lower	220	36%	14.1	67%	712	44%
Initial capital	10% higher	519	84%	18.4	87%	1,494	92%
	10% lower	720	117%	24.4	115%	1,756	108%

Additionally, the cash flow sensitivity has been tested by varying the metals prices according to three prices scenarios, namely:

- Scenario A: the Base Case
- Scenario B: Three year trailing averages for metal prices according to the US Securities and Exchange Commission Guidelines as of November 2012: copper, 3.64 US$/lb; gold, 1,455 US$/oz; silver, 28.06 US$/oz
- Scenario C: copper, 3.0 US$/lb; gold, 1,200 US$/oz; silver, 25 US$/oz

Table 22.4 gives the sensitivity response as percent variation of financial KPI's.

Table 22.4 Cash Flow Sensitivity to Copper price

KPI --->	**NPV, MUS$**			**IRR, %**
Price Scenario	**@ 0% discount rate**	**@ 6% discount rate**	**@ 8% discount rate**	
Scenario A	**1,625**	**796**	**617**	**21.2**
Scenario B	5,579	2,874	2,342	41.9
Scenario C	2,842	1,436	1,148	28.6

23 ADJACENT PROPERTIES

AMC has not verified the information on the following adjacent property. The mineralization on this property is not necessarily indicative of the mineralization on the Property.

23.1 Inguaran Project

The Inguaran Valley Porphyry Copper-Tungsten Project is held by Rome Resources Ltd. (Rome) of Surrey. The Project is located in the Inguaran Valley and consists of concessions that total 10,209 ha. All concessions are wholly owned by Roma Recursos de Mexico, S.A. de C.V., a wholly owned subsidiary of Rome, with the following exceptions that:

- La Verdosa is subject to a 1.0% NSR payable to the Mexican government.
- Concepcion and San Jose are under option.

Rome’s Inguaran Valley Porphyry Copper-Tungsten Project surrounds, but does not include the Inguaran Copper Mine. The Inguaran Copper Mine was mined by block- cave and Wowas operated by ASARCO from the 1970s to early 1980s. Production reported was about 10 Mt averaging 1.2% Cu.

A diamond drilling program in December 2004 of 12 holes, totalled 2,681 m, and eleven trenches totaled 350 m. Results showed some encouraging copper grades at reasonable thicknesses.

A 3,000 m diamond drilling program from November 2006 to January 2007 mainly tested geochemical and geophysical anomalies in the El Toro and Esmeralda areas in the northeastern portions of the area held by Rome. Results indicated mainly strong copper veins in the El Toro area, which are considered characteristic of mineralization peripheral to major porphyry mineralization.

24 OTHER RELEVANT DATA AND INFORMATION

There is no additional information or explanation required to make the report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

The present study indicates that La Verde is an attractive mining opportunity. The key financial indicators, based on reasonable future copper prices and capital and operating cost estimates, justify advancing the Property to the next stage.

There are, however, key areas of risk or uncertainty that need to be addressed in moving the Project forward:

- Additional drilling to improve the robustness of the geological model and upgrade the substantial component of Inferred Resources to Indicated or Measured Resources
- Management of high arsenic grades and arsenic byproducts, both from an economic and an environmental point of view.
- Availability of reliable and convenient power and water sources.
- Identification of TMF locations for the substantial projected LOM shortfall in tailings dam capacity.
- Acquisition of the land required for TMF locations.
- Social license. The initiation of systematic social and environmental assessment of the water access, infrastructure and tailings disposal assumptions contained in the study is required. Continued development of good relationships with the local stakeholders, particularly with the nearby communities is required.

26 RECOMMENDATIONS

1. Advance the Project to a Pre-Feasibility level.
2. Initiate formal discussions with the relevant Mexican government agencies to investigate the feasibility of the proposed TMF solution outlined in this study.
3. Initiate formal discussions with the relevant Mexican government agencies to investigate a reliable power supply for the power consumption levels outlined in this study.
4. Strengthen Company involvement with local communities and initiate conversation with the relevant stakeholders regarding the extension and considerations of the project. Particularly relevant is the community engagement respect the project's water and land uses.
5. Initiate environmental and social baseline studies. On the environmental side, include the potential for contamination by arsenic from waste dumps and stockpiles from the earliest evaluation stages. See Table 26.1 for cost estimate.
6. In the next mine planning exercise, investigate options to deplete West Hill pit earlier to provide an opportunity for backfilling the pit with tailings or waste material. For costing purposes, it is included in the PFS.
7. Initiate a program investigating the geo-metallurgical variability across the deposits to develop a geo-metallurgical map and appropriate geo-metallurgical domains. See Table 26.1 for cost estimate.
8. Investigate options to improve precious metals recovery, especially gold in West Hill sulphides. See Table 26.1 for cost estimate.
9. Undertake a trade-off study on the use of high pressure grinding rolls (HPGR). See Table 26.1 for cost estimate.
10. Carry out concentrate roaster testwork and associated preliminary design studies. See Table 26.1.
11. With respect to pit wall planning:
    e. Undertake a geotechnical drilling program using orientated, NQ3 or HQ3 diamond drill core.
    f. Develop a geotechnical and structural model at pit scale to help with the modelling of the pit wall stability.
    g. Conduct a hydrogeological study to assess the presence, nature, and depth of the water table, and how this may be affected by mining.
    h. Undertake an assessment of the rock mass strength and carry out a slope stability analysis at bench, inter-ramp and overall scale to assess the slope parameters for each pit.

Table 26.1 Estimated Cost of Recommendations

Activity	Cost Estimate (C$m)
Pre-Feasibility study	1.0
Additional drilling to upgrade Resources + first Geomechanic studies	2.9
Environmental and social baseline studies	1.1
Geo-metallurgical investigations	0.8
HPGR Testing	0.1
Concentrate Roasting testwork	0.2

27 REFERENCES

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28 CERTIFICATES

P R Stephenson

1 I, Patrick R Stephenson, P.Geo., BSc (Hons), FAusIMM (CP), MAIG, MCIM, of Vancouver, British Columbia, do hereby certify that I am General Manager and a Principal Geologist with AMC Mining Consultants (Canada) Limited, Suite 202 – 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2 I graduated with a BSc (Hons) in Geology from Aberdeen University in Scotland in 1971.

3 I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia.

4 I have worked as a Geologist and Manager for a total of 40 years since my graduation from university and have relevant experience in geology, exploration and mineral resource estimation for base and precious metal deposits and in public reporting of mineral assets.

5 I have read the definition of “Qualified Person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

6 I am responsible for the preparation of Sections 1 to 5, 15, 16, and 19 to 26 of the “La Verde Copper Project Michoacán State, Mexico, Technical Report for Catalyst Copper Corp”, dated effective 30 September 2012 (the “Technical Report).

7 I have not visited the Property.

8 I am independent of Minera Hill 29 SA de CV and Catalyst Copper Corp as defined by Section 1.5 of the Instrument.

9 I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

10 As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 17 January 2013

Original signed and sealed by

Patrick R Stephenson

A Riles

1. I, Alan Riles, MAIG, BMet (Hons), Grad Dipl Business Management, do hereby certify that I am Associate Principal Consultant Metallurgist with AMC Mining Consultants (Canada) Ltd, Suite 202 – 200 Granville Street, Vancouver, British Columbia V6C 1S4, Canada.
2. I graduated with a Bachelor of Metallurgy (Hons Class 1) from Sheffield University, UK in 1974.
3. I am a registered member of the Australian Institute of Geoscientists.
4. I have practiced my profession continuously since 1974, with particular experience in study management, and both operational and project experience in precious and base metal deposits.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Sections 13, 17 and 19, parts of 18 to 21, and contributed to Section 25 of the "La Verde Copper Project Michoacán State, Mexico, Technical Report for Catalyst Copper Corp", dated effective 30 September 2012 (the "Technical Report).
7. I visited the Property from 11–12 April 2012.
8. I am independent of Minera Hill 29 SA de CV and Catalyst Copper Corp as defined by Section 1.5 of the Instrument.
9. I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
10. As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 17 January 2013

Original signed and sealed by

Alan Riles

M Molavi

1 I, Mo Molavi, P.Eng., M Eng, B Eng, do hereby certify that I am a Principal Mining Engineer with AMC Mining Consultants (Canada) Limited, Suite 202 – 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2 I graduated with a B Eng in Mining Engineering from the Laurentian University in Sudbury Ontario in 1979, and an M Eng in Mining Engineering specializing in Rock Mechanics and mining methods from the McGill University of Montreal in 1987.

3 I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of Saskatchewan and a Member of the Canadian Institute of Mining, Metallurgy and Petroleum.

4 I have worked as a Mining Engineer for a total of 30 years since my graduation from university and have relevant experience in project management, feasibility studies and technical report preparations for mining projects in North America.

5 I have read the definition of “Qualified Person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

6 I am responsible for the preparation of Section 18 of the “La Verde Copper Project Michoacán State, Mexico, Technical Report for Catalyst Copper Corp”, dated effective 30 September 2012 (the “Technical Report).

7 I have not visited the Property.

8 I am independent of Minera Hill 29 SA de CV and Catalyst Copper Corp as defined by Section 1.5 of the Instrument.

9 I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

10 As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 17 January 2013

Original signed and sealed by

Mo Molavi

Michael F. O'Brien, MSc, PR.SCI.NAT., FGSSA, FAUSIMM, FSAIMM
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1 I, Michael F. O'Brien, MSc, Pr.Sci.Nat., GSSA, FAusIMM, FSAIMM, do hereby certify that I am Chief Geologist for Tetra Tech WEI Inc., 800 – 555 West Hastings Street, Vancouver, British Columbia V6B 1M1.

2 I graduated with a BSc (HONS) in Geology from the University of Natal, South Africa in 1978.

3 I am a registered Professional Natural Scientist (Geological Scientist) in good standing of the South African Council for Natural Scientific Professions (South Africa, 400295/87).

4 I have practiced my profession continuously for 33 years. My experience is in operations and mineral project assessment, and I have the experience relevant to Mineral Resource estimation of metal deposits.

5 I certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101 under the Accepted Foreign Associations and Membership Designations (Appendix A).

6 I am responsible for the preparation of Section 14 for the "La Verde Copper Project Michoacán State, Mexico, Technical report for Catalyst Copper Corp", dated effective 30 September 2012 (Technical Report).

7 I am independent of Minera Hill 29 SA de CV as defined by Section 1.5 of the Instrument.

8 I have not conducted a personal inspection of the Property and I have no prior involvement with the Property that is the subject of the Technical Report.

9 I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

10 As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th day of January, 2013

Original signed and sealed by

Michael F. O'Brien, M.Sc., Pr.Sci.Nat.,
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1 I, **Margaret Harder, MSc, P.Geo.**, do hereby certify that I am a Geologist with Tetra Tech WEI Inc., 800 – 555 West Hastings Street, Vancouver, British Columbia V6B 1M1.

2 I graduated with a BSc in Geology from the University of Saskatchewan in 2002, and an MSc in Geology from the University of British Columbia in 2004.

3 I am a registered member of the the Association of Professional Engineers and Geoscientists of British Columbia (#32139).

4 I have practiced my profession in geology for a total of eight years with experience in exploration, advanced evaluation, operations, resource estimation, and technical report writing

5 I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6 I am responsible for the preparation of Sections 6 to 12 and 14 of the "La Verde Copper Project Michoacán State, Mexico, Technical report for Catalyst Copper Corp", dated effective 30 September 2012 (the "Technical Report).

7 My most recent personal inspection of the Property was 21 August 2012.

8 I am independent of Minera Hill 29 SA de CV as defined by Section 1.5 of the Instrument.

9 I have no prior involvement with the Property that is the subject of the Technical Report.

10 I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

11 As at the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th day of January, 2013

Original signed and sealed by

Margaret Harder, MSc, P.Geo.

Mat Hal

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