M3-PN 110127

MacArthur Copper Project

Amended NI 43-101 Technical Report

Preliminary Economic Assessment

Lyon County, Nevada, USA

Issue Date: 17 January 2014
Effective Date: 23 May 2012
Prepared For:

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DATE AND SIGNATURES PAGE

The Qualified Persons contributing to this report are noted below. The Certificates and Consent forms of the qualified persons are located in Appendix A, Certificate of Qualified Persons ("QP") and Consent of Authors.

- Mr. Myron R. Henderson. P.E.; Project Manager with M3 Engineering & Technology Corporation; principal author of this technical report and responsible for Sections 1 through 3, Sections 17 through 19, Sections 21 and 22, and Sections 24 through 26.
- Dr. Rex C. Bryan, Ph.D.; Senior Geostatistician with Tetra Tech MM, Inc.; responsible for Sections 4 through 12, Section 14, Section 15, and Section 23.
- Mr. Herbert E. Welhener, MMSA-QPM; Vice President of Independent Mining Consultants, Inc.; responsible for Section 16 – Mining Methods.
- Dr. Richard W. Jolk, P.E., Ph.D., Principal Mine Engineer, Metallurgical Engineer, and Certified Minerals Appraiser with Tetra Tech MM, Inc.; responsible for Section 13 - Mineral Processing and Metallurgical Testing.
- Mr. Mark A. Willow, M.Sc., SME-RM, Practice Leader with SRK Consulting (U.S.), Inc.; responsible for Section 20 – Environmental Studies, Permitting and Social or Community Impact.

This Technical Report is current as of May 23, 2012.

(Signed) "Myron R. Henderson"	January 17, 2014
Myron R. Henderson, P.E.	Date
(Signed) "Rex C. Bryan"	January 17, 2014
Rex C. Bryan, Ph.D.	Date
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Richard W. Jolk, P.E., Ph.D.	Date
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Note: This Amended Report was issued on 17 January 2014 with the following edits:

1. **The report's title was changed to "Amended Preliminary Economic Assessment."**
2. **All disclaimers not permitted by item 3 of the Form were removed from Section 3**
3. **Amended Section 23 to include SGS properties.**
4. **Cautionary language was added to Section 22 indicating that mineral resources that are not mineral reserves do not have demonstrated economic viability.**
5. **Amended Sections 6 and 23 to include information required by s.2.4 of NI 43-101.**
6. **The use of the term "ore" was modified to indicate "mineralized material," to clarify that the project does not currently have mineral reserves.**
7. **Mark Willow's Certificate of Qualified Person was amended to clarify his qualifications.**
8. **Qualified Person assignment for Sections 15 and 23 was clarified.**
9. **Unnecessary Appendices were removed.**

These modifications did not change project economics.

TABLE OF CONTENTS

SECTION **PAGE**

MACARTHUR COPPER PROJECT

FORM 43-101F1 PRELIMINARY ECONOMIC ASSESSMENT

LIST OF FIGURES AND ILLUSTRATIONS

FIGURE DESCRIPTION PAGE

LIST OF TABLES

TABLE DESCRIPTION **PAGE**

LIST OF APPENDICES

1 SUMMARY

Quaterra Alaska, Inc. (Quaterra), a wholly owned subsidiary of Quaterra Resources, Inc. commissioned M3 Engineering and Technology Corporation (M3) to prepare a Canadian National Instrument 43-101 (NI 43-101) compliant Preliminary Economic Assessment (PEA) for the MacArthur Copper Project in Lyon County, Nevada. Tetra Tech Inc. (Tt) and Independent Mining Consultants, Inc. (IMC) prepared several sections of the PEA. This report includes an update of the January 2011 MacArthur technical report and reflects changes to the resource estimate as a result of the 2011 exploration drilling and continued geologic investigations. This amended report replaces and supersedes the previous PEA for the MacArthur Copper Project in its entirety. The previous PEA was filed on the SEDAR website on June 29, 2012 and had an effective date of May 23, 2012. The effective date of this report remains as of May 23, 2012.

The Qualified Person for Sections 4 through 12, 14, 15 and 23 of this report is Mr. Rex Bryan, Ph.D., Senior Geostatistician for Tetra Tech, Golden Colorado. The Qualified Person for Section 13 of this report is Mr. Richard W. Jolk, P.E., PhD, Principal Minerals Engineer for Tetra Tech, Golden Colorado. The Qualified Person for Section 16 of this report is Mr. Herb Welhener, Principal Mining Engineer for Independent Mining Consultants, Inc., Tucson, Arizona.

The MacArthur Copper Property is located near the geographic center of Lyon County, Nevada, USA along the northeastern flank of the Singatse Range approximately seven miles northwest of the town of Yerington, Nevada. The property is accessible from Yerington by approximately five miles of paved roads and two miles of maintained gravel road. Topographic coverage is on US Geological Survey “Mason Butte” and “Lincoln Flat” 7.5’ topographic quadrangles. The nearest major city is Reno, Nevada approximately 75 miles to the northwest.

The Preliminary Economic Assessment within this Technical Report is based upon the oxide / chalcocite portion of the updated resource. This oxide / chalcocite portion includes a measured and indicated resource of 159.1 million tons averaging 0.21% Cu (percent total copper or TCu) containing 676 million pounds of copper at a 0.12% Cu cutoff and an inferred resource of 243 million tons averaging 0.20% Cu at a 0.12% Cu cutoff containing 980 million pounds of copper.

This PEA is preliminary in nature and includes discussion of mineral resources including inferred mineral resources that are too speculative geologically to have economic considerations applied to them. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

A future opportunity exists to integrate the residuals from the Quaterra Resources 2011 acquisition of the historic neighboring Yerington copper mine, which could provide a positive impact on the economics of the MacArthur Project. The residuals consist of oxide-copper bearing sub-grade material representing stripped material from the Yerington mine, vat leach tailings representing oxide tailings from copper oxide vat leaching, and partially leached tailings and mineralized material previously mined by Arimetco.

1.1 PROPERTY DESCRIPTION AND OWNERSHIP

The MacArthur Copper Property is located near the geographic center of Lyon County, Nevada, USA along the northeastern flank of the Singatse Range approximately seven miles northwest of the town of Yerington, Nevada. The property is accessible from Yerington by approximately five miles of paved roads and two miles of Lyon County maintained gravel road. Topographic coverage is on US Geological Survey “Mason Butte” and “Lincoln Flat” 7.5’ topographic quadrangles. The nearest major city is Reno, Nevada approximately 75 miles to the northwest.

The property consists of 470 unpatented lode claims totaling approximately 9700 acres on lands administered by the US Department of Interior - Bureau of Land Management (BLM). All required annual payments to the BLM and Lyon County have been paid in a timely manner and the claims are current.

1.2 HISTORY

Over the history of the project, previous operators have contributed more than 300 holes to the current drill hole database. Table 1-1 summarizes the exploration history of the MacArthur area. Of the historic holes, 280 of those holes drilled by the Anaconda Company (Anaconda) during 1972-73 have been deemed acceptable under NI 43-101 standards and have been used during the resource estimation.

Table 1-1: Exploration Drilling History

MACARTHUR PROJECT February 2009			
Operator	**Drill Program Date Range**	**Number of Holes Drilled**	**Feet Drilled**
U.S. Bureau of Mines	1947-50	8	3,414
Anaconda Company	1955-57	14	3,690
Bear Creek Mining Company	1963-??	~14	Unknown
Superior Oil Company	1967-68	11	13,116
Anaconda Company	1972-73	280	55,809
Pangea Explorations, Inc.	1987-1991	15	2,110
Arimetco International, Inc.	Unknown	Unknown	Unknown
Total		**~342**	**~78,139**

1.3 GEOLOGY AND MINERALIZATION

The MacArthur property is one of several copper deposits and prospects located near the town of Yerington that collectively comprise the Yerington Mining District. The property is underlain by Middle Jurassic granodiorite and quartz monzonite intruded by west-northwesterly-trending, moderate to steeply north-dipping quartz porphyry dike swarms. These dikes host a large portion of the primary copper mineralization at the nearby Yerington mine and are associated with all porphyry copper occurrences in the district.

The MacArthur copper deposit consists of a 50-150 foot thick, tabular zone of secondary copper (oxides and/or chalcocite) covering an area of approximately two square miles. This mineralized zone has yet to be fully delineated and remains open to the west and north. Limited drilling has also intersected underlying primary copper mineralization open to the north, but only partially tested to the west and east.

Oxide copper mineralization is most abundant and particularly well exposed in the walls of the MacArthur pit. The most common copper mineral is chrysocolla; also present is black copper wad (neotocite) and trace cuprite and tenorite. The flat-lying zones of oxide copper mirror topography, exhibit strong fracture control and range in thickness from 50 to 100 feet. Secondary chalcocite mineralization forms a blanket up to 50 feet or more in thickness that is mixed with and underlies the oxide copper. Primary chalcopyrite mineralization has been intersected in several locations mixed with and below the chalcocite. The extent of the primary copper is unknown as many of the drill holes bottomed at 400 feet or less. The primary copper is currently not included in the mine plan for the PEA.

The MacArthur deposit is part of a large, partially defined porphyry copper system that has been complicated by complex faulting and post-mineral tilting. Events leading to the current geometry and distribution of known mineralization include 1) Middle Jurassic emplacement of primary porphyry copper mineralization by quartz monzonite dikes intruding the Yerington Batholith; 2) Late Tertiary westward tilting of the porphyry deposit 60-90° by Basin and Range extensional faulting; 3) secondary (supergene) enrichment resulting in the formation of a widespread, tabular zone of secondary chalcocite mineralization below outcrops of oxidized rocks called leached cap; 4) oxidation of outcropping and near-surface parts of this chalcocite blanket, as well as oxidation of the primary porphyry sulfide system.

1.3.1 Geophysics

Quaterra contracted three surveys at the MacArthur Project in 2011 and 2012. A borehole geophysical survey and a surface IP/resistivity (IPR) survey were carried out by Zonge International in 2011, and a detailed helicopter magnetic survey was flown by Geosolutions Pty. Ltd. in 2012. These surveys supplement previous geophysical work on the property that includes: a 2009 IPR survey carried out by Zonge; a 2007 helicopter magnetic survey carried out by EDCON-PRJ; a series of historic aeromagnetic surveys (1966 to 1975) available in analog form from the Anaconda Archives; and a series of historic IPR surveys (1963 – 1964) carried out by Kennecott Exploration Services/Bear Creek Mining Company and Superior Oil.

The mineralized system at MacArthur has an anomalous IP and resistivity response first detected in the Kennecott and Superior Oil IPR surveys in the 1960's. The Quaterra 2009 and 2011 IPR surveys confirmed the reliability of the earlier surveys and further defined the depth extent of the IP anomalies. The 2009 and 2011 Quaterra surveys confirmed that the 1963-64 Kennecott data is of good quality and is useful for mapping anomalous IP zones within the upper 1,000-1,200 feet from the surface. Below this depth, the older data cannot effectively resolve the bottom of the IP anomalies nor determine if any of the anomalies extend to great depths.

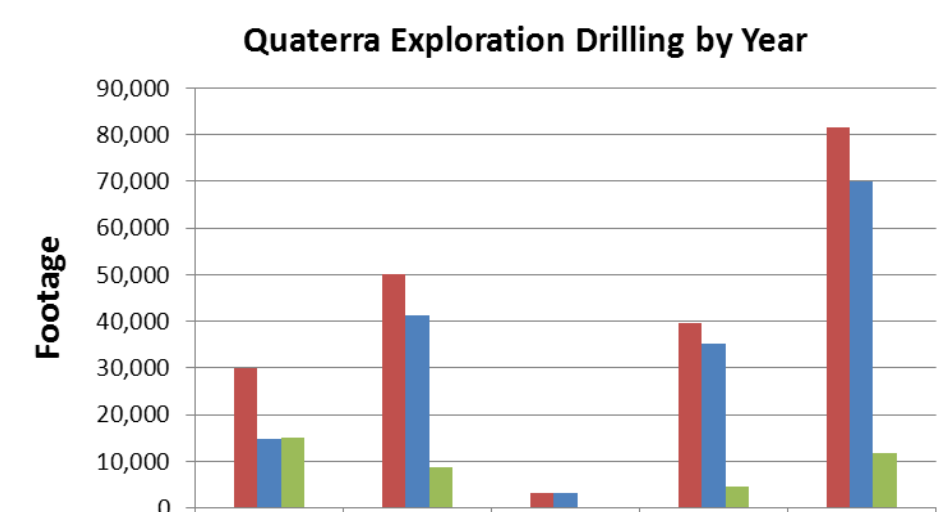
The 2009 and 2011 data sets show this increased depth of exploration is important. Portions of the IP response are flat lying with limited depth extent. However, both the 2009 and 2011 surveys have identified anomalous IP responses with depth extent in excess of 2000 feet and possibly feeder zones of the near surface zones. In 2011, two borehole IP surveys were run that demonstrate Quaterra's ability to explore for deep sulfide responses below the depth of exploration of surface techniques. The modern data maps subtle low resistivity features which are interpreted to be porphyry alteration systems and have identified anomalous IP responses that extend under post-mineral volcanic cover to the north and west of the main MacArthur system. These buried anomalies are high priority drill targets.

Two high resolution helicopter magnetic surveys were flown over the MacArthur Project in 2007 (EDCON-PRJ) and 2012 (Geosolutions). The modern, high resolution data has a broad frequency bandwidth and will be used for 3D modeling and exploring beneath the magnetic volcanic cover.

1.4 EXPLORATION STATUS

1.4.1 Exploration Drilling Program

Quaterra has completed 204,656 feet of drilling in 401 holes since beginning drilling in 2007. Core holes total 40,233 feet in 58 holes and reverse circulation holes total 164,423 feet in 343 holes. (Note that one previously listed, but abandoned 115 foot drill hole, has now been removed from the database and reported totals). Figure 1-1 show Quaterra's yearly exploration drilling footage by year.



	2007	2008	2009	2010	2011
Total Drilling	29,921	50,216	3,210	39,658	81,651
RC Drilling	14,790	41,425	3,210	35,108	69,890
Core Drilling	15,131	8,791	0	4,550	11,761

Figure 1-1: Quaterra Exploration Drilling by Year

Quaterra's initial objective was to verify and expand the MacArthur oxide resource as had been defined by the 1972-1973 Anaconda drilling program and, importantly, to follow up chalcocite intercepts in several Anaconda holes as well as in a few outlying early 1960's holes drilled by Bear Creek Mining Company, in late 1960's drilling by Superior Oil, and holes drilled by the US Bureau of Mines in 1950. Quaterra's drilling through 2010 successfully expanded the oxide mineralization outbound from the MacArthur pit and encountered a widespread, underlying tabular blanket of mixed oxide-chalcocite mineralization as well as primary copper intercepts that remain incompletely tested.

During 2011, exploration and infill drilling focused north of the MacArthur pit where earlier drilling encountered better grades of oxide and chalcocite mineralization. Holes were angled both southerly and northerly to test high angle fractures common in the west-northwest structural grain. Strong chalcocite and chalcopyrite mineralization was intersected in several holes in the North Ridge zone including QM-183: 1.37% Cu over 40 feet and QM-187: 1.66% Cu over 90'. These results were followed by a tightened drill spacing from 500 feet to 250 feet over an approximate 2,500 feet by 2,500 feet area north of the MacArthur pit, forming the basis for the 2011 resource.

Deep drilling north of the North Ridge Zone intersected significant primary sulfide mineralization grading 1.32% Cu over 64 feet in hole QM-164 which is open to the north and partially open to the west and east. Although the mineralization at MacArthur has yet to be completely closed off to the west and north, the 2011 drilling program expanded and in-filled earlier drill results and defined the footprint for the mineral resource estimation published in this document.

1.5 RESOURCE ESTIMATE

An updated mineral resource estimate has been generated using drill hole sample assays results and the interpretation of a geologic model which relates to the spatial distribution of copper in the MacArthur deposit. Interpolation characteristics have been defined based on geology, drill hole spacing and geostatistical analysis of the data.

This PEA is preliminary in nature and includes discussion of mineral resources including inferred mineral resources that are too speculative geologically to have economic considerations applied to them. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized

1.5.1 Block Model Definition

The block model parameters for MacArthur were defined to best reflect both the drill hole spacing and current geologic model. Table 1-2 shows the block model parameters used for the 2011 estimates.

Table 1-2: MacArthur Model Parameters

MacArthur East Model Parameters	X (Columns)	Y (Rows)	Z (Levels)
Origin (lower left corner):	2,429,300	14,685,800	2,800
Block size (feet)	25	25	20
Number of Blocks	548	400	150
Rotation	0 degrees azimuth from North to left boundary		
Composite Length	10 feet (Zone)		

1.5.2 Assay Database

An Excel database provided by Quaterra contained the pertinent drill hole and assay information for the MacArthur Copper deposit. The database contained 737 drill holes of which 676 drill holes from Quaterra and Anaconda (sometimes referred to as the Metech holes) were used. The 61 holes removed included holes with limited or no information on the assays (Pangea Gold 1991, Superior, USBM 1952, Anaconda 1955-57), and six Quaterra holes outside the model limits. Of the 676 holes used, there are 280 Anaconda (Metech) RC holes and 396 Quaterra holes (58 core and 338 RC holes). These drill holes traversed 257,895 feet, producing 51,258 total copper sample assay values at a nominal five feet in length.

A total of 151 drill holes totaling 80,800 feet were added to the database used for the resource estimation. These included two holes for which data was unavailable at the time of the last estimate, but did not include three 2011 holes which were outside the model limits.

The variables available in the database are for total copper from Quaterra and Anaconda intervals, and acid-soluble copper, a limited number of ferric sulfate soluble (QLT) copper assays and a very limited number of cyanide leach copper assays from Quaterra holes.

1.5.3 Compositing

The assay data was composited using a 10-foot "zone method". The zone method is a variant of down hole compositing, with the distinction that the composite begins as the drill interval enters a rock code zone. This method tends to reduce averaging composites across zones. The process first used DataMine® to assign a MinZone to each 25x25x20-foot block within the model specified in Table 1-3. When the majority of a block fell within the interpreted MinZone wireframe it was assigned the appropriate code. These coded blocks were then imported into MicroModel® and used to "back-mark" each composite using a simple majority rule. No capping was applied. Table 1-3 presents the MinZone codes used in the model. Initial codes of alluvium, oxide, oxide and chalcocite mix, and sulfide were 10, 20 and 30 respectively. These codes were altered by the addition of 1 if the assays, composites or blocks fell within a 0.12% Cu grade envelope predicted by indicator kriging. The codes were also altered by the addition of 100 if the data was within a modeled dike.

Table 1-3: MinZone Codes and Density

MinZone Code	Description	Density (cu.ft/ton)
0	Air and previously mined pit	Air (0) and Mined (12.5)
5, 6, 105, 106	Alluvium	12.5
10, 11, 110, 111	Oxide zone	12.5
20, 21, 120, 121	Chalcocite mix zone	12.5
30, 31, 130, 131	Sulfide zone	12.5
9999	Undefined	12.5

1.5.4 Geostatistical Analysis and Variography

A total of twenty-two (21 directional and an omni-directional) variograms were calculated using MicroModel® for each MinZone within each area. The program searches along each direction for data pairs within a 12.5-degree window angle and 5-feet tolerance band. All experimental variograms are inspected so that spatial continuity along a primary, secondary and tertiary direction can be modeled.

Each variogram model was then validated using the "jackknifing" method. This method sequentially removes values and then uses the remaining composites to krige the missing value using the proposed variogram.

1.5.5 Kriging and Resource Classification

Table 1-4 presents the search and kriging parameters employed in the resource model. The composite and block codes were used to determine which composites were selected to estimate a particular block.

Table 1-4: Kriging and Search Parameters

Matching Codes			Anisotropy				MIF Search Ranges[5]						Variogram Parameters			
Composite Code	Block Codes	Zone Name	Axis	Anisotropy Axis Length (m)	Anisotropy Rotation	Type[3]	Resource Class[5]	Pass[8]	Resource Code[2]	Maximum Search Range	MaxPts / Sector / Pts Single Drillhole	Min Pts Required to Estimate	Rotation[6]	Nested	Model Type[4]	Parameters[7]
0, 1	0, 1	Indicator > 0.12 Grade	Primary	300	note 6	Az	na	1	na	500	5/2	8	note 6	model 1	Sph	note 7
			Second	400	note 6	Dip	na	na	na	na	na	na	note 6	model 2	Sph	note 7
			Tertiary	100	0	Tilt	na	na	na	na	na	na	0	model 3	Sph	note 7
10	10, 110	< 0.12 Grade Oxide	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	note 6	Dip	I	2	2	260	4/2	6	note 6	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
20	20, 120	< 0.12 Grade Mixed	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	note 6	Dip	I	2	2	260	4/2	6	note 6	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
30	30, 130	< 0.12 Grade Sulfide	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	45	Dip	I	2	2	260	4/2	6	45	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
11	11, 111	>= 0.12 Grade Oxide	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	note 6	Dip	I	2	2	260	4/2	6	note 6	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
21	21, 121	>= 0.12 Grade Mixed	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	note 6	Dip	I	2	2	260	4/2	6	note 6	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
31	31, 131	>= 0.12 Grade Sulfide	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	45	Dip	I	2	2	260	4/2	6	45	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
110, 111, 120, 121, 130, 131	110, 111, 120, 121, 130, 131	Dike[9]	Primary	na	note 6	Az	na	na	na	500	5/2	16	note 6	model 1	Sph	note 7
			Second	na'	-60	Dip	na	na	na	na	4/2	na	-60	model 2	Sph	note 7
			Tertiary	na	0	Tilt	na	na	na	na	4/2	na	0	model 3	Sph	note 7

All measurements in feet, all directions in degrees azimuth

Untized Spherical Variogram Structural Parmeters: C0 = 0.140; Sill1 = 0.300; Sill2 = 0.140; Sill3 = 0.42

Cu estimate is done in three passes (measured is first pass, indicated is second and inferred is third), a kriging error adjust is use post-kriging

Notes:
1 Indictor grade envelop IK uses absolute variogram; Copper grade OK uses Unified Relative Variogram converted from correlograms.
2 Kriging Error is used to adjust preliminary class 1,2,3 to 1,2,3 & 4 by post-kriging filter at 0.75 Maximum Kriging Error
3 Az=Azimuth is clockwise (CW) from North, Dip is positive when downward, Tilt rotates CW around primary axis.
4 Sph=Spherical, Lin=Linear, Exp=Exponential, Gau=Gaussian
5 MIF is the accronym for M=Measured, I=Indicated, F=Inferred
6 Dynamic Kriging (Zone 10,11,20,21 rotation and dip defined for each block "on-the-fly" -- see figure 1; Zone 30, 31 rotation "on-the-fly")
7 Dynamic Kriging (Variogram range parameters defined for each block "on-the-fly")
8 Pass: Estimation is done sequentially from shortest search range to largest. Previously estimated blocks are not overwritten.
9 Dike Estimation: Dikes are estimated first with non-dyke codes and then overwritten with Dike coded data. Any prior MIF codes are retained.

Tt used a two-part approach to classify the total copper resources. This approach takes into account the spatial distribution of the drilling, the distance to the nearest data points used to estimate a block, and finally the relative kriging error generated by the estimate. Tt has found this approach to be very robust and provide highly reproducible results. The following points detail this approach:

1. A measured block requires 16 samples, with a maximum of five samples per sector in a six sector search pattern and a maximum of 2 composites coming from a single drill hole. This implies that in most cases, for a block to be classified as measured there must be at least 8 drill holes in four cardinal directions.

2. The constraints for an indicated block are not as stringent for a measured block. An indicated block requires a minimum of 6 samples, with a maximum of 4 samples per sector in a sector search pattern and a maximum number of 2 samples coming from a single drill hole. This implies that for most cases an indicated block must have at least 3 drill holes in three of the four cardinal directions.

3. Relaxing the constraints even more, a inferred block requires a minimum of 2 samples, with a minimum of 2 samples per sector in a sector search pattern and a maximum of 2 composites from a single drill hole. This implies that an inferred block must have a least one drill hole from one of the four cardinal directions.

In addition to the search parameters, kriging error comes into play when determining if a block falls into a particular class. Tt has found that by plotting the kriging error as a log-probability plot, there is a natural break in the distribution which signifies when the error is too great to allow a block to be classified as measured or indicated. In the case of the MacArthur deposit, any block with a kriging error of 0.75 or greater was classified as inferred.

1.5.6 Estimated Resources

Table 1-5 and Table 1-6 present the measured and indicated resources, and Table 1-8 presents the inferred resources. The base case cutoff grade for the leachable resource is 0.12% Cu (or TCu) while the base case cutoff grade for the primary sulfide resources is 0.15% Cu. Both of these values are representative of actual operating cutoff grades in use as of the date of this report. It is Tt’s opinion that the MacArthur Mineral Resources meet current CIM definitions for classified resources.

This PEA is preliminary in nature and includes discussion of mineral resources including inferred mineral resources that are too speculative geologically to have economic considerations applied to them. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

Table 1-5: Measured Copper Resources

MEASURED RESOURCES MACARTHUR COPPER PROJECT –YERINGTON, NEVADA May 2012				
	Cutoff Grade %TCu	**Tons (x1000)**	**Average Grade %TCu**	**Contained Copper (lbs x 1000)**
Oxide and Chalcocite Material (MinZone 10 and 20)	0.5	1,444	0.675	19,491
	0.4	3,196	0.548	35,041
	0.35	5,074	0.483	49,025
	0.3	8,633	0.417	71,930
	0.25	15,929	0.35	111,599
	0.2	33,472	0.283	189,518
	0.18	43,753	0.261	228,566
	0.15	58,388	0.237	276,993
	0.12	**71,829**	**0.218**	**313,174**
Primary Material (MinZone 30)	0.5			
	0.4			
	0.35			
	0.3			
	0.25			
	0.2			
	0.18			
	0.15	N/A	N/A	N/A

Table 1-6: Indicated Copper Resources

INDICATED COPPER RESOURCES MACARTHUR COPPER PROJECT –YERINGTON, NEVADA May 2012				
	Cutoff Grade %TCu	**Tons (x1000)**	**Average Grade %TCu**	**Contained Copper (lbs x 1000)**
Oxide and Chalcocite Material (MinZone 10 and 20)	0.5	1,957	0.753	29,484
	0.4	3,533	0.615	43,442
	0.35	5,018	0.543	54,516
	0.3	7,618	0.468	71,259
	0.25	13,930	0.379	105,478
	0.2	31,949	0.29	185,049
	0.18	45,554	0.26	236,607
	0.15	67,271	0.229	308,639
	0.12	**87,264**	**0.208**	**362,320**
Primary Material (MinZone 30)	0.5	98	0.72	1,411
	0.4	193	0.586	2,263
	0.35	273	0.523	2,857
	0.3	354	0.478	3,382
	0.25	507	0.416	4,216
	0.2	670	0.369	4,938
	0.18	796	0.34	5,414
	0.15	**1,098**	**0.292**	**6,408**

Table 1-7: Measured + Indicated Copper Resource

MEASURED + INDICATED RESOURCES MACARTHUR COPPER PROJECT –YERINGTON, NEVADA May 2012				
	Cutoff Grade %TCu	**Tons (x1000)**	**Average Grade %TCu**	**Contained Copper (lbs x 1000)**
Oxide and Chalcocite Material (MinZone 10 and 20)	0.5	3,401	0.72	48,974
	0.4	6,730	0.583	78,485
	0.35	10,092	0.513	103,544
	0.3	16,251	0.441	143,171
	0.25	29,859	0.364	217,075
	0.2	65,421	0.286	374,601
	0.18	89,306	0.26	465,106
	0.15	125,659	0.233	585,822
	0.12	**159,094**	**0.212**	**675,513**
Primary Material (MinZone 30)	0.5	98	0.72	1,411
	0.4	193	0.586	2,263
	0.35	273	0.523	2,857
	0.3	354	0.478	3,382
	0.25	507	0.416	4,216
	0.2	670	0.369	4,938
	0.18	796	0.34	5,414
	0.15	**1,098**	**0.292**	**6,408**

Table 1-8: Inferred Copper Resources

INFERRED COPPER RESOURCES MACARTHUR COPPER PROJECT –YERINGTON, NEVADA May 2012				
	Cutoff Grade %TCu	**Tons (x1000)**	**Average Grade %TCu**	**Contained Copper (lbs x 1000)**
Oxide and Chalcocite Material (MinZone 10 and 20)	0.50	4,294	0.657	56,423
	0.40	9,656	0.538	103,899
	0.35	15,357	0.477	146,444
	0.30	25,851	0.414	213,788
	0.25	43,695	0.356	311,108
	0.20	82,610	0.293	483,929
	0.18	109,920	0.267	587,412
	0.15	166,930	0.232	774,889
	0.12	**243,417**	**0.201**	**979,510**
Primary Material (MinZone 30)	0.50	10,644	0.819	174,413
	0.40	18,442	0.653	240,742
	0.35	23,316	0.594	277,181
	0.30	33,831	0.511	345,415
	0.25	53,060	0.423	449,312
	0.20	89,350	0.341	609,188
	0.18	101,375	0.323	654,680
	0.15	**134,900**	**0.283**	**764,074**

1.6 METALLURGY

Considering both recent and historical test work, along with information from previous mining operations at the MacArthur site, the design basis for this PEA considers a ROM heap leach operation with processing of the pregnant leach solution (PLS) through traditional solvent extraction / electrowinning (SX/EW). Copper extraction is predicted to range between 60 and 70 percent depending on material type. Acid consumption projections range between 30 and 35 pounds of acid per ton of material. The historic MacArthur Pit contains 133 million tons of oxide material which is predicted to yield 70% copper extraction with acid consumption of 30 pounds of acid per ton of material leached. Material from the MacArthur pit is predominately mined and processed over the first 7 years of operation.

The leach pad will be constructed using an HDPE liner system meeting Nevada requirements (NR 455). Conventional solvent extraction will be used. Electrowinning will include permanent mother blank stainless steel technology and harvesting of Grade A copper cathode on a 7 day pull schedule. All process facilities will incorporate proven industry standard designs and equipment.

It is recommended that additional metallurgical test work be performed for the pre-feasibility study (PFS), to better understand the metallurgy of this project. A preliminary test program design for the PFS is discussed in Section 26 of this PEA.

The MacArthur Project has a long history of metallurgical testing from 1976 through 2011 including bottle roll and column leach testing and full scale heap leach operations. Anaconda performed the first test work in 1976 and multiple subsequent owners continued test work through 2011. The most comprehensive test work was performed by the current owner, Quaterra Alaska, Inc., during 2010 and 2011. Quaterra ran a substantial number of bottle roll leach tests along with 32 column leach tests on 26 new PQ size core drill holes. These drill holes provided reasonable representivity of the MacArthur Project mineral resources. The testwork, both historic and that most recently performed, shows the mineralized material is amenable to standard heap leaching with good copper extraction.

The MacArthur Project deposits generally consist of oxidized copper caps transitioning through a mixed oxide/secondary sulfide interface into primary sulfides at depth. Of the 271 million tons of acid soluble material, 185 million tons is classified as oxide, and 86 million tons is classified as mixed or secondary sulfide mineralization.

Arimetco operated a run of mine (ROM) heap leach/solvent extraction / electrowinning facility from 1989 through 1998 leaching low grade oxide stockpile material. Additionally, 6.1 million tons of ROM oxide material from the historic MacArthur pit was leached by Arimetco at the Yerington Site.

1.7 ECONOMIC ASSESSMENT

The mine and process facilities include a heap leach pad, solvent extraction / electrowinning facilities, a sulfuric acid plant with power plant, and the necessary infrastructure to support the mine and process facilities. The initial capital cost for the mine and process facilities are estimated to be $232.75 million with an additional $147.57 million in sustaining capital. The sustaining capital includes a phased expansion of the heap leach pad, additional mine equipment, and mobile equipment replacement throughout the life of mine. Closure and reclamation costs at the end of the 18 year mine life are estimated to be an additional $82.96 million including a salvage value for equipment and materials at mine closure.

The overall life of mine operating cost for the facilities is $1.89 per pound of recovered copper and includes mining, solvent extraction / electrowinning, sulfuric acid plant, general and administrative cost, and transportation cost to transport the final cathode copper product to market.

The Net Present Value (NPV) was calculated based on an average annual copper production of 41.5 million pounds of copper per year and a price of copper of $3.48 per pound. The Project will generate after tax NPV of $201.57 million at a discount rate of 8% with an Internal Rate of Return of 24.2% and a payback period of 3.1 years.

This PEA is preliminary in nature and includes discussion of mineral resources including inferred mineral resources that are too speculative geologically to have economic considerations applied to them. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Therefore, there is no certainty that the PEA will be realized.

1.8 CONCLUSIONS AND RECOMMENDATIONS

Quaterra intends to develop the MacArthur Copper Project as a stand-alone project; however, other mineral resources owned by Singatse Peak Services (SPS), a subsidiary of Quaterra Alaska Inc. at the Yerington mine may be added to the development as appropriate. The MacArthur Copper Project has shown potential for development as a large scale copper oxide heap leach operation.

The following additional work is recommended as part of a pre-feasibility study to advance the project.

a) Additional exploration and delineation drilling to better define the resource, particularly in the area north of the MacArthur pit and at depth, reduce technical risk and increase the project resources.

b) Update the project resource model with the additional drilling information.

c) Optimize the mine plan based on the new resource model.

d) Additional metallurgical test work to confirm the extraction rates and acid consumption as outlined in Section 26.

e) Confirm the design parameters for the heap leach pad, including lift height, irrigation rate and inter-lift liners.

2 INTRODUCTION

2.1 GENERAL

Quaterra Alaska, Inc.'s parent company, Quaterra Resources, Inc. (NYSE Amex: QMM; TSX-V: QTA), with headquarters located in Vancouver, British Columbia, Canada, is a mineral exploration company focused on making significant base and precious metals discoveries in North America. The company also has a local office located in Yerington, Nevada.

Quaterra requested a number of consultants to provide a Preliminary Economic Assessment Technical Report, compliant with Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects, for the MacArthur Copper Project located in Lyon County, Nevada, approximately 75 miles southeast of Reno, Nevada. Tetra Tech MM, Inc. of Golden, Colorado, was commissioned to prepare an update of the resource estimate and provide a review of the metallurgical test work. Independent Mining Consultants, Inc. (IMC) of Tucson, Arizona, was commissioned to provide the mining methods and pit design. SRK Consulting (U.S.), Inc. of Reno, Nevada, was commissioned to provide the environmental and permitting review; and M3 Engineering & Technology Corporation of Tucson, Arizona, was commissioned to provide the process and infrastructure, capital and operating costs, and the economic assessment for the project.

2.2 PURPOSE OF REPORT

The purpose of this report is to present updated mineral resource information and a mine production plan, process and metallurgical testing information, infrastructure, capital and operating costs, a preliminary economic analysis, and other relevant data and information for the MacArthur Copper Project since the issuance of the updated mineral resource estimate 43-101 Technical Report dated January 21, 2011. It is the intent of Quaterra to continue to develop the MacArthur Copper Project with possible integration of other resources within the Yerington mining district.

The effective date of this report is May 23, 2012.

2.3 SOURCES OF INFORMATION

This report is based on data supplied by Quaterra with the use of historic data from Anaconda, Pangea Explorations (Pangea), North Exploration LLC (North), Bear Creek Mining Company, The Superior Oil Company (Superior), U.S. Bureau of Mines, and Arimetco International, Inc. (Arimetco). Drilling and sampling at the MacArthur site started in 1955 with Anaconda and has continued through November 2011 with Quaterra's last exploration program.

The information presented, opinions and conclusions stated, and estimates made are based on the following information:

- Source documents used for this report as summarized in Section 27,
- Assumptions, conditions, and qualifications as set forth in this report,

- Data, reports, and opinions from prior owners and third-party entities, and
- Personal inspection and reviews.

Tetra Tech, in the preparation of its sections, has not independently conducted any title or other searches, but has relied upon Quaterra for information on the status of claims, property title, agreements, permit status, and other pertinent conditions. In addition, Tetra Tech has not independently conducted any sampling, mining, processing, economic studies, permitting or environmental studies on the property.

Information provided by Quaterra includes:

- Assumptions, conditions, and qualifications as set forth in the report,
- Drill hole records,
- Property history details,
- Sampling protocol details,
- Geological and mineralization setting,
- Data, reports, and opinion from prior owners and third-party entities, and
- Copper and other assays from original records and reports.

Additional information provided by third-party entities includes a Preliminary Column Leach Study prepared by METCON Research, dated December 2011 and a Scoping Study dated March 2012 for the Re-mining and Processing of Residual Ore Stockpiles and Tailings at Yerington prepared by SRK Consulting.

2.4 CONSULTANTS AND QUALIFIED PERSONS

Quaterra contracted a number of consultants, including M3 Engineering & Technology Corporation, to provide a review of prior and new work on the project and to prepare technical and cost information to support a Preliminary Economic Assessment (PEA) and this Technical Report. M3 Engineering & Technology Corporation was responsible for defining the process facilities, infrastructure, capital cost, operating cost, preliminary financial assessment, and integrating the work by other consultants into a final Technical Report compliant with the Canadian National Instrument 43-101 standards.

Mr. Myron R. Henderson, P.E., of M3 Engineering and Technology Corporation is the principal author and Qualified Person responsible for preparation of this report. Mr. Henderson visited the site on November 30, 2011 and December 1, 2011 to review the physical conditions and the existing infrastructure at site. Other contributing authors and Qualified Persons responsible for preparing sections of this report include Dr. Rex C. Bryan of Tetra Tech, Dr. Richard W. Jolk, P.E. of Tetra Tech, Mr. Herbert E. Welhener of Independent Mining Consultants, Inc. (IMC), and Mr. Mark Willow of SRK Consulting (U.S.) Inc.

Dr. Rex C. Bryan, Ph.D., of Tetra Tech is the Qualified Person responsible for preparation of the property description, property history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation and security, data verification, and description of adjacent properties. These sections of the report were taken or updated from the MacArthur Copper Project NI 43-101 Technical Report, Lyon County, Nevada, USA dated January 21, 2011 prepared by Mr. John W. Rozelle, P.G., Principal Geologist of Tetra Tech. Dr. Bryan was also responsible for preparation of the updated resource estimate. Dr. Bryan visited the site in September 2011 for a physical review of sample preparation and security procedures, as well as discussions with geologists and individuals regarding data handling and project geology. It is Dr. Bryan's opinion that there were no deficiencies in the company's protocols or procedures.

Mr. Herbert E. Welhener, MMSA-QPM, of IMC was responsible for preparation of the mining methods. Mr. Welhener visited the site on November 30, 2011 and December 1, 2011 to inspect the physical conditions at the site and the existing MacArthur pit.

Dr. Richard W. Jolk. P.E., Ph.D., of Tetra Tech was responsible for the review of the new and historical metallurgical test work and preparation of the mineral processing and metallurgical testing section of this report. Dr. Jolk visited the site on February 20, 2012, March 19, 2012, and April 17, 2012.

Mr. Mark A. Willow, M.Sc., SME-RM, of SRK Consulting was responsible for the preparation of the environmental studies, permitting and social impact section of this report. Mr. Willow visited the site on January 30, 2012 and April 17, 2012.

The QP responsibilities mentioned above are summarized as follows:

Table 2-1: Qualified Person Responsibilities

Qualified Person	Registration	Company	Sections of Responsibility	Site Visit
Myron R. Henderson	P.E.	M3	Sections 1, 2, 3, 17, 18, 19, 21, 22, 24, 25, 26, and 27.	November 30, 2011 and December 1, 2011
Rex C. Bryan	Ph. D.	Tetra Tech	Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 15, and 23.	September 2011
Herbert E. Welhener	MMSA-QPM	IMC	Section 16.	November 30, 2011 and December 1, 2011
Richard W. Jolk	P.E., Ph.D.	Tetra Tech	Section 13.	February 20, 2012, March 19, 2012, and April 17, 2012
Mark A. Willow	M.Sc., SME-RM	SRK	Section 20.	January 30, 2012 and April 17, 2012

2.5 DEFINITION OF TERMS USED IN THIS REPORT

Unless explicitly stated, all units presented in this report are in the Imperial System (i.e. short tons, miles, feet, inches, pounds, percent, parts per million, and troy ounces). All monetary values are in United States (US) dollars unless otherwise stated.

Common units of measure and conversion factors used in this report include:

Linear Measure:

1 inch = 2.54 centimeters

1 foot = 0.3048 meter

1 yard = 0.9144 meter

1 mile = 1.6 kilometers

Area Measure:

1 acre = 0.4047 hectare

1 square mile = 640 acres = 259 hectares

Capacity Measure (liquid):

1 US gallon = 4 quarts = 3.785 liter

1 cubic meter per hour = 4.403 US gpm

Weight:

1 short ton	= 2000 pounds	= 0.907 tonne
1 pound	= 16 oz	= 0.454 kg
1 oz (troy)	= 31.103486 g	

Analytical Values:

	percent	**grams per metric tonne**	**troy ounces per short ton**
1%	1%	10,000	291.667
1 gm/tonne	0.0001%	1.0	0.0291667
1 oz troy/short ton	0.003429%	34.2857	1
10 ppb			0.00029
100 ppm			2.917

Frequently used acronyms and abbreviations:

ac-ft	=	acre feet
ACu or AsCu	=	Acid Soluble Copper Assay
Ag	=	silver
Au	=	gold
Ag oz/t	=	troy ounces silver per short ton (oz/ton)
Au oz/t	=	troy ounces gold per short ton (oz/ton)
BADCT	=	Best Available Demonstrated Control Technology
BLM	=	Bureau of Land Management
CIM	=	Canadian Institute of Mining, Metallurgical, and Petroleum
CNCu	=	Cyanide Soluble Copper Assay
EPA or USEPA	=	United States Environmental Protection Agency
EIS	=	Environmental Impact Statement
°F	=	degrees Fahrenheit
FA	=	Fire Assay
ft	=	foot or feet
ft2	=	square foot or feet
ft3	=	cubic foot or feet
GCL	=	Geosynthetic Clay Liner
g	=	gram(s)
gpl	=	grams per liter
gpm	=	gallons per minute
h	=	hour
HDPE	=	High Density Polyethylene
km	=	kilometer
kV	=	kilovolts
kWh	=	Kilowatt hour
kWh/t	=	Kilowatt hours per ton
l	=	liter
lb(s)	=	pound(s)
lbs/ft^3	=	pounds per cubic foot
LME	=	London Metal Exchange
MW	=	megawatts

NDEP	=	Nevada Division of Environmental Protection
NEPA	=	National Environmental Policy Act
NSR	=	net smelter return
PEA	=	Preliminary Economic Assessment
PFS	=	Preliminary Feasibility Study
PLS	=	Pregnant Leach Solution
PoO	=	Plan of Operations
ppm	=	parts per million
ppb	=	parts per billion
QLT	=	Quick Leach Test, also Ferric Soluble Copper
RC	=	reverse circulation drilling method
ROD	=	Record of Decision
SCFM	=	standard cubic feet per minute
SX/EW	=	Solvent extraction & electrowinning
TCu	=	Total Copper Assay
ton	=	short ton(s)
tph	=	tons per hour
tpy	=	tons per year
tpm	=	tons per month
tpd	=	tons per day
tph	=	tons per hour
µm	=	micron(s)
VLT	=	Vat Leach Tailings
%	=	percent

Abbreviations of the Periodic Table

actinium = Ac	aluminum = Al	americium = Am	antimony = Sb	argon = Ar
arsenic = As	astatine = At	barium = Ba	berkelium = Bk	beryllium = Be
bismuth = Bi	bohrium = Bh	boron = B	bromine = Br	cadmium = Cd
calcium = Ca	californium = Cf	carbon = C	cerium = Ce	cesium = Cs
chlorine = Cl	chromium = Cr	cobalt = Co	copper = Cu	curium = Cm
dubnium = Db	dysprosium = Dy	einsteinum = Es	erbium = Er	europium = Eu
fermium = Fm	fluorine = F	francium = Fr	gadolinium = Gd	gallium = Ga
germanium = Ge	gold = Au	hafnium = Hf	hahnium = Hn	helium = He
holmium = Ho	hydrogen = H	indium = In	iodine = I	iridium = Ir
iron = Fe	juliotium = Jl	krypton = Kr	lanthanum = La	lawrencium = Lr
lead = Pb	lithium = Li	lutetium = Lu	magnesium = Mg	manganese = Mn
meltnerium = Mt	mendelevium = Md	mercury = Hg	molybdenum = Mo	neodymium = Nd

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by M3 and co-authored by the QPs listed in Section 2. The information, conclusions, opinions and estimates contained herein are based on:

- Information available to the authors of this report up to and including the effective date of this report ;
- Assumptions, conditions and qualifications as set forth in this report; and
- Data, reports and other information supplied by Quaterra and other third party sources.

Reports received from other experts have been reviewed for factual errors by Quaterra and M3. The statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false or misleading at the date of these reports.

M3 and Tetra Tech. relied upon Quaterra for property ownership data and have not verified ownership or underlying agreements.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The MacArthur Copper Property is located near the geographic center of Lyon County, Nevada, USA along the northeastern flank of the Singatse Range approximately seven miles northwest of the town of Yerington, Nevada (Figure 4-1 and Figure 4-2). The property is accessible from Yerington by approximately five miles of paved roads and two miles of maintained gravel road. Topographic coverage is on US Geological Survey "Mason Butte" and "Lincoln Flat" 7.5' topographic quadrangles. The nearest major city is Reno, Nevada approximately 75 miles to the northwest.

4.2 PROPERTY OWNERSHIP

The property consists of 470 unpatented lode claims totaling approximately 9700 acres on lands administered by the US Department of Interior - Bureau of Land Management (BLM) (Figure 4-3). Sixty one claims are held by Quaterra by means of a mineral lease with option to purchase, executed on August 27, 2005, followed by three amendments dated January 16, 2007, August 6, 2007, and January 9, 2011. The agreement gives Quaterra the right to purchase the claims from North Exploration by making three annual payments of $524,000 (option balance) plus interest at the rate of six percent per annum by January 15, 2013. Quaterra's purchase is subject to a two percent Net Smelter Return (NSR) royalty with a royalty buy down option of $1,000,000 to purchase one percent of the NSR, leaving a perpetual one percent NSR. The agreement with North Exploration is in good standing. The remaining 409 claims were staked as lode mining claims by Quaterra. These claims are in good standing with all annual payments to the BLM and Lyon County having been paid.

A portion of the MacArthur claim group is also included in the area referred to as the "Royalty Area" in Quaterra Resource's purchase agreement for the acquisition of Arimetco's Yerington properties. Under this agreement, MacArthur claims within this area (as well as the Yerington properties) are subject to a two percent NSR production royalty derived from the sales of mineralized material, minerals and materials mined and marketed from the property up to $7,500,000. The northern-most limit of the Royalty Area is shown in Figure 4-3.

Quaterra's claims are located in sections 2 and 3, Township 13 North, Range 24 East; in sections 10-15, 22-27, and 34-36, Township 14 North, Range 24 East; and in sections 18- 20 and 29-31, Township 14 North, Range 25 East, Mount Diablo Base & Meridian. Claim outlines and boundaries are displayed on Figure 4-2 and Figure 4-3.

4.3 MINERAL TENURE AND TITLE

All claims on the project are unpatented lode-mining claims, and as such require a Federal annual maintenance fee of $140 each, due by 12:00 PM (noon) of September 1 of each year. Further, each lode claim staked in Nevada requires an Intent to Hold fee of $10.50 each, plus a $4.00 filing fee, due 60 days after September 1 of each year. All fees for all claims within the project have been paid in a timely manner and all claims are current.

All claims were staked by placing a location monument (two- by two-inch wood post) along the center line of each claim and two- by two-inch wood posts at all four corners, with all posts properly identified in accordance with the rules and regulations of the BLM and the State of Nevada. Maximum dimension of unpatented lode claims is 600 feet x 1500 feet. The author observed various location monuments and claim corners during the field examination. No legal survey of the claims has been undertaken.

4.4 RELEVANT INFORMATION

Quaterra's 2007-2008 core and reverse circulation exploration drilling programs were approved by the BLM at the Notice of Intent level supported by posting of a $37,075 bond (File Name: NVN-083324, 3809, (NV-033)). Quaterra is currently conducting exploration under a BLM Plan of Operations / Environmental Assessment (File name 3809 (NV923Z), BLM Bond Number NVB001150) and under Reclamation Permit #0294 with the Nevada Division of Environmental Protection.

Figure 4-1: General Location Map

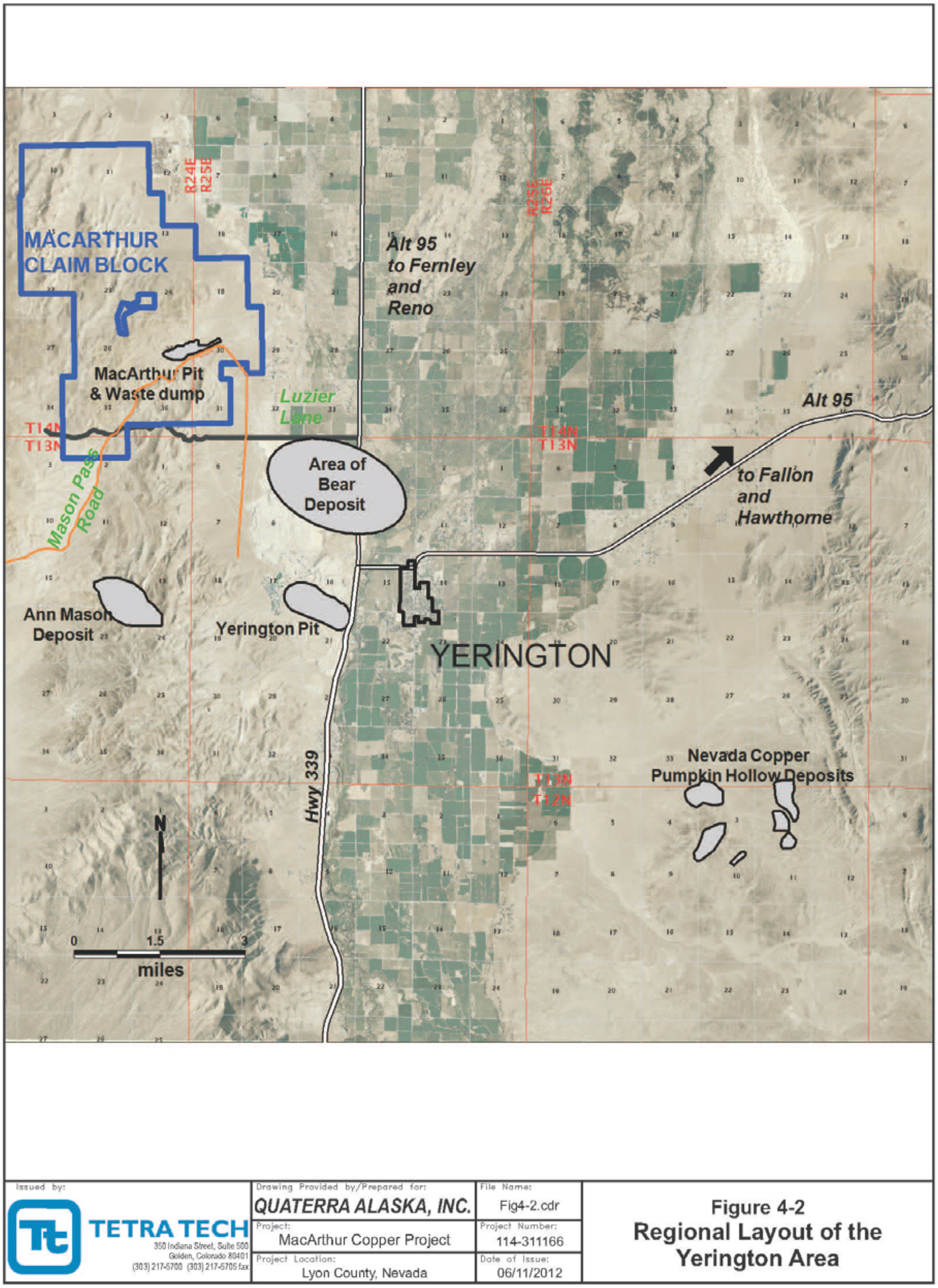


Figure 4-2: Regional Layout Map

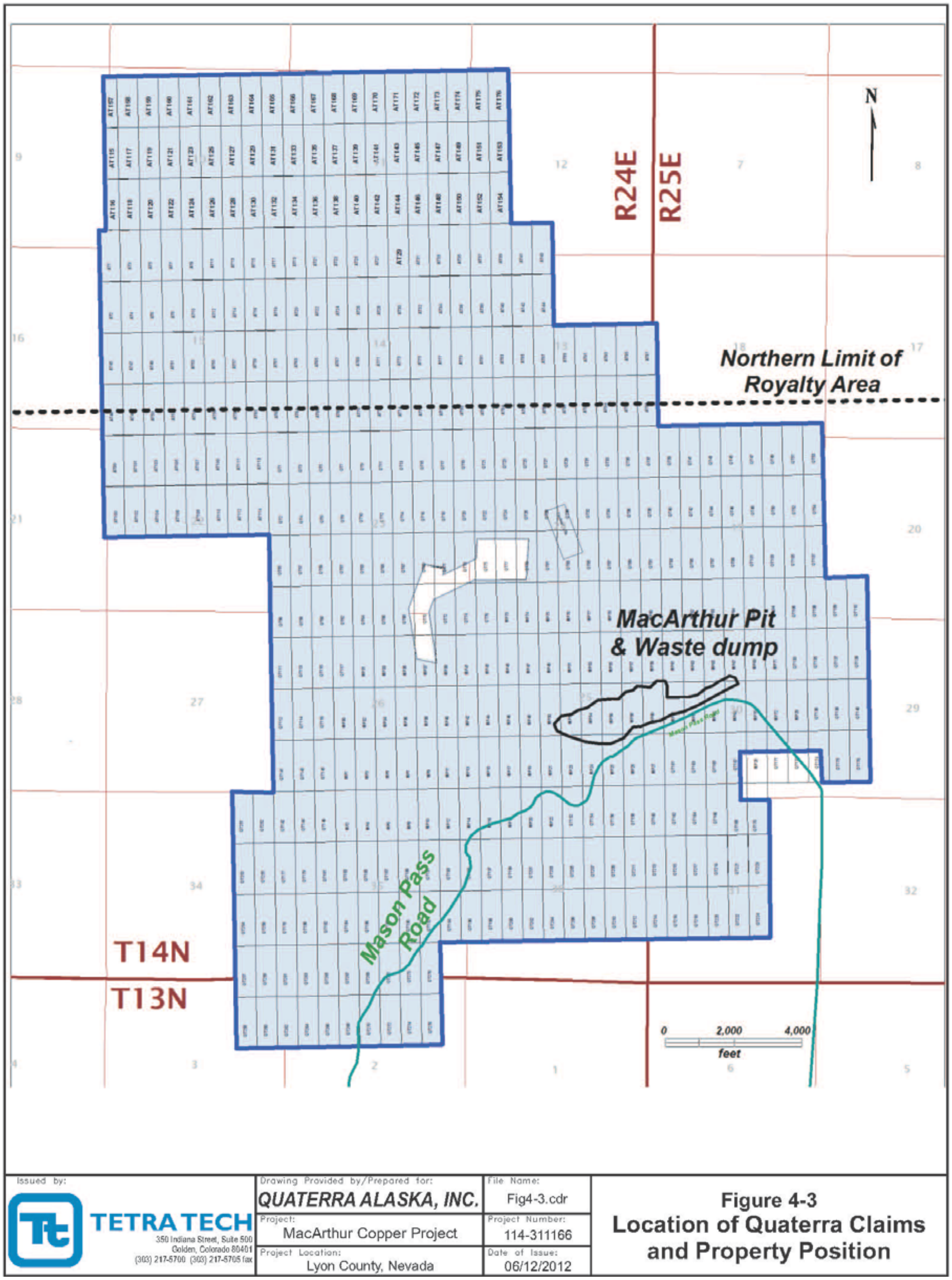


Figure 4-3: MacArthur Property Map

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

Access to the property from the town of Yerington is approximately three miles north along US Highway ALT 95 to Luzier Lane, then west approximately two miles by pavement to the Mason Pass road, an improved county gravel road leading two miles northerly to the property (Figure 4-2). Property entry is along a 100-foot wide improved gravel mine road that accessed the MacArthur open pit copper mine during the 1990s. Beyond the MacArthur pit area are several historic two-track dirt roads that provide access throughout the property.

5.2 CLIMATE

Elevations on the property range from 4,600 to 5,600 feet as low-rolling to moderately steep terrain, sparsely covered by sagebrush interspersed with low profile desert shrubs. There are no active streams or springs on the property. All gulches that traverse the property are dry year-round. The climate is temperate, characterized by cool winters with temperatures between zero and 50 °F and warm to hot summers with temperatures between 50 and 100 °F. Average annual precipitation is estimated at three to eight inches per year, with a significant part of this total precipitation falling as snow and increasing with elevation. Work can be conducted throughout the year with only minor stoppage during winter months due to heavy snowfall or unsafe travel conditions when roads are particularly muddy.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The nearest incorporated town is the agricultural community of Yerington located seven miles to the southeast along improved gravel roads and pavement. Formerly an active mining center from 1953 to 1978 when Anaconda operated the Yerington copper mine and from 1995 to 1997 when Arimetco operated the MacArthur oxide copper mine, Yerington now serves as a base for three active exploration groups: Quaterra Resources, Inc. (MacArthur and Yerington copper properties held by Quaterra Alaska and Singatse Peak Services, respectively), Entrée Gold Inc. (Ann Mason copper-molybdenum property), and Nevada Copper Corporation (Pumpkin Hollow Copper Project) as displayed on Figure 4-2. Yerington hosts a work force active in, qualified for, and familiar with mining operations within a one-hour drive to the property.

Yerington offers most necessities and amenities including police, hospital, groceries, fuel, regional airport, hardware, and other necessary items. A propane-fired 220 megawatt electrical generating power plant, operated by NV Energy, is located approximately 12 road miles north of Yerington accessed off State Highway 95A. The Wabuska railhead is located approximately ten miles north of Yerington along State Highway 95A, two miles north of the turnoff to the power plant. Drilling supplies and assay laboratories can be found in Reno, a 1.5-hour drive from Yerington. Reverse circulation drilling contractors are found in Silver Springs, Nevada 33 miles north of Yerington and in Winnemucca and Elko, Nevada areas, from three to five driving hours from Yerington.

During the Arimetco operating period, approximately 6.1 million tons of leach mineralized material mined from the MacArthur pit was trucked approximately five miles south to the former Anaconda Yerington mine site onto leach pads (with approved liners). Leach pad sites and ancillary facilities for the MacArthur Project are proposed on unpatented claims controlled by Quaterra located northeast of the MacArthur pit, as discussed in Section 18 of this report (Figure 18-1).

6 HISTORY

6.1 PROPERTY HISTORY

Following the early 1860's bonanza silver discoveries along the Comstock Lode in the Virginia City mining district, prospectors stepped out 30 miles to the southeast to investigate the colorful oxide copper showings along the Singatse Range within the present-day Yerington mining district (Figure 6-1). A majority of the early work (earliest recorded date of 1883) concentrated on contact-metamorphic replacement copper deposits hosted in limestone or limey sedimentary rocks clustered from four to six miles south-southwest of the MacArthur property (Moore, 1969). These contact copper deposits were mined on a small scale, shipping 2,000 to 1.7 million tons of copper mineralized material. Most of this early activity took place before and during World War I. Tingley, et al (1993) estimates production from the Yerington district at over 85 million pounds of copper from 1905 to 1920, ostensibly with very little contribution from the shallow prospects in the MacArthur area.

After the 1920s, only minor copper production is recorded from the contact replacement prospects and mines (Moore, 1969). The largest nearby operation, located in the Buckskin mining district approximately five miles northwest of the MacArthur property, was the Minnesota Mine where copper was mined in the early 1920s, but sizeable production of skarn (contact) magnetite iron mineralized material began in 1952 with approximately four million tons of mineralized material produced by the end of 1966.

During the 1940s, Anaconda geologists investigated copper showings over the MacArthur property and conducted pre-development drilling over the present day Yerington Mine. US Government-funded strategic minerals exploration in the early 1950s supported Anaconda's initial development of the Yerington mine (fully funded by Anaconda following expiration of strategic minerals funding in the late 1950s). During 1953 to 1978, Anaconda produced 162 million tons of 0.55% Cu mineralized material amounting to over 1.75 billion pounds of copper from a single open pit mine known as the Yerington Mine located five miles south of the MacArthur property (Tingley, et al, 1993). Oxide and sulfide copper mineralized material, hosted in a Middle Jurassic porphyry system of granodiorite, quartz monzonite, and quartz monzonite porphyry dike swarms, and were extracted from the Yerington Mine.

Anaconda, the US Bureau of Mines, Bear Creek Mining Company, Superior Oil and others conducted mineral exploration campaigns at the MacArthur property from the mid-1940s through the early 1970s. The most significant program was conducted in 1972 to 1973 by Anaconda following an extensive trenching and drilling program that resulted in 13 million tons of plus 0.4% Cu mineralization (Heatwole, 1978).

A qualified person has not done sufficient work to classify the historical estimate as a current mineral resource or reserve and Quaterra does not treat the historical estimate as a current mineral resource or reserve.

During the late 1980s, Arimetco permitted heap leaching sites at the Yerington mine site with feed sourced from Yerington mine dumps, oxide stockpiles, and vat leach tailings. Arimetco expanded their operations to include approximately 6.1 million tons grading about 0.30% Cu mined from 1995 to 1997 from what is now the present day MacArthur pit. Based on 1972 and 1973 Anaconda drilling, Arimetco published a reserve of 29 million tons of 0.28% copper mineralized material remaining in the planned MacArthur pit (MineMarket.com, 2000).

A qualified person has not done sufficient work to classify the historical estimate as a current mineral resource or reserve and Quaterra does not treat the historical estimate as a current mineral resource or reserve.

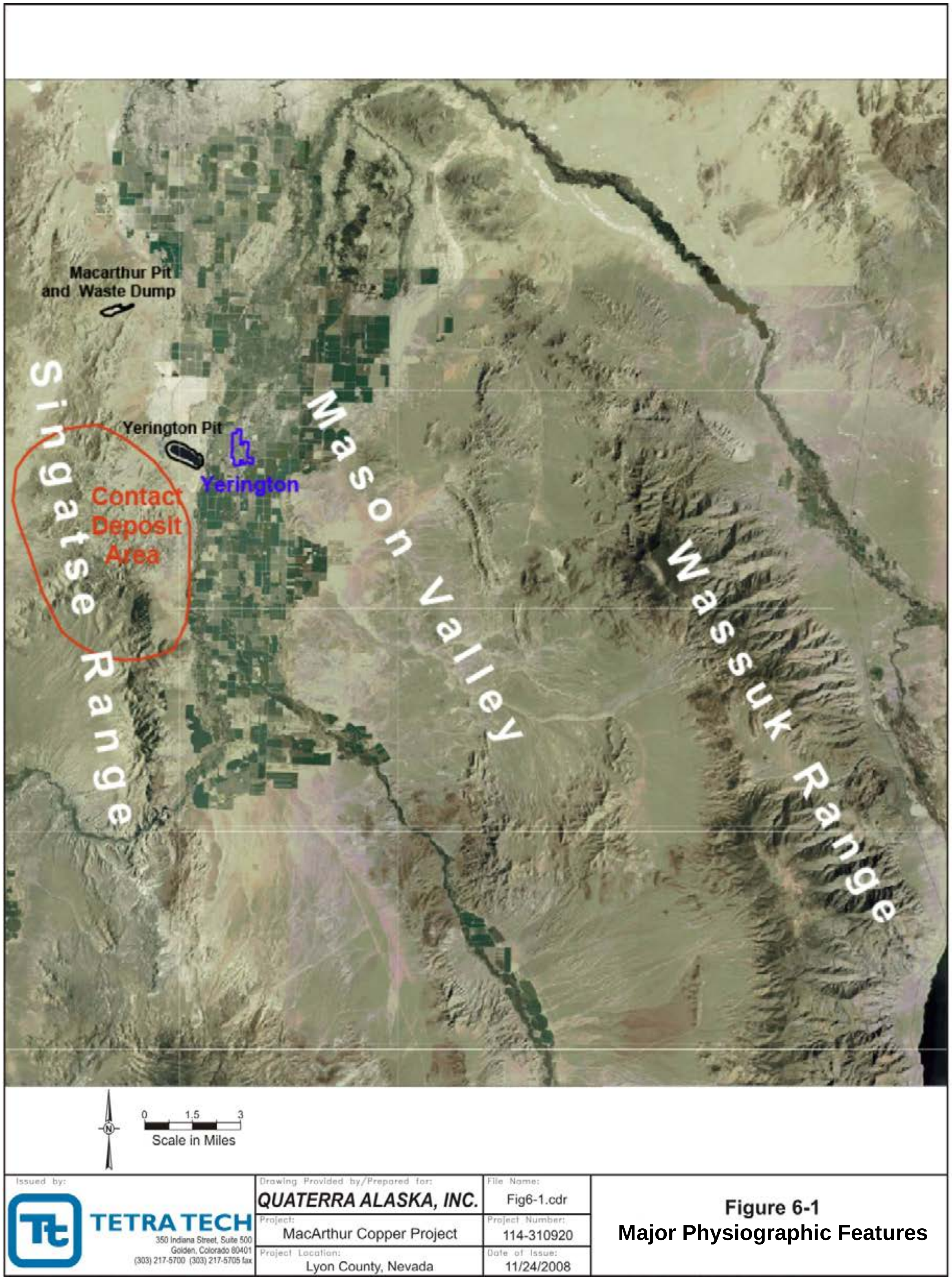


Figure 6-1: Major Physiographic Features

6.2 HISTORICAL RESOURCES

Anaconda, the US Bureau of Mines, Bear Creek Mining Company, Superior Oil and others conducted mineral exploration campaigns at the MacArthur property from the mid-1940s through the early 1970s. The most significant program was conducted in 1972 to 1973 by Anaconda following an extensive trenching and drilling program that resulted in an estimate of 13 million tons of plus 0.4% Cu mineralization (Heatwole, 1978).

A qualified person has not done sufficient work to classify the historical estimate as a current mineral resource or reserve and Quaterra does not treat the historical estimate as a current mineral resource or reserve.

During the late 1980s, Arimetco permitted heap leaching sites at the Yerington mine site with feed sourced from Yerington mine dumps, oxide stockpiles, and vat leach tailings. Arimetco expanded their operations to include approximately 6.1 million tons grading about 0.30% Cu mined from 1995 to 1997 from what is now the present day MacArthur pit. Based on 1972 and 1973 Anaconda drilling, Arimetco published a reserve of 29 million tons of 0.28% copper mineralized material remaining in the planned MacArthur pit (MineMarket.com, 2000).

A qualified person has not done sufficient work to classify the historical estimate as a current mineral resource or reserve and Quaterra does not treat the historical estimate as a current mineral resource or reserve.

6.3 HISTORIC MINING

The MacArthur Project area has seen limited historic mining activity, and there is no indication of any historic, small-scale, artisanal mining activity. The most recent activity occurred between 1995 and 1997, when Arimetco mined a limited tonnage of surface oxide copper for heap leaching at the historic Yerington Mine site. No consistent, large-scale mining has occurred on the site.

6.4 HISTORIC METALLURGICAL TESTWORK AND MINERAL PROCESSING

The metallurgical testwork performed on material from the MacArthur property is dated and focused on leach performance of material typical of what was historically mined from the MacArthur pit. Anaconda, Bateman Engineering (Bateman), and Mountain States R&D International (Mountain States) have all performed various metallurgical testwork for the MacArthur property.

Anaconda completed bottle roll and vat leaching tests on crushed mineralized material. Anticipated recoveries ranged from 82 to 85% of total copper while consuming 4 to 5 pounds acid per pound recovered copper. Bateman ran 18 and 24-inch diameter 20-foot high column leach tests on run-of-mine mineralized material and achieved 50 to 60% recovery of total copper while consuming 3 to 4 pounds acid per pound copper. Mountain States testing consisted of crushed un-treated mineralized material and acid-cured mineralized material column leach testing at 1.5 and 2.5 inch sizes. Mountain States estimated recoveries for the un-treated mineralized material at approximately 70% of soluble copper at a 2.5 inch crushed mineralized material size with only slightly better recovery at a 1.5 inch size. Acid consumption was approximately 3 pounds acid per pound copper. Recoveries for the acid-cured mineralized material were increased by 5 to 10%, and the indicated acid consumption was reduced by approximately 1 pound acid per pound copper. Acid-cured mineralized material also leached faster than the un-treated mineralized material, with recovery times going from 30 to 60 days down to less than 30 days.

A more detailed discussion concerning the historical metallurgy in light of the current metallurgical work can be found in Section 13, Mineral Processing and Metallurgical Testing.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The MacArthur Project area is located within the western Basin and Range Province in Nevada on the east side of the Sierra Nevada Mountains. Within the Basin and Range, north trending normal faults have down-dropped basins on either side of upland ranges. In a similar setting in western Nevada, the Singatse Range and Wassuk Range form the western and eastern boundaries, respectively, of Mason Valley. The MacArthur property, in the Yerington mining district, is located in the west-central portion of Mason Valley along the eastern slopes of the Singatse Range.

The regional geology is displayed on Figure 7-1 (Proffett and Dilles, 1984). The oldest rocks in the Yerington area of Mason Valley consist of an approximate 4,000-foot thick section of Late Triassic, intermediate and felsic metavolcanics and lesser sedimentary rocks, the McConnell Canyon Formation, associated with volcanic arc development along the North American continent during the Mesozoic.

This sequence is disconformably overlain by a series of Upper Triassic carbonates, clastic sediments, and volcaniclastics that are in turn overlain by the Norian (aka Mason Valley) Limestone, a massive limestone nearly 1,000 feet thick. During the Upper Triassic – Lower Jurassic, a section of limestones, clastic sediments, tuffs, and argillites, in part correlative with the Gardnerville Formation, were deposited. The Ludwig Limestone, containing gypsum, sandstone, and arkose, overlies the Gardnerville Formation.

Mesozoic plutonism, possibly related to the igneous activity that formed the Sierra Nevada Mountains, followed during the Middle Jurassic with emplacement of the Yerington batholith of granodiorite (field name) composition and the Bear quartz monzonite. Mesozoic plutonism, emplaced approximately 169 Ma (Proffett and Dilles, 1984), was closely followed by Middle Jurassic quartz monzonite porphyry dikes and dike swarms related to the Luhr Hill granite porphyry. Andesite and rhyolite dikes represent the final phase of Mesozoic igneous activity.

Mesozoic rocks were deeply eroded and then overlain by Mid-Tertiary tuffs and lesser sedimentary rocks. Coarser grained andesite dikes are tabbed as Tertiary. The entire package was subsequently faulted along north-trending, down-to-the-east dipping faults that resulted in extension and major westerly tilting.

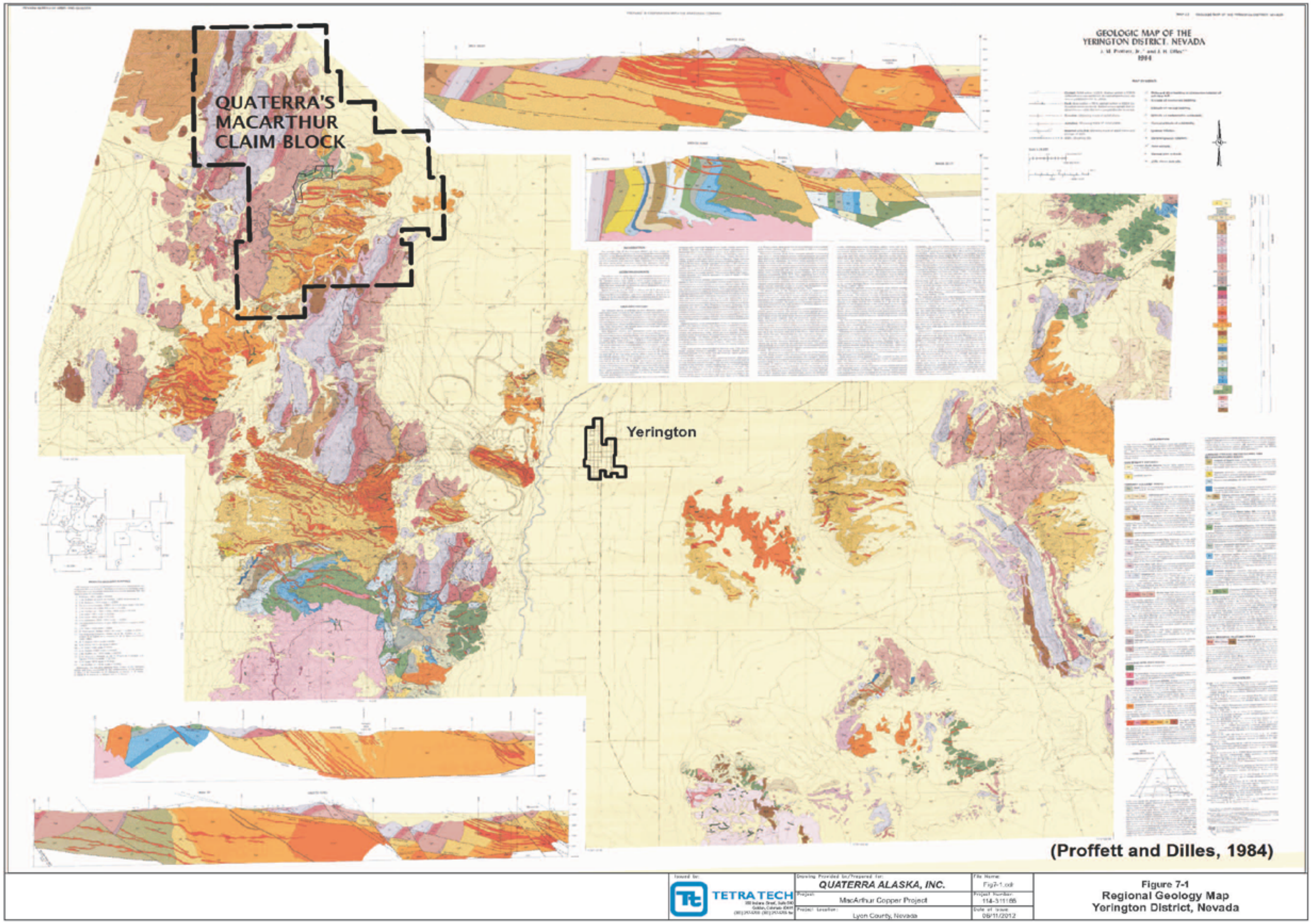


Figure 7-1: Regional Geology

7.2 LOCAL GEOLOGY

The MacArthur Copper Property is one of several copper deposits and occurrences hosted in or related to Middle Jurassic intrusive rocks within the Yerington Mining District, Lyon County, Nevada. The Yerington area is underlain by early Mesozoic volcanic and sedimentary rocks now exposed along uplands in the Singatse Range to the west and the Wassuk Range to the east. These Mesozoic rocks were intruded by three Middle Jurassic batholiths, the oldest known as the McLeod Hill Quartz Monzodiorite (field map name granodiorite), followed by the Bear Quartz Monzonite that comprise the majority of outcropping rocks on the MacArthur property. A finer grained phase of the Bear Quartz Monzonite, known as the Border Phase Quartz Monzonite, occurs at the contact between the McLeod Hill Quartz Monzonite and the Bear Quartz Monzonite. These batholiths were subsequently intruded during the Middle Jurassic by the Luhr Hill Granite, the source of quartz monzonitic (or granite) porphyries, consisting of moderately to steeply north dipping quartz-biotite-hornblende porphyry dike swarms, responsible for copper mineralization, striking west-northwesterly across the MacArthur property as well as across the entire mining district.

The geologic record is absent until the middle Tertiary when basalt and voluminous ash flow tuffs were deposited over the Mesozoic rocks.

During advent of Basin and Range normal faulting, ca 18-17 Ma, this entire package of rocks was down-dropped to the east along northerly striking, east dipping, low-angle faults that flatten at depth creating an estimated 2.5 miles of west to east dilation-displacement (Proffett and Dilles, 1984). Such extension rotated the section such that the near vertically-emplaced batholiths were tilted westerly to an almost horizontal position. Pre-tilt, flat-lying younger volcanics now crop out as steeply west dipping units in the Singatse Range west of the MacArthur property. Easterly extension thus created a present-day surface that in plan view actually represents a cross-section of the geology.

7.3 PROPERTY GEOLOGY

The MacArthur property is underlain by two Middle Jurassic batholiths, granodiorite (McLeod Hill Quartz Monzodiorite) intruded by quartz monzonite, (Bear Quartz Monzonite) both of which are intruded by Middle Jurassic quartz porphyry hornblende and quartz porphyry biotite (hornblende) dikes. The north dipping porphyry dike swarms follow penetrative west-northwest and east-west structural fabrics. Narrow (<10 feet) fine grained andesite and rhyolite dikes, post porphyry diking, follow the same structural fabrics.

The McLeod Hill Quartz Monzodiorite, aka granodiorite, weathers as an irregularly orange stained, medium olive green, fine to medium grained rock underlying most of the northern and western parts of Quaterra's claim block. Greenish epidote and minor orange limonite staining are present to common. Megascopic rock constituents include ~50% plagioclase, ~20% orthoclase, <20% quartz, 5 to 20% mafics (hornblende and biotite), 1 to 10% epidote, and minor magnetite and other opaques.

The quartz monzonite, formal designation as Bear Quartz Monzonite, cropping out along the east part of the claim block and underlying the MacArthur pit, is beige to light gray to off white, fine to medium grained, hard but well-fractured, with minor textural variants. Megascopic constituents include ~30% orthoclase, ~30% plagioclase, ~ 20% quartz, and 5- to 10-percent hornblende. In bench walls at the MacArthur Pit, quartz monzonite hosts conspicuous light brown limonite banding (averaging 4 to 6 per foot) sub-parallel to the steeply north dipping, west-northwest trending quartz porphyry dikes. Along the eastern portions of the property, including the eastern third of the MacArthur pit, quartz monzonite assumes a light gray color due to widespread sodic-calcic alteration.

A phase known as the "border-phase quartz monzonite" is found at the top of the Bear Quartz Monzonite pluton (Proffett and Dilles, 1984) and is often mapped at the contact between the granodiorite and the quartz monzonite. The border-phase is finer-grained than the quartz monzonite and contains more abundant potassium feldspar.

Quartz-hornblende / biotite porphyry dikes, originating from the Jurassic Luhr Hill Granite intrude both granodiorite and quartz monzonite at the MacArthur property and are recognized in dike swarms regionally throughout the Yerington mining district. Porphyry dikes hosted a large portion of the primary copper mineralization at Anaconda's Yerington mine and are associated with all copper occurrences in the district. Not all porphyry dikes host copper mineralization, be it sulfide or oxide. At the MacArthur property, porphyry dikes strike west-northwesterly, dipping moderate to steeply north, typically as ridge-formers with widths to 50 feet or more. Porphyry dikes at MacArthur are classified by dominant mafic minerals as quartz biotite porphyry and quartz hornblende porphyry, each subdivided further based on composition and alteration. Dikes contain feldspar crystals and either hornblende or biotite crystals set in an aphanitic matrix. MacArthur pit walls offer excellent exposures of the dikes that host (fracture-controlled) oxide copper mineralization. The following descriptions originate from Quaterra's surface mapping and from core and chip logging:

- Quartz biotite porphyry: contains 2 to 4 mm, generally euhedral, blackish biotite "books" (5 to 10%) and 2 to 8 mm cloudy quartz phenocrysts ("quartz eyes") 2 to 5%. Hornblende is rare to absent. Feldspars commonly 3 to 5 mm. May host sulfide or oxide copper. May or may not have indigenous limonite. If hornblende is present and altered to secondary biotite, the dike is mapped as QMpb-2, otherwise mapped as QMpb-1.

- Quartz hornblende porphyry: contains acicular hornblende crystals, typically thin, "needle-like" to 5 mm long; feldspars vary from 2 to 5 mm. Variety QMph-1 contains 1-5% sulfide (mostly pyrite) with or without indigenous limonite and 3-5% quartz phenocrysts (2 to 5 mm). Variety QMph-2 contains 2-3% sulfides (common) and always has indigenous glass (resinous) limonite derived from primary oxidized chalcopyrite, it also contains oxide copper, and quartz phenocrysts (2-5 mm) present to 2-5%. Variety QMph-3 commonly contains large (to 10 mm) epidote "splotches" (phenocrysts or "epidotization") with 0% to trace fine grained (~1 mm) quartz phenocrysts, 0% to trace sulfides. Any oxide copper is transported from nearby copper-bearing rocks and not oxidized from the porphyry itself.

The best exposures of Jurassic age andesite dikes are found in the walls of the MacArthur Pit where the typically soft- to medium-hard, recessive, olive-greenish dikes can be traced from bench to bench and in some cases followed across the pit floors. Andesite dikes are commonly very fine grained, plagioclase-bearing porphyries that pinch and swell as they fill fractures. Fist-sized pillows may be a weathering product. Andesite dikes intrude the hornblende and biotite quartz porphyry dikes, again best exposed in MacArthur pit walls. Andesite dikes commonly contain oxide copper derived from nearby copper-bearing rocks rather than from the andesite dikes themselves.

Jurassic age rhyolite dikes are also well exposed within the MacArthur Pit walls. The rhyolite is a white to gray, dense, siliceous rock. Rhyolite dikes contain approximately 5% mafic minerals (hornblende and biotite) and rare (1-2%) quartz phenocrysts. Within the MacArthur pit the rhyolite can contain oxide copper mineralization; elsewhere on the property it is barren.

Tertiary hornblende andesite dikes have also been identified on the MacArthur property. These dikes are similar, but coarser grained than the Jurassic andesite dikes, containing abundant, acicular, black hornblende phenocrysts and occasionally plagioclase phenocrysts up to 5-10 mm in long dimension. Tertiary hornblende andesite dikes are frequently observed intruding Basin & Range fault structures. These dikes occasionally contain exotic oxide copper mineralization.

The Mesozoic intrusive rocks are unconformably overlain by a series of nine, moderate to steeply west dipping Mid-Tertiary ash flow tuff units with minor mafic flows and tuffaceous sediments dated at 27.1 to 25.1 Ma (Proffett and Proffett, 1976). The volcanic units make up the uplands in this part and throughout the Singatse Range and cover alteration and structure in the Jurassic igneous rocks.

The dominant west-northwest (N60°W to N80°W) structural fabric recognized throughout the Yerington District is manifested at the MacArthur property as porphyry dike swarms and as high angle shears, faults, and joints along which andesite dikes developed. Structure played a key role in localizing copper oxide mineralization around the historic pit area, principally along the west-northwest fabric and, secondarily, along generally orthogonal northeast structure bearing N20°E to N40°E.

The MacArthur fault, a low angle, easterly striking, north dipping, normal fault is the largest structure recognized on Quaterra's claims. The hanging wall of the fault displaces the basal unit of the Tertiary ash flow tuff sequence approximately 2,000 feet to the east. The displacement of Jurassic intrusive as defined by the offset of the contact of the border quartz monzonite with granodiorite is on the order of 4,000 feet to the east. The MacArthur fault is one of few faults in the Yerington district known to have been active in both Jurassic and Tertiary time.

Chalcocite/oxide mineralization has a close spatial relation to the trace of the MacArthur fault north and west of the MacArthur pit. Gouge in the fault frequently contains chalcocite and/or copper oxide suggesting a structural mineralizing trap.

7.3.1 Alteration

Alteration types recognized at the MacArthur property represent those found in mineralized porphyry copper systems. A generalized distribution of the MacArthur alteration types is displayed in Figure 7-2. The following descriptions are derived from field observation and from drill core and chip logging.

7.3.1.1 Propylitic Alteration

Propylitic alteration is common throughout the MacArthur property in the granodiorite, quartz monzonite, quartz monzonite porphyries, and in the Jurassic andesite. This alteration type occurs as chlorite replacing hornblende, and especially epidotization as veining, coatings, and or flooding on the granodiorite. Calcite veining is present but not common, observed largely in core or drill cuttings. Feldspars are commonly unaltered. Propylitic alteration frequently overprints or occurs with the alteration types described below.

7.3.1.2 Quartz-Sericite-Pyrite (QSP) or Phyllic Alteration

Phyllic alteration is most frequently characterized by tan or light green sericite partially or completely replacing hornblende and/or biotite sites. When phyllic alteration becomes more intense, plagioclase and/or K-feldspar sites are also replaced by sericite. Maroon limonite, hematite, and trace sulfide (chalcocite, pyrite, and chalcopyrite) accompany sericite. However, these minerals do not replace mafic or felsic sites. Sericitic altered zones are often quite siliceous; however, it is unclear if it is due to quartz addition or simply the destruction of other primary minerals.

Phyllic alteration is most pervasive and intense in the Gallagher area and in the northeastern part of the deposit, around hole QM-072. Weak and less pervasive phyllic alteration is found just west of the MacArthur pit and in limited areas around the MacArthur fault. The alteration type does not show preference with rock type and has been described in the granodiorite, quartz monzonite, and quartz monzonite porphyries.

7.3.1.3 Potassic Alteration

Potassic alteration occurs as shreddy, fine-grained biotite replacing hornblende and rarely as pinkish potassium feldspar flooding or in vein haloes, along with disseminated magnetite.

Potassic alteration as shreddy secondary biotite is most obvious in the western and central areas of the MacArthur pit. However, there is occasional biotite replacing hornblende in the northwestern and western portions of the MacArthur property, but is usually less than 20%. K-feldspathization is conspicuous at the base of the mineralized drill intercept of drill hole QM-100. Potassic alteration of some degree has been identified in the granodiorite, quartz monzonite, and quartz monzonite porphyries.

7.3.1.4 Sodic-Calcic Alteration

Pervasive sodic-calcic alteration has been identified within the eastern portions of the MacArthur pit and as broad zones in the far northeastern portion of the district and south of the MacArthur pit. This type of alteration most frequently occurs as albite replacing K-feldspar and as chlorite replacing hornblende in the quartz monzonite, Sodic-calcic alteration has also been identified in the granodiorite and quartz monzonite porphyries. Epidote staining and phenocrysts as well as sphene crystals are ubiquitous. Actinolite replaces hornblende in the more intense zones of sodic-calcic alteration occurring most commonly in the Albite Hills east of the MacArthur pit.

7.3.1.5 Silicification

Silicification occurs as a wholesale replacement of the rock, but only occurs as small and irregular zones that are less than 200 feet across. Typically silification is confined as a narrow halo (less than five feet) along structure and quartz veining. Silicification is present in the western portion of the district, around the Gallagher area and as isolated occurrences within the MacArthur pit.

7.3.1.6 Multiple alteration types

Multiple alteration types are common throughout the area and tend to occur together. Shreddy chlorite has been identified in the MacArthur pit, which likely represents propylitic alteration overprinting potassic alteration. Zones of QSP and propylitic alteration have been identified between the Gallagher area and the MacArthur pit.

7.3.1.7 Supergene alteration

Sulfuric acid (H_2SO_4), formed by the oxidation of sulfides, has altered feldspars and mafic minerals to clay and sericite. At the Gallagher area and north of the MacArthur pit, supergene alteration has formed leached capping which is underlain by chalcocite mineralization.

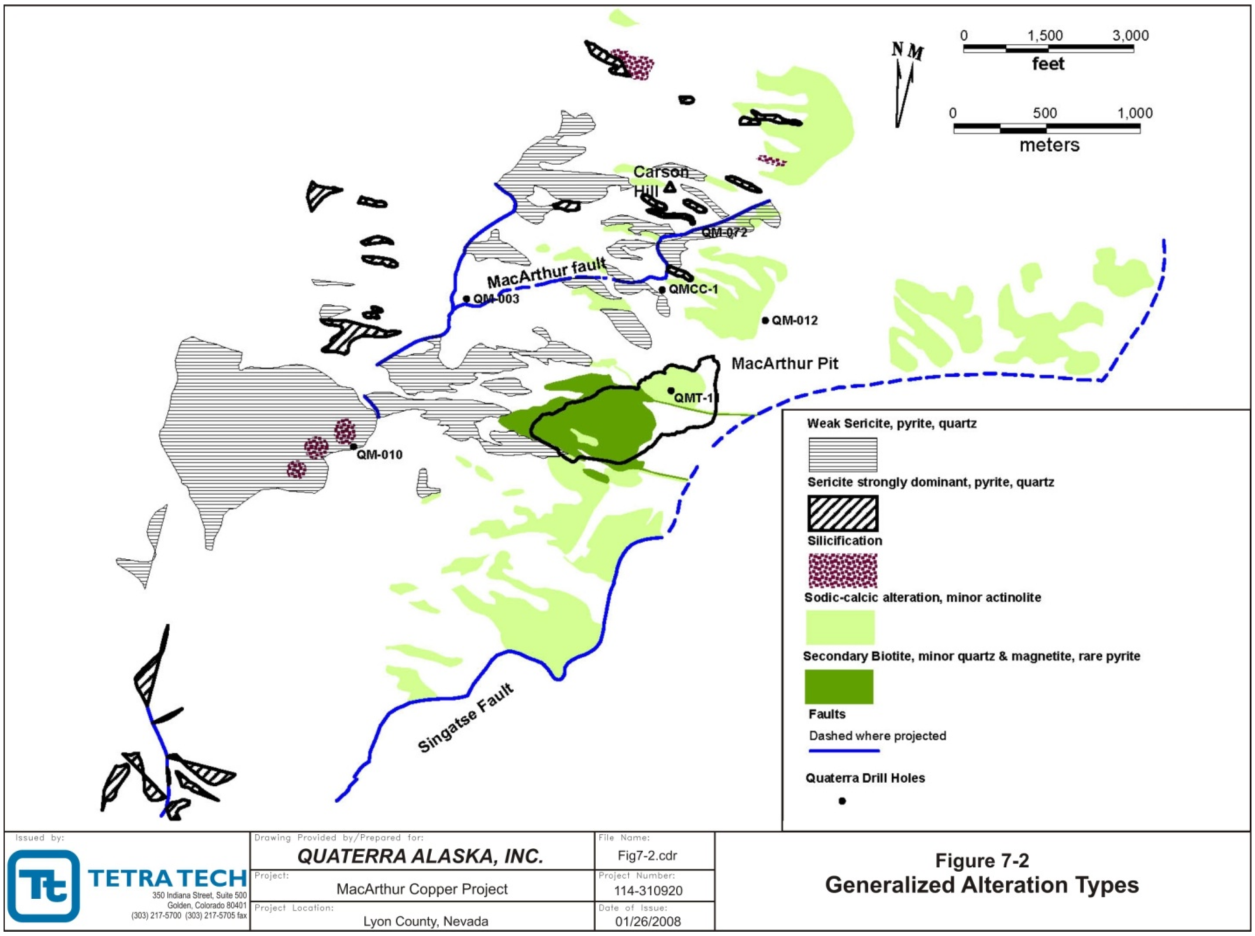


Figure 7-2: Generalized Alteration Types

7.4 MINERALIZATION

Copper mineralization has been identified across nearly the entire area investigated by Quaterra's drilling program at MacArthur and gives every indication of extending well beyond. As currently defined by drilling, copper mineralization covers an area of approximately two square miles as defined by drill holes on 500 feet to 250 feet spacing north of the MacArthur pit to approximately 150 feet spacing within the pit.

Oxide, chalcocite, and primary copper mineralization is hosted in both granodiorite and quartz monzonite, and in quartz biotite-hornblende (quartz monzonite) porphyry dikes all of middle Jurassic age. An insignificant percentage of oxide copper is also hosted in northwest striking andesite dikes that make up less than approximately one to two percent of the host rocks on the property. Fracturing and favorable ground preparation supplied the passage ways for the copper to migrate.

Copper oxide minerals are exposed throughout Quaterra's MacArthur property, in MacArthur pit walls as primarily green and greenish-blue chrysocolla $CuSiO_3 \cdot 2H_20$ along with black neotocite, aka copper wad (Cu, Fe, Mn) SiO_2, with very minor azurite $Cu_3(OH_2)(CO_3)$ and malachite

$Cu_2(OH_2)CO_3$, while tenorite (CuO) was identified with the electron microprobe (Schmidt, 1996). Copper-enriched limonite was identified by Anaconda as the mineral delafossite ($CuFeO_2$). Chalcocite has been identified in drill holes below and north of the MacArthur pit and in drilling throughout the property. The sulfides digenite (Cu_9S_5) and covellite (CuS) have been identified petrographically in drill cuttings. Bornite (Cu_5FeS_4) has also been identified petrographically in the Gallagher area. The oxide copper mineralization is fracture controlled, coating joint and fracture surfaces and within shears and faults. Both green and black copper oxides are frequently found on 1-5 millimeter fractures, as coatings and selvages and may be mixed with limonite. The fractures trend overall N60°W to N80°W (bearing 300° to 280° azimuth) and generally dip to the north. Limited turquoise is found on the property, mainly in small veinlets. On a minor scale, oxide copper mineralization replaces feldspar phenocrysts in the igneous host units, favoring andesite.

A significant amount of chalcocite has been intersected in drill holes. Chalcocite is seen on drill chips or drill core coating pyrite and replacing chalcopyrite as tiny, blackish “dustings” and thin to thick coatings, strongest when occurring on and near the MacArthur fault. Chalcopyrite is present as disseminations and veinlets, with or without chalcocite. As much of the historic and current drilling was stopped at shallow (<400 to 500 foot) depths, the scope and extent of chalcopyrite mineralization has not been fully defined.

Both copper oxide and chalcocite mineralization occur over approximately 10,000 feet east-west by 5,000 feet north-south. Copper oxides are structurally controlled coating fractures, joint surfaces, and developed as green or black “streaks” within shears and faults over several feet. Oxide mineralization occurs as tabular, flat-lying shapes extending with good continuity 150 feet below surface and less continuously up to 600 feet below surface. Chalcocite mineralization in tabular geometry ranges to 50 feet or more in thickness, mixed with or below oxide mineralization.

Primary chalcopyrite mineralization occurs as porphyry style disseminations or as veinlets in quartz monzonite associated with potassic alteration below both the oxide and chalcocite mineralization. Significant chalcopyrite mineralization was intersected from 4,500 feet to 5,500 feet north of the MacArthur pit in core holes QM-100 (0.58% Cu over 65 feet) and QM-164 (1.32% Cu over 64 feet) respectively. Both intercepts represent veinlet and disseminated primary mineralization, open to the north. Quaterra’s drilling program in the Gallagher area has delineated a zone of chalcopyrite mineralization that extends over a north-south distance of 2,500 feet. The primary sulfide zone has a defined width of 500 feet and extends to a depth of approximately 650 feet. The chalcopyrite mineralization is not included in the current mine plan of the PEA.

Petrographic study of drill core from holes QM-100 and QM-164 describes veinlet and disseminated copper mineralization as well as copper-bearing sheared, milled quartz veining underlain by potassic feldspar flooding.

8 DEPOSIT TYPES

The MacArthur deposit is a supergene enriched, oxidized porphyry copper system. Although the porphyry system likely developed in near-vertical geometry, regional studies by Proffett and Dilles (1984) suggest the MacArthur area is tilted westerly approximately 60 to 90 degrees from its original vertical position and extended to the east so that the map view is actually a structural cross section. The original northwest strike of the near vertical porphyry dikes resulted in a northerly dip of the structures with the post mineral tilting (Figure 8-1).

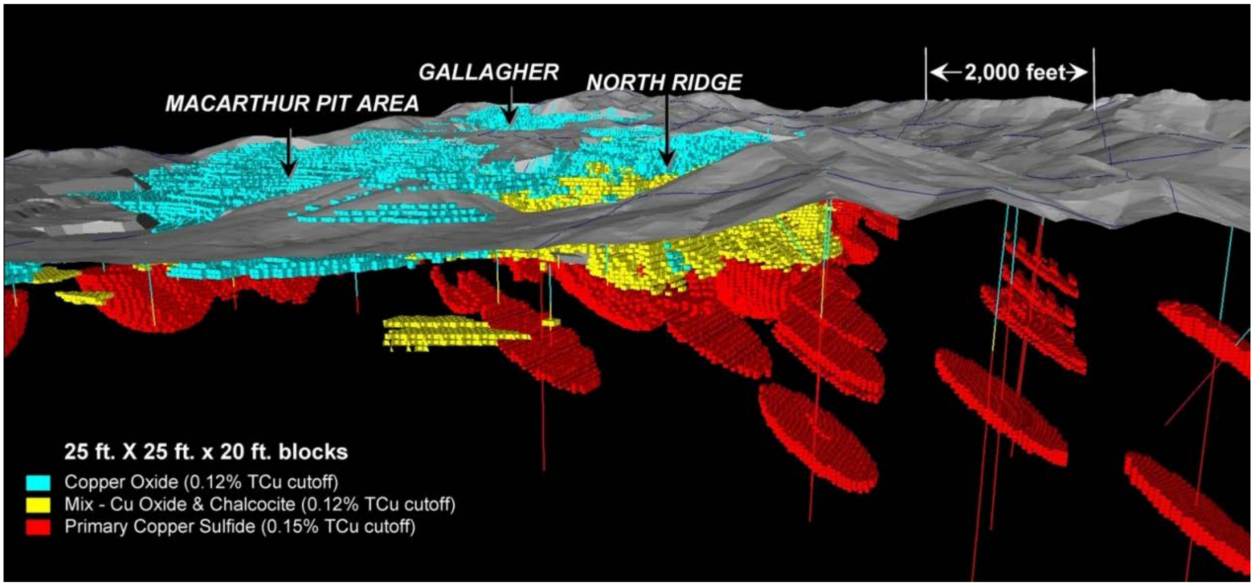


Figure 8-1: Datamine© View of Resource Block Model Looking West

(Figure 8-1 View of Datamine© resource block model with planned open pits looking West below the North Ridge showing the northerly dip of primary sulfide mineralization (red).)

The alteration visible in outcrops and drill samples is consistent with the west tilted, near horizontal orientation of the porphyry system. Phyllic alteration from the upper portion of the porphyry system dominates to the west. The alteration grades to potassic in the central MacArthur pit area and pervasive sodic-calcic alteration dominates in the eastern portions of the MacArthur pit and in the far northeastern portion of the district.

Copper occurrences in the MacArthur pit area are related to primary copper sulfides associated with the porphyry copper center. The primary chalcopyrite ($CuFeS_2$) was enriched by supergene chalcocite (Cu_2S) and later exposed to oxidation forming chrysocolla ($CuSiO_3$) and black copper wad (Cu,Fe,Mn SiO_2). In the North Ridge area the chalcocite blanket shows only minor oxidation. The supergene blanket follows current topography except to the north of 14,691,501E (approximately) where it has a shallow dip to the north (Figure 8-2 and Figure 8-3).

Primary porphyry copper sulfides have also been intersected north of the North Ridge area in drill thicknesses up to 100 feet and in the Gallagher area. These intercepts maybe related to the MacArthur pit porphyry center or a new, yet to be discovered porphyry copper deposit.

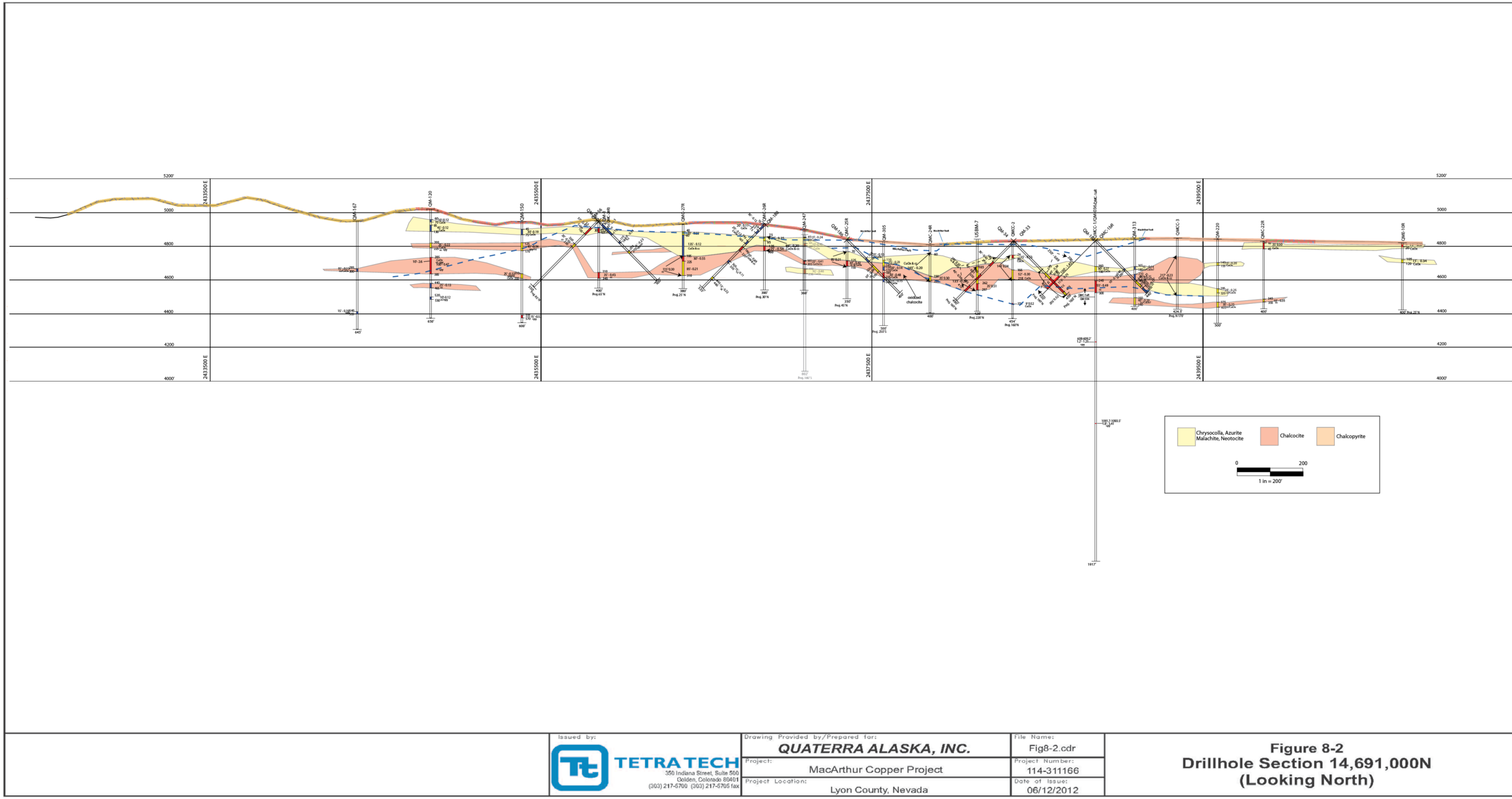


Figure 8-2: East-West Section 14,691,000N (Looking North)

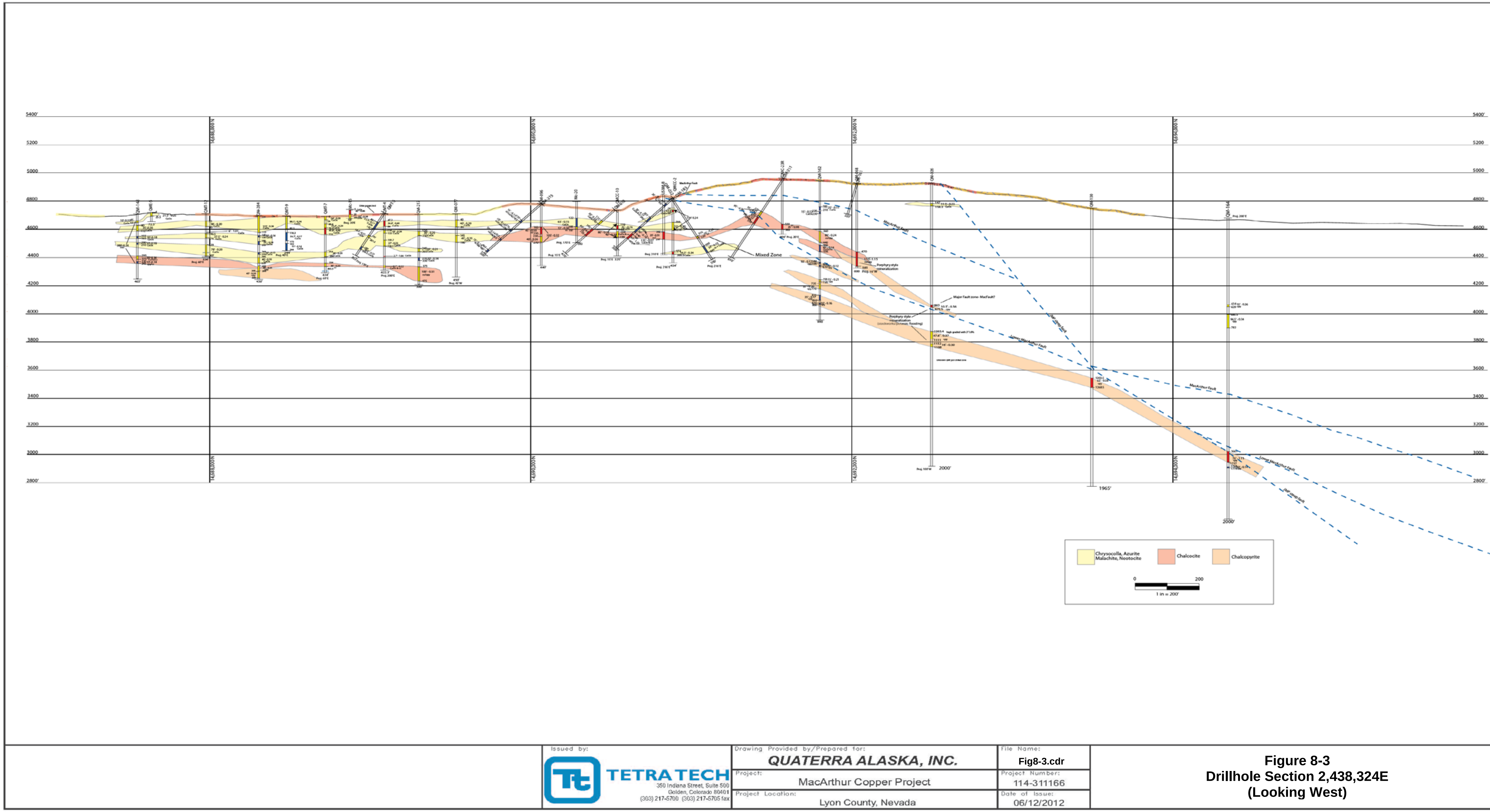


Figure 8-3: North- South Section 2,438,324 (Looking West)

8.1 EXPLORATION

Starting in April 2007 and continuing through October 2008, and from December 2009 through November 2011, Quaterra completed extensive reverse circulation and core drilling at the MacArthur property. Drill results through October 2008 coupled with 1972-1973 Anaconda drilling provided the data for Tetra Tech to publish the February 2009, revised March 2009, MacArthur NI43-101 Technical Report. An additional 77 drill holes completed through September 2010 formed the basis for the January 2011 NI 43-101 Technical Report. During 2011 an additional 152 holes were completed, and are the basis of this updated Technical Report.

There are three different mineralization zones encountered at MacArthur. All three mineralization zones - oxide, mixed chalcocite/oxide, and primary sulfide - have grown with additional drilling and none are yet entirely closed off.

8.2 OXIDE ZONE EXPLORATION

Extents of the oxide mineralization on the property remain open to the west and are only partially defined to the south.

Five thousand feet west of the MacArthur pit, Quaterra holes QM-133 and QM-153 intersected 0.27% Cu over 235 feet and 0.16% Cu over 125 feet, respectively, of oxide and acid soluble copper. The mineralization is open 1,000 feet farther to the west.

Southeast of the MacArthur pit, holes spaced from 500 to 1,000 feet apart contain 0.1 to 0.3% Cu intercepts. Drill holes QM-142, QM-108, and QM-140 encountered 0.21% Cu over 50 feet to 0.31% Cu over 10 feet in an area that remains untested for 3000 feet to the Shuman area (3,500 feet south of the MacArthur pit) where oxide intercepts of 0.24% Cu over 45 feet and 0.39% Cu over 30 feet were encountered from surface. Mineralization in these holes (referred to as the Shuman drill holes) is open in all directions, but obscured to the south by Tertiary volcanic cover.

8.3 CHALCOCITE/OXIDE ZONE EXPLORATION

Chalcocite/oxide mineralization remains open to various degrees in all directions, in light of the 500 foot drill spacing. Chalcocite mineralization is partially open to the northwest.

8.4 PRIMARY SULFIDE ZONE EXPLORATION

Primary, porphyry-style copper mineralization has been encountered at the North Porphyry Target area and is described in the following paragraph. In the Gallagher area, primary copper mineralization occurs from 450 feet depth in QM-10, with 0.43% Cu over 155 feet to 0.74% over 76 feet in QM-46 from 1,279 feet depth as chalcopyrite disseminations and veinlets. Additional drilling to target primary sulfide mineralization is warranted for as there are only eight holes exceeding 800 feet depth over an approximate one half square mile area.

Quaterra's drilling program at the North Porphyry Target, some 3,000 feet north of the MacArthur pit, encountered 115 feet of mineralization (partially enriched with chalcocite) averaging 1.15% Cu at a depth of 470 feet in drill hole QM-68. A similar section of mineralization in QM-070 (500 feet east of QM-068) averaged 1.02% Cu over a thickness of 45 feet at a depth of 435 feet. Together with mineralized intercepts in QM-072, (500 feet east of QM-070) which cut 15 feet of 1.2% Cu, the results indicated a possible porphyry center in the foot wall of the MacArthur fault. In 2010 this concept was favorably tested 1,500 feet north of QM-68 where drill hole QM-100 intersected 0.58% Cu over 65’ from 1203.5 feet. During 2011, QM-100 was offset 1,000 feet north by QM-164 returning 1.32% Cu over 64 feet from 1,673 feet depth. These primary sulfide intercepts define a 6,000 foot mineralized zone (corridor), including the oxide mineralization at the MacArthur Pit north to the sulfide intercepts in QM-164, untested 500 feet east and west of QM-100 and QM-164 and open to the north.

9 EXPLORATION

9.1 GEOPHYSICS

Quaterra contracted three surveys at the MacArthur Project in 2011 and 2012. A borehole geophysical survey and a surface Induced Polarization/Resistivity (IPR) survey were carried out by Zonge International in 2011. A detailed helicopter magnetic survey was flown by Geosolutions Pty. Ltd. in 2012. These surveys supplement previous geophysical work on the property that includes: a 2009 IPR survey carried out by Zonge; a 2007 helicopter magnetic survey carried out by EDCON-PRJ; a series of historic aeromagnetic surveys (1966 to 1975) available in analog form from the Anaconda Archives; and a series of historic IPR surveys (1963 – 1964) carried out by Kennecott Exploration Services/Bear Creek Mining Company and Superior Oil.

The 2009 and 2011 IPR surveys were designed to confirm the reliability of the earlier surveys and to further define the depth extent of the IP anomalies. The 1963-64 data indicate that the zone of anomalous IP response is typically flat-lying with a thickness of less than 1,000 feet and does NOT extend beyond a depth of 1500 feet. Comparison of data from the surveys done more than 45 years apart is that the 1963-64 Kennecott data is of good quality and can be used effectively to define IP anomalous zones within the upper 1,000-1,200 feet of the subsurface. However beyond that depth the 1963-64 data cannot effectively resolve the bottom of the IP anomalies nor determine if any of the anomalies extend to great depths. The modern data sets show this increased depth of exploration is important. For example a portion of the anomalous IP zone on line 306075E is depth limited however the anomalies on the north and south ends of the lines extend much deeper, to a depth exceeding 2,000 feet.

The 2007 EDCON-PRJ high-resolution, helicopter-borne aeromagnetic survey was flown over the MacArthur Copper Project. The survey was designed so that data from historic Anaconda surveys (1966 to 1975) could be merged with the new data. The historic surveys were recovered from the Anaconda Archive collection maintained by the American Heritage Center, University of Wyoming. EDCON-PRJ digitized the historic survey data from the paper maps, as no digital data was available for those surveys.

Note that all modern geophysical surveys have been run on Nad27 UTM Zone 1N metric grids, but for purposes of consistency, depths and distances are given in feet.

9.1.1 IP/Resistivity Surveys

9.1.1.1 2011 Work

Surface and down hole IP/resistivity (IPR) surveys were run at the MacArthur Project area in 2011. The surface IPR survey was conducted by Zonge International in February of 2011. The primary purpose was to continue to cover the project area with modern high quality IPR data to replace the historic data collected by Kennecott and Superior Oil in the early 1960's. The goal of the survey was to map sulfide and alteration response at depth within the central alteration zone (Figure 9-1) and beneath volcanic cover adjacent to the alteration zone. Four lines of surface IPR were run in 2011. The quality of the data recorded is good. Previous interpretations are supported and a number of new targets have been identified. Of particular interest are targets identified beneath volcanic cover to the north and west of the main alteration zone and low resistivity/high IP phase anomalies which continue to depth indicating possible fluid feeder zones from depth.

Figure 9-2 through Figure 9-7 show the pseudo-sections and inversion models for three of the 2011 IPR lines. In each figure the top panel shows the inversion model for IP or resistivity. The observed data is shown in the middle panel and the bottom panel is the calculated pseudo-section generated from the inversion model.

The IP models and pseudo-sections for lines 4300 (Figure 9-2) and 4900 (Figure 9-4) run over the Gallagher Zone and continue into the volcanic cover to the north. Both lines show deeper IP response continuing under the near surface volcanic cover (see black arrow). Although the amplitude of the deeper response is lower, 20 to 30 milliradian (mrads) at depth versus up to 80 mrads at the surface this may not accurately reflect sulfide content at depth. Also note that the base of the IP response is poorly resolved and there is some evidence of continuation to depth on all four lines (L4300, L4900, L5300 and L7500) runs in 2011. Figure 9-3 and Figure 9-5 on line L4300 and L4900 respectively show lower resistivity zones associated with the higher IP responses. These low resistivity zones are interpreted to indicate alteration associated with mineralization. A possible interpretation of the deeper IP and resistivity anomalies is that they are feeder zones for the shallower, flat lying mineralization.

Figure 9-6 and Figure 9-7 show IPR models and pseudo-sections for line L7500 located to the east of the North Porphyry target. The top of the IP response is approximately 800 feet (250 meters) depth and the response is weak. A low resistivity zone surrounds the IP response. This response may indicate a porphyry system style of zonation.

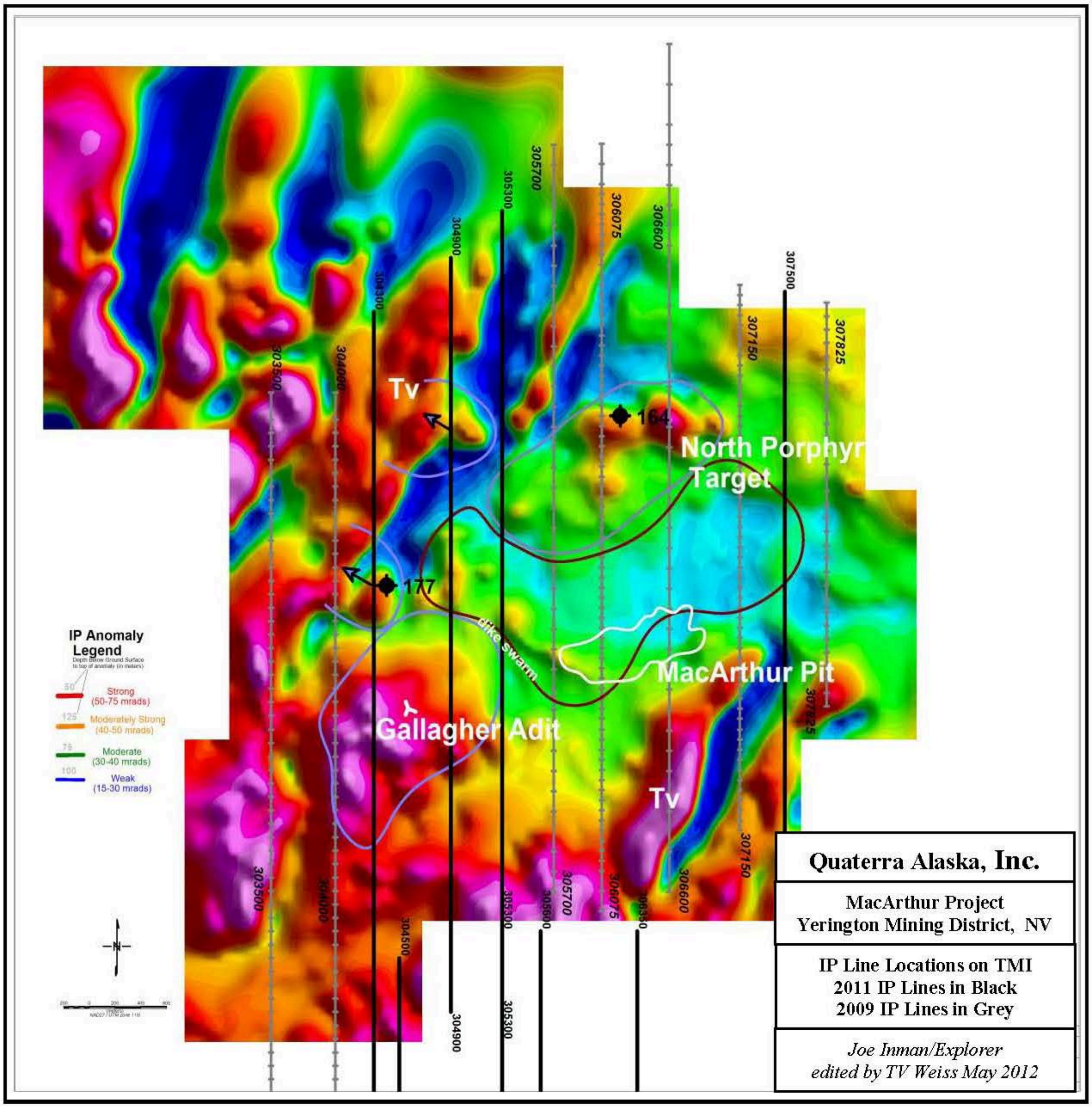


Figure 9-1: IPR line locations over the central MacArthur Project area.

(Figure 9-1: The 2011 lines are shown in black and the 2009 lines in grey. The lines are plotted on an image of the Reduced to Pole - Total Magnetic Intensity (RTP) data acquired by EDCON-PRJ in 2007. The magnetic low located between the Gallagher Adit and the North Porphyry target is interpreted to represent the central alteration zone which is targeted by the IPR data set. The location of drill holes QM-164 and QM-177 used for borehole IP surveys are shown and are further discussed within the text).

The second ground geophysical program carried out at the MacArthur Project in 2011 was a down hole IP/resistivity survey utilizing drill holes QM-164 and QM-177. This work was carried out by Zonge International in August 2011. The surveys were designed to explore for sulfide response at depths greater than can be resolved using surface arrays. The goal was to determine which direction from the drill hole sulfides occur. The drill hole acts as the pathway to place an electrode to depth. The technique maps sulfides at or above the level of the buried electrode. Drill holes QM-164 and 177 are of moderate depth and were used to test the process for emplacing electrodes in holes that are difficult to keep open (Figure 9-8 and Figure 9-9).

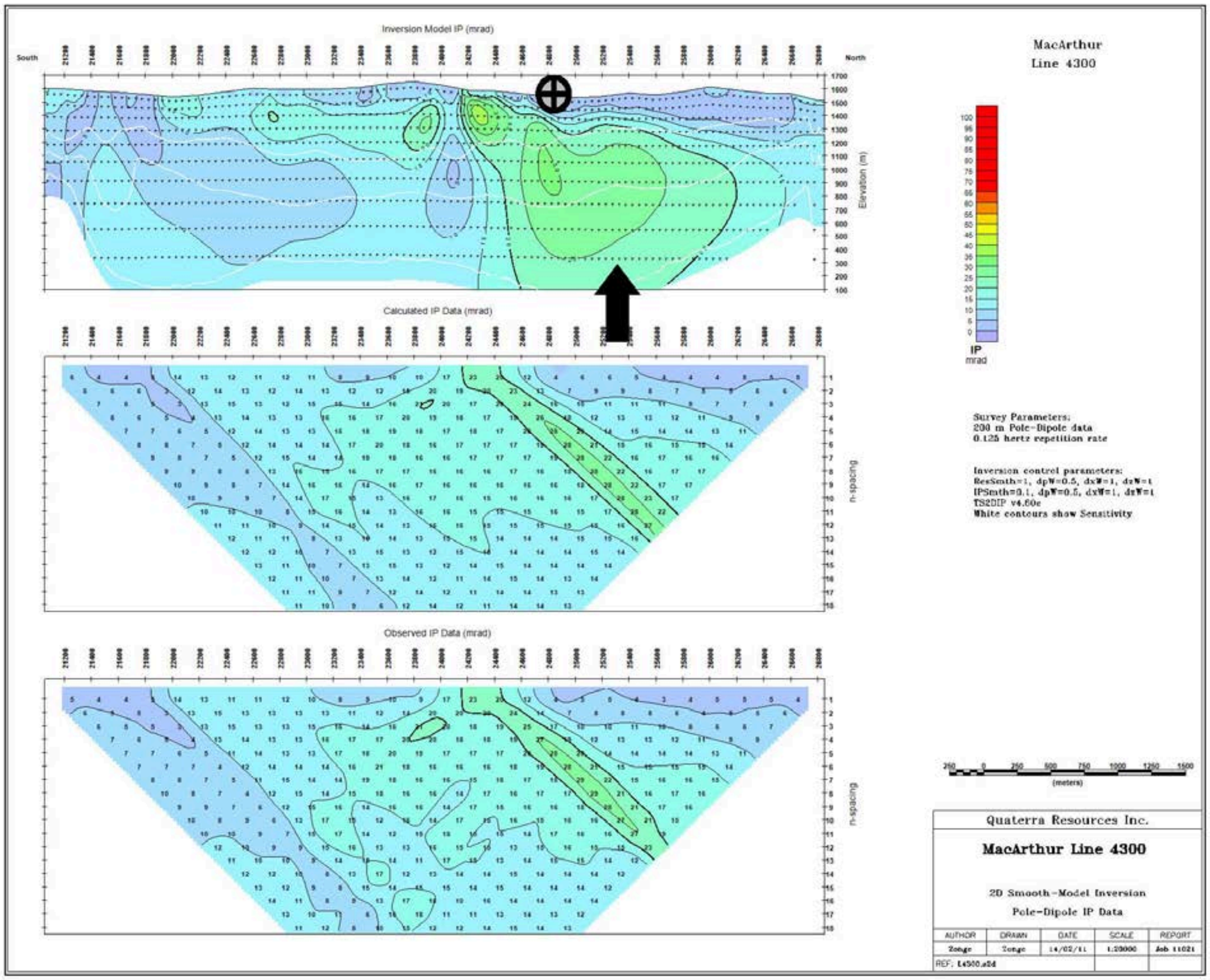


Figure 9-2: Line 4300 (304300E) IP pseudo-section and inverted phase/depth model

(Figure 9-2 - Line 4300 runs over the western side of the Gallagher target area. The primary feature of interest on this line is extension of the near surface phase anomaly to depth (800-1000 feet (200 to 300 meters)) under volcanic cover to the north (see black arrow). Approximate location of QM-177(328 feet (100 meters) to east) is shown as circled + sign.)

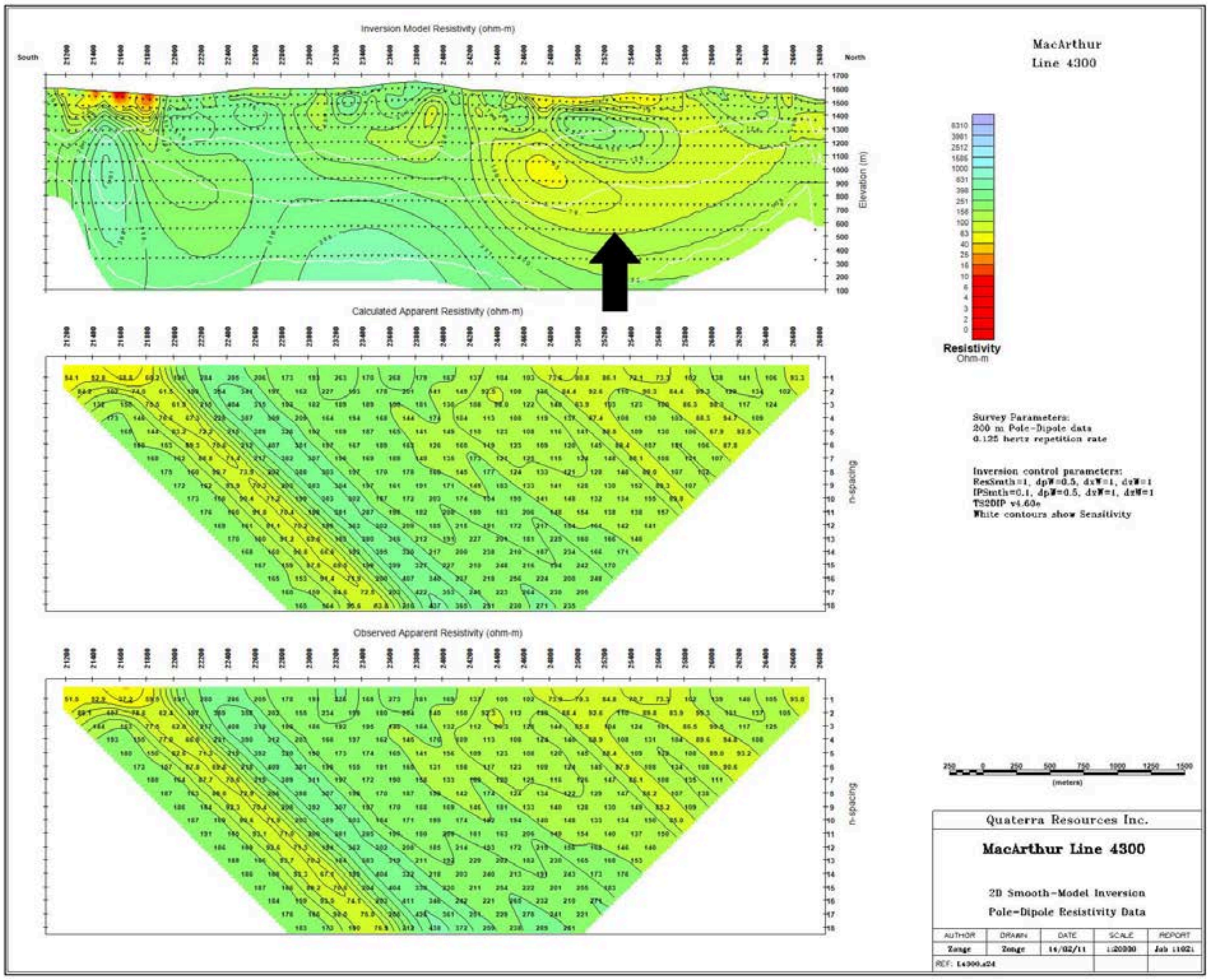


Figure 9-3: Line 4300 Resistivity pseudo-section and inverted resistivity/depth model

(Figure 9-3 - Line 4300 (304300E) The weak low resistivity zone may be associated with alteration coincident with the buried sulfide system (see black arrow).

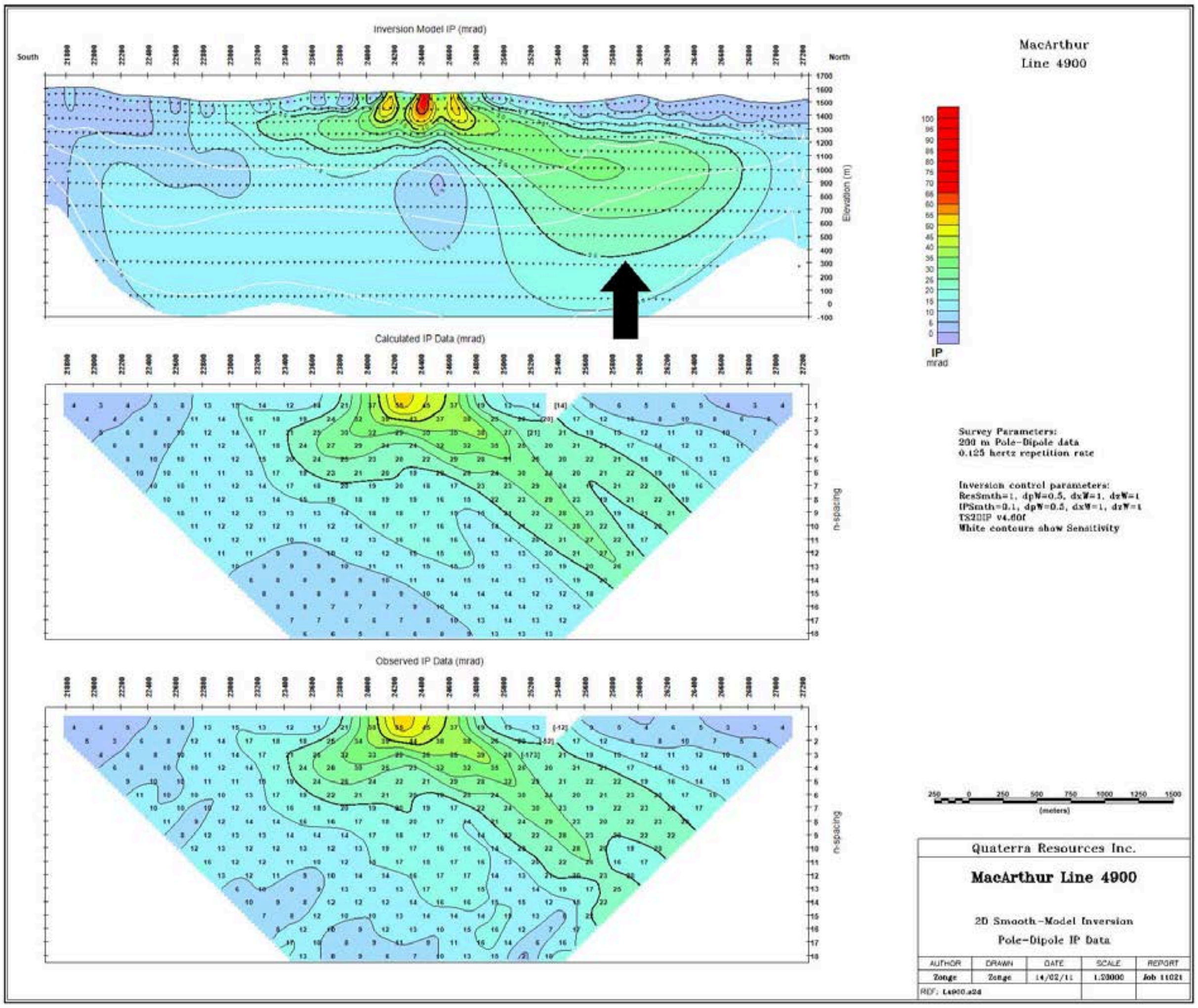


Figure 9-4: Line 4900 IP pseudo-section and inverted phase/depth model

Figure 9-4 - Line 4900 (304900E) runs over the eastern side of the Gallagher target area. Note phase response beneath volcanic cover (black arrow).

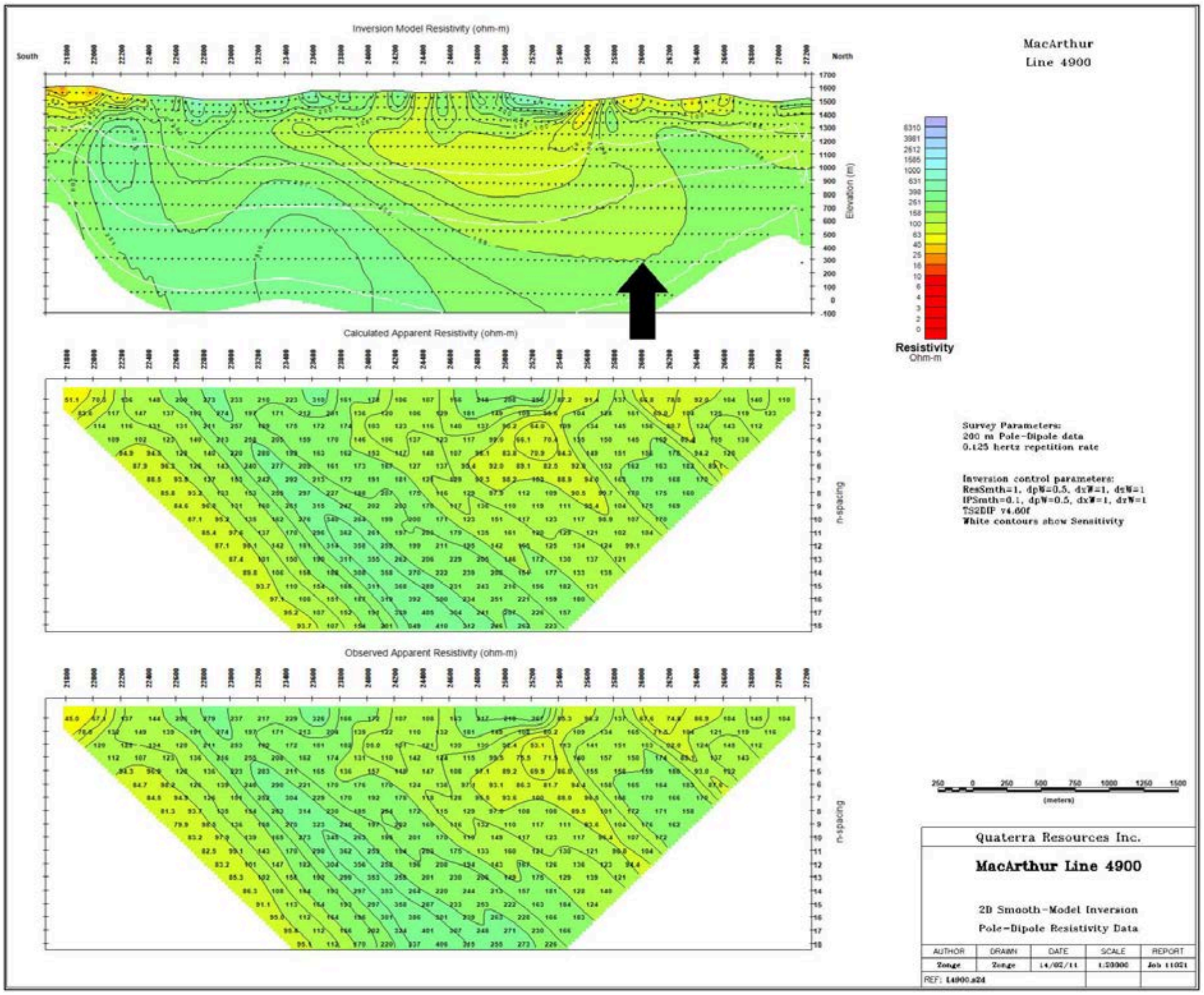


Figure 9-5: Line 4900 Resistivity pseudo-section and inverted resistivity/depth model.

(Figure 9-5 - Line 4900 (304900E) The weak low resistivity zone is possible alteration coincident with the buried sulfide system (black arrow)).

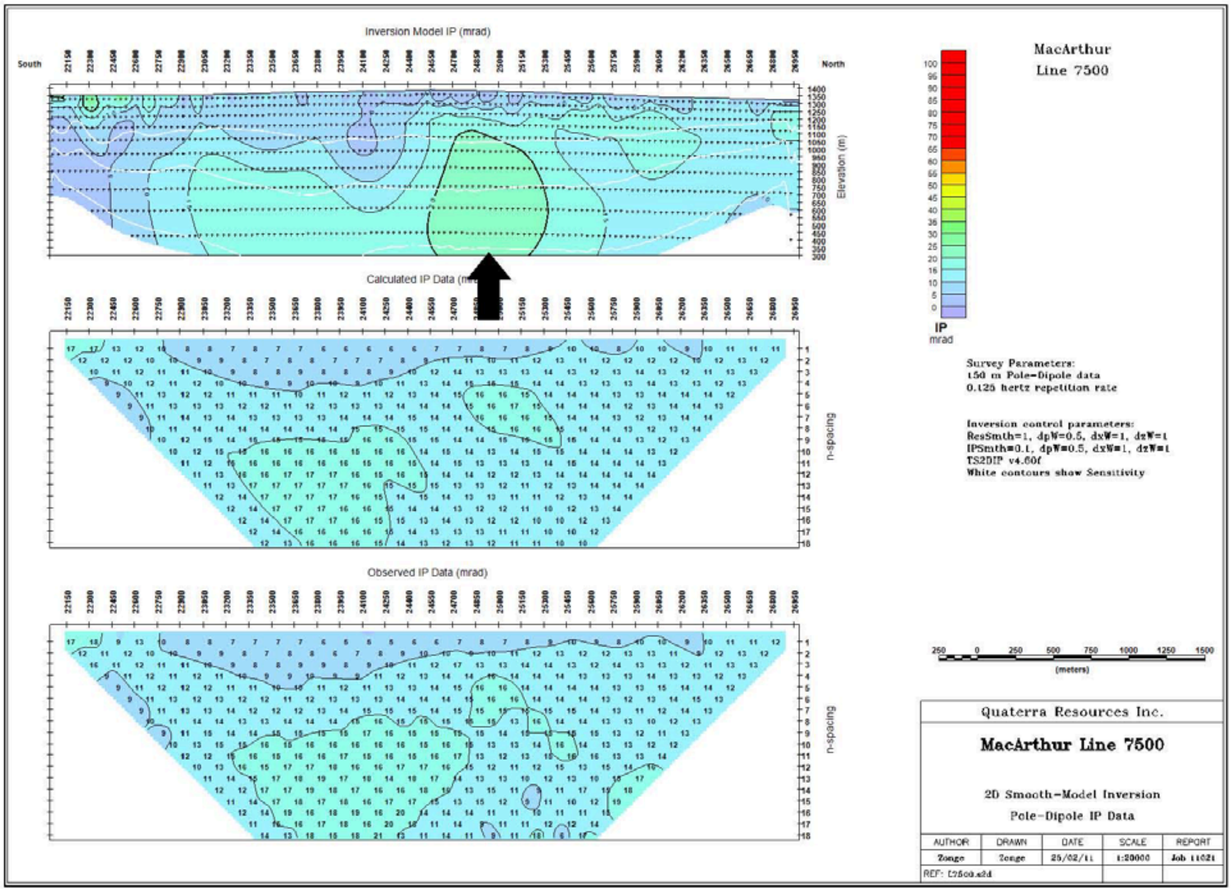


Figure 9-6: Line 7500 IP pseudo-section and inverted phase/depth model

(Figure 9-6 - Line 7500 (307500E) cuts off the MacArthur Pit North Target to the east. The weak phase anomaly (20+ mrads) indicates the continuation of sulfides to depth.)

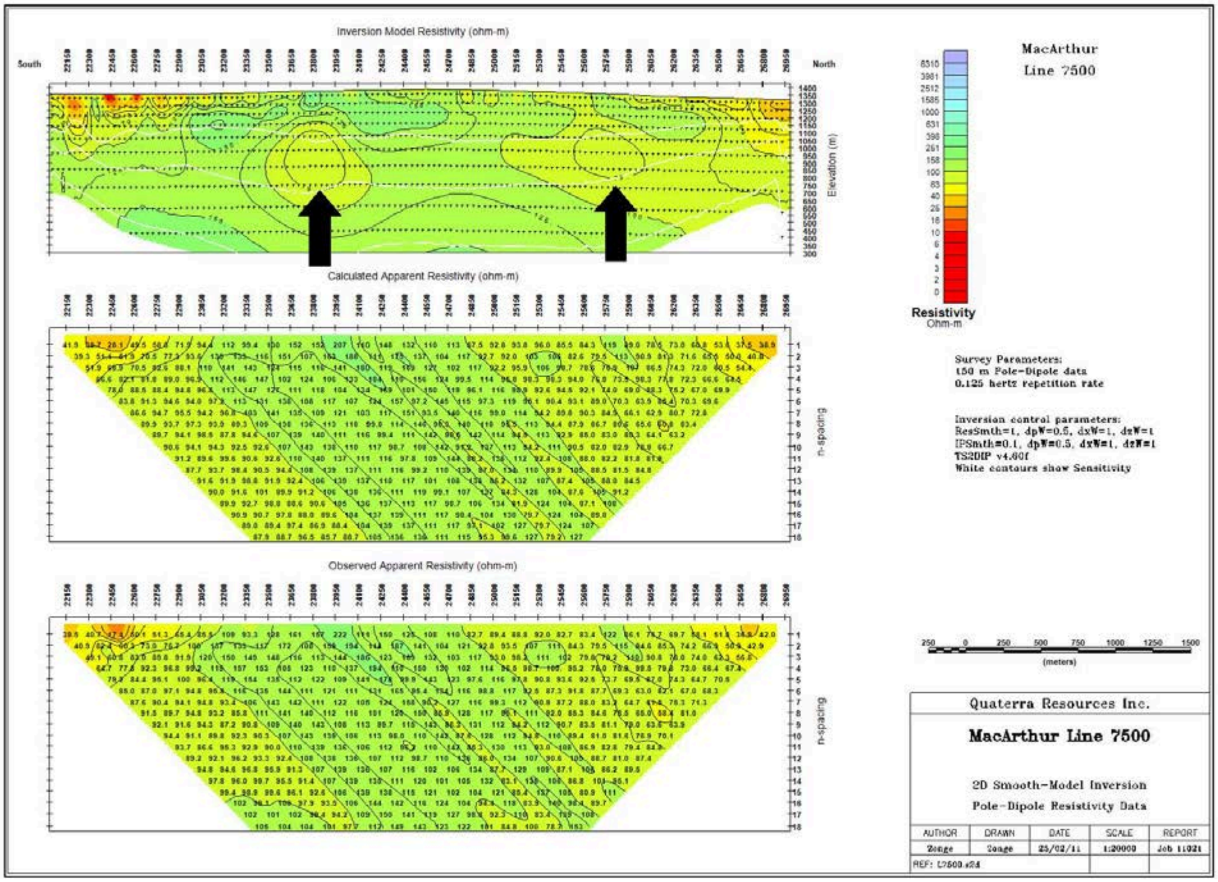


Figure 9-7: Line 7500 Resistivity pseudo-section and inverted resistivity/depth model

(Figure 9-7 - Line 7500 (307500E) The weak resistivity lows surround the IP response and may indicate a classic porphyry alteration zonation.)

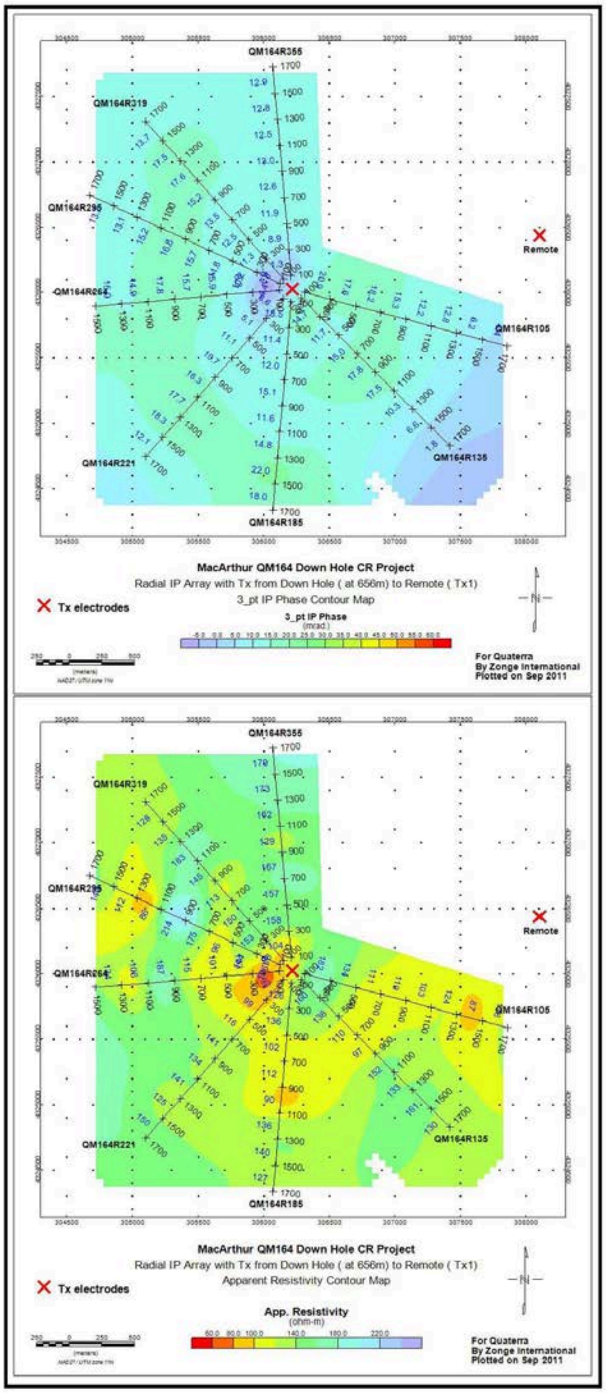


Figure 9-8: QM-164 down hole electrode to remote electrode transmitter pair

(Figure 9-8 – IP response plotted in upper image and resistivity response in lower image. No significant IP response was detected in this drill hole.)

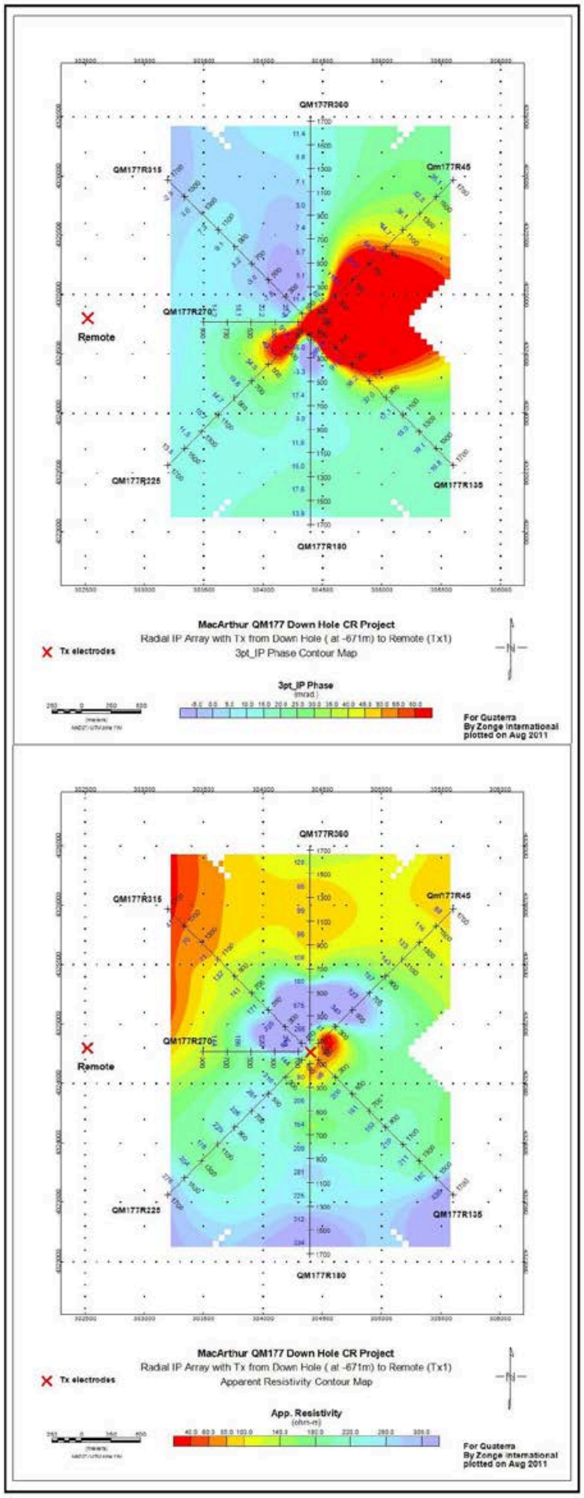


Figure 9-9: QM-177 down hole electrode to remote electrode transmitter pair

(Figure 9-9 - IP response plotted in upper image and resistivity response in lower image. A significant IP response is located to the east of the drill hole in this image.)

9.1.1.2 2009 and older IP/Resistivity (IPR) Surveys

Seven lines of surface IPR were run in 2009. These lines together with the 2011 IPR lines make up the modern data set which replaces the historic data sets in this area. The purpose of this survey was to confirm the results from historic early 1960's surveys run by Kennecott and Superior Oil. Those surveys although useful in initially detecting sulfide response were recorded on old generation analog systems and have limited depth extent and unknown quality.

Figure 9-10 shows the location of the IPR lines conducted in 1963-64 by Kennecott Exploration Services (KES) for the Bear Creek Mining Company. KES collected 11 lines of IPR data which are plotted in black. The Superior Oil IP lines have not been used in this compilation as the data quality and line locations are questionable.

The 2009 Zonge data confirmed the results of this previous work, explored to greater depth with higher quality data and essentially replaced the older data. Seven (7) lines were surveyed and are plotted in white (Figure 9-10).

To put the 2009 and 2011 surveys in perspective it has been observed that high quality IPR surveys are capable of sensing and mapping metallic sulfide concentrations of pyrite and/or chalcopyrite as low as 1-2% by volume. A significant volume of rock containing 3-5% pyrite/chalcopyrite will result in an IP anomaly exceeding 30-40 milliradians, whereas 7-10% metallic sulfides will result in anomalies exceeding 75 milliradians. (Nelson and Van Voorhis, 1983) Both the 2009 and 2011 surveys are of this high quality.

A number of gaps existed in the 2009 data set which were later filled in by the 2011 survey (described above) or are yet to be filled in by future work.

Figure 9-11 is a summary interpretation of the historic and 2009 IP data sets plotted on a magnetic susceptibility inversion image. It is important to note that the stronger amplitude IP responses, shown as red bars along the lines, generally reflect shallow responses. The fact that the deeper responses are lower amplitude may not reflect relative sulfide content accurately. This is important as the exploration program under volcanic cover develops.

Note that the Central Zone (outlined in brown) is characterized by low magnetic susceptibility, probably destruction of magnetite, and high IP effect due to increased sulfides (Figure 9-1 and Figure 9-11). Where the NW trending Qmp dike swarm occurs at the SW edge of the Central Zone (Figure 9-1) the magnetic susceptibility increases due to magnetite in the dikes. The IP effect is high in this area as well.

The North Porphyry Target, located NNE of the Central Zone (outlined in grey, Figure 9-1 and Figure 9-11) is characterized by high IP effect and increased magnetic susceptibility due to quartz porphyry dikes. The strongest IP anomalies are coincident with the more intense magnetic highs.

The Gallagher Adit area occurs to the SW of the Central Zone. Similar to the North Porphyry target, the northern edge of the Gallagher Adit area is characterized by a zone of moderate magnetic susceptibility (Qmp) with zones of moderately strong to strong IP anomalies.

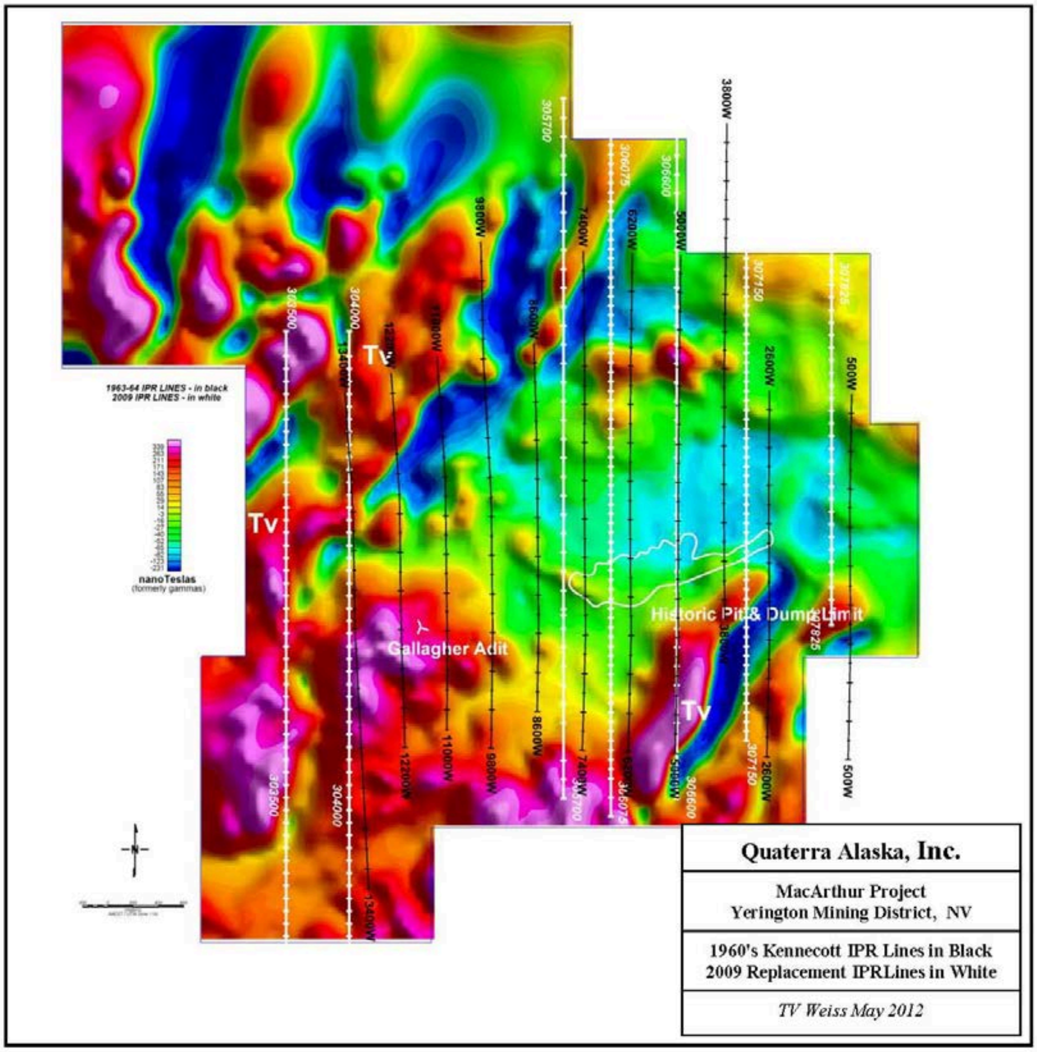


Figure 9-10: Line location of the 1960's Kennecott lines (in black) and the 2009 replacement line (in white).

The NW and NW Gallagher Targets are shown by the grey outlines and arrows pointing under the Tertiary volcanic cover (Figure 9-1 and Figure 9-11). Alteration and copper mineralization as well as zones of coincident high magnetic susceptibility and IP response continue to the contact with the post-mineral Tertiary volcanic front in the western portion of Quaterra's claim block. Lines 4300 and 4900 from the 2011 survey indicate those IP responses continue under the volcanic cover.

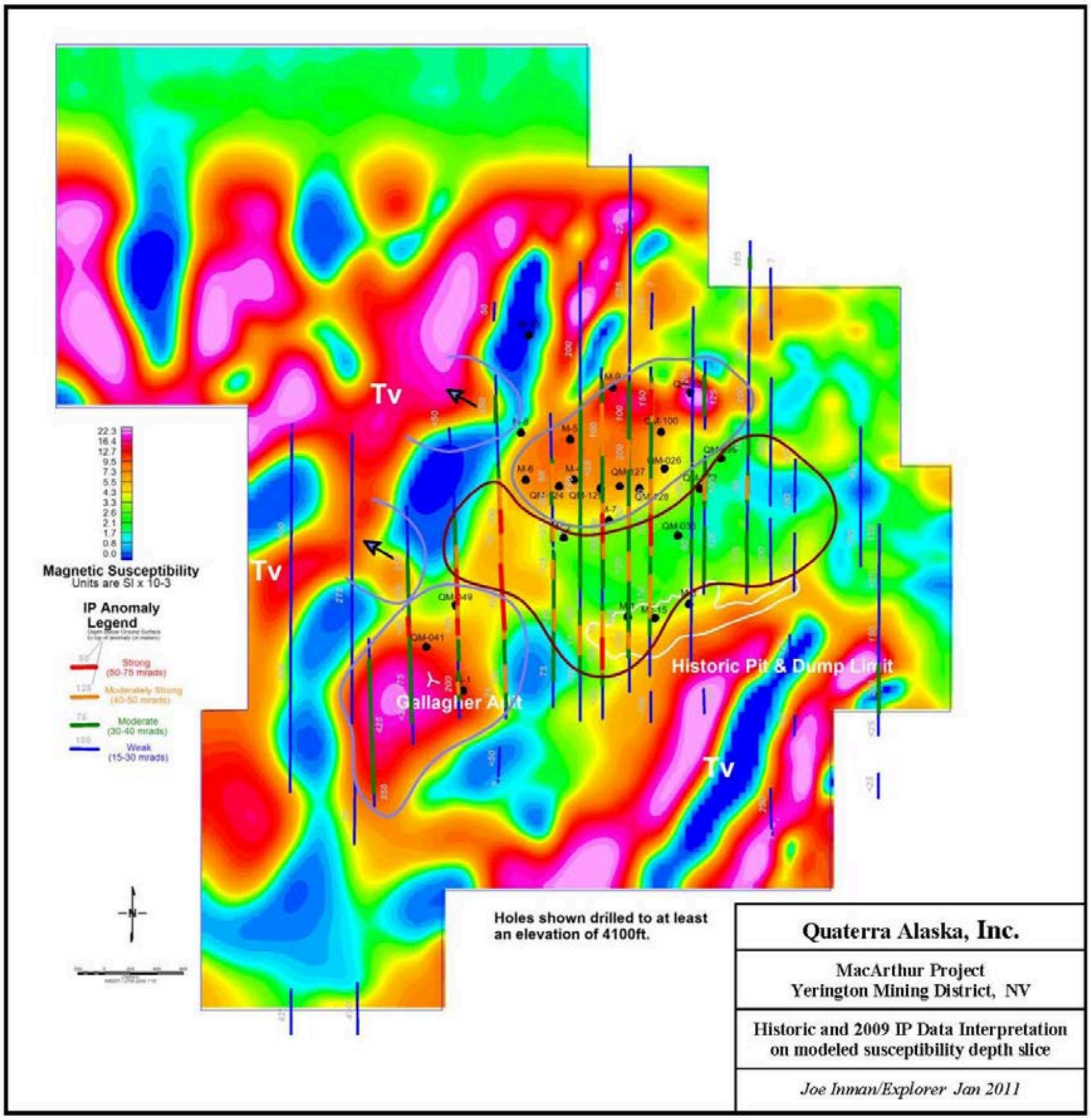


Figure 9-11: Historic and 2009 IP data on a modeled magnetic susceptibility depth slice

(Figure 9-11 A qualitative interpretation of the historic and 2009 IP data plotted on a modeled magnetic susceptibility depth slice. Note all of the surface IP work at MacArthur, including the 2011 survey (not shown here) cover this central zone between the Gallagher Adit and the North Porphyry target.)

Figure 9-12 shows the inverted IP and resistivity model for line 6075 (306075E) recorded in 2009. The line runs directly over the MacArthur pit to North Porphyry target area. The IP model shows a flat lying near surface response with deep responses to the north and south of the Central zone. The resistivity model shows the low resistivity alteration pattern associated with the modeled IP response. This is a good example of strong IP response surrounded by low resistivity due to alteration. The deep, vertical features may be reflecting feeder zones continuing to depth.

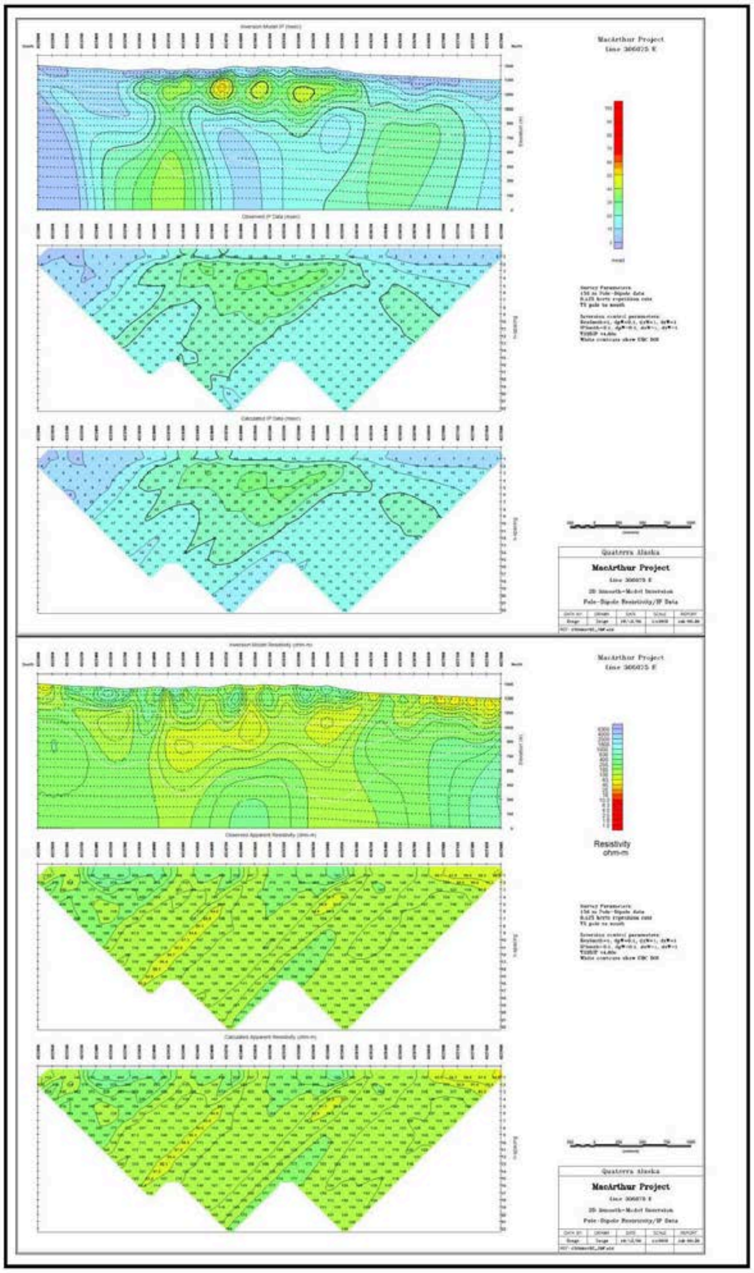


Figure 9-12: Inversion model and pseudo-sections for line 6075 recorded in 2009.

Figure 9-12 Line 6075 (306075E) The line runs directly over the MacArthur pit to North Porphyry target area. The importance of this line is that it indicates that both the IP response and low resistivity alteration zones are open to depth. One interpretation is that these deep features are feeder zones for mineralizing fluids coming from depth.

9.1.2 Airborne Magnetic Surveys

9.1.2.1 2012 Survey

A small 428 line kilometer airborne magnetic survey was flown over the northern extension of the MacArthur Project area in April of 2012 by Geosolution's Pty Ltd. The block is located to the north of the previous survey as shown in Figure 9-13. The flight line direction is N-S, line spacing 164 feet (50 meters) and the attempted sensor terrain clearance is 98 feet (30 meters) although the mean sensor clearance is somewhat higher, 148 feet (45 meters) due to steep topography which required the pilot to fly higher in some areas.

The objectives of the survey were: 1) to map the contact between batholithic intrusive and sedimentary basement; 2) to explore for magnetite rich skarn bodies along this contact; 3) to map quartz porphyry dikes and other intrusives; and 4) to map alteration of volcanic rocks, destruction of magnetite, associated with porphyry style mineralization in this area. Interpretation of this data set is currently in progress and results will not be discussed in this report.

The particular magnetometer system that was used was designed to maintain high frequency information that will allow 3D modeling of the broad band magnetic data set. The 3D model will be used to explore beneath volcanic cover which masks the magnetic response of deeper units of interest.

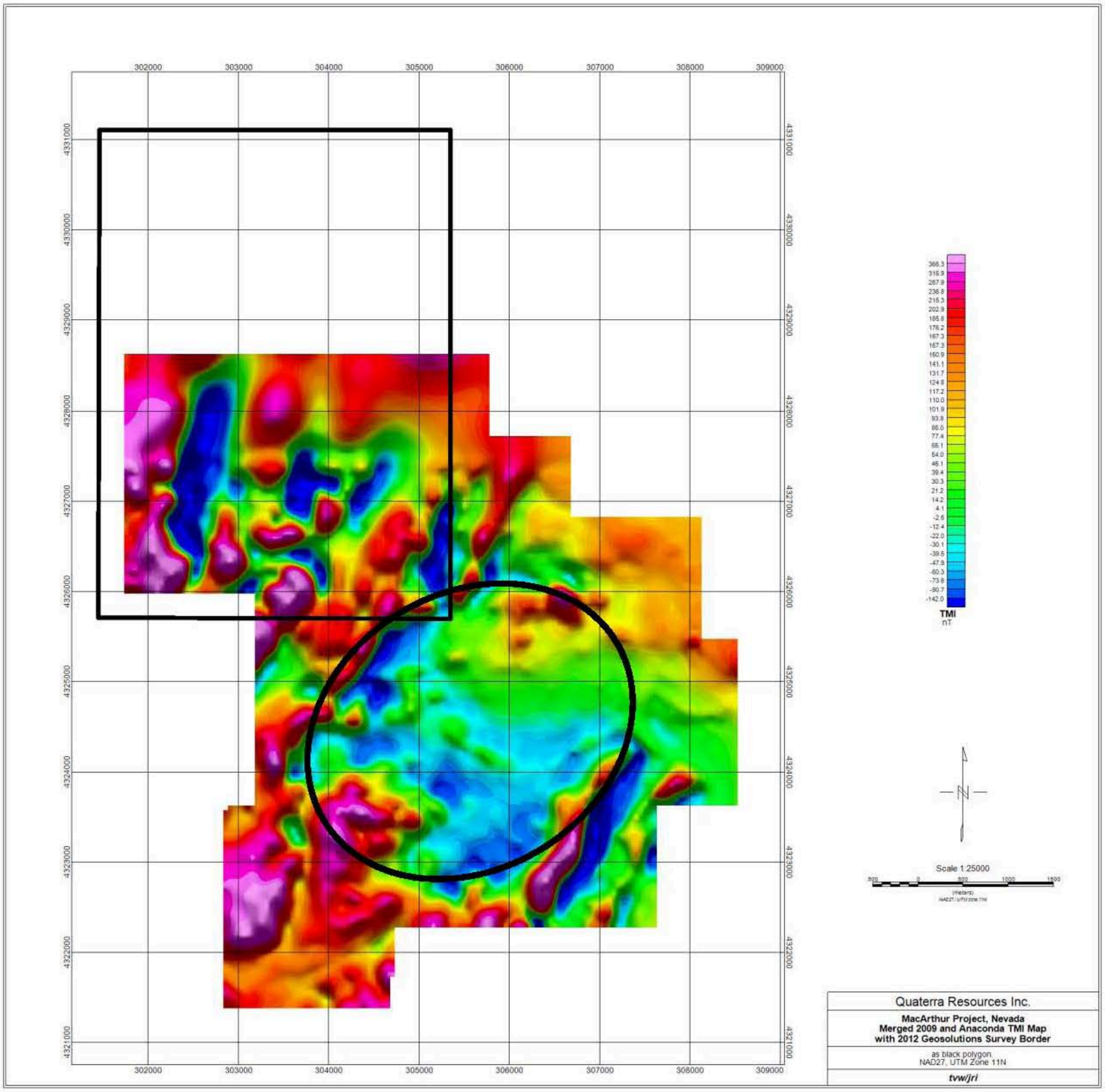


Figure 9-13: Location of the 2012 detailed helicopter magnetic survey

(Figure 9-13 – Location of the 2012 detailed helicopter magnetic survey (black polygon) with respect to the previous magnetic and IP/resistivity work. Note the NE-SW striking ellipse is interpreted to be associated with the mineralized alteration zone.)

9.1.2.2 2007 Survey

The 2007 detailed magnetic data set was flown over the MacArthur land block by Edcon-PRJ (see Figure 9-1 and Figure 9-13). The data set was flown with a stinger mounted system at a terrain clearance of 328 feet (100 meters). The combination of a stinger mounted system and the large terrain clearance results in the removal of the high frequency information required in 3D modeling. Because of the frequency content difference between the 2007 and 2012 data sets the initial modeling program will not use the 2007 data. Ultimately the data sets will be merged and modeled together but as a second pass at the modeling.

The 2007 MacArthur dataset (Figure 9-1) illustrates several features that correlate to the geology, alteration and mineralization at MacArthur. The magnetic field in the MacArthur area is dominated by intense highs and lows caused by Tertiary volcanic rocks. The northwest quarter and the southeast corner of the MacArthur claim block contain highly magnetic volcanic units. These areas are denoted in the figure by “Tv”. The intense magnetic lows (deep blue in Figure 9-1) correlate with specific geologic units within the Tertiary volcanic sequence. Some of these units have very strong remnant magnetization which has a major component in the opposite direction to the current magnetic field. Hence the strong magnetic field lows.

The area between the two Tertiary volcanic “fronts” contains the altered and mineralized MacArthur hydrothermal system. It appears as a zone of moderately suppressed magnetism but not the intense lows associated with remnant magnetization. This zone is approximately 3 miles long, NE-SW and 2 miles wide, NW-SE. Alteration, favorable Jurassic dikes, and mineralization extend to the edges of Tertiary volcanic rocks, and likely continue under the post- mineralized material ‘volcanic cover’ in some areas.

10 DRILLING

10.1 EXPLORATION & DRILLING HISTORY

Although the MacArthur area is dotted with numerous shallow pits and prospects, there is little available published information. Over the history of the project, several operators have contributed to the current drill hole database of more than 300 holes. Table 10-1 summarizes the exploration history of the MacArthur area prior to Quaterra's entry. Figure 10-1 shows the location of all historical drill holes.

Table 10-1: Historic Exploration Drilling

MACARTHUR PROJECT February 2009			
Operator	**Drill Program Date Range**	**Number of Holes Drilled**	**Feet Drilled**
U.S. Bureau of Mines	1947-50	8	3,414
Anaconda Company	1955-57	14	3,690
Bear Creek Mining Company	1963-??	~14	Unknown
Superior Oil Company	1967-68	11	13,116
Anaconda Company	1972-73	280	55,809
Pangea Explorations, Inc.	1987-1991	15	2,110
Arimetco International, Inc.	Unknown	Unknown	Unknown
Total		**~342**	**~78,139**

During the late 1940s, Consolidated Copper Mines consolidated various claims into a single package that became known as MacArthur, and then attracted the interest of the US Bureau of Mines during their investigation and development of domestic mineral resources. The Bureau of Mines completed 7,680 feet of trenching in 1948 and followed up with eight diamond drill holes for 3,414 feet in 1950 (Matson, 1952). Five of the US Bureau of Mines' holes (#1-5) fall within the northern segment of the present day MacArthur open pit (Table 10-2). Holes #6-8 were collared in an area of widespread iron oxide staining approximately 2,000 feet north of the MacArthur pit within Quaterra's mineral resource mine plan footprint. Oxide copper was intersected in the southern holes #1-5 while secondary, sooty, chalcocite enrichment was found in the northern holes #6-8. Following the US Bureau of Mines exploration and drilling programs, Consolidated Copper abandoned their claims.

Table 10-2: U.S. Bureau of Mines 1947-1950 Drilling Highlights

MACARTHUR PROJECT Feb-09			
Hole ID	**Total Depth (feet)**	**Key Intercepts (Interval or thickness in feet and % Cu)**	**Notes**
Hole 1	220	110+: 0.2%	Bottomed in +0.2% Cu
Hole 2	556 (-45º)	509-556: 0.55%	Bottomed in 0.55% Cu
Hole 3	428	245-286: 0.40%	
Hole 4	469 (-45º)	79-114: 0.82%, av. 0.2+/-%	Lost hole
Hole 5	510	291+: 0.25%; av. 0.2+/-%	Bottomed in 0.25% Cu
Hole 6	409	241-303: 0.61%. 303+: ~0.15%	Bottomed in 0.2% Cu
Hole 7	428	262-297: 0.51%	
Hole 8	394	250-299: 0.36%	Lost hole

During the middle 1950s, Anaconda, by then operating the Yerington Mine, acquired leases and began investigations at MacArthur including 33 shallow drill holes (only 11 exceeding 100 feet) during 1955, 1956, and 1957. Six Anaconda holes (#'s 12, 14-17, and 19) fall within the current MacArthur pit limits. Key interval assay results from the holes exceeding 100 feet in depth are shown in Table 10-3 (Anaconda Collection-American Heritage Center). Anaconda, likely searching for shallow oxide feed for their Yerington mine, abandoned the claims sometime after 1957.

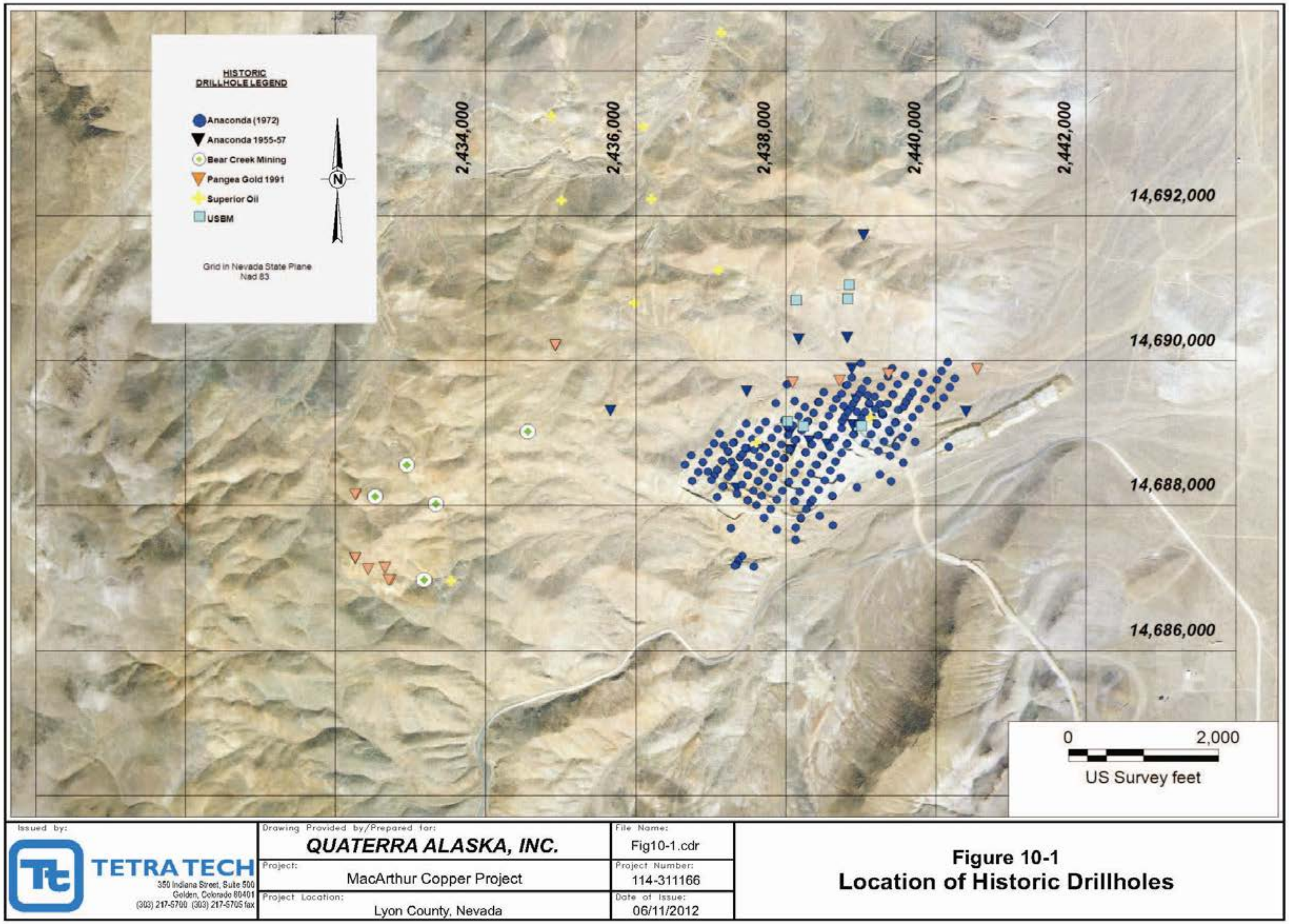


Figure 10-1: Location of Historic Drill holes

Table 10-3: Anaconda Company 1955-1957 Drilling Highlights

MACARTHUR PROJECT Feb-09			
Hole ID	**Total Depth (ft)**	**Key Intercepts (Interval in feet and % Cu)**	**Notes**
Mc 9	388	153-188: 0.52% Cu	Bottomed in <0.1% Cu
Mc10	350	139-161: 0.44% Cu	Bottomed in 0.09% Cu
Mc 11	299	144-178: 0.32% Cu	Bottomed in 0.2% Cu
Mc 12	471	267-273: 1.0% Cu	
Mc 13	292		Bottomed in <0.1% Cu
Mc 17	152		Bottomed in 0.12% Cu
Mc 18	493	306-380: 0.35% Cu	Bottomed in 0.13% Cu
Mc 19	347	65-150: 0.22% Cu	Bottomed in 0.08% Cu
Mc 20	292		Bottomed in 0.06% Cu
Mc 21	252		Bottomed in 0.05% Cu
Mc 22	263	235-245 1.02% Cu	Bottomed in 0.15% Cu

In 1963, Bear Creek Mining Company (Bear Creek) optioned claims on the MacArthur property that included leases on the Gallagher area to the west (within Quaterra's current claim position) as well as staking additional claims. Bear Creek completed large-scale geologic mapping, rock chip (and float) grid sampling, alteration mapping, Induced Polarization/Resistivity (IPR) and audio magneto-telluric geophysical surveys, and drilled at least fourteen air rotary holes, the deepest to 663 feet. At least four holes totaling 1,237 feet were drilled to satisfy claim staking location work. Exploration drilling was targeted on limonite cappings and on IP anomalies. Bear Creek drilled north and west of the MacArthur pit boundaries (within Quaterra's current claim position), focusing most of their attention and drilling in the Gallagher area.

During 1967 to 1968, The Superior Oil Company (Superior) optioned the claims formerly held by Bear Creek and drilled 13,116 feet in eleven holes as rotary pre-collar, core finish, to test the concept that a deep primary sulfide-bearing porphyry copper mineralized material shell might underlie the MacArthur oxide mineralization that had previously been tested no deeper than 663 feet. Two of Superior's holes were collared along the current north margin of the MacArthur pit while the remainder fall within Quaterra's claim boundaries. St. Joe Minerals optioned Superior's claims and drilled at least 1,833 feet of core, prospecting for shallow copper mineralization southwest of the present MacArthur pit. Other than collar coordinates and depth, Quaterra has been unable to find any data from St. Joe's drilling.

During the early 1970s, with mining of the Yerington mine oxide mineralized material nearly complete, Anaconda acquired a land position and launched an extensive trenching and rotary drilling program (more than 225 rotary holes for approximately 46,000 feet in 1972 and 55 rotary holes for approximately 9,809 feet in 1973) over and adjacent to the present day MacArthur pit. The result was an oxide resource approaching 13 million tons of plus 0.4% Cu (1972 data only and not NI 43-101 compliant), described as an oxidized low-grade copper deposit which has been locally enriched by exotic copper (Heatwole, 1978). Anaconda's resource calculations were developed into the mine plan supporting the 5.0 million tons at 0.30% Cu mined from the MacArthur pit by Arimetco during 1995-1997.

During 1987 to 1991, Pangea Explorations, Inc. located 304 unpatented lode claims and conducted an aggressive gold evaluation of the MacArthur area from the present day MacArthur pit westerly to the Gallagher area. Pangea's program included over 549 rock chip samples, geologic and alteration mapping, followed by trenching two target areas (Adams, 1987). Eight trenches totaling over 1,420 feet were cut and sampled in the Gallagher area and four additional trenches totaling over 720 feet located in an undefined "north target." Table 10-4 details some of Pangea's exploration drilling results. Anomalous gold values (41 samples exceeding 0.015 Au oz/ton) led to a 15-hole / 2,110-foot reverse circulation drilling program with 1,310 feet in seven holes testing the Gallagher area. Pangea found the drilling results discouraging (best assay value of 0.026 Au oz/ton over 5 feet) and abandoned the property.

Table 10-4: Pangea Exploration 1987-1991 Drilling Highlights

MACARTHUR PROJECT Feb-09			
Hole ID	**Interval (ft)**	**Interval Length (ft)**	**Gold Grade (Au oz/ton)**
	20-45	25	0.012
MAC 91-1	165-175	10	0.013
	100-110	10	0.012
MAC 91-2	130-145	15	0.016
MAC 91-3	75-90	15	0.013
	45-55	10	0.011
MAC 91-4	145-155	10	0.015
MAC 91-5	90-100	10	0.011
	85-95	10	0.021
	100-110	10	0.014
MAC 91-6	85-110	25	0.014
	5-15	10	0.015
MAC 91-7	55-75	20	0.016
MAC 91-8	105-115	10	0.016
MAC 91-9	75-85	10	0.015
MAC 91-10	60-80	20	0.014
MAC 91-11	20-30	10	0.011

During the late 1980s through the late 1990s, Arimetco consolidated a major land position in the Yerington mining district consisting of over 8,500 acres including 85 patented claims. Arimetco entered the district to extract copper by heap leaching methods, with initial production from the Anaconda Yerington mine dumps, oxide stockpiles and Yerington mine vat leach tailings. Arimetco's leach pads were located on the Yerington mine property approximately five miles south of the MacArthur property. During evaluation and mining of the MacArthur mine, Arimetco drilled an unknown number of holes as a check on Anaconda's 1972 to 1973 drilling. Anaconda's drilling and resource calculations provided the mine planning data for Arimetco's MacArthur mine. Due to rising costs and depressed copper prices, Arimetco was forced to abandon their claim position and file for bankruptcy in 1999.

In 2004, North Exploration located unpatented claims covering portions of the MacArthur property and the MacArthur pit that were leased to Quaterra in 2005. Subsequently, Quaterra has staked additional claims, bringing the current total to 470 unpatented lode claims over the project area. Quaterra's current land position is displayed on Figure 4-2.

10.2 HISTORIC MINING

The MacArthur Project area has seen limited historic mining activity. The most recent activity occurred between 1995 and 1997, when Arimetco mined a limited tonnage (estimated 6.1 million tons) of surface oxide copper for heap leaching at the historic Yerington Mine site. No consistent, large-scale mining has occurred on the site.

10.3 CURRENT DRILLING

Although 2011 step out RC drilling at 500 foot centers continued to intersect acid soluble and primary copper mineralization, Quaterra focused mid-year on defining a resource that could support a positive mine plan. Drilling was therefore centered on an approximate one-half square mile area from North Ridge south to the MacArthur Pit, and the Gallagher area located west of the existing MacArthur Pit. Drill spacing was reduced to 250 foot centers on several drill fences. South-bearing angle holes tested the WNW, north dipping structural / mineralized grain and east- and west-bearing angle holes tested orthogonal structure resulting in the upgraded resource calculation reported Section 14 of this TR.

During 2011, 81,651 feet of exploration drilling in 152 holes were completed including 69,890 feet in 146 RC holes and 11,761 feet in six core holes. (See Figure 10-2)

Also during 2011, 3,274.8 feet of PQ size core (3.35 inches) were drilled at 26 sites for metallurgical testwork. PQ holes twinned existing Quaterra RC and core holes. PQ holes were prefixed by "PQ-11" followed by the ID of the twinned hole.

10.4 SURVEYING DRILL HOLE COLLARS

Drill hole locations are surveyed by Quaterra staff using a Trimble XHT unit, and are shown in Figure 10-2.

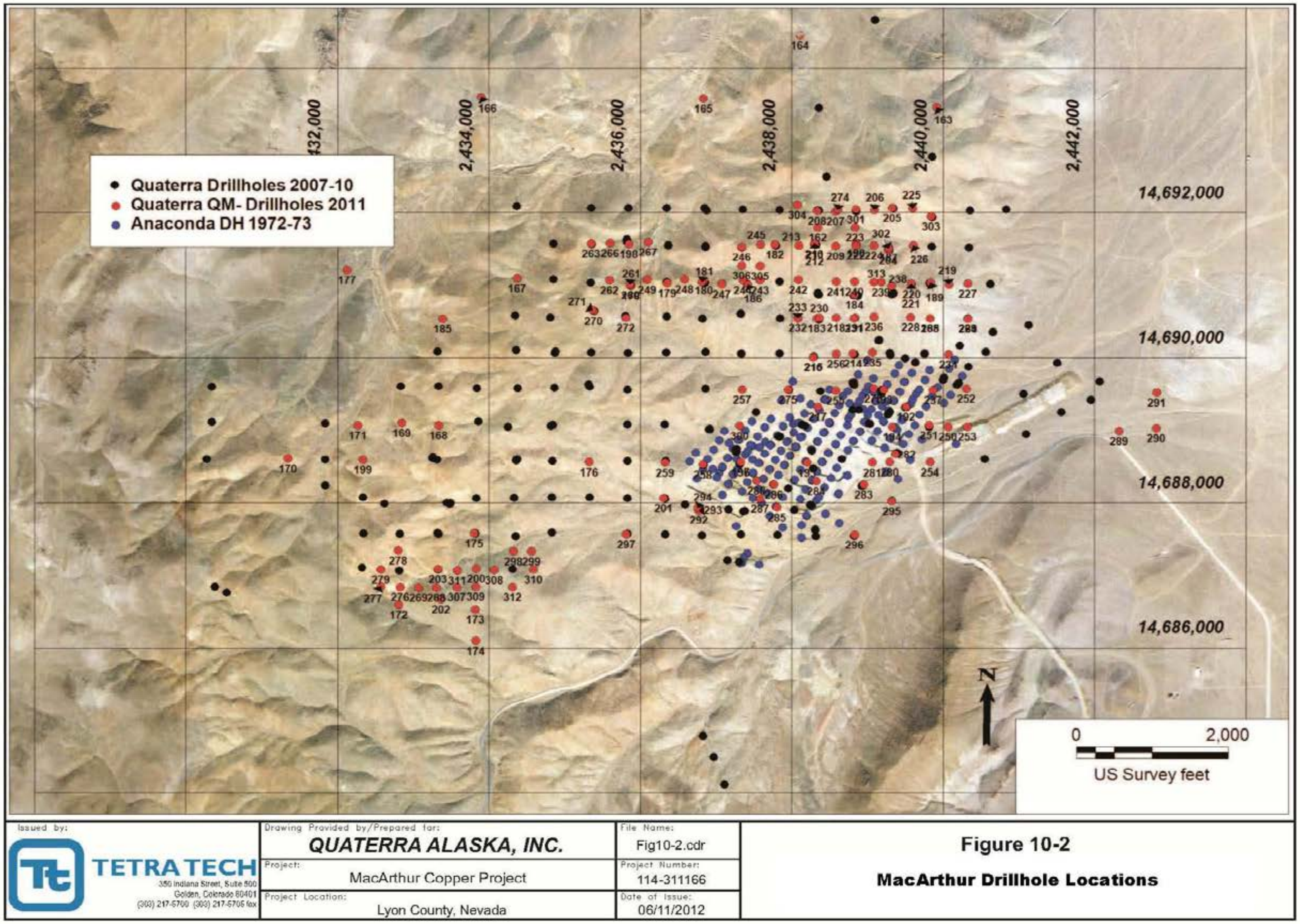


Figure 10-2: Drill hole Location Map

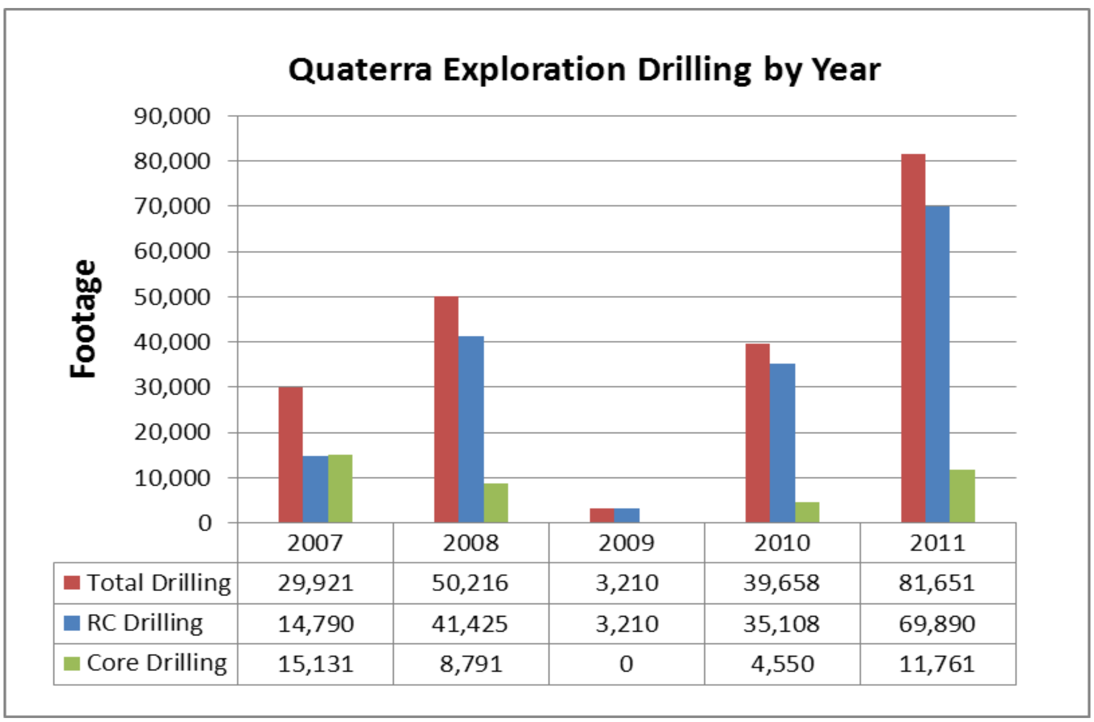
10.5 DOWNHOLE SURVEYS

During 2011 five holes were downhole surveyed by International Directional Services, Elko, Nevada USA operating a surface recording Gyroscope. Downhole surveyed holes included: QM-163, 164, 165, 166, and 177. To date, downhole surveys have been completed on 57 of the Quaterra drill holes, and are now routinely done for holes greater than 1000 feet deep.

10.6 CURRENT DRILLING METHODS AND DETAILS

Quaterra has explored the MacArthur property with both reverse circulation (RC) and diamond core drilling methods. Reverse circulation holes have been drilled by Diversified Drilling LLC, Missoula, Montana, USA, DeLong Construction Inc., Winnemucca, Nevada, USA and by Leach Drilling Inc., Silver Springs, Nevada, USA. During 2007-2008 the core drilling was contracted to Kirkness Diamond Drilling of Dayton, Nevada, USA and Kirkness Brothers Diamond Drilling (aka KB Drilling Co, Inc.) of Carson City, Nevada, USA. Major Drilling America, Inc., Salt Lake City, Utah, conducted core drilling during 2009-2010. Core drilling during 2011 was contracted to Ruen Drilling Inc., Clark Fork, Idaho, USA. The RC crews ran one 10-12 hour shift per day; the core drill crews operated 24 hours per day.

Quaterra has completed 204,656 feet of drilling in 401 holes since beginning drilling in 2007. Core holes total 40,233 feet in 58 holes and reverse circulation holes total 164,423 feet in 343 holes. (Note that one previously listed, but abandoned 115 foot drill hole, has now been removed from the database and reported totals). Figure 10-3 show Quaterra's yearly exploration drilling footage by year.



	2007	2008	2009	2010	2011
Total Drilling	29,921	50,216	3,210	39,658	81,651
RC Drilling	14,790	41,425	3,210	35,108	69,890
Core Drilling	15,131	8,791	0	4,550	11,761

Figure 10-3: Quaterra Exploration Drilling by Year

During 2011, 69,890 feet in 146 RC holes and 11,761 feet in six core holes were drilled resulting in over 16,300 samples collected for analysis for total copper, acid soluble copper, ferric sulfate soluble (“QLT”), gold (selected samples), and trace elements (selected samples). Selected core was used to calculate rock quality designation (RQD) and measure bulk density. In addition 3,274.8’ of PQ size core was drilled in 26 holes for metallurgical testwork. The total area covered by the MacArthur drilling is approximately 11,000 feet east-west by 6,000 feet north-south at approximate drill spacing of 500 feet. Drill spacing reduces to approximately 250 feet within an approximate 1,500 feet east-west by 1,000 feet north-south within the northeast portion of the MacArthur pit and reduces to 250 foot spacing over portions of a 5000 foot square area north of the MacArthur pit. Historic Anaconda drilling spacing is 125 feet in the MacArthur pit.

10.7 REVERSE CIRCULATION DRILLING SAMPLING METHOD

All reverse circulation (RC) drilling is conducted with water added to eliminate dust. A percussion hammer with interchange sampling system has been used by the RC drill. Samples are collected in a conventional manner via a cyclone and standard wet splitter in 17-inch by 26-inch cloth bags placed in five-gallon buckets to avoid spillage of material. Sample bags are pre-marked by Quaterra personnel at five-foot intervals and also include a numbered tag bearing the hole number and footage interval. Collected samples, weighing approximately 15 to 20 pounds each, are wire tied, and then loaded onto a ten-foot trailer with wood bed allowing initial draining and drying. Each day, Quaterra personnel, or the drillers at end of their shift, haul the sample trailer from drill site to Quaterra’s secure sample preparation warehouse in Yerington, Nevada. Geologic logging samples are collected at the drill site in a mesh strainer, washed, and placed in standard plastic chip trays collected daily by Quaterra personnel.

10.8 CORE DRILLING SAMPLING METHOD

For 2011 exploration drilling core diameter was HQ (approximately 2.75-inch diameter. Following convention, at the drill site core was placed in wax-impregnated, ten-foot capacity cardboard boxes. Sample intervals vary from less than one foot to six feet, dependent upon rock consistency. Sample boxes were delivered to Quaterra’s secure sample warehouse in Yerington, Nevada by the drill crew following each 12-hour shift.

PQ core drilling for metallurgical testwork followed similar protocol as exploration drilling. PQ core was placed in wax-impregnated, five-foot capacity cardboard boxes and delivered to Quaterra’s secure sample warehouse by the drill crew following each 12-hour shift.

Special treatment was required for PQ core metallurgical samples to avoid undue oxidation prior to column testwork. PQ core was “quick-logged” without removing the core from core box or without breaking up the core. Prior to photographing, magnetic susceptibility and RQD measurements were collected. The entire core box was then sealed with plastic wrap to avoid oxidation. Shrink-wrapped core boxes were stacked on pallets, secured with plastic wrap and steel banding for shipment to METCON Research Laboratories in Tucson, Arizona USA. Time from core arrival at the core shed to plastic wrap of the core box was less than 24 hours.

10.9 DRILLING, SAMPLING, AND RECOVERY FACTORS

No factors were shown that could materially impact the accuracy and reliability of the above results. With few exceptions, core recovery exceeded 80% while RC recovery is estimated to be greater than 95%.

10.10 SAMPLE QUALITY

It is Tt's opinion that Quaterra's samples of the MacArthur Project are of high quality and are representative of the property. This statement applies to samples used for the determination of grades, lithologies, densities, and for planned metallurgical studies.

It is the opinion of the author that during the period in 1972 to 1973 when Anaconda explored and drill tested the MacArthur property, the drill samples taken by Anaconda were representative of the deposit and the methodologies commonly used by the industry at that time. This statement applies to samples used for the determination of grades, lithology, and densities, as well as metallurgical performance, supported by similar determinations and conditions being carried out at that time at Anaconda's Yerington mine operation and as referenced below in an internal Anaconda report (Heatwole, 1972), portions of which follow:

"*From March to November, 1972, over 225 holes were drilled... Approximately 33,000 feet of vertical hole and 13,000 feet of angle hole were drilled using percussion and rotary methods.*"

The majority (62%) of the drilling, which was supervised by Anaconda's Mining Research Department, was accomplished using Gardner-Denver PR123J percussion drills. The percussion drill was fitted with a sampling system designed by the Mining Research Department, which collected the entire sample discharged from the hole. The remainder of the drilling was done by Boyles Brothers Drilling Company using rotary and down-the-hole percussion equipment. The sampling system used by Boyles, especially during the early stages of drilling is not considered to be as accurate as the system designed by Mining Research.

While no details are available regarding Anaconda's exact assaying protocol and quality control during drilling at the MacArthur property, an interview conducted by Quaterra personnel in October 2008 with Mr. Henry Koehler, Anaconda's Chief Chemist during the 1960s and 1970s, confirmed that the techniques and procedures implemented conformed to industry standards for that era. Mr. Koehler was employed in Anaconda's analytical laboratory from 1952 to mine closure in 1978. He currently resides in Yerington, Nevada.

October 24, 2008

I am Henry Koehler. I was employed at the analytical laboratory of the Anaconda Company in Yerington, Nevada during the years 1952 through 1978, and was Chief Chemist during the years of exploration on the MacArthur project.

From 1971 through 1973, samples were delivered from the MacArthur prospect for assay.

Samples were delivered to the laboratory where they were blended, pulverized, and a 2gm sample was extracted for assay.

Samples were received and assayed for total copper and oxide copper, according to standard wet chemistry procedures.

Reports were hand written and issued over my signature with 3 carbon copies and one original, which was given to management.

Henry Koehler
425 Pearl St.
Yerington Nv.

Figure 10-4: Letter from Mr. Henry Koehler

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Tt has reviewed all of the Quaterra sample preparation, handling, analyses, and security procedures. It is Tt's opinion that the current practices meet NI 43-101 and CIM defined requirements. Following a Tt recommendation, standards are stored in a locked and secured area

11.1 RC SAMPLE PREPARATION AND SECURITY

RC sample bags, having been transported on a ten-foot trailer by Quaterra personnel from the drill site to the secure sample warehouse, are unloaded onto suspended wire mesh frames for further drying. Diesel-charged space heaters assist in drying during winter months. Once dry, sets of three samples are combined in a 24- by 36-inch woven polypropylene transport ("rice") bag, wire tied, and carefully loaded on plastic lined pallets. Each pallet, holding approximately 13 to 15 rice bags, is shrink-wrapped and further secured with wire bands. Quaterra's samples were shipped via UPS Freight to Skyline Assayers & Laboratories (Skyline), Tucson, Arizona USA through 2008. During the 2009-2010 drill campaign, Skyline dispatched a transport truck from Tucson to collect samples. In 2011, Skyline established a sample preparation facility in Battle Mountain, Nevada, from which trucks were dispatched to pick up Quaterra's drill samples under a chain of custody protocol. Following sample preparation in the Battle Mountain facility, Skyline ships a representative pulp sample to the Skyline laboratory in Tucson, Arizona for analysis.

Complying with earlier recommendations from Tt, Quaterra now weighs each shrink-wrapped pallet of samples prior to departure from Yerington. Rejects and pulps are returned to Quaterra and stored under cover in a secure location.

11.2 CORE SAMPLE PREPARATION AND SECURITY

Drill core, having been transported at end of each shift by the drill crew to Quaterra's secure sample warehouse, is logged by a Quaterra geologist who marks appropriate sample intervals (one to nominal five feet) with colored flagging tape. Each core box, bearing a label tag showing drill hole number, box number, and box footage interval, is then photographed. Rock quality designations (RQD), magnetic susceptibility, and recovery measurements are taken. Core preceding drill hole QMCC-20 was sawed in half by Quaterra personnel; core holes QM-026, QM-036, QM-041, QM-046, and QM-049 were split in half using a hydraulic powered blade at the warehouse by Quaterra personnel (for approximately six months through core hole QMCC-20, core was sawed rather than hydraulically split). One half of the split was bagged in 11- by 17-inch cloth bags for assay while the other half was returned to the appropriate core box for storage in the sample warehouse. Approximately five to six cloth sample bags are combined in a larger 24- by 36-inch transport polypropylene ("rice") bag, wire tied, and carefully loaded on plastic lined pallets. Each pallet, holding approximately 13 to 15 rice bags, was shrink-wrapped and further secured with wire bands for shipment to Skyline in Tucson. The same chain of custody protocol is used for both RC and core samples.

Following geologic logging and RQD measurements, the core portions of holes QM-99, QM-100, and QM-109 (2009-2010) and QM-163, QM-164, QM-165, QM-177 and QM-185 (2011 program) were strapped and shrink wrapped on pallets for shipment to ALS Minerals laboratory in Reno, Nevada. Core samples were picked up from the warehouse by a Reno, Nevada-based ALS Minerals driver, and sample pallets were weighed upon receipt by the laboratory. ALS personnel sawed the core in half, one half for assay at the ALS laboratory, storing the other half in the core box for return to Quaterra. Chain of custody procedures for ALS Minerals follow the format described for Skyline.

Following geologic logging, magnetic susceptibility and RQD measurements, and photography, PQ core for metallurgical testing was shrink-wrapped in its cardboard core box, stacked on pallets, shrink-wrapped together, wire banded, and weighed. Pallets were shipped to METCON Research Laboratories, Tucson, Arizona via UPS Ground. Chain of Custody was signed upon departure from Yerington and receipt in Tucson.

11.3 SAMPLE ANALYSIS

During 2007, 12 drill holes (core) were analyzed at American Assay Laboratories (AAL) in Sparks, Nevada, USA. AAL is ISO/UEC 17025 certified as well as a Certificate of Laboratory Proficiency PTP-MAL from the Standards Council of Canada.

With sample submission-to-reporting time exceeding two months at AAL, Quaterra elected to use Skyline Assayers & Laboratories (Skyline) and ISO certified assay lab in Tucson, Arizona, USA for all further analytical work. Samples submitted to AAL were re-assayed (pulps or rejects) by Skyline for consistency of the data set.

Core from drill holes QM-99, QM-100, and QM-109 (2009-2010) and QM-163, QM-164, QM-165, QM-166, QM-177 and QM-185 (2011 program) were submitted to ALS Minerals, Sparks, Nevada, USA. ALS Minerals is an ISO registered and accredited laboratory in North America. Quaterra samples arrive at Skyline via UPS truck freight and in 2009-2010 by a transport truck dispatched from Tucson by Skyline. A Quality Assurance and Quality Control Assay Protocol have been implemented by Quaterra where one blank and one standard are inserted with every 18 drill hole samples going into the assay stream. The Skyline assay procedures are as follows:

- For Total Copper: a 0.2000 to 0.2300 gram (g) sample is weighed into a 200-milliliter (ml) flask in batches of 20 samples plus two checks (duplicates) and two standards per rack. A three-acid mix, 14.5 ml total is added and heated to about 250°C for digestion. The sample is made to volume and read on an ICP/AAS using standards and blanks for calibration.

- For Acid Soluble Copper: a 1.00 to 1.05 g sample is weighed into a 200 ml flask in batches of 20 samples plus two checks (duplicates) and two standards per rack. Sulfuric acid (2.174 l) in water and sodium sulfite in water are mixed and added to the flask and allowed to leach for an hour. The sample is made to volume and read on an ICP/AAS using standards and blanks for calibration.

- For Ferric Soluble Copper (QLT): uses an assay pulp sample contacted with a strong sulfuric acid-ferric sulfate solution. The sample is shaken with the solution for 30 minutes at 75ºC, and then filtered. The filtrate is cooled, made up to a standard volume, and the copper determined by AA with appropriate standards and blanks for calibration.

- For Sequential Copper Leach: consists of four analyses: Total Copper, Acid Soluble Copper, Cyanide Soluble Copper, and the difference, or Residual. Following analysis for Total Copper and Acid Soluble Copper, the residue from the acid soluble test is leached (shake test) in a sodium cyanide solution to determine percent cyanide soluble minerals. The Sequential Copper Leach is a different approach to the Ferric Soluble Copper (QLT) leach, with possible greater leaching of certain sulfides (e.g. chalcocite or bornite) during the cyanide leach step.

Beginning in 2009, Quaterra requested 34-element trace element geochemistry from Skyline on selected samples which were analyzed by ICP.OES Aqua Regia Leach.

During 2009-2010 Quaterra core samples were picked up at Quaterra's warehouse facility by ALS Minerals personnel and transported to ALS Minerals laboratory in Sparks, Nevada, USA. ALS Minerals personnel sawed the core, saving one-half for return to Quaterra. ALS assayed core for trace element geochemistry with 48-element Four Acid "Near-Total" Digestion.

In keeping with Tt recommendations, beginning in 2009, Quaterra began a program to re-assay selected samples when blanks, standards, or repeat assays exceeded or were below the expected values by 15%, or blanks returned an assay of >.015% Cu. The QC program now re-assays standards outside +/- 2 standard deviations of the expected value, repeat assays +/- 15% of the original assay, and blanks greater than .015% Cu.

11.4 LEACH ASSAY ANALYSIS

Both Sequential copper leach assays and QLT leach assays, when combined with column leach tests can be indicative of actual heap leach recoveries. Historically, sequential copper leach assays were not performed on samples at MacArthur. Section 6.4 discusses the problems encountered by previous operators while leaching mineralized material from the MacArthur pit. Since previous operators were unable to explain the longer leach times and low solution head grades they encountered, Tt recommended that Quaterra perform sequential copper leach assays on some of the available sample coarse rejects. While only early results were available for the 2009 TR, Table 11-1 shows a January 2011 updated summary of the total copper, acid-soluble copper (ACu), and cyanide-soluble copper (CNCu) quantities categorized by mineralized zones. The acid-soluble fraction of total copper is greatest in the oxide zone. The cyanide-soluble fraction of total copper is greatest in the chalcocite/oxide zone where the dominant species of copper mineral is chalcocite. In the primary sulfide zone, both acid- and cyanide-soluble fractions of total copper are low due to high levels of chalcopyrite.

Table 11-1: Sequential Copper Leach Assay Results

QUATERRA ALASKA, INC. – MACARTHUR PROJECT January 2011							
	Average Values						
Mineralized Zone	**%Cu**	**%ACu**	**%CNCu**	**ACu : Cu**	**CNCu : Cu**	**% Soluble Cu**	**# of Samples**
Oxide	0.185	0.103	0.015	0.56	0.08	64%	213
Chalcocite/Oxide	0.252	0.065	0.084	0.26	0.33	59%	281
Primary	0.186	0.019	0.030	0.10	0.16	27%	60

Tt proposed that Quaterra performed either, standard CU assays, warm H_2SO_4 assay, and QLT or standard sequential copper leach assays on all drill hole samples that exceed 0.10% Cu for all future drilling programs. This data will help Quaterra to better understand potential mineralogical differences between the oxide, secondary, and primary mineral zones as well as help link column leach test composites with in situ material to better predict heap leach performance.

Beginning with drill hole QM-086 in December 2009, Quaterra continued to request analyses for total copper (Cu or TCu) from all drill samples. Analyses for acid soluble copper (ACu) and for ferric sulfate leach aka Quick Leach Tests (QLT) were requested for drill samples (plus an additional 50 feet downhole) containing visible green or black copper or containing chalcocite. These analyses were completed by Skyline Laboratories, Tucson, Arizona for all reverse circulation drilling. A high acid soluble copper to total copper ratio indicates that leachable oxide copper is present. QLT minus acid soluble offers an estimate of acid soluble (leachable) sulfide copper, i.e. chalcocite. The Cu-ACu-QLT analysis combination is an alternative approach, rather than using the Sequential Leach analysis (Cu, ACu, cyanide soluble copper, then calculate residual).

Skyline Laboratories performed QLT analyses from both core and reverse circulation drill samples at the onset of Quaterra's exploration program in 2007. In order to complete the QLT analyses through drill hole QM-085, Skyline analyzed an additional 2,747 pulps representing both core and reverse circulation drill footages.

Table 11-2 summarizes the results of the Cu, ACU, and QLT testing for those intervals containing all three assays and where the Cu value is >0.1% Cu. It should be noted that no ACU or QLT assays were included in the data received for the Anaconda drilling within the MacArthur oxide deposit. The data shown in Table 11-2 therefore reflects only the Quaterra drilling and reflects averages over the entire project area.

Table 11-2: Ferric Sulfate Leach (QLT) Assay Results

QUATERRA ALASKA, INC. - MACARTHUR PROJECT January 2012							
	Averages for samples with Cu>.1%Cu with ACU and QLTCU Assays						
Mineralized Zone	**%Cu**	**ACu**	**QLTCu**	**ACu : Cu**	**QLTCu:Cu**	**% Soluble Cu**	**# of Samples**
Oxide	0.236	0.116	0.128	0.425	0.471	47%	1585
Chalcocite/Oxide	0.341	0.066	0.143	0.215	0.401	40%	2576
Primary	0.315	0.029	0.063	0.096	0.171	17%	486

Further metallurgical work is expected to provide a better understanding of the differences noted when comparing the QLT and sequential leach method results.

11.5 QUALITY CONTROL

As part of the quality control program, 675 standards and 622 blanks were submitted (Table 11-3) along with 15,063 individual drill hole samples to Skyline Laboratories. Additionally, 87 standards and 85 blanks were submitted along with 1,748 core samples to ALS Mineral Labs in Reno.

Lot failure criteria were established as any standard assaying beyond two standard deviations of the expected value, or any blank assay greater than 0.015% Cu. Failed lots were reviewed and lot samples were selected for reassay. Results indicated that all original assays, with the exception of 4, in which sample numbers had been switched, would be accepted as originally received.

Table 11-3: MacArthur 2011 QA/QC Program Results

	Skyline Labs	**ALS Mineral Labs**
Total Drill Hole Samples	15,063	1,748
Submitted Standards	675	87
Failed Standards	27	3
% Standards Failure	4.0%	3.4%
Submitted Blanks	622	85
Failed Blanks	7	0
% Blank Failure	1.1%	0%

Check assays from ALS Mineral Labs compared well with Skyline assays, providing additional confidence in the assay database, as shown in Figure 11-1.

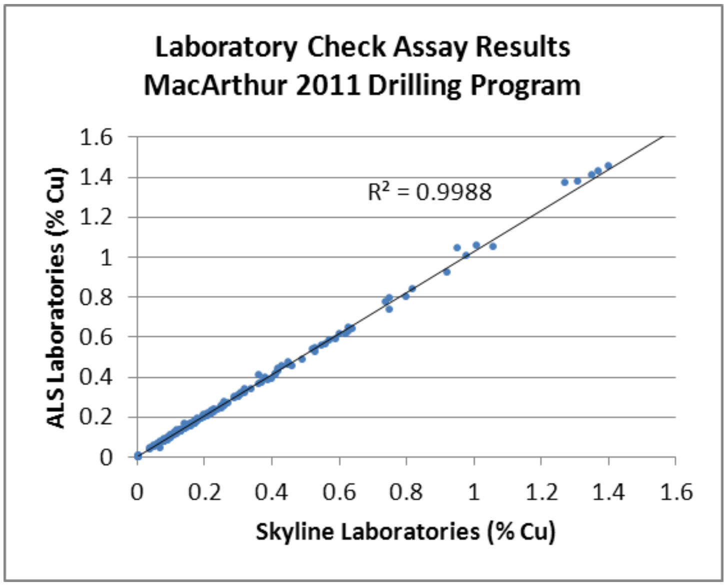


Figure 11-1: MacArthur Check Assay Results

11.6 REVIEW OF ADEQUACY OF SAMPLE PREPARATION, ANALYSES, AND SECURITY

During the visit to the project in 2011, Dr. Bryan observed geologic logging and data entry of drill data following an established protocol (Figure 11-2), and procedures for manually creating geologic sections from the drill data (Figure 11-3).

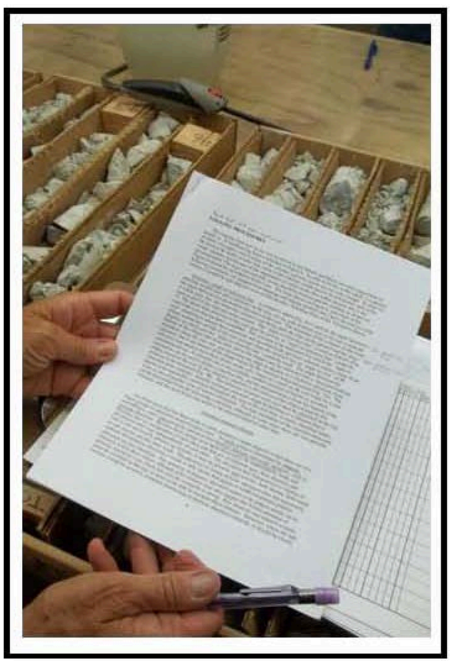


Figure 11-2: Reviewing Established Protocol for Data Entry

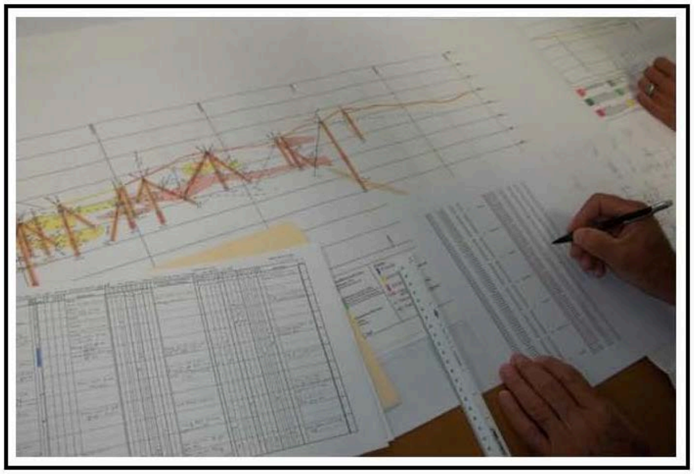


Figure 11-3: Manually Creating Geologic Sections from the Drill Data

12 DATA VERIFICATION

Dr. Rex Bryan of Tt conducted a site visit to the MacArthur Project area and Quaterra's field office in Yerington, Nevada on September 9, 2011. During this visit Quaterra staff discussed the history of the project, presented all requested data, answered questions posed by Tt, presented the current geologic interpretation of the MacArthur deposit, and guided Dr. Bryan on a field examination through the MacArthur property which included observing drill sample collection during reverse circulation drilling. This section details the results of Tt's verification of existing data for the MacArthur Project.

12.1 HISTORIC DATA CHECK

Tt did not collect independent samples to corroborate historic data. It is Tt's opinion that the previous owners of the MacArthur Project area were competent established companies that followed industry standard practices for drilling, sampling, and assaying according to the industry standards in place at the time of the work. However, Quaterra has completed verification work on the historic data by re-assaying, when material was available, and twin hole drilling.

As an assay check on the historic Anaconda drilling within the confines of the current MacArthur pit, Quaterra twinned nineteen Anaconda holes using both reverse circulation and core drilling methods (Table 12-1). The attached histogram (Figure 12-1) contains information on 57 total holes: 38 Quaterra and 19 Anaconda. It provides a comparison of average copper grades between the 1972-1973 Anaconda drilling (all as dry drilling, capturing 100% of the dry sample) and Quaterra's twin holes (wet sample recovery for all Quaterra reverse circulation drilling). Some of the twin holes drilled by Quaterra are angled whereas the corresponding Anaconda hole was drilled vertically. For these twin angle-drilled holes, the intercept displayed in Figure 12-1 is the length-weighted average over the projected vertical interval. The abbreviations Q-aRC and Q-bRC are first and second twins of existing holes.

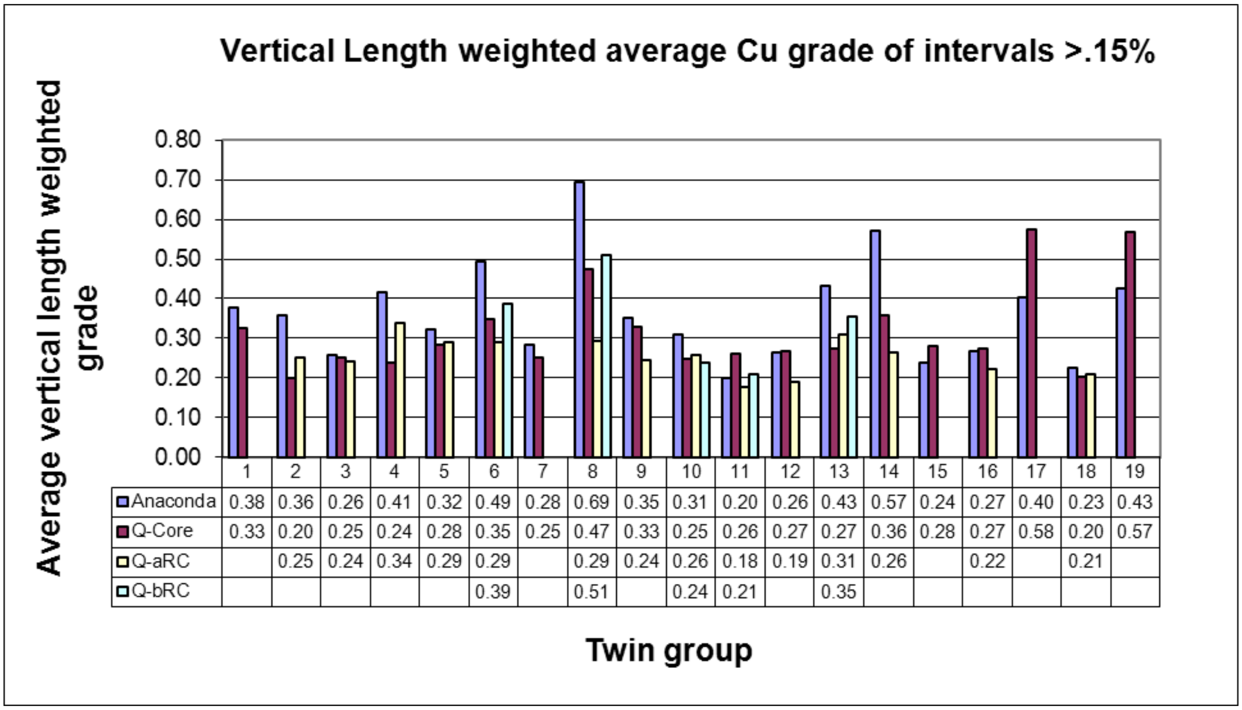
A complete discussion of the twin hole program is available in the "MacArthur Copper Project NI 43-101 Technical Report", dated Feb 17, 2009.

12.2 CURRENT DATA CHECK

Tt has made several data checks and verifications of Quaterra work that has been performed for the MacArthur Project. These checks include validation of assays from Skyline and comparing geologic field logs with drill hole data. No discrepancies have been found.

Table 12-1: List of Twin Holes Drilled By Quaterra

QUATERRA ALASKA, INC. – MACARTHUR PROJECT February 2009				
Twin Group	**Anaconda Hole**	**Quaterra Twin Core Hole**	**Quaterra Twin aRC Hole**	**Quaterra Twin bRC Hole**
1	M120-C50-1	QMT-4		
2	M120-C50-2	QMT-5	QMT-5aR	
3	M165-K-1	QMT-11	QMT-11aR	
4	M172.5-I-1	QMT-8	QMT-8aR	
5	M195-M-1	QMT-13	QMT-13aR	
6	M195-M-2	QMT-14	QMT-14aR	QMT-14bR
7	M205-G-2	QMT-6		
8	M210-K-1	QMT-10	QMT-10aR	QMT-10bR
9	M210-O-1	QMT-15	QMT-15aR	
10	M270-Q-1	QMT-17	QMT-17aR	QMT-17bR
11	M270-S-1	QMT-18	QMT-18aR	QMT-18bR
12	M30-K-1	QMT-12	QMT-12aR	
13	M45-C1-1	QMT-1	QMT-1aR	QMT-1bR
14	M45-C1-2	QMT-2	QMT-2aR	
15	M75-I-1	QMT-9		
16	M90-B-1-2	QMT-3	QMT-3aR	
17	M-90-G-4	QMT-19		
18	M90-O-1	QMT-16	QMT-16aR	
19	M95-G-1	QMT-7		



	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Anaconda	0.38	0.36	0.26	0.41	0.32	0.49	0.28	0.69	0.35	0.31	0.20	0.26	0.43	0.57	0.24	0.27	0.40	0.23	0.43
Q-Core	0.33	0.20	0.25	0.24	0.28	0.35	0.25	0.47	0.33	0.25	0.26	0.27	0.27	0.36	0.28	0.27	0.58	0.20	0.57
Q-aRC		0.25	0.24	0.34	0.29	0.29		0.29	0.24	0.26	0.18	0.19	0.31	0.26		0.22		0.21	
Q-bRC						0.39		0.51		0.24	0.21		0.35						

Figure 12-1: Twin Hole Charted Results

12.2.1 Adequacy of Data

It is Tetra Tech’s opinion that the data collection of both historic and modern data by Quaterra is adequate for the use of a 43-101 resource for the following reasons:

- The sampling is representative of the deposit in both survey and geological context
- The drill hole cores have been archived and are available for further checking

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The MacArthur deposit generally consists of an oxidized copper capping transitioning through a mixed oxide/secondary copper interface into primary sulfides at depth. Essentially all metallurgical testwork to date has been conducted on the copper oxide resources with a few tests having been performed on mixed oxide/sulfide material.

The MacArthur Project has a long history of metallurgical bottle roll and column testwork from 1976 through 2011. Historical test work by Anaconda in 1976 included bottle roll and column leach tests on samples collected from surface trenches. Arimetco performed a number of bottle and column leach tests on surface samples between 1992 and 1995 using several different metallurgical laboratories. Quaterra performed bottle roll and column tests between 2010 and 2011 through METCON Research in Tucson, Arizona. A list summarizing this testwork can be found in Table 13-1 at the end of this section.

Of significance, Anaconda operated a vat leach facility processing oxide mineralized material from the Yerington Pit, the results from which were documented over the many years of operation. Arimetco also operated a number of leach pads between 1989 and 1995 treating oxide and transition mineralized material mined from the Yerington Pit. However, between 1994 and 1997, approximately 6.1 million tons of mineralized material was mined from the MacArthur Pit and hauled Run of Mine (ROM) to the Arimetco pads for processing. This commercial operational database for both the vat and heap leach operations was significant since both Yerington and MacArthur mineralized material deposits are very similar in origin, geology and mineralization. A summary of several years of data from the vat leach operation is available for review.

A review of the METCON metallurgical test work shows good copper extraction but variable acid consumption spatially throughout the deposit. METCON column test work (32 columns) conducted in 2011 using material from 32 different PQ core drill holes (rather than material taken during surface sampling) was completed for the PEA. The drill holes provided reasonable spatial representivity of the MacArthur resources in all the deposit area. The METCON column study completed in 2011 is available for review.

Combined, the 2011 METCON study, the Anaconda vat leaching data, and the Arimetco commercial leach pad data provided sufficient metallurgical information to gain a preliminary level of confidence used in developing this PEA Study. However, to achieve the level of confidence for a prefeasibility study (PFS), additional metallurgical test work is necessary to better understand acidification techniques and the resultant copper extraction spatially in mineralized resource contained within the mine plan. This metallurgical test work will be undertaken during the PFS. Recommendations for this test program are provided in Section 26 of this report.

The following sections discuss the criteria upon which generation of the heap leach design parameters for the PEA were made. The design parameters are summarized in sub-section 13.9 Heap Leach Design Criteria.

13.1 OXIDE MATERIAL COPPER EXTRACTION

Predicted copper extraction and acid consumption was derived from the existing metallurgical data base, METCON columns and Arimetco historical information. Figure 13-1 below shows column copper extraction versus grade during a 120 day leach cycle. The 32 METCON columns average 60% extraction, which is globally near the extraction achieved by Arimetco at a similar copper grade. Half of the columns averaged 0.11% copper head grade which likely provides a downward bias on copper extraction. As grade increases, copper extraction increases. Assuming a 0.15% copper cutoff grade, Figure 13-1 shows column copper extraction of 65%. Using a permanent heap leach pad, extraction is predicted to increase during residual leaching of the overlaid pads, greater than offsetting solution copper inventory buildup in the pad.

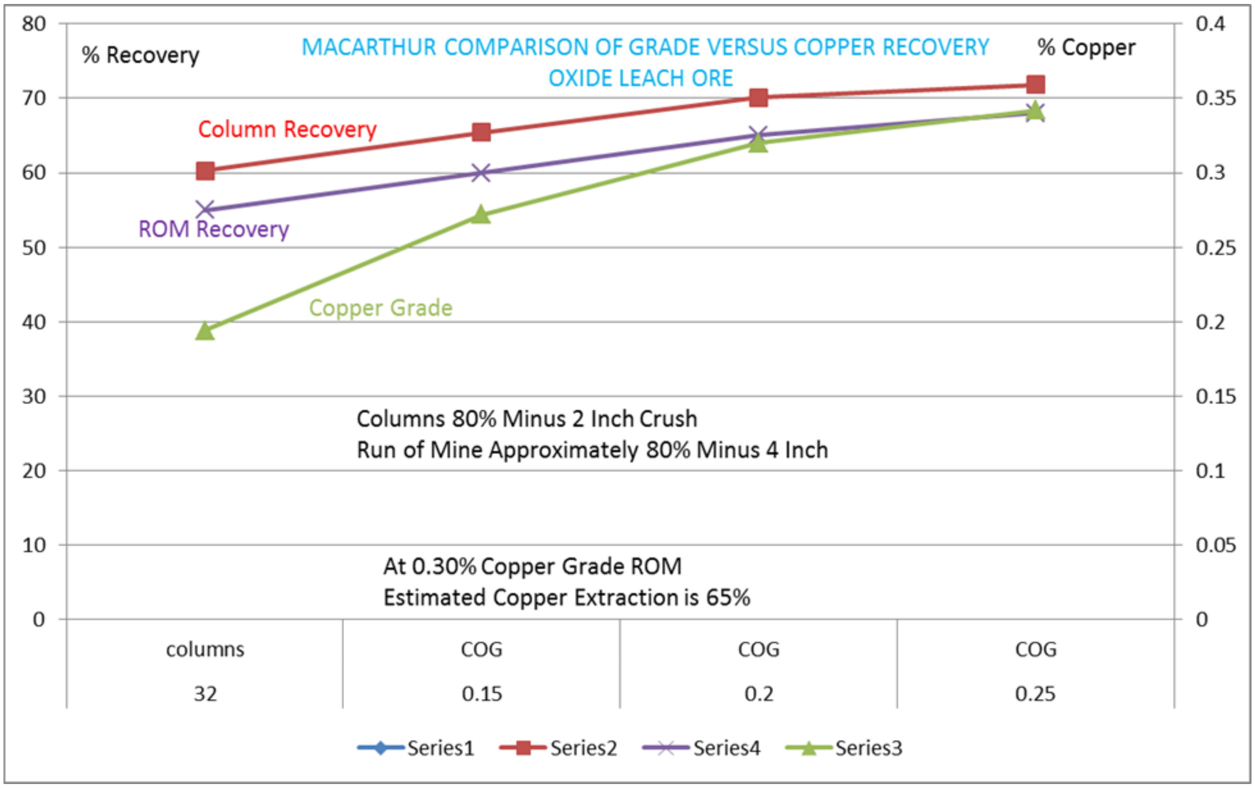


Figure 13-1: Comparison of Grade versus Copper Recovery Oxide Leach Material

The 32 METCON column tests showed that the leaching performance in the old MacArthur Pit area provided higher copper extraction and lower acid consumption compared to both the Gallagher Pit area and the North and Northwest MacArthur Pit areas. The variation in leach performance is not fully understood and will be addressed with future drilling and metallurgical test work in the PFS (discussed in Section 26).

The nine columns from the historic MacArthur Pit area averaged 83.9% extraction. Based on the 65% average column extraction of the 32 columns using a 0.15% cutoff grade, MacArthur Pit resources were conservatively predicted to achieve 70% copper extraction while all other pit areas were predicted to achieve 65% extraction. Acid consumption for the MacArthur Pit material only was also reduced from the estimated global acid consumption of 35 pounds of acid per ton of oxide material to 30 pounds per ton. Oxide material from the other pit areas was projected at 35 pounds of acid per ton of mineralized material. Mixed oxide/secondary sulfide material was projected to consume 30 pounds of acid per ton of mineralized material.

The project life of mine (LOM) mine plan shows that, of the 271 million tons of mineralized material, 132 million consists of the MacArthur Pit oxide material (49%). The plan also shows that 68.4% of the total mineralized material is oxide mineralization, mixed oxide and secondary sulfide making up the remaining 31.6 percent.

Of importance to project economics are the timing and sequence of mineralized material production. MacArthur Pit oxide material constitutes 90 percent of the material leached during the first 7 years of mine life. MacArthur oxide material has the highest copper leach extraction, the lowest acid consumption, and the lowest strip ratio in these first years' works to optimize project economics.

13.2 OXIDE MINERALIZED MATERIAL ACID CONSUMPTION

Column test work assumed the use of an acid cure application followed by continued acidification during leaching/rinsing of the columns. During the cure stage, 31.59 pounds of acid per ton of mineralized material was added. Following the acid cure, leaching of most columns consumed almost an equal amount of additional acid during the 120 day leach cycle. Most columns were operated between 1.5 and 1.6 pH during this leach cycle. It is probable that all 32 columns were over acidified both during the acid cure and leaching which resulted in excess acid consumption, averaging 57.3 pounds of sulfuric acid per ton of mineralized material processed.

Using leach test results from only the 16 columns at a cut-off grade of 0.15% copper, and disregarding the test work results from the Gallagher Pit zone, acid consumption was determined to be 45.4 pounds of sulfuric acid per ton of mineralized material. These test work results, taken in conjunction with qualified opinion predicts that acid consumption may be reduced 20% to 36.3 pounds of acid per ton of mineralized material considering the column over acidification that was realized combined with shortening of the leach cycle time to 90 days. Arimetco added 25 to 30 pounds of acid per ton of mineralized material with 7.7 pounds of acid consumed per pound of copper produced. However, since acid consumption appears to increase on the periphery of the MacArthur Pit, average acid consumption for all mineralized material was estimated at 35 pounds of acid per ton of mineralized material.

13.3 TRANSITION MATERIAL EXTRACTION AND ACID CONSUMPTION

Research for prediction of copper leach extraction from secondary sulfides (chalcocite) in the transition material is limited to one METCON column. A number of bottle roll tests with high levels of secondary copper were also run but bottle roll tests are considered index tests and do not produce data with an acceptable level of confidence on their own for heap leach design purposes. The total grade of the METCON column #4 was 0.363% copper with a cyanide soluble copper of 0.203% (secondary sulfide). Leach extraction of the secondary copper values was 56%, the extraction kinetics being slower than the oxide columns which is typical of secondary sulfide leaching. Leach extraction after 120 days was still significant and would continue in practice through the residual leaching of lifts as this material is overlaid by fresh mineralized material.

The total head iron content was 3.87% Fe with a tail residue of 3.32% Fe, showing an iron leach extraction of 6.22%. Test results from this column showed the least continuing acid consumption and iron extraction. Acid added during the cure was 32.5 pounds of acid per ton of mineralized material. A total of 45.56 pounds of acid per ton of mineralized material were consumed during this 120 day column test. The pH of the leach solution on day one of the column test was 0.43 indicating that the column was likely over acidified.

During the leach cycle the column pH ran between 1.45 and 1.55 and was much easier to maintain at this level.

Ferric iron concentration was 14.3 g/l the first day of rinsing which supplies ferric iron for chalcocite leaching. The ferrous iron was near zero after about 20 days of leaching showing that first stage chalcocite leaching was complete. The solution oxidation/reduction potential (ORP) remained about 650 mV after 20 days, ideal for second stage chalcocite leaching.

The head screen analysis of the one secondary sulfide column tested was coarser than the materials in the other 31 columns. This column also showed minimal chemical degradation. The head screen analysis was significantly coarser than the column averages and very little chemical degradation occurred. Chalcocite may tend to be more disseminated within the host rock than oxide material. Although the copper grade in the column is not high, some acid will be generated during residual leaching as the second stage of chalcocite (covellite) is slowly leached resulting in elemental sulfur formation. Therefore, considering a shorter leach cycle time, acid consumption for secondary sulfide material leaching was predicted to be 30 pounds of sulfuric acid per ton of mineralized material. Copper leach extraction with residual leaching is predicted at 60 percent.

13.4 LEACH CYCLE TIME

A review of the column test work shows that copper extraction was nearing completion after 75 to 90 days. Overall extraction kinetic curves of the 32 columns were marginally slower than typical oxide leach columns, perhaps due to some oxide mineral dissemination within the host rock. A 90 day leach cycle was selected. This cycle time was also chosen to minimize the continuing acid consumption with little copper extraction occurring over the remaining 30 days of the 120 day leach as realized in the columns. Residual leaching beneath an overlaid lift will not see the 6 to 8 g/l acid concentration of a new lift. Once overlaid, the lift will see between 2 and 4 grams per liter, still promoting leaching but at significantly lower acid consumption, thereby optimizing overall copper extraction and acid consumption globally.

13.5 LEACH SOLUTION APPLICATION RATE

Considering the chemical degradation experienced during the column leach tests and with the Arimetco leach pads, a leach application rate of 0.0035 gpm per square foot was selected. With multiple lift overlays, the leach pad will see continuing consolidation by chemical degradation and mineralized material column weight. Thus permeability will tend to decrease probably limiting the leach application rate to 0.0035 gallons per minute per square foot. The new leach pads can still be leached at higher application rates for the first 20 to 30 days to optimize copper extraction kinetics while the bulk of the pads can be operated at a lower application rate.

13.6 PAD HEIGHT

Production lift height was selected to be 20 feet even though a 15 foot lift may be preferable metallurgically. The pad footprint, leach flow rate, and capital and operating costs increase proportionally with reduced lift heights. Extraction performance at the grade of this mineralized material with a 15 foot high lift height would likely not justify these significant capital and operating costs. Additional test work in the pre-feasibility stage will supply better data to refine the lift height.

13.7 PLS FLOW RATE AND PLS GRADE

Assuming no intermediate leach solution recycle, the PLS flow rate is estimated at 10,400 gallons per minute with a PLS grade of 1.0 g/l copper, including raffinate recycle of 0.1 g/l copper considering 90% SX recovery of copper. M3 Engineering confirmed the flowrate of 10,400 gpm and PLS grade of 1 g/liter based on leach duration, irrigation rates and lift height. The PLS grade was determined by copper recovery and PLS flow rate.

13.8 PARTICLE SIZE TO HEAP LEACH

Historic test work provides limited ROM data for copper extraction and acid consumption. However, MacArthur ROM mineralized material was successfully processed by Arimetco with good copper extraction and acid consumption, supporting the ROM leaching approach. The proposed Phase II PFS metallurgical program will address particle size vs. copper extraction and acid consumption (refer to section 26 for additional detail).

13.9 HEAP LEACH DESIGN CRITERIA

As a result of studying and analyzing all metallurgical test work to date, the following design criteria were developed for the MacArthur Project PEA.

- Annual leach material mining rate -15,000,000 tons
- Daily leach material mining rate - 41,095 tons
- ROM truck dumping direct to the leach pad
- Acidification procedures to be determined during the pre-feasibility test work
- Leach pad slope at 1.75 to 1 with step backs between each lift
- One inner-lift liner at mid-point of the final pad elevation
- Leach cycle time-90 days
- Leach solution application rate-0.0035 gallon per minute per square foot
- Lift height-20 feet
- Individual leach module lift size- 250 feet by 600 feet by 20 feet deep

- PLS grade-1.0 grams per liter
- PLS flow rate- 10,400 gallons per minute
- Mineralized material bulk density- 125 pounds per cubic foot
- MacArthur Pit oxide material
  - Copper extraction- 70%
  - Acid consumption 30 pounds of acid per ton of mineralized material
- Other oxide material
  - Copper extraction- 65%
  - Acid consumption- 35 pounds of acid per ton of mineralized material
- Transition sulfide material
  - Copper extraction- 60%
  - Acid consumption-30 pounds of acid per ton of mineralized material

Table 13-1: MacArthur Historical Test Work

Date	Laboratory	Prepared for	Sample type	Head Grade % Tcu	Head Grade % AsCu	Treatment	% Recovery Tcu	% Recovery AsCu	Cumulative PLS Grade gpl	Cure Acid, lb/ton mineralized material	Acid Consumption lb./ton mineralized material	Acid Consumption lb acid / lb Cu Extracted	Irrigation Rate gpm/ft.2	Remarks
1995	Leach Inc., Tucson,AZ	Arimetco, Inc.	Bulk 2,450 lb from MacArthur	0.302		BM-1 Crush-6" 24"x 8.2' column	65.5%		0.77	20.5	34.0	8.6	Initial 0.0045 gpm/ft.2	Cu present as Chrysocolla.
1995	Leach Inc., Tucson,AZ	Arimetco, Inc.	Bulk 2,450 lb from MacArthur	0.348		BM-2 Crush-3" 12"x9.3' column	75.0%		0.52	24.5	35.5	6.8	for 42 days , then reduced	No malachite, azutite or Cuprite
1995	Leach Inc., Tucson,AZ	Arimetco, Inc.	Bulk 2,450 lb from MacArthur	0.347		BM-3 Crush-1"; 8"x 9.9' column	84.6%		1.32	36.2	59.3	10.1	to 0.0030 gpm/ft2 for 18 days	All samples - acid cure 7 days
1995											55			Acid Consumption 8 hour test of sample, with pH of 1.8
				Calculated Head Grades							Acid Consumption Calculate			
												Acid consumption after SX/EW Credit		
1992	McClelland, Sparks, NV-1 Summary Progress Report	Mine Development Associates	Bulk 2,300 lbs	0.8315	0.6285	Crush-6",24"x10' column,no acid agglom	19.5%	25.9%	N/A	0	17.5	7.6	N/A	One bulk sample split to 2 tests. Tests were in progress 51 days.
1992	McClelland, Sparks, NV-2 Summary Progress Report	Mine Development Associates	Bulk 2,180 lbs	0.8315	0.6285	Crush-6",24'x10' column, acid agglom	54.2%	71.7%	N/A	30	33.4	5.5	N/A	Tests were in progress and not completed. Recoveries are a projections on total copper. Acid cure was for 5 days before leaching.
				Total Cu										
1990	Mountain States, Tucson, AZ - Final Report	MacArthur Mining & Processing Company	Bulk 520lb BM-14	0.56		Crush-2 1/2' 10"x10' Column	81.6%		N/A	30	30.2	3.3	0.004 gpm/ft2	Test period was 48 days Acid consumption "30lbs/ton ore". Cure time was 7 days
1990	Mountain States, Tucson, AZ - Final Report	MacArthur Mining & Processing Company	Bulk 520lb BM-15	0.51		Crush-2 1/2' 10"x10' Column	75.6%		N/A	30	27.7	3.6	0.004 gpm/ft3	Test period was 48 days Acid consumption "30lbs/ton ore". Cure time was 7 days
				Calculated Head Grade			Recovery after 48 days							
1989	Bateman Metallurgical	Timberline Minerals, Inc.	Phase 1 Trench-8 tons TMI-A	1.114	0.862	Phase 1 Bottle Roll, Crush - 2" (average of 2 runs)	79.2%	84.1%	N/A	175	85.9	4.9		Head grade is average of calculated head grades. Acid consumption =lbs acid /ton divided by lbs recovered Cu
1989	Bateman Metallurgical	Timberline Minerals, Inc.	Phase 1 TMI-B	0.334	0.269	Phase 1 Bottle Roll, Crush - 2" (average of 2 runs)	38.0%	51.1%	N/A	144	49.8	19.6		Head grade is average of calculated head grades. Acid consumption =lbs acid /ton divided by lbs recovered Cu
1989	Bateman Metallurgical	Timberline Minerals, Inc.	Phase 2 Trench-8 tons TMI-A 22 lbs	1.108	0.862	Phase 2 Bottle Roll, Crush - 2" (average of 4 runs)	67.9%		N/A	70 & 105	81.75	5.4		Head grade is average of calculated head grades. Acid consumption =lbs acid /ton divided by lbs recovered Cu
1989	Bateman Metallurgical	Timberline Minerals, Inc.	Phase 2 TMI-B 22 lbs	0.345	0.269	Phase 2 Bottle Roll, Crush - 2" (average of 3 runs)	48.2%		N/A	57.5	35.8	10.8		Head grade is average of calculated head grades. Acid consumption =lbs acid /ton divided by lbs recovered Cu
1989	Bateman Metallurgical	Timberline Minerals, Inc.	Phase 3 TMI-B 22 lbs	0.338	0.269	Phase 3 Bottle Roll, Crush - 2" (Run #6)	40.6%	51.1%	N/A	57.5	36.1	13.2		Head grade is average of calculated head grades. Acid consumption =lbs acid /ton divided by lbs recovered Cu
1989	Bateman Metallurgical	MacArthur Mining & Processing	Bulk Sample A (TMI-A) 2,100 lbs duplicate samples	0.675	0.485	Crushed to minus 3 inches, column leach 18 inch dia by 15 ft	44.2%	61.6%	N/A	50 & 100 gpl acid preconditioning	32.85	5.5	.004 gpm/ft2	Mixed Copper Ores. Head Grade is calculated and averaged.
1989	Bateman Metallurgical	MacArthur Mining & Processing	Sample B (TMI-B) 3,500 lbs duplicate samples	0.335	0.26	Crushed to minus 3 inches, column leach 24 inch dia by 15 ft	22.8%	29.3%	N/A	51 & 100 gpl acid preconditioning	10.95	7.2	.004 gpm/ft3	Predominantly chrysocolla mineralization. Head grade is calculated and averaged
1989	Bateman Metallurgical	MacArthur Mining & Processing	sample B with chloride ion	0.216	0.167		49.3%	64.0%	N/A	57.5	38.1	17.9		
				Calculated Head Grades										
1992	Arimetco	Arimetco	Samples from existing trenches	N/A		ROM? 12'"x10' column A, Leached 93 days		82.7%	N/A	N/A		7.8		Recovery = % of ASCu; acid consumption excludes EW Credit
1992	Arimetco	Arimetco	Samples from existing trenches	N/A		ROM? 12'"x10' column B, leached 58 days		91.6%	N/A	N/A		8.1		Recovery = % of ASCu; acid consumption excludes EW Credit
1976	Anaconda		Composite Samples by ore type by Anaconda MRD 76-118-A	0.48	0.4	Bottle Role Crushed to -3/8 in., 2 kg samples leached for 120 hours	83.24%		N/A	N/A	80.7	12.5		Rock mineralogy black Cu WAD in quartz monzonite, andesite, limonite and quartz monzonite, quartz monzonite porphyry and limonite or quartz monzonite and andesite.
1976	Anaconda		Composite Samples by ore type by Anaconda MRD 76-118-B	0.85	0.74	Bottle Role Crushed to -3/8 in., 2 kg samples leached for 120 hours	87.0%		N/A	N/A	86.0	9.4		Samples were chrisocolla with minor amounts of malachite in quartz monzonite, quartz monzonite and limonite, or andesite.
1976	Anaconda		Composite Samples by ore type by Anaconda MRD 76-118-C	0.14	0.08	Bottle Role Crushed to -3/8 in., 2 kg samples leached for 120 hours	63.3%		N/A	N/A	72.0	38.8		Sampes were limonite in quartz monzonite
1976	Duplicate of above tests done in Tucson		MRD 76-118-A	1	0.88		89.1%		N/A	N/A	134.0	6.4		
1976	Duplicate of above tests done in Tucson		MRD 76-118-B	2.17	2.16		96.5%		N/A	N/A	114.0	2.6		
1976	Duplicate of above tests done in Tucson		MRD 76-118-C	0.23	0.15		65.3%		N/A	N/A	64.0	14.1		
													Calculated Values	
	NOTE:Acid consumption in tests give as Lbs/Ton ore. Lb acid/Lb Cu calculated as ((head gradeX20)X% recovery)/Lbs Acid/Ton													

14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

An updated resource model has been prepared for the MacArthur Copper Project, located near Yerington Nevada, that supersedes previous estimates reported in the January 2011 Technical Report (Tetra Tech, 2011). The previous report included data obtained only through year 2010. Updated mineral resource estimates have been generated by incorporating new exploration drilling and sampling conducted as part of the 2011 exploration program conducted by Quaterra Alaska. Interpolation characteristics have been defined based on the geology, drill hole spacing and geostatistical analysis of the data. The mineral resources have been classified by their proximity to the sample locations and are reported, as required by NI 43-101 guidelines, according to the CIM standards on Mineral Resources and Reserves.

A total of 151 drill holes totaling 80,800 feet were added to the database used for the resource estimation. These included two holes for which data was unavailable at the time of the last estimate, but did not include three 2011 holes which were outside the model limits. This model differs from the previous resource estimate in the following ways:

- The physical dimensions of the overall block model were increased to include the new drilling. The 2010 model (Tetra Tech, 2011) contained only 512 blocks in the x-direction whereas the current updated model contains 548. The other dimension of 400 blocks in the y-direction and 150 levels are unchanged. The individual block size utilized is 25x25x20 feet.
- The interpretation of the mineralized zones for the oxide, mixed (transition) and sulfide mineralization was updated based on the assay data obtained in 2011.
- Dikes were added to the model, steeply dipping to the north, and modified search conditions were used in grade estimations.
- Indicator kriging (IK), based on total copper grades above and below 0.12%, was employed to modify search conditions used in grade estimation.
- More codes were assigned to blocks based on conditions of oxide-mixed-sulfide mineralization, the IK grade envelopes, and whether a block was within a dike or not.
- Dynamic kriging was used to alter the direction of the search ellipsoid for each block sub-areas such as the SE-pit area and areas north and south of the “Hinge Line” that had distinct azimuth and directions of the search ellipsoids
- Search parameters such as the maximum number of samples per sector, the number of samples per drill hole and the minimum samples required were modified.
- Compositing was changed from 20 foot-level style to a 10 foot-zone style.
- Correlograms were used to model the spatial structure used in kriging.
- Some modifications to the search parameters were done on criteria which determined whether an estimated block was to be classified as measured, indicated or inferred.

- New jackknife studies were done to determine the required kriging parameters for block class.

14.2 MACARTHUR RESOURCE ESTIMATION

This section describes the methodology used in developing the mineral resource estimate for contained copper resources in the MacArthur deposit. Recent drilling on the MacArthur property, which further defines a significant amount of copper, coupled with updated geologic and mineral zone interpretations, provides the basis for an updated mineral resource estimate. Figure 14-1 details the drill holes used in the updated estimation of the MacArthur deposit.

The MacArthur mineral resource estimate was prepared in the following manner:

- Data from an additional 151 holes was added for this report.
- The density values for each rock code based on the previous studies are unchanged from the previous model.
- The resource estimate was broken into two areas: the southeast historical pit area (variously called SE or SE-Pit area in this report) and the northwest area (variously called NW or NW-Out in this report).
- Quaterra provided cross-sections with interpreted geology, lithology units, mineral zones (MinZones) and dikes. The MinZones were digitized by Quaterra and Tetra Tech (Tt) to produce wireframes surfaces.
- Dike intercepts were used to create dike blocks, oriented east-west or N70^{o} W, dipping 60^{o} to the north to allow separate grade interpolation within those blocks.
- Statistics for drill hole five-foot interval assays were analyzed for each of the MinZone codes broken out by the southeast and northwest areas and by drill holes completed by Metech and Quaterra.
- The interval assays were composited to a ten-foot zone-length. Statistics for the composites were analyzed for each of the rock codes within the southeast and northwest areas. As with the five-foot interval data, analyses were done separately on the Metech (Anaconda) and Quaterra data.
- Geostatisitcal analysis was done on the ten-foot composite data. Unitized General Relative variograms (UGR) were generated. The directional variograms were modeled with the spherical function using a nugget and up to three nested structures.
- The quality of the variogram models was checked using a model-validation technique called "jackknifing". The method helps determine the best variogram parameters to be used for the theoretical model, and to determine the best kriging parameters (range, direction and search parameters).
- The resource model used multiple pass ordinary kriging (OK) to estimate total copper within each MinZone. The kriged grades were checked by comparing block, composite and assay histograms.

- The block model values were visually inspected in multiple sections and plan maps. These values were compared to the drill hole traces that contain both interval assay data and composite data;
- A resource classification of measured, indicated and inferred was developed based on a combination of minimum required data points, jackknifing and kriging error analysis.
- The MacArthur copper resource was tabulated for volume, tonnage and contained metal for the measured, indicated and inferred classes.
- The resource estimate was broken into two areas for evaluation: the southeast historical pit area and the northwest area.

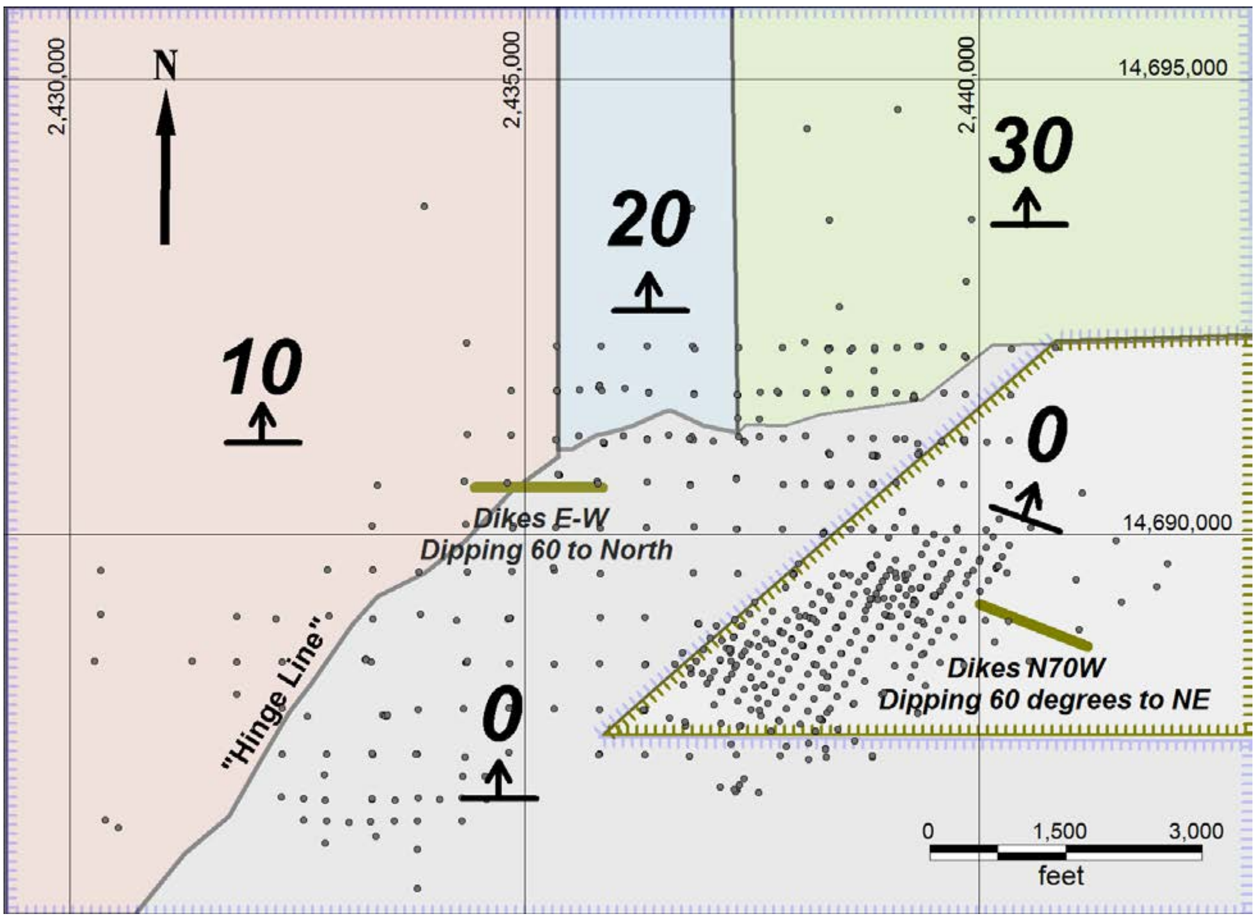


Figure 14-1: Drill Location and Search Zones for the MacArthur 2011 Model

14.3 MACARTHUR BLOCK MODEL

Block model parameters for the MacArthur Copper Project were defined to best reflect both the drill spacing and current geologic interpretations. Table 14-1 shows the MacArthur block model parameters.

Table 14-1: MacArthur Model Parameters

MacArthur East Model Parameters	X (Columns)	Y (Rows)	Z (Levels)
Origin (lower left corner):	2,429,300	14,685,800	2,800
Block size (feet)	25	25	20
Number of Blocks	548	400	150
Rotation	0 degrees azimuth from North to left boundary		
Composite Length	10 feet (Bench)		

An Excel database provided by Quaterra contained the pertinent drill hole and assay information for the MacArthur Copper deposit. The database contained 737 drill holes of which 676 drill holes from Quaterra and Anaconda (Metech) were used. The 61 holes removed included holes with limited or no information on the assays (Pangea Gold 1991, Superior, USBM 1952, Anaconda 1955-57), and six Quaterra holes which were outside the model limits. Of the 676 holes used, there are 280 Anaconda (Metech) RC holes and 396 Quaterra holes (58 core and 338 RC holes). These drill holes traversed 257,895 feet, producing 51,258 total copper sample assay values at a nominal five feet in length.

Table 14-2 shows the MinZone codes, which can be considered levels of oxidation from topography changing with depth. Ideally, the top zone is the oxide zone with the chalcocite mix at a deeper level until a sulfide zone is encountered at depth.

These zones were modeled as strata determined by Quaterra geologists by inspecting the mineralogy of samples from core and RC cuttings. The transition from air (MinZone 0) to the oxide zone/chalcocite mix transition was modeled as MinZone 10. The transition from the oxide zone to the sulfide was modeled as MinZone 20. The MinZone code below the chalcocite to sulfide zones was given the code MinZone 30. Finally, any undefined zones were given the code 9999.

By creating and then combining boundary lines on sections, these transition lines were used to generate MinZone transition surfaces. Then by using wireframe techniques the model produced 3-D MinZone volumes (Tetra Tech used MicroModel® and Quaterra used DataMine®). These initial zones codes were modified by the addition of 100 if the material was within a dike and the addition of 1 if indicator kriging defined the material to be within a higher grade zone.

Table 14-2: MinZone Codes and Density

MinZone Code	Description	Density (cu.ft/ton)
0	Air and previously mined pit	Air (0) and Mined (12.5)
5, 6, 105, 106	Alluvium	12.5
10, 11, 110, 111	Oxide zone	12.5
20, 21, 120, 121	Chalcocite mix zone	12.5
30, 31, 130, 131	Sulfide zone	12.5
9999	Undefined	12.5

Table 14-3 shows the count of the described MinZones of the 5-foot intervals. The table is broken into two-parts. Note that the term “POLYGON” which designates a subset of the drill holes that has been isolated for statistical work. In the second section of Table 14-3 the drill hole data from the “NW-Out Area” (Northwest area) has been segregated for the count. Some of the counted assays are now above the current post-mine topography and are coded with a MinZone Code of 0. Even though these particular samples are above the current topography their assay values contain geostatistical information that was used in estimating remaining resources.

Table 14-3: MinZone Interval Data Count and Drill hole Assay Statistics

```
     Quaterra-RC-2009, Quaterra-Core-2009, Metech-RC, Metech-RC-Twin, QM-Core-Twin-2010
     QM-RC-Twin-2010, Quaterra-RC-2010, Quaterra-Core-2010, QM_2011
   ------------------------------------------------------------------------------------
      POLYGON:  None
      ALL Areas
      TOTAL DRILL HOLES      676
      TOTAL LENGTH      257895.1

                        EASTING      NORTHING ELEVATION  AZIMUTH    DIP   DEPTH
      MINIMUM         2430272.0    14686098.0      4563.0      0.0   45.0     0.0
      MAXIMUM         2442063.2    14694670.0      5491.0    357.1   90.0  2685.5
      AVERAGE         2437574.3    14689347.1      4818.6     64.1   75.3   381.5
      RANGE             11791.2        8572.0       928.0    357.1   45.0  2685.5

      AVERAGE VALUES OF SELECTED DATA
      LABEL      NUMBER       AVERAGE  STD DEVIATION     MIN. VALUE     MAX. VALUE  # MISS.
      FROM-TO    51550        5.00202        0.96702        0.40000       55.90002        0
      Cu%        51258        0.12289        0.19754        0.00000       13.80000      292
      asCu%      36206        0.03203        0.08982        0.00100        4.30000    15344
      QltCu%     18388        0.05205        0.13421        0.00500        6.16000    33162
      cnCu%        578        0.05150        0.14173        0.00500        2.43000    50972
   ------------------------------------------------------------------------------------
      POLYGON: SE-PIT
      INSIDE SE-PIT AREA  DRILL HOLES            405
      TOTAL LENGTH      104865.7
      AVERAGE VALUES OF SELECTED DATA
      LABEL      NUMBER       AVERAGE  STD DEVIATION     MIN. VALUE     MAX. VALUE  # MISS.
      FROM-TO    21006        5.00028        0.75582        0.40000       54.80000        0
      Cu%        20885        0.17185        0.17696        0.00500        3.84000      121
      asCu%       9615        0.05856        0.12198        0.00300        2.30000    11391
      QltCu%      4202        0.07795        0.12791        0.00500        2.30000    16804
      cnCu%        180        0.02864        0.05539        0.00500        0.42000    20826
     ----------------------------------------------------------------------------------
  -
      POLYGON: NW-OUT
      OUTSIDE PIT AREA DRILL HOLES              271
      TOTAL LENGTH      153029.4
      AVERAGE VALUES OF SELECTED DATA
      LABEL      NUMBER       AVERAGE  STD DEVIATION     MIN. VALUE     MAX. VALUE  # MISS.
      FROM-TO    30544        5.00323        1.08874        0.40001       55.90002        0
      Cu%        30373        0.08927        0.20380        0.00000       13.80000      171
      asCu%      26591        0.02244        0.07252        0.00100        4.30000     3953
      QltCu%     14186        0.04437        0.13508        0.00500        6.16000    16358
      cnCu%        398        0.06183        0.16573        0.00500        2.43000    30146
```

MinZone Interval Data Count (continued)

```
 Quaterra-RC-2009, Quaterra-Core-2009, Metech-RC, Metech-RC-Twin, QM-Core-Twin-2010
 QM-RC-Twin-2010, Quaterra-RC-2010, Quaterra-Core-2010,
POLYGON:  None
Drill Data before 2011
                   EASTING      NORTHING ELEVATION  AZIMUTH    DIP   DEPTH
MINIMUM          2430272.0   14684114.0     4533.3      0.0   45.0    35.0
MAXIMUM          2442821.8   14694670.0     5491.0    357.1   90.0  2000.0
AVERAGE          2437692.1   14689147.7     4801.0     55.3   78.3   338.3
RANGE              12549.8      10556.0      957.7    357.1   45.0  1965.0

TOTAL COUNT            531
TOTAL LENGTH      179649.1


AVERAGE VALUES OF SELECTED DATA
LABEL      NUMBER        AVERAGE  STD DEVIATION     MIN. VALUE     MAX. VALUE  # MISS.
FROM-TO    17682         5.01859        1.39921        0.40001       55.90002        0
Cu%        17547         0.09064        0.19562        0.00000        8.85000      135
asCu%      16326         0.02135        0.06956        0.00100        3.56000     1356
QltCu%      3979         0.06491        0.15691        0.00500        4.81000    13703
cnCu%        398         0.06183        0.16573        0.00500        2.43000    17284

-------------------------------------------------------------------------------------
    QM_2011
POLYGON:  None
Drill Data 2011


                   EASTING      NORTHING ELEVATION  AZIMUTH    DIP   DEPTH
MINIMUM          2431342.5   14686098.0     4613.9      0.0   45.0     0.0
MAXIMUM          2440325.0   14694458.0     5301.7    270.0   90.0  2685.5
AVERAGE          2437248.7   14689948.0     4876.0     92.3   64.7   535.1
RANGE               8982.5       8360.0      687.7    270.0   45.0  2685.5


TOTAL COUNT              151
TOTAL LENGTH         80806.0

AVERAGE VALUES OF SELECTED DATA
LABEL      NUMBER        AVERAGE  STD DEVIATION     MIN. VALUE     MAX. VALUE  # MISS.
FROM-TO    12862         4.98220        0.34916        0.59998       20.00000        0
Cu%        12826         0.08739        0.21453        0.00000       13.80000       36
asCu%      10265         0.02417        0.07696        0.00100        4.30000     2597
QltCu%     10207         0.03636        0.12464        0.00500        6.16000     2655
```

14.4 ASSAY DATA

The assay data was assigned MinZones using the interpreted wireframes. The process first used DataMine® to assign a MinZone to each 25x25x20-foot block within the model specified in Table 14-1. When the majority of a block fell within the interpreted MinZone wireframe it was assigned the corresponding code. These coded blocks were then imported into MicroModel® and used to "back-mark" each sample using a simple majority rule. Table 14-4 gives the count of MinZone for composites. The table is divided into three sections. The first section gives a count of the MinZone codes for assays from all drill hole types, and no limiting polygon. For example the count of assays above the current topography (Code 0) is 7,733. The second section gives the statistics for the assays. For example the mean Cu grade in % in MinZone 0 is 0.327. The average for all zones, combined SE and SW areas is 0.125. The third section is a classic histogram plotted with a log scale.

Table 14-4: Statistics of Cu Assay Data (All Areas)

MINZONE COUNT FOR SAMPLES

All Classes: 1 = Quaterra RC 2 = Quaterra Core 3 = Metech RC 4 = Metech RC-Twin 5 = QMT-Core-Twin 6 = QMT-RC-Twin 11 = quaterra-2010 13 = quaterra-core-2010 14 = QM-2011
POLYGON LIMITING FILE USED: None

CODE*	COUNT	MINCOL	MAXCOL	MINROW	MAXROW	MINLEV	MAXLEV
0	7733	39	507	12	355	1	130
5	701	44	511	32	253	81	135
6	95	167	426	55	231	89	117
10	14118	42	511	12	269	68	134
11	10153	50	491	30	250	71	130
20	5343	39	511	12	307	70	118
21	2622	82	462	51	253	70	119
30	3624	102	455	52	253	47	114
31	487	158	441	72	253	48	108
105	39	122	439	90	231	89	124
106	16	273	302	210	232	102	108
110	2468	42	474	52	269	75	126
111	1167	119	431	35	233	74	112
120	1257	39	474	39	251	73	114
121	537	119	437	72	250	74	115
130	939	122	455	72	251	58	114
131	43	261	400	191	253	68	83
9999	208	184	425	307	347	1	1
TOTAL	51550						

- 2010 43-101 base codes of 5, 10, 20 and 30 have been modified such that:
  All Dike Material has had a 100 added to the base code.
  All composites within a grade shell of 0.12% Cu have a 1 added to the base code.
  Codes 0 and 9999 are exceptions.

All Cu Assay Statistics (continued)

DH CLASS LIMITED BY All Holes
DATA TYPE IS SAMPLE
MINIMUM CUT-OFF ENTERED = 0.005000

STATISTICS FOR LABEL : Cu%
MAXIMUM CUT-OFF ENTERED = 15.000000

ROCK TYPE	SAMPLE COUNT: MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS: MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS: LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED: MEAN	COEF. OF VAR.
0	0	0	0	1083	0.01000	2.8600	0.32744	0.05706	0.23887	0.7295	-1.3312	0.4637	0.6810	0.3331	0.7681
5	2	0	0	641	0.00500	0.40000	0.04271	0.00201	0.04488	1.0508	-3.5473	0.7740	0.8798	0.0424	1.0809
6	0	0	0	93	0.01000	0.80000	0.22656	0.02336	0.15283	0.6746	-1.7566	0.6625	0.8140	0.2404	0.9694
10	45	0	0	14075	0.00500	2.4400	0.06605	0.00458	0.06766	1.0245	-3.0572	0.7348	0.8572	0.0679	1.0417
11	26	0	0	10104	0.00500	3.8900	0.24121	0.03925	0.19812	0.8213	-1.6335	0.4120	0.6419	0.2399	0.7140
20	7	0	0	5321	0.00500	1.2800	0.06776	0.00612	0.07824	1.1546	-3.0903	0.8152	0.9029	0.0684	1.1223
21	0	0	0	2640	0.00500	13.800	0.28725	0.18901	0.43475	1.5135	-1.6692	0.8003	0.8946	0.2811	1.1073
30	9	0	0	3615	0.00500	2.6700	0.05788	0.00820	0.09057	1.5648	-3.3498	0.9396	0.9693	0.0561	1.2485
31	0	0	0	491	0.01000	5.8500	0.27614	0.20353	0.45115	1.6338	-1.7865	0.9592	0.9794	0.2706	1.2687
105	0	0	0	30	0.00500	0.25000	0.05750	0.00241	0.04907	0.8534	-3.1330	0.5856	0.7653	0.0584	0.8923
106	0	0	0	18	0.04000	0.31000	0.14611	0.00943	0.09708	0.6644	-2.1473	0.4611	0.6790	0.1471	0.7654
110	3	0	0	2452	0.00500	1.8700	0.07990	0.01196	0.10934	1.3685	-3.0281	1.0056	1.0028	0.0800	1.3167
111	8	0	0	1146	0.00500	2.2700	0.19537	0.04252	0.20621	1.0555	-1.9700	0.7180	0.8474	0.1997	1.0249
120	4	0	0	1252	0.00500	1.5300	0.09204	0.01608	0.12679	1.3775	-2.9668	1.2297	1.1089	0.0952	1.5557
121	1	0	0	539	0.00500	8.8500	0.20833	0.21505	0.46373	2.2259	-2.2175	1.2372	1.1123	0.2021	1.5640
130	4	2	0	1058	0.00500	2.7500	0.05164	0.01442	0.12009	2.3254	-3.7098	1.2872	1.1345	0.0466	1.6194
131	0	0	0	36	0.01000	0.78000	0.17694	0.03833	0.19578	1.1065	-2.3864	1.5726	1.2540	0.2019	1.9542
9999	173	631	0	6041	0.00500	5.2900	0.05999	0.02967	0.17226	2.8713	-3.6190	1.2186	1.1039	0.0493	1.5435
ALL	282	633	0	50635	0.00500	13.800	0.12459	0.03937	0.19842	1.5925	-2.7221	1.3615	1.1668	0.1299	1.7035

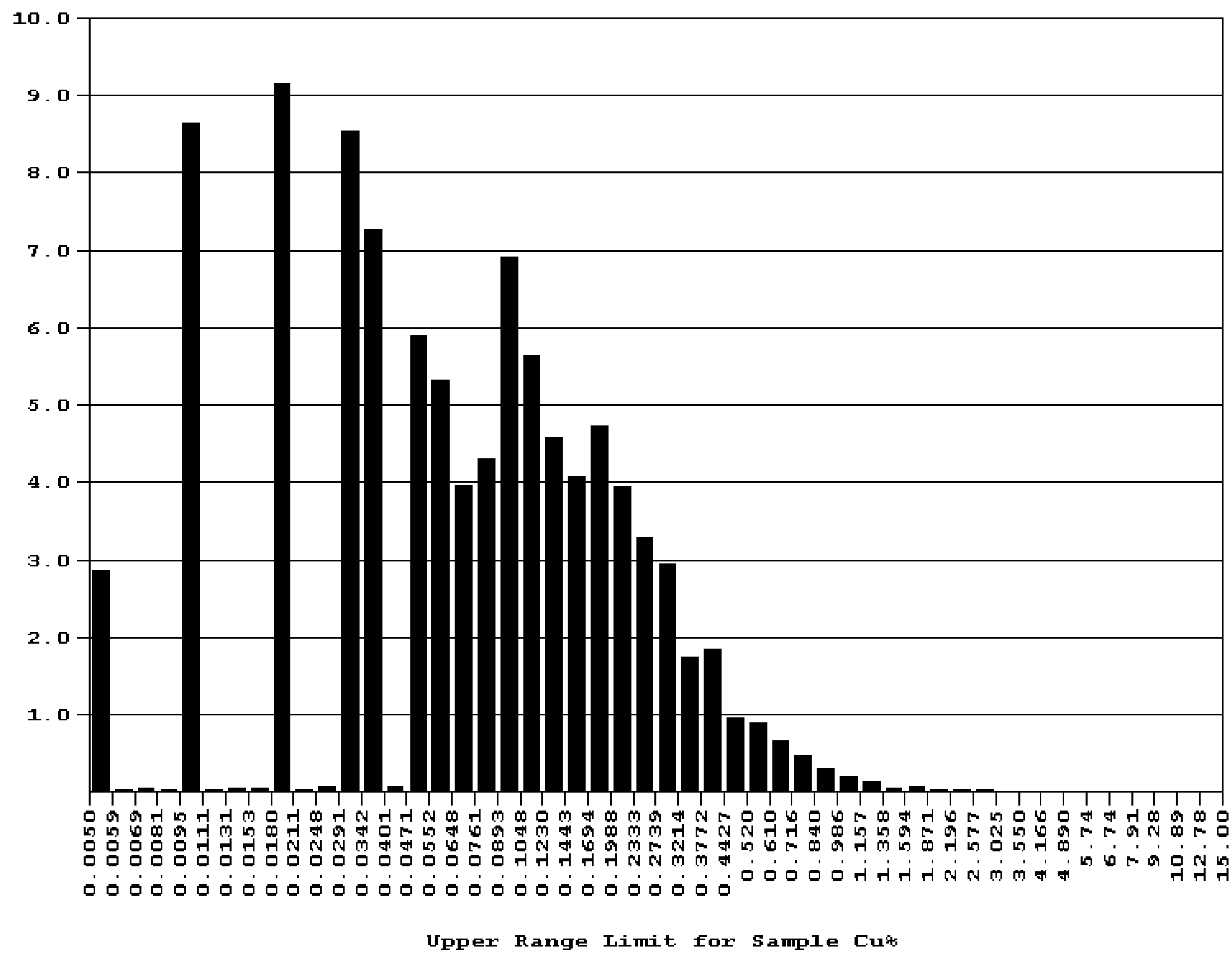


Table 14-5 and Table 14-6 show the statistics in the SE area and NW areas respectively.

The SE data is "lognormal-like", in that it generally follows a bell shaped curve with some notable deviations and an average total copper grade of 0.172. The NW data has a mean grade of 0.091 with a highly skewed distribution.

Table 14-5: SE-Pit Area Cu Assay Statistics

POLYGON SE-PIT-AREA
DH CLASS LIMITED BY All Holes
DATA TYPE IS SAMPLE
MINIMUM CUT-OFF ENTERED = 0.005000

STATISTICS FOR LABEL : Cu%
MAXIMUM CUT-OFF ENTERED = 15.000000

ROCK TYPE	SAMPLE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
0	0	0	0	1073	0.01000	2.8600	0.32999	0.05688	0.23849	0.7227	-1.3153	0.4384	0.6621	0.3342	0.7417
5	0	0	0	466	0.00500	0.40000	0.03811	0.00181	0.04257	1.1169	-3.6728	0.7675	0.8760	0.0373	1.0744
6	0	0	0	79	0.05000	0.80000	0.25266	0.02215	0.14881	0.5890	-1.5651	0.4274	0.6538	0.2589	0.7303
10	10	0	0	6200	0.00500	1.1300	0.08079	0.00392	0.06257	0.7745	-2.7500	0.5246	0.7243	0.0831	0.8305
11	16	0	0	8910	0.00500	3.8400	0.24465	0.03657	0.19124	0.7817	-1.6033	0.3722	0.6101	0.2424	0.6715
20	1	0	0	847	0.00500	0.68000	0.06155	0.00317	0.05634	0.9154	-3.1787	0.9232	0.9608	0.0661	1.2318
21	0	0	0	585	0.01000	1.0300	0.22508	0.02372	0.15403	0.6843	-1.6934	0.4302	0.6559	0.2280	0.7332
30	3	0	0	559	0.00500	0.31000	0.05137	0.00200	0.04475	0.8713	-3.3046	0.7224	0.8499	0.0527	1.0292
31	0	0	0	114	0.01000	0.71000	0.19470	0.01500	0.12248	0.6291	-1.8707	0.6045	0.7775	0.2084	0.9112
105	0	0	0	14	0.01000	0.25000	0.06929	0.00442	0.06650	0.9598	-3.0323	0.7204	0.8488	0.0691	1.0273
106	0	0	0	0	0.	0.	0.	0.	0.	0.0000	0.0000	0.0000	0.0000	0.	0.0000
110	1	0	0	478	0.00500	1.2100	0.10558	0.01662	0.12893	1.2212	-2.8218	1.3444	1.1595	0.1165	1.6840
111	8	0	0	875	0.00500	2.2700	0.21099	0.04730	0.21748	1.0307	-1.8785	0.6824	0.8261	0.2150	0.9893
120	0	0	0	62	0.00500	0.32600	0.06992	0.00396	0.06291	0.8998	-3.1674	1.3651	1.1684	0.0833	1.7077
121	0	0	0	30	0.00500	0.74000	0.17350	0.05083	0.22546	1.2995	-2.5303	1.6382	1.2799	0.1807	2.0361
130	0	0	0	143	0.00500	0.46000	0.03021	0.00250	0.05002	1.6557	-4.1170	1.0319	1.0158	0.0273	1.3440
131	0	0	0	0	0.	0.	0.	0.	0.	0.0000	0.0000	0.0000	0.0000	0.	0.0000
9999	82	0	0	408	0.00500	0.49500	0.06564	0.00662	0.08135	1.2393	-3.4334	1.5612	1.2495	0.0705	1.9403
ALL	121	0	0	20843	0.00500	3.8400	0.17195	0.03134	0.17704	1.0296	-2.1888	1.0166	1.0082	0.1863	1.3280

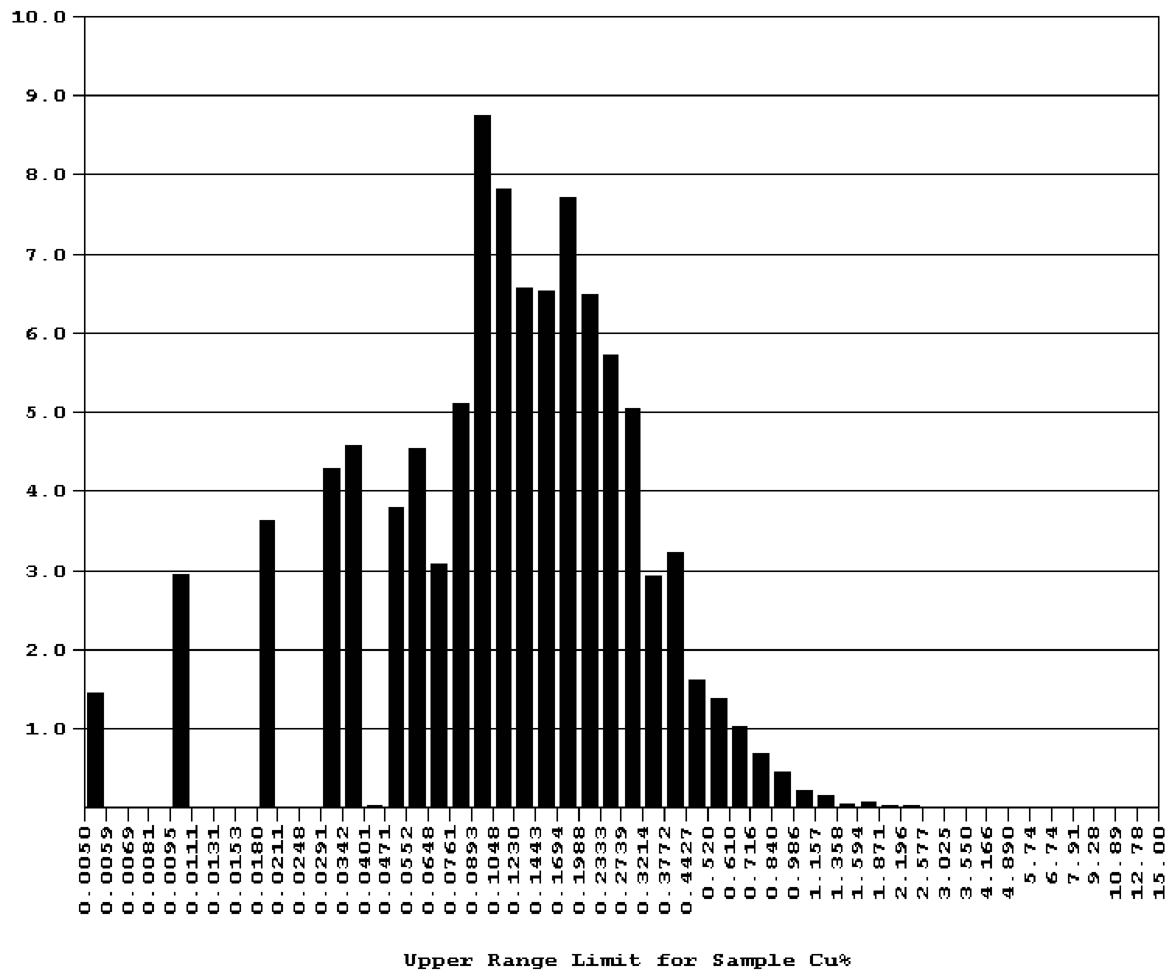


Table 14-6: NW Area TCu Assay Statistics

POLYGON: NW-OUT-AREA
DH CLASS LIMITED BY All Holes
DATA TYPE IS SAMPLE
STATISTICS FOR LABEL : Cu%
MINIMUM CUT-OFF ENTERED = 0.005000
MAXIMUM CUT-OFF ENTERED = 15.000000

ROCK TYPE	SAMPLE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
0	0	0	0	10	0.02000	0.09000	0.05400	0.000649	0.02547	0.4717	-3.0360	0.2540	0.5040	0.0545	0.5377
5	2	0	0	175	0.00500	0.30000	0.05497	0.00236	0.04858	0.8837	-3.2132	0.6381	0.7988	0.0553	0.9449
6	0	0	0	14	0.01000	0.27000	0.07929	0.00493	0.07022	0.8856	-2.8371	0.6146	0.7840	0.0797	0.9214
10	35	0	0	7875	0.00500	2.4400	0.05443	0.00479	0.06924	1.2720	-3.2991	0.7670	0.8758	0.0542	1.0739
11	10	0	0	1194	0.00500	3.8900	0.21556	0.05854	0.24195	1.1224	-1.8591	0.6514	0.8071	0.2158	0.9583
20	6	0	0	4474	0.00500	1.2800	0.06894	0.00667	0.08168	1.1848	-3.0736	0.7931	0.8906	0.0688	1.1001
21	0	0	0	2055	0.00500	13.800	0.30495	0.23468	0.48444	1.5886	-1.6623	0.9054	0.9515	0.2983	1.2136
30	6	0	0	3056	0.00500	2.6700	0.05907	0.00933	0.09659	1.6351	-3.3581	0.9791	0.9895	0.0568	1.2892
31	0	0	0	377	0.01000	5.8500	0.30076	0.25812	0.50805	1.6892	-1.7611	1.0637	1.0314	0.2925	1.3773
105	0	0	0	16	0.00500	0.10000	0.04719	0.000580	0.02408	0.5103	-3.2211	0.4510	0.6716	0.0500	0.7550
106	0	0	0	18	0.04000	0.31000	0.14611	0.00943	0.09708	0.6644	-2.1473	0.4611	0.6790	0.1471	0.7654
110	2	0	0	1974	0.00500	1.8700	0.07368	0.01063	0.10312	1.3996	-3.0781	0.9107	0.9543	0.0726	1.2190
111	0	0	0	271	0.01000	1.6500	0.14491	0.02387	0.15452	1.0663	-2.2654	0.7186	0.8477	0.1487	1.0255
120	4	0	0	1190	0.00500	1.5300	0.09320	0.01668	0.12917	1.3859	-2.9564	1.2205	1.1048	0.0957	1.5456
121	1	0	0	509	0.00500	8.8500	0.21039	0.22477	0.47410	2.2535	-2.1991	1.2075	1.0989	0.2028	1.5314
130	4	2	0	915	0.00500	2.7500	0.05499	0.01621	0.12731	2.3150	-3.6461	1.2972	1.1389	0.0499	1.6307
131	0	0	0	36	0.01000	0.78000	0.17694	0.03833	0.19578	1.1065	-2.3864	1.5726	1.2540	0.2019	1.9542
9999	91	631	0	5633	0.00500	5.2900	0.05959	0.03134	0.17704	2.9712	-3.6325	1.1910	1.0913	0.0480	1.5134
ALL	161	633	0	29792	0.00500	13.800	0.09146	0.04232	0.20572	2.2493	-3.0955	1.2624	1.1236	0.0851	1.5918

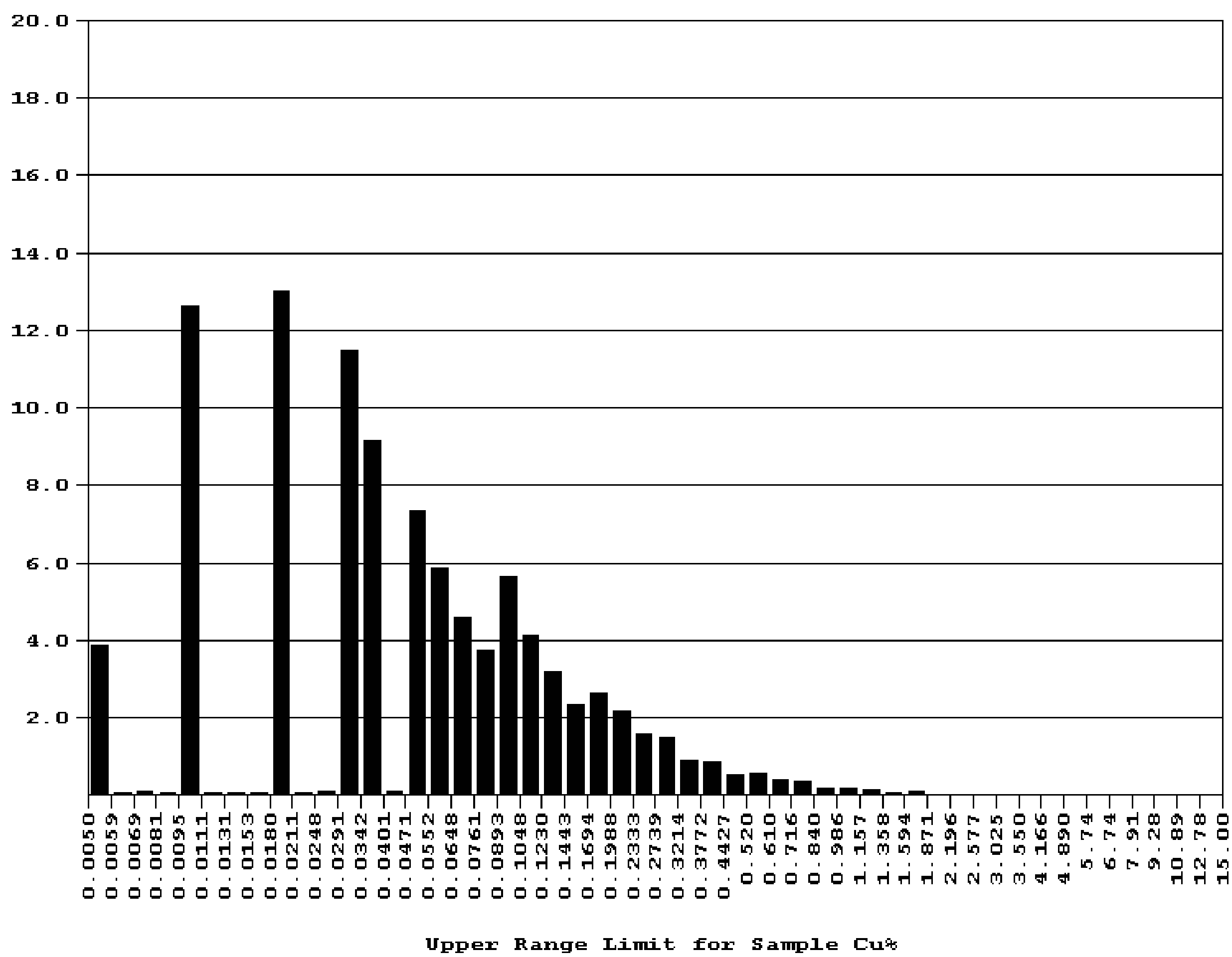


Figure 14-2 shows the SE and NW copper assay grades together in a side-by-side format. The length of the histogram bars is proportional to the total count of assays from each area. Note the enhanced grade of the SE-Pit area (blue bars) is quite apparent with respect to the NW-Out values (green bars).

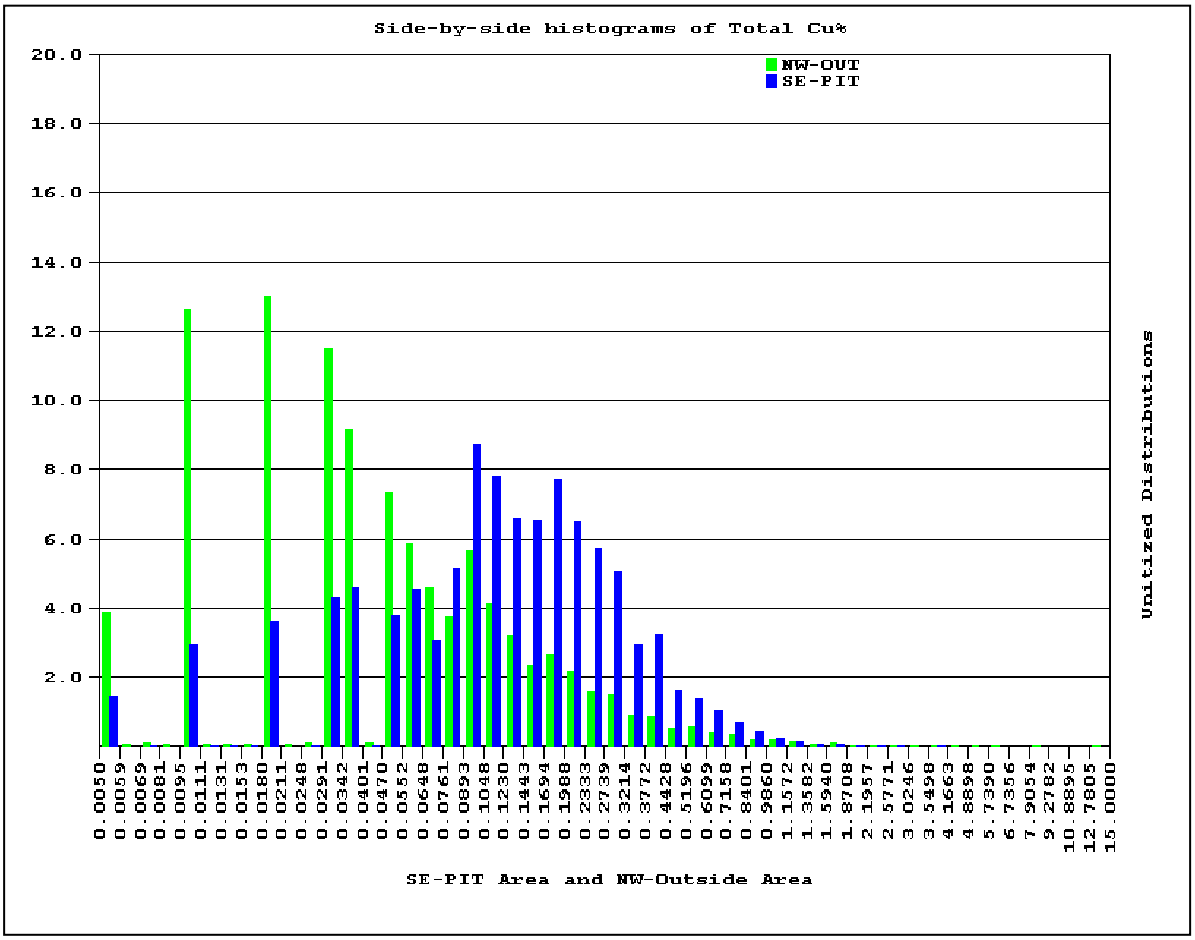


Figure 14-2: Side-by-Side Histograms – TCu% Assay SE-PIT area and NW-OUT area

14.5 COMPOSITE DATA

The assay data was composited using a 10-foot “zone method”. The zone method is a variant of down hole compositing, with the distinction that the composite begins as the drill interval enters a rock code zone. This method tends to reduce averaging composites across zones. The process first used DataMine® to assign a MinZone to each 25x25x20-foot block within the model specified in Table 14-1. When the majority of a block fell within the interpreted MinZone wireframe it was assigned the code. These coded blocks were then imported into MicroModel® and used to “back-mark” each composite using a simple majority rule. No capping was applied. Table 14-7 gives the count of MinZone for composites. Note that the plotted histograms shown in Table 14-8 through Table 14-10 are more lognormal-like than the original assay data. Also note that the average values of the composites are quite similar to the averages shown for assays, and the coefficient of variation (CV) has been reduced.

Table 14-7: MinZone Composite Count (All Areas)

MINZONE COUNT FOR COMPOSITES (10-foot Zone)

Quaterra & Metech Class: 1 = Quaterra RC 2 = Quaterra Core 3 = Metech RC 4 = Metech RC-Twin
5 = QMT-Core-Twin 6 = QMT-RC-Twin 11 = quaterra-2010 13 = quaterra-core-2010 14 = QM-2011
POLYGON LIMITING FILE USED: None

CODE*	COUNT	MINCOL	MAXCOL	MINROW	MAXROW	MINLEV	MAXLEV
0	560	223	421	87	250	91	108
5	347	44	541	32	253	81	135
6	47	167	426	55	231	90	117
10	7118	42	541	12	269	68	134
11	5084	50	541	30	250	72	130
20	2669	39	541	12	307	70	118
21	1317	82	462	51	253	70	119
30	1816	102	455	52	253	47	114
31	244	158	441	72	253	48	108
105	15	122	437	90	209	90	124
106	9	273	302	210	232	102	108
110	1227	42	474	52	269	75	126
111	565	119	431	35	233	74	112
120	624	39	474	39	251	73	114
121	268	119	437	72	250	74	115
130	525	122	455	72	251	66	114
131	18	261	400	191	253	68	83
9999	3577	39	541	1	355	1	130
TOTAL	26030						

- 2010 43-101 base codes of 5, 10, 20 and 30 have been modified such that:
  All Dike Material has had a 100 added to the base code.
  All composites within a grade shell of 0.12% Cu have a 1 added to the base code.
  Codes 0 and 9999 are exceptions.

Table 14-8: All Cu Assay Statistics for Quaterra Composites

POLYGON: NONE
DH CLASS LIMITED BY All Holes
DATA TYPE IS COMPOSITE   STATISTICS FOR LABEL : cCu%
MINIMUM CUT-OFF ENTERED   =   0.005000
MAXIMUM CUT-OFF ENTERED   =   15.000000

	COMPOSITE COUNT				UNTRANSFORMED STATISTICS						LOG-TRANSFORMED STATS			LOG-DERIVED	
ROCK TYPE	MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG MEAN	LOG VAR.	LOG STD.DEV	MEAN	COEF. OF VAR.
0	0	0	0	560	0.01500	2.4600	0.32875	0.05337	0.23101	0.7027	-1.3110	0.4282	0.6544	0.3339	0.7311
5	0	0	0	347	0.00500	0.29000	0.04070	0.00171	0.04136	1.0162	-3.5822	0.7348	0.8572	0.0402	1.0416
6	0	0	0	47	0.02000	0.70000	0.22619	0.02077	0.14413	0.6372	-1.7259	0.5785	0.7606	0.2377	0.8851
10	13	0	0	7105	0.00500	1.2500	0.06571	0.00358	0.05980	0.9100	-3.0258	0.6580	0.8112	0.0674	0.9649
11	6	0	0	5078	0.00750	3.3200	0.24119	0.03198	0.17882	0.7414	-1.6004	0.3415	0.5844	0.2394	0.6380
20	1	0	0	2668	0.00500	0.77000	0.06786	0.00466	0.06828	1.0062	-3.0294	0.6987	0.8359	0.0686	1.0056
21	1	0	0	1316	0.00500	9.8650	0.28720	0.15200	0.38987	1.3575	-1.5957	0.6440	0.8025	0.2798	0.9509
30	11	0	0	1805	0.00500	1.6300	0.05785	0.00631	0.07940	1.3727	-3.2732	0.7978	0.8932	0.0565	1.1049
31	0	0	0	244	0.01400	4.8100	0.27693	0.15982	0.39978	1.4436	-1.6691	0.7038	0.8390	0.2679	1.0107
105	0	0	0	15	0.01250	0.18500	0.05750	0.00189	0.04344	0.7554	-3.0874	0.4728	0.6876	0.0578	0.7774
106	0	0	0	9	0.05000	0.27000	0.14611	0.00863	0.09290	0.6358	-2.1222	0.4154	0.6445	0.1474	0.7176
110	1	0	0	1226	0.00500	1.7550	0.07958	0.01027	0.10133	1.2732	-2.9884	0.9194	0.9589	0.0798	1.2280
111	0	0	0	565	0.00500	1.9653	0.19323	0.03145	0.17733	0.9177	-1.9320	0.6246	0.7903	0.1980	0.9314
120	0	0	0	624	0.00500	1.1900	0.09097	0.01249	0.11176	1.2284	-2.9137	1.1130	1.0550	0.0947	1.4295
121	0	0	0	268	0.00500	5.8600	0.20928	0.17454	0.41778	1.9963	-2.1526	1.1174	1.0571	0.2031	1.4342
130	2	0	0	523	0.00500	1.5050	0.04966	0.00837	0.09146	1.8419	-3.6329	1.1367	1.0662	0.0467	1.4548
131	0	0	0	18	0.01000	0.55500	0.17694	0.03162	0.17783	1.0050	-2.3624	1.5776	1.2560	0.2073	1.9605
9999	120	232	0	3225	0.00500	3.1400	0.05572	0.01821	0.13496	2.4221	-3.6303	1.1993	1.0951	0.0483	1.5224
ALL	155	232	0	25643	0.00500	9.8650	0.12331	0.03249	0.18026	1.4618	-2.6926	1.2893	1.1355	0.1290	1.6218

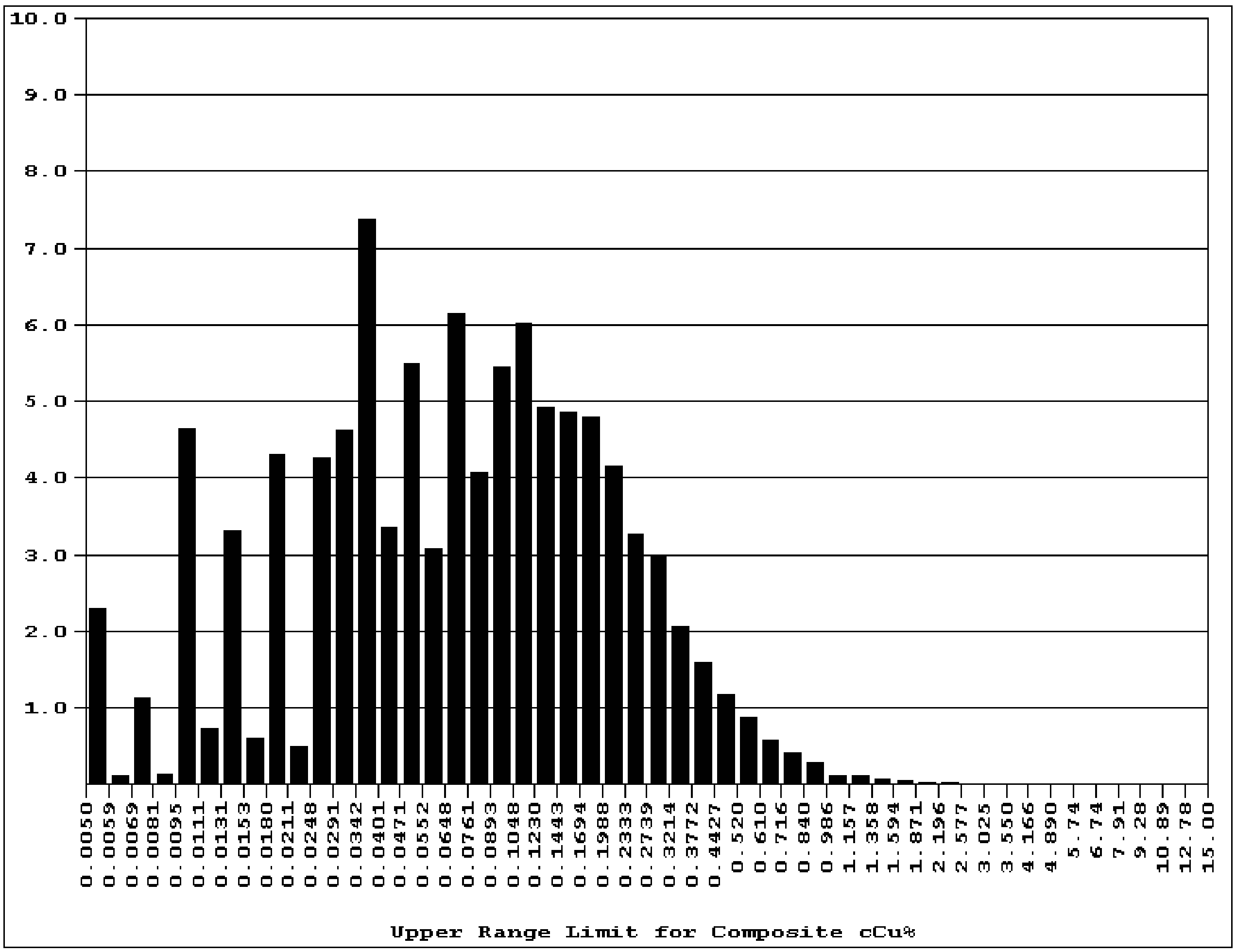


Table 14-9: SE Area Cu Assay Statistics for Quaterra Composites

```
POLYGON:  SE-Pit
CLASS LIMITED BY All Holes
DATA TYPE IS COMPOSITE            STATISTICS FOR LABEL : cCu%
MINIMUM CUT-OFF ENTERED             =        0.005000
MAXIMUM CUT-OFF ENTERED             =       15.000000
```

ROCK TYPE	COMPOSITE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
0	0	0	0	555	0.01500	2.4600	0.33123	0.05315	0.23055	0.6961	-1.2955	0.4031	0.6349	0.3349	0.7046
5	0	0	0	191	0.00500	0.29000	0.04471	0.00172	0.04143	0.9267	-3.4289	0.6395	0.7997	0.0446	0.9463
6	0	0	0	40	0.05500	0.70000	0.25189	0.01929	0.13889	0.5514	-1.5389	0.3561	0.5967	0.2564	0.6540
10	3	0	0	3133	0.00500	1.0100	0.08028	0.00316	0.05622	0.7003	-2.7257	0.4566	0.6757	0.0823	0.7607
11	4	0	0	4469	0.00750	3.3200	0.24443	0.02994	0.17302	0.7078	-1.5756	0.3142	0.5606	0.2421	0.6076
20	0	0	0	492	0.00500	0.49000	0.06703	0.00306	0.05531	0.8251	-3.0029	0.6778	0.8233	0.0697	0.9846
21	1	0	0	308	0.04000	0.88750	0.22401	0.01735	0.13174	0.5881	-1.6465	0.3019	0.5495	0.2241	0.5937
30	6	0	0	324	0.00500	0.67500	0.05768	0.00381	0.06170	1.0697	-3.1746	0.6138	0.7834	0.0568	0.9206
31	0	0	0	61	0.02520	0.58500	0.20124	0.01180	0.10863	0.5398	-1.7617	0.3685	0.6071	0.2065	0.6676
105	0	0	0	7	0.01500	0.18500	0.06929	0.00347	0.05891	0.8502	-2.9788	0.6455	0.8035	0.0702	0.9524
106	0	0	0	0	0.	0.	0.	0.	0.	0.0000	0.0000	0.0000	0.0000	0.	0.0000
110	1	0	0	246	0.00500	0.85500	0.10382	0.01392	0.11799	1.1365	-2.7932	1.2230	1.1059	0.1129	1.5483
111	0	0	0	430	0.00500	1.9653	0.20863	0.03472	0.18633	0.8931	-1.8417	0.6018	0.7758	0.2142	0.9085
120	0	0	0	51	0.00500	0.28750	0.06583	0.00238	0.04876	0.7407	-3.0381	0.8604	0.9276	0.0737	1.1680
121	0	0	0	15	0.00750	0.74000	0.17477	0.05001	0.22363	1.2796	-2.4695	1.5222	1.2338	0.1812	1.8927
130	1	0	0	70	0.00500	0.29000	0.02897	0.00169	0.04112	1.4192	-4.0750	0.9180	0.9581	0.0269	1.2265
131	0	0	0	0	0.	0.	0.	0.	0.	0.0000	0.0000	0.0000	0.0000	0.	0.0000
9999	0	0	0	215	0.00500	0.44500	0.07360	0.00659	0.08118	1.1030	-3.1769	1.2602	1.1226	0.0783	1.5893
ALL	16	0	0	10607	0.00500	3.3200	0.17148	0.02708	0.16457	0.9597	-2.1492	0.8971	0.9472	0.1826	1.2052

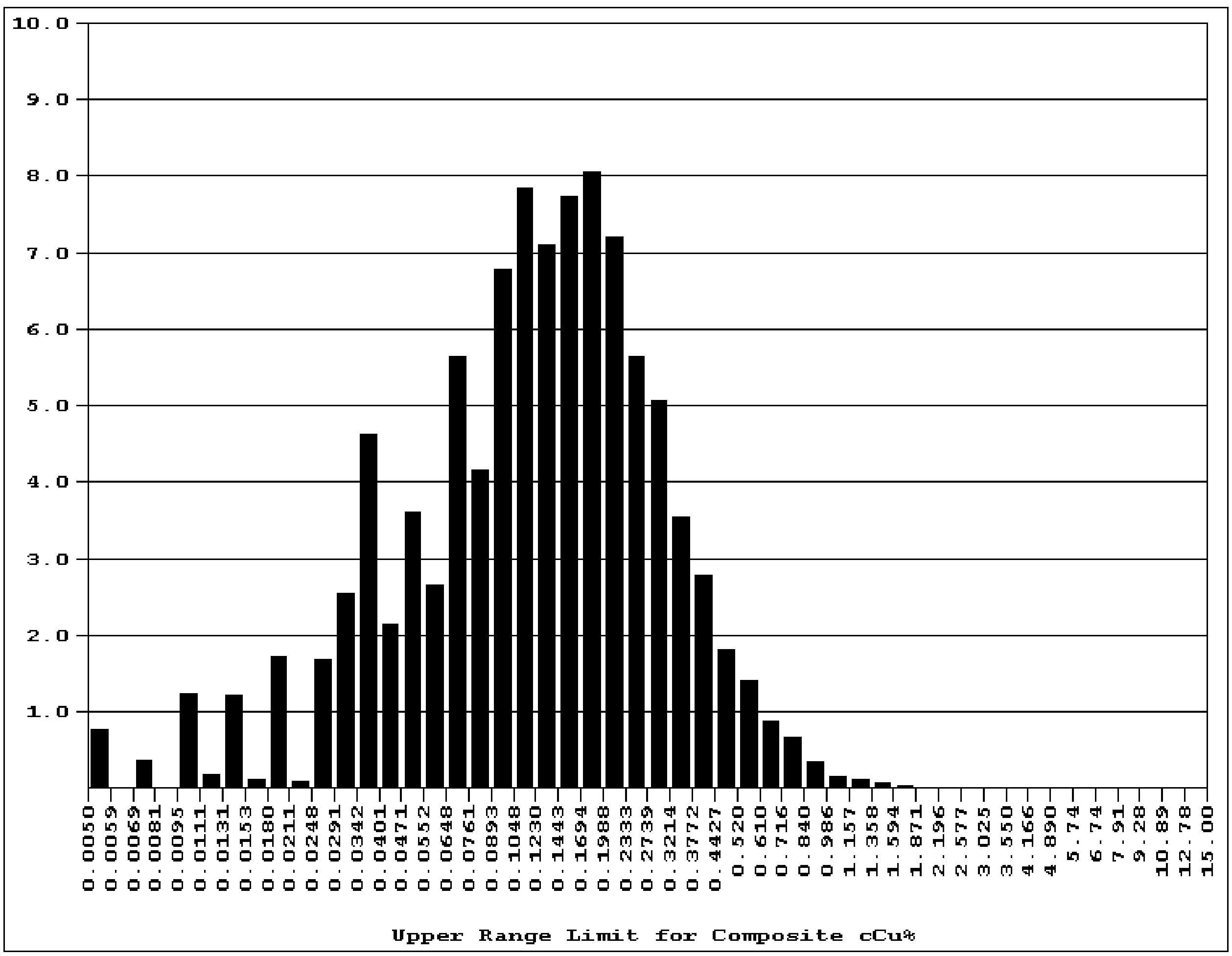


Table 14-10: NW Area Cu Assay Statistics for Quaterra Composites

POLYGON: NW_Area
DH CLASS LIMITED BY ALL
DATA TYPE IS COMPOSITE STATISTICS FOR LABEL : cCu%
MINIMUM CUT-OFF ENTERED = 0.005000
MAXIMUM CUT-OFF ENTERED = 15.000000

	COMPOSITE COUNT				UNTRANSFORMED STATISTICS						LOG-TRANSFORMED STATS			LOG-DERIVED	
ROCK TYPE	MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG MEAN	LOG VAR.	LOG STD.DEV	MEAN	COEF. OF VAR.
0	0	0	0	5	0.02500	0.08000	0.05400	0.000693	0.02632	0.4873	-3.0295	0.2373	0.4872	0.0544	0.5175
5	0	0	0	88	0.01000	0.26500	0.05498	0.00212	0.04605	0.8376	-3.1748	0.5379	0.7334	0.0547	0.8441
6	0	0	0	7	0.02000	0.21000	0.07929	0.00430	0.06554	0.8266	-2.7942	0.5086	0.7131	0.0789	0.8142
10	10	0	0	3933	0.00500	1.2500	0.05450	0.00357	0.05974	1.0961	-3.2478	0.6706	0.8189	0.0543	0.9774
11	2	0	0	597	0.01500	3.1700	0.21586	0.04604	0.21458	0.9940	-1.7882	0.5029	0.7092	0.2151	0.8084
20	1	0	0	2238	0.00500	0.77000	0.06908	0.00509	0.07136	1.0329	-3.0092	0.6679	0.8172	0.0689	0.9747
21	0	0	0	1026	0.00500	9.8650	0.30464	0.18856	0.43424	1.4254	-1.5824	0.7384	0.8593	0.2973	1.0453
30	5	0	0	1530	0.00500	1.6300	0.05900	0.00714	0.08449	1.4321	-3.2794	0.8381	0.9155	0.0573	1.1455
31	0	0	0	188	0.01400	4.8100	0.30083	0.20243	0.44992	1.4956	-1.6398	0.8090	0.8994	0.2907	1.1161
105	0	0	0	8	0.01250	0.09000	0.04719	0.000538	0.02320	0.4917	-3.1824	0.3022	0.5498	0.0483	0.5940
106	0	0	0	9	0.05000	0.27000	0.14611	0.00863	0.09290	0.6358	-2.1222	0.4154	0.6445	0.1474	0.7176
110	0	0	0	988	0.00500	1.7550	0.07333	0.00912	0.09548	1.3021	-3.0378	0.8267	0.9093	0.0725	1.1340
111	0	0	0	135	0.01000	1.0200	0.14419	0.01802	0.13424	0.9310	-2.2197	0.5886	0.7672	0.1458	0.8953
120	0	0	0	593	0.00500	1.1900	0.09208	0.01294	0.11375	1.2354	-2.9023	1.1004	1.0490	0.0952	1.4161
121	0	0	0	253	0.00500	5.8600	0.21133	0.18207	0.42670	2.0192	-2.1339	1.0871	1.0427	0.2039	1.4021
130	1	0	0	453	0.00500	1.5050	0.05285	0.00933	0.09657	1.8272	-3.5645	1.1356	1.0656	0.0500	1.4536
131	0	0	0	18	0.01000	0.55500	0.17694	0.03162	0.17783	1.0050	-2.3624	1.5776	1.2560	0.2073	1.9605
9999	120	232	0	2866	0.00500	3.1400	0.05663	0.01977	0.14060	2.4827	-3.5876	1.1066	1.0519	0.0481	1.4226
ALL	139	232	0	14935	0.00500	9.8650	0.09066	0.03365	0.18344	2.0234	-3.0388	1.1571	1.0757	0.0854	1.4767

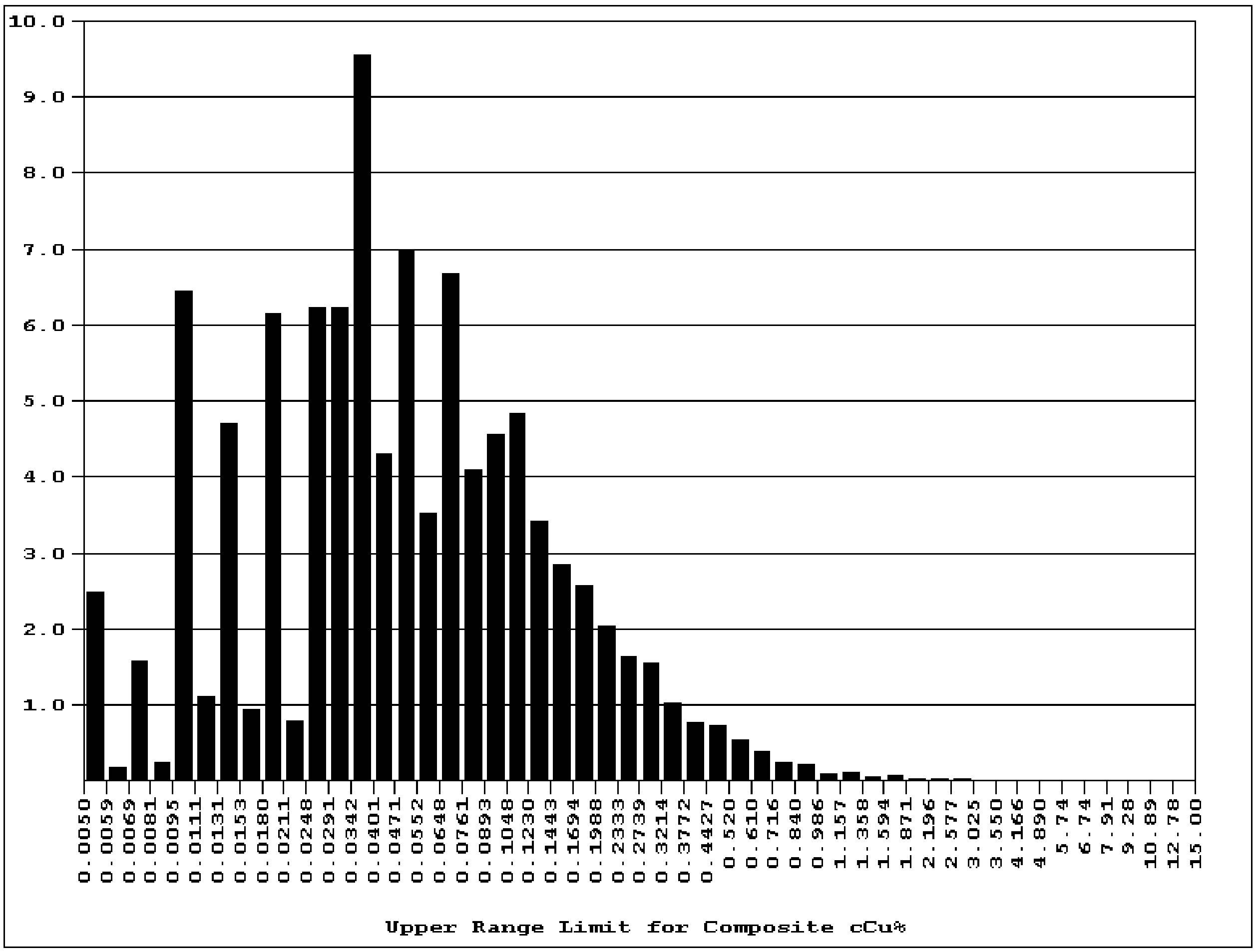


Figure 14-3 shows the SE and NW copper composite grades together in a side-by-side format. The length of the histogram bars is proportional to the total count of assays from each area. Again the enhanced grade of the SE-Pit area (blue bars) is quite apparent with respect to the NW-Out values (green bars).

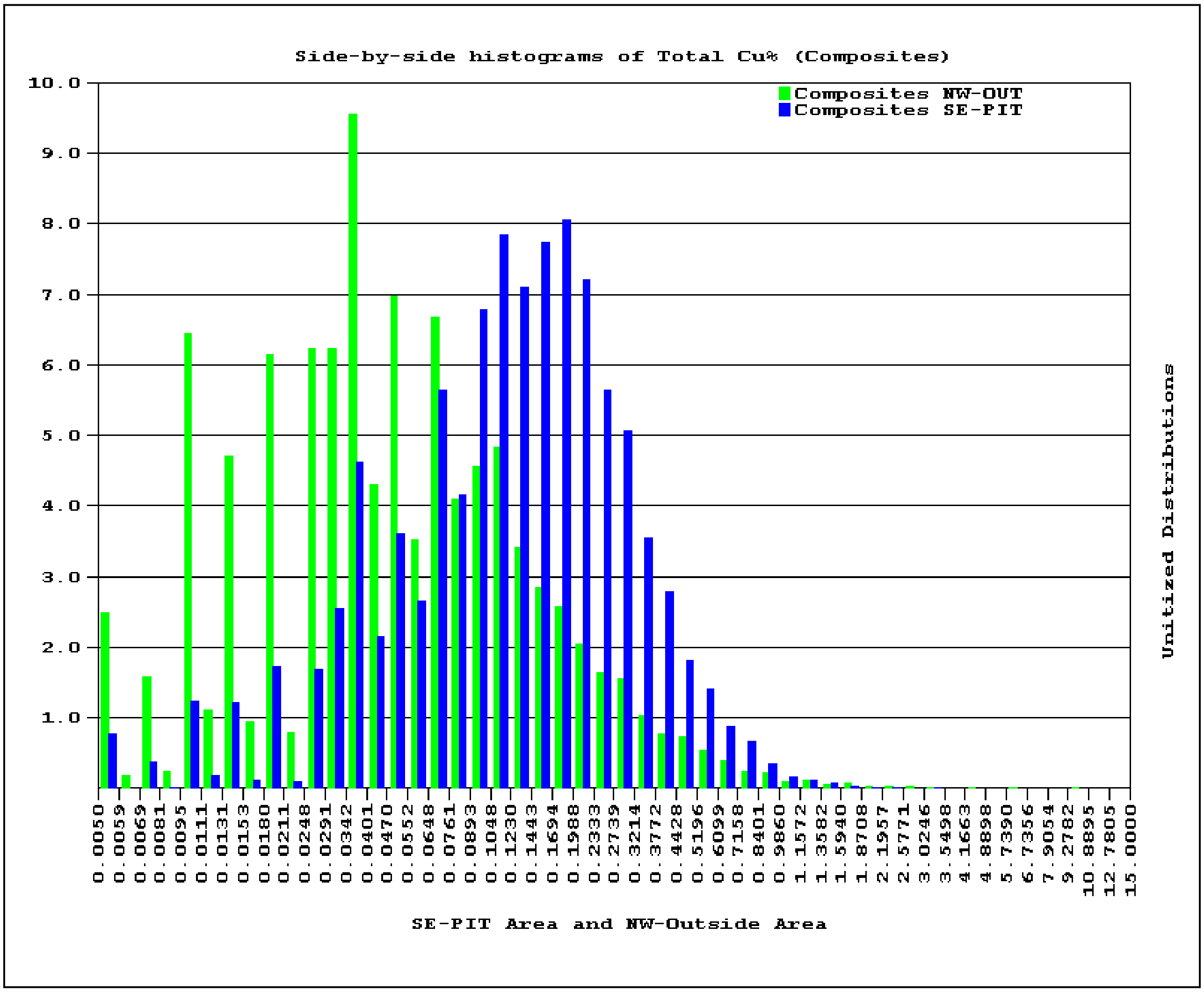


Figure 14-3: Side-by-Side Histograms – TCu% Composites SE-PIT area & NW area

14.6 GEOSTATISTICAL ANALYSIS AND VARIOGRAPHY

A total of twenty-two (21 directional and one omni-directional) variograms were calculated using MicroModel® for each MinZone within each area. The program searches along each direction for data pairs within a 12.5-degrees window angle and 5-feet tolerance band. All experimental variograms are inspected so that spatial continuity along a primary, secondary and tertiary direction can be modeled.

Each variogram model was then validated using the "jackknifing" method. This method sequentially removes values and then uses the remaining composites to krige the missing value using the proposed variogram.

Figure 14-4 shows the horizontal omni-variogram of indicator 0-1 values of total copper grades equal to and below 0.12% Cu (indicator=0) and above (indicator=1). A nugget and three spherical model structures are modeled. Indicator kriging was used to partition the blocks into "high (1)" and "low (0)" grade sub-areas. These sub-areas were used to recode the blocks. For example, a block with a 10 code within a high grade area was recoded with an "11" code.

The second panel of Figure 14-5 shows two figures containing experimental correlograms of total copper in various directions. A correlogram can be considered as a variation of a variogram, with the graph plotting the correlation of data at increasing separation distances. A perfect correlation plots as a "1.0". A nugget effect is when two samples at nearly the same location have a correlation less than 1. When samples are no longer correlated (0.0), the distance is considered to be the "range" of the data.

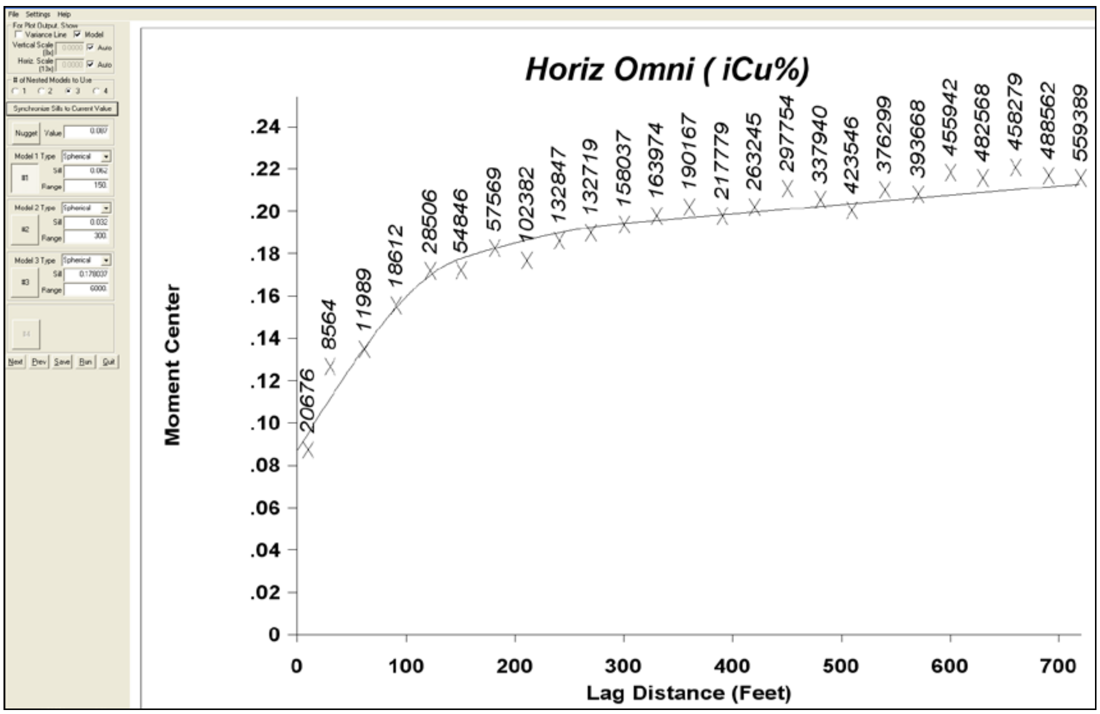


Figure 14-4: 0.12% Indicator Variograms (Omni Direction) For NW-Out and SE-Pit Areas

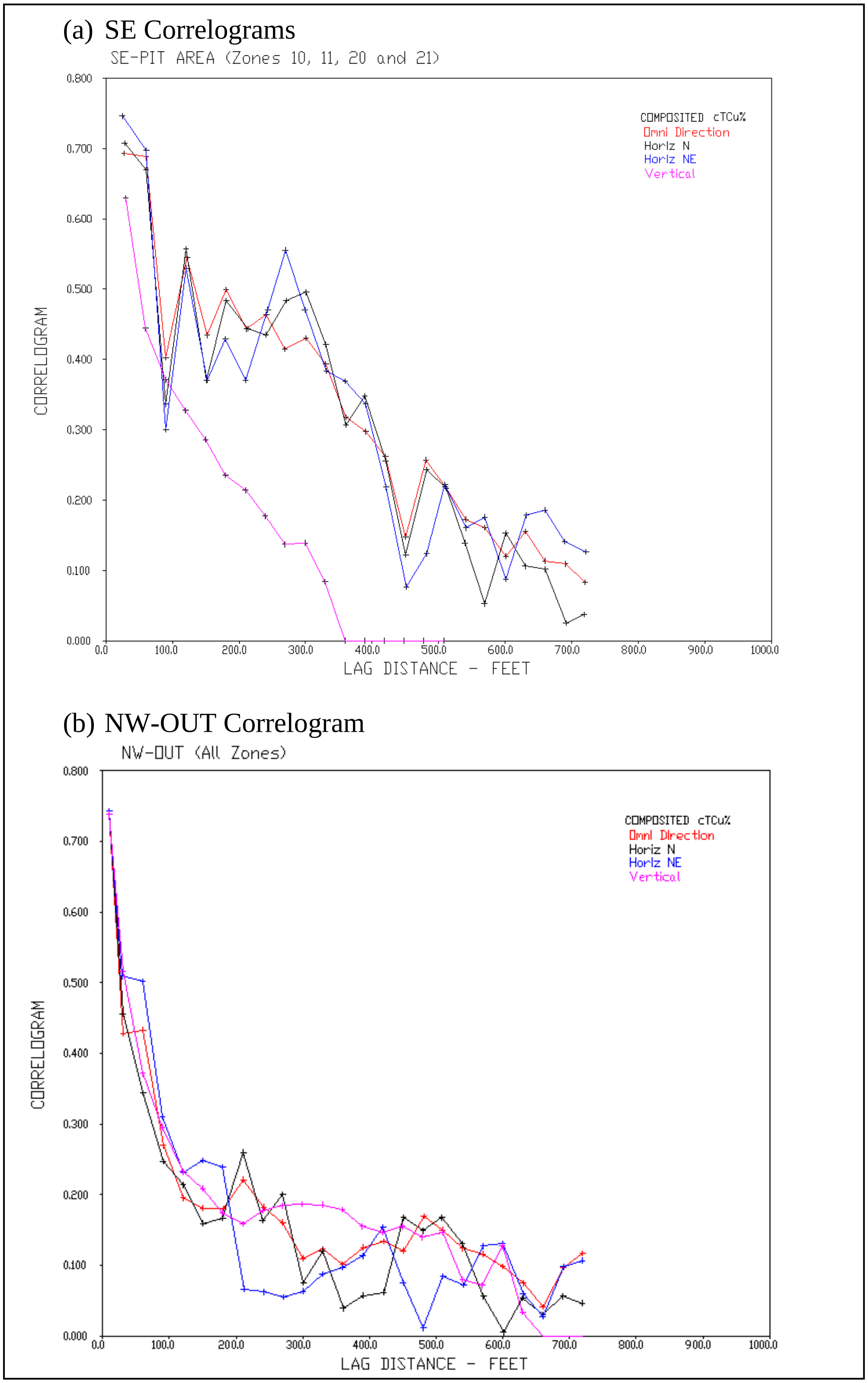


Figure 14-5: Selected Cu% Correlograms For SE-Pit And NW-Out Areas

14.7 KRIGING

Kriging requires not only a variogram model but other search parameters. Table 14-11 shows the search parameters and variogram parameters used for block kriging of total copper. As discussed, the initial MinZone codes of 10, 20 and 30 have been modified. Zone codes within modeled dikes have been increased by 100. Zones within areas considered to be of higher grade using a 0.12% indicator have been increased by 1. Dynamic kriging was used in this estimate; the method changes both the search and variogram parameters for every estimate block. The dip parameter for all 30 and 100 codes are defined in the table.

Table 14-11: Variogram and Search Parameters

Matching Codes			Anisotropy				MIF Search Ranges[5]						Variogram Parameters			
Composite Code	Block Codes	Zone Name	Axis	Anisotropy Axis Length (m)	Anisotropy Rotation	Type[3]	Resource Class[5]	Pass[8]	Resource Code[2]	Maximum Search Range	MaxPts / Sector / Pts Single Drillhole	Min Pts Required to Estimate	Rotation[6]	Nested	Model Type[4]	Parameters[7]
0, 1	0, 1	Indicator > 0.12 Grade	Primary	300	note 6	Az	na	1	na	500	5/2	8	note 6	model 1	Sph	note 7
			Second	400	note 6	Dip	na	na	na	na	na	na	note 6	model 2	Sph	note 7
			Tertiary	100	0	Tilt	na	na	na	na	na	na	0	model 3	Sph	note 7
10	10, 110	< 0.12 Grade Oxide	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	note 6	Dip	I	2	2	260	4/2	6	note 6	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
20	20, 120	< 0.12 Grade Mixed	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	note 6	Dip	I	2	2	260	4/2	6	note 6	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
30	30, 130	< 0.12 Grade Sulfide	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	45	Dip	I	2	2	260	4/2	6	45	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
11	11, 111	>= 0.12 Grade Oxide	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	note 6	Dip	I	2	2	260	4/2	6	note 6	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
21	21, 121	>= 0.12 Grade Mixed	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	note 6	Dip	I	2	2	260	4/2	6	note 6	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
31	31, 131	>= 0.12 Grade Sulfide	Primary	300	note 6	Az	M	1	1	100	5/2	16	note 6	model 1	Sph	note 7
			Second	400	45	Dip	I	2	2	260	4/2	6	45	model 2	Sph	note 7
			Tertiary	100	0	Tilt	F	3	3	500	4/2	2	0	model 3	Sph	note 7
110, 111, 120, 121, 130, 131	110, 111, 120, 121, 130, 131	Dike[9]	Primary	na	note 6	Az	na	na	na	500	5/2	16	note 6	model 1	Sph	note 7
			Second	na'	-60	Dip	na	na	na	na	4/2	na	-60	model 2	Sph	note 7
			Tertiary	na	0	Tilt	na	na	na	na	4/2	na	0	model 3	Sph	note 7

All measurements in feet, all directions in degrees azimuth

Untized Spherical Variogram Structural Parmeters: C0 = 0.140; Sill1 = 0.300; Sill2 = 0.140; Sill3 = 0.42

Cu estimate is done in three passes (measured is first pass, indicated is second and inferred is third), a kriging error adjust is use post-kriging

Notes:
1 Indictor grade envelop IK uses absolute variogram; Copper grade OK uses Unified Relative Variogram converted from correlograms.
2 Kriging Error is used to adjust preliminary class 1,2,3 to 1,2,3 & 4 by post-kriging filter at 0.75 Maximum Kriging Error
3 Az=Azimuth is clockwise (CW) from North, Dip is positive when downward, Tilt rotates CW around primary axis.
4 Sph=Spherical, Lin=Linear, Exp=Exponential, Gau=Gaussian
5 MIF is the accronym for M=Measured, I=Indicated, F=Inferred
6 Dynamic Kriging (Zone 10,11,20,21 rotation and dip defined for each block "on-the-fly" -- see figure 1; Zone 30, 31 rotation "on-the-fly")
7 Dynamic Kriging (Variogram range parameters defined for each block "on-the-fly")
8 Pass: Estimation is done sequentially from shortest search range to largest. Previously estimated blocks are not overwritten.
9 Dike Estimation: Dikes are estimated first with non-dyke codes and then overwritten with Dike coded data. Any prior MIF codes are retained.

For example, for MinZone 10 in the SE area, the search ellipse of 300x400x100 feet will be oriented so its primary axis has an azimuth of 20 degrees north, a dip of 0 degrees. Conditions regarding the minimum number of drill holes to be used for each resource class and how an additional condition that the kriging error must be within certain bounds may also impact a resource classification are discussed further in Section 14.8.

Table 14-12 gives the count of potentially estimated blocks for each of the MinZones in the SE and NW areas. It should be noted that not all of these blocks will be estimated. Table 14-13 through Table 14-15 give the statistics for the kriged blocks within the SE and NW Areas, respectively.

Figure 14-6(a) shows the block values in the SE-Pit area (Zones 10 and 20) for measured + indicated (M&I, green bars) and inferred (red bars) in side-by-side format. Note that M&I have sharply defined the higher grade population. The inferred distribution is more complex, perhaps reflecting a mix of several grade populations. Figure 14-6(b) shows a lesser grade enhancement in the NW zones 10 and 20 for the M&I blocks versus the inferred. In both cases, the enhancement is due to the geologic modeling.

Table 14-12: MinZone Block Count (All Areas)

ROCK COUNT FOR BLOCK MODEL (R200)
NUMBER OF ROCK TYPES FOUND = 18

CODE	COUNT	MINCOL	MAXCOL	MINROW	MAXROW	MINLEV	MAXLEV
0	9822321	1	548	1	400	80	150
5	109536	1	548	1	400	79	145
6	1602	52	455	23	244	88	132
10	3766110	1	548	1	400	59	145
11	133858	47	548	10	359	69	133
20	1186141	1	548	1	400	48	118
21	85279	58	488	39	271	72	119
30	17478986	1	548	1	400	1	114
31	40182	106	485	55	263	54	114
105	1581	32	495	31	309	85	130
106	146	160	447	58	233	88	120
110	67377	22	499	31	354	66	130
111	10970	111	483	34	265	75	122
120	37011	22	499	51	354	62	118
121	10739	94	446	55	256	72	119
130	86867	31	492	56	354	1	114
131	2073	108	418	69	256	63	114
9999	39221	1	548	1	400	1	136
TOTAL	32880000						

Table 14-13: SE-Pit and NW Areas Cu Block Statistics

POLYGON: NONE
CURRENT LABEL : (G101) Kriged Grade kCu%
MINIMUM CUT-OFF ENTERED = 0.005000 MAXIMUM CUT-OFF ENTERED = 15.000000

ROCK TYPE	BLOCK COUNT MISSING	BLOCK COUNT BELOW LIMITS	BLOCK COUNT ABOVE LIMITS	BLOCK COUNT INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
5	56156	0	0	53380	0.00677	0.66483	0.04568	0.00190	0.04357	0.9539	-3.4888	0.8069	0.8982	0.0457	1.1139
6	4	0	0	1598	0.07295	0.49797	0.21242	0.00507	0.07120	0.3352	-1.6091	0.1267	0.3559	0.2132	0.3675
10	3057581	11	0	708518	0.00500	0.71106	0.06831	0.00186	0.04309	0.6307	-2.8786	0.4274	0.6537	0.0696	0.7302
11	880	0	0	132978	0.02949	1.3921	0.21203	0.00494	0.07031	0.3316	-1.5920	0.0748	0.2734	0.2113	0.2786
20	763863	6	0	422272	0.00500	2.0317	0.07869	0.00432	0.06575	0.8356	-2.8418	0.6435	0.8022	0.0805	0.9504
21	0	0	0	85279	0.04866	5.3773	0.26258	0.02320	0.15232	0.5801	-1.4455	0.1899	0.4358	0.2591	0.4573
30	16018515	59757	0	1400714	0.00500	1.9775	0.06203	0.00798	0.08935	1.4405	-3.2416	0.8403	0.9167	0.0595	1.1476
31	137	0	0	40045	0.01489	1.3064	0.23893	0.02052	0.14324	0.5995	-1.5936	0.3424	0.5852	0.2411	0.6390
105	233	0	0	1348	0.00891	0.53346	0.07742	0.00277	0.05267	0.6804	-2.7838	0.4994	0.7067	0.0793	0.8048
106	0	0	0	146	0.03000	0.38161	0.13956	0.00403	0.06349	0.4549	-2.0904	0.2787	0.5279	0.1421	0.5670
110	9505	0	0	57872	0.00700	0.79600	0.07895	0.00285	0.05342	0.6766	-2.7401	0.4220	0.6496	0.0797	0.7246
111	62	0	0	10908	0.01100	0.88415	0.16795	0.00666	0.08161	0.4859	-1.9037	0.2683	0.5180	0.1704	0.5548
120	411	0	0	36600	0.00500	1.0800	0.09871	0.00713	0.08443	0.8553	-2.6755	0.8560	0.9252	0.1057	1.1635
121	0	0	0	10739	0.00500	1.9200	0.17942	0.02387	0.15449	0.8610	-2.0299	0.7053	0.8398	0.1869	1.0122
130	5664	3667	0	77536	0.00500	1.4077	0.06278	0.00520	0.07209	1.1483	-3.2262	0.8873	0.9420	0.0619	1.1953
131	0	0	0	2073	0.00800	0.75274	0.14852	0.01095	0.10462	0.7044	-2.1865	0.6213	0.7882	0.1532	0.9281
ALL	19952232	63441	0	3042006	0.00500	5.3773	0.08175	0.00858	0.09265	1.1334	-2.9311	0.8909	0.9439	0.0833	1.1989

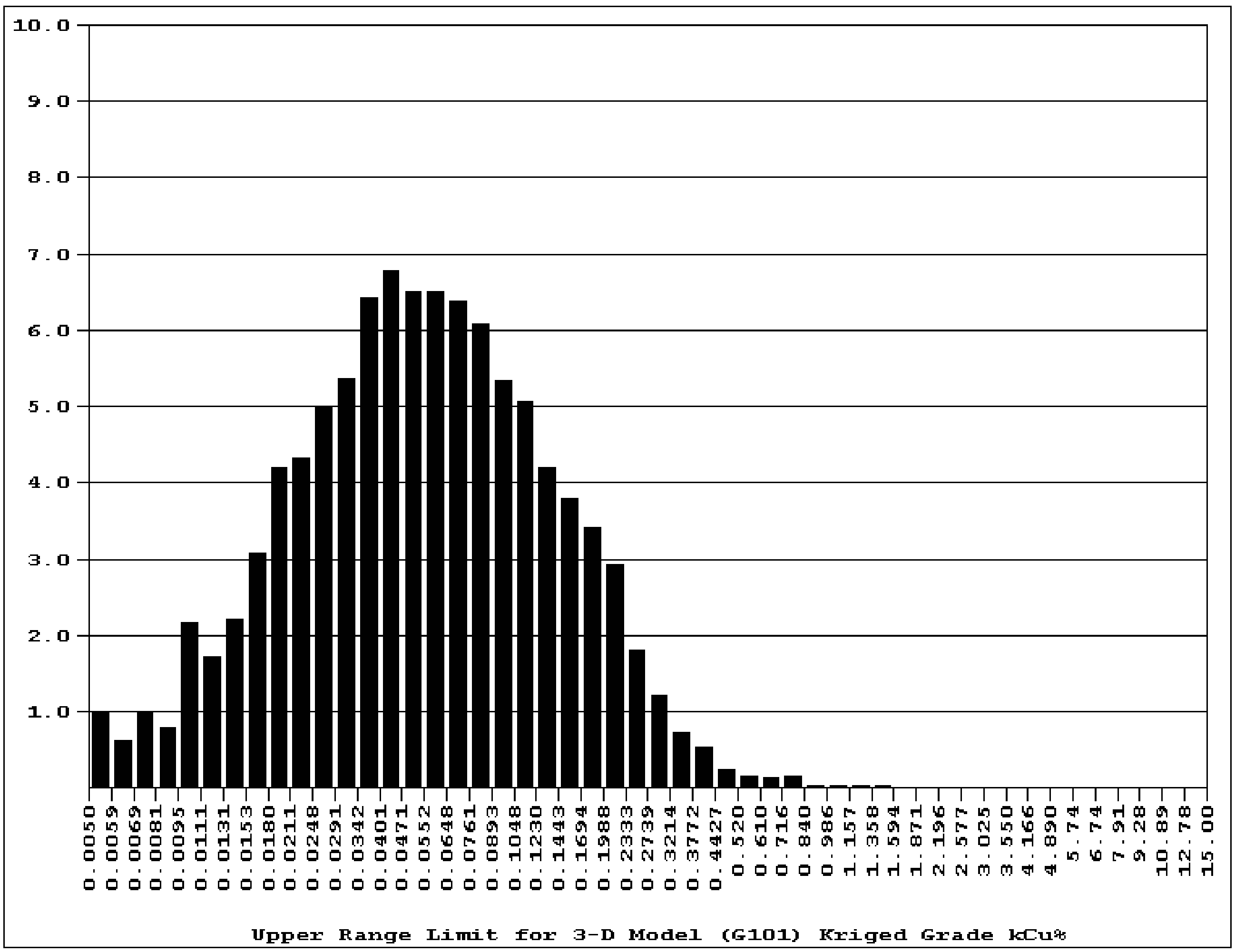


Table 14-14: SE Area Cu Block Statistics

POLYGON: SE-PIT AREA
CURRENT LABEL : (G101) Kriged Grade kCu%
MINIMUM CUT-OFF ENTERED = 0.005000 MAXIMUM CUT-OFF ENTERED = 15.000000

ROCK TYPE	BLOCK COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
5	19516	0	0	38699	0.00677	0.45581	0.03913	0.00174	0.04175	1.0669	-3.6901	0.8274	0.9096	0.0378	1.1346
6	0	0	0	932	0.11429	0.49797	0.22113	0.00323	0.05680	0.2569	-1.5385	0.0566	0.2379	0.2209	0.2413
10	308433	0	0	185401	0.00689	0.60958	0.08138	0.00207	0.04550	0.5591	-2.7003	0.4719	0.6870	0.0851	0.7766
11	396	0	0	102320	0.04206	1.2254	0.21104	0.00437	0.06607	0.3131	-1.5927	0.0669	0.2587	0.2103	0.2631
20	73051	6	0	53582	0.00500	0.63016	0.07629	0.00286	0.05346	0.7008	-2.8956	0.8784	0.9372	0.0857	1.1861
21	0	0	0	21279	0.10751	0.63863	0.20800	0.00414	0.06435	0.3094	-1.6135	0.0844	0.2905	0.2078	0.2968
30	3121676	439	0	200404	0.00500	0.40630	0.06947	0.00310	0.05565	0.8010	-2.9873	0.7579	0.8706	0.0737	1.0648
31	0	0	0	13109	0.02645	0.41835	0.22382	0.00806	0.08976	0.4011	-1.5723	0.1520	0.3898	0.2239	0.4051
105	103	0	0	489	0.00891	0.25200	0.08826	0.00332	0.05758	0.6524	-2.6999	0.6656	0.8159	0.0938	0.9725
106	0	0	0	80	0.07600	0.24900	0.16337	0.00106	0.03258	0.1994	-1.8334	0.0468	0.2164	0.1637	0.2189
110	621	0	0	9004	0.00700	0.58200	0.10003	0.00306	0.05532	0.5530	-2.4709	0.3838	0.6195	0.1024	0.6840
111	58	0	0	7234	0.01400	0.79900	0.17265	0.00463	0.06804	0.3941	-1.8323	0.1616	0.4020	0.1735	0.4188
120	36	0	0	3052	0.00500	1.0800	0.10589	0.00864	0.09296	0.8779	-2.5371	0.6105	0.7813	0.1073	0.9172
121	0	0	0	913	0.01100	1.0800	0.21136	0.02124	0.14575	0.6896	-1.8197	0.6308	0.7942	0.2222	0.9376
130	991	0	0	8273	0.00500	0.33769	0.05180	0.00269	0.05185	1.0010	-3.3877	0.8624	0.9287	0.0520	1.1700
131	0	0	0	384	0.02628	0.21790	0.20159	0.00137	0.03706	0.1838	-1.6308	0.0812	0.2849	0.2039	0.2908
ALL	3536222	445	0	645155	0.00500	1.2254	0.10385	0.00653	0.08080	0.7781	-2.6223	0.8911	0.9440	0.1134	1.1991

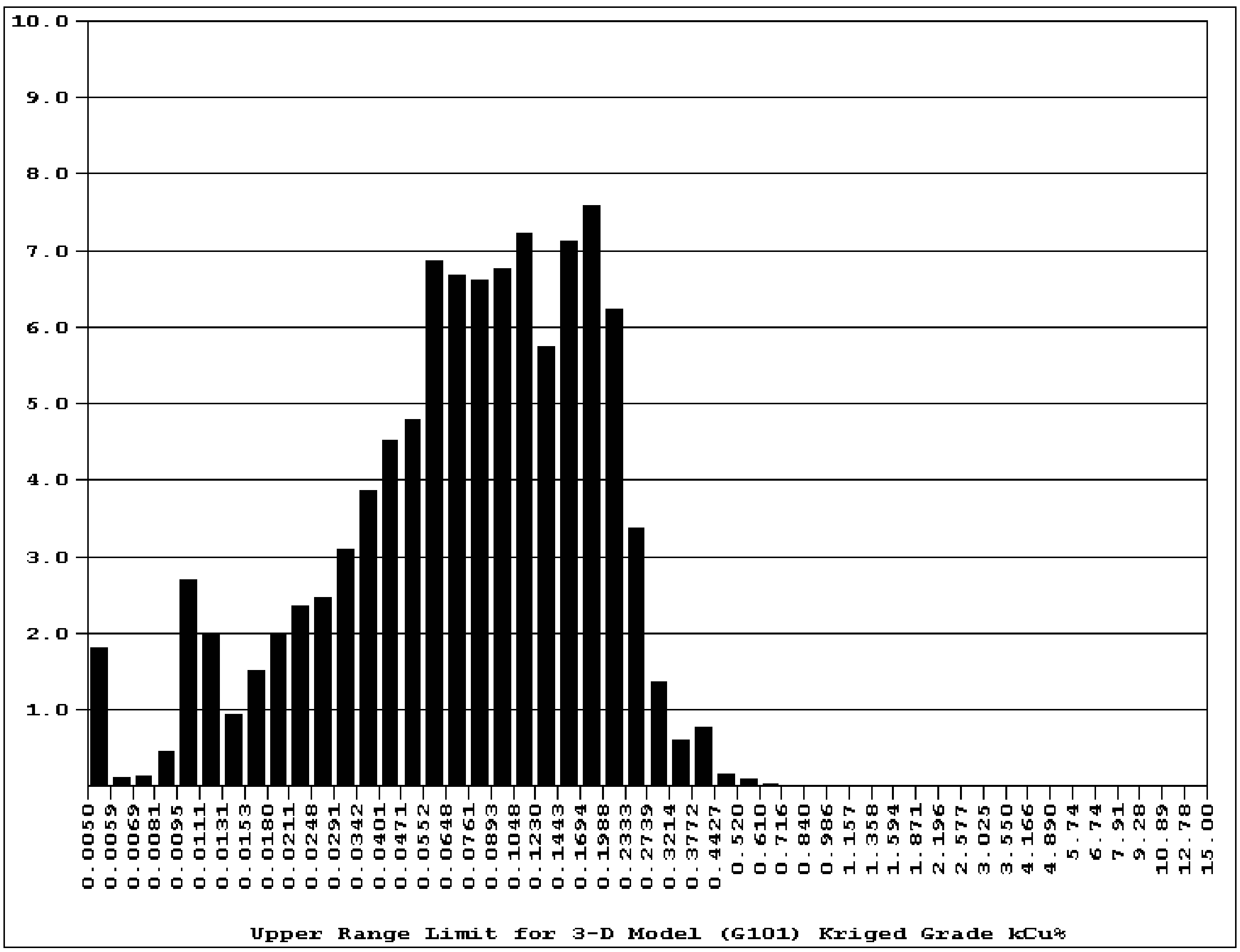
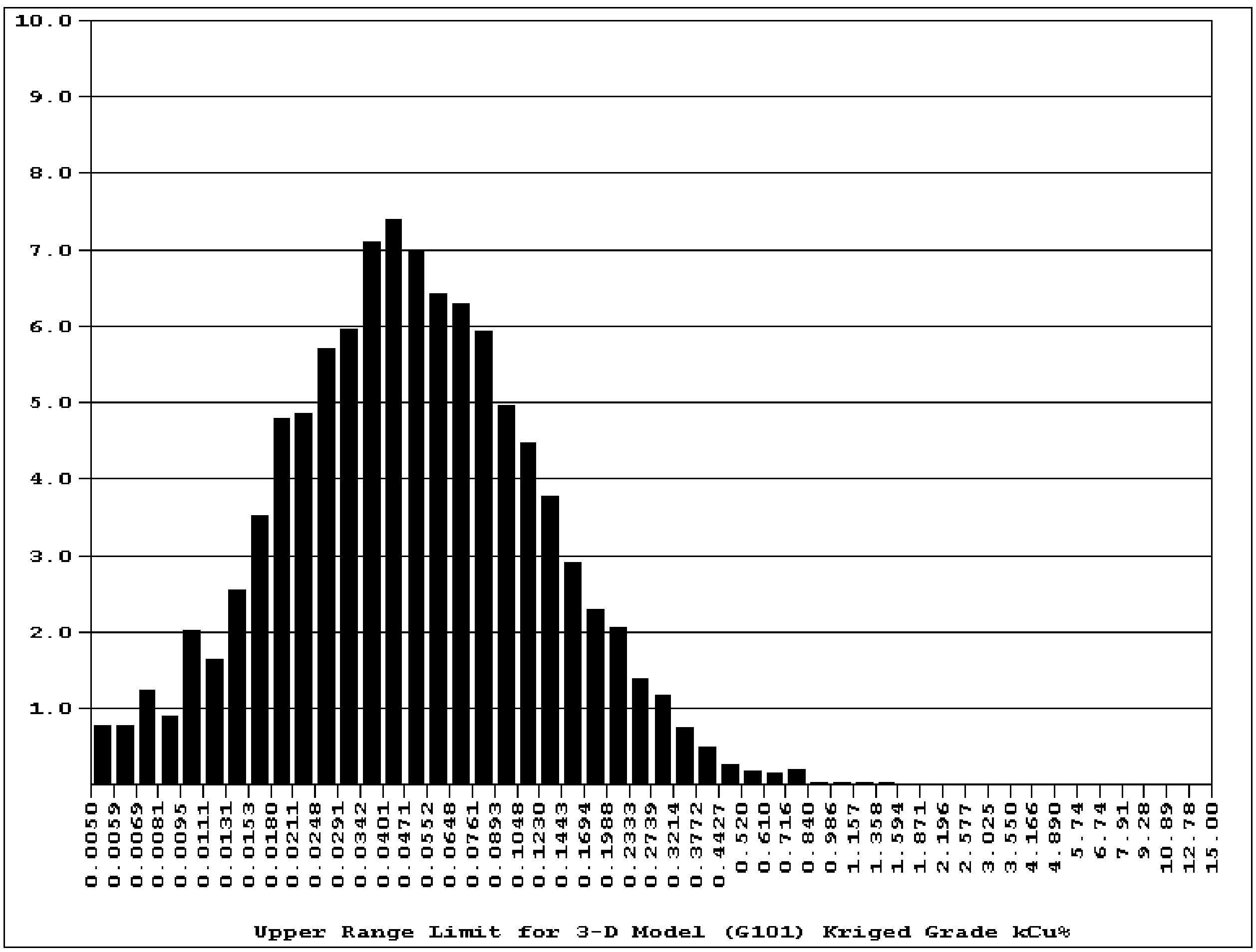
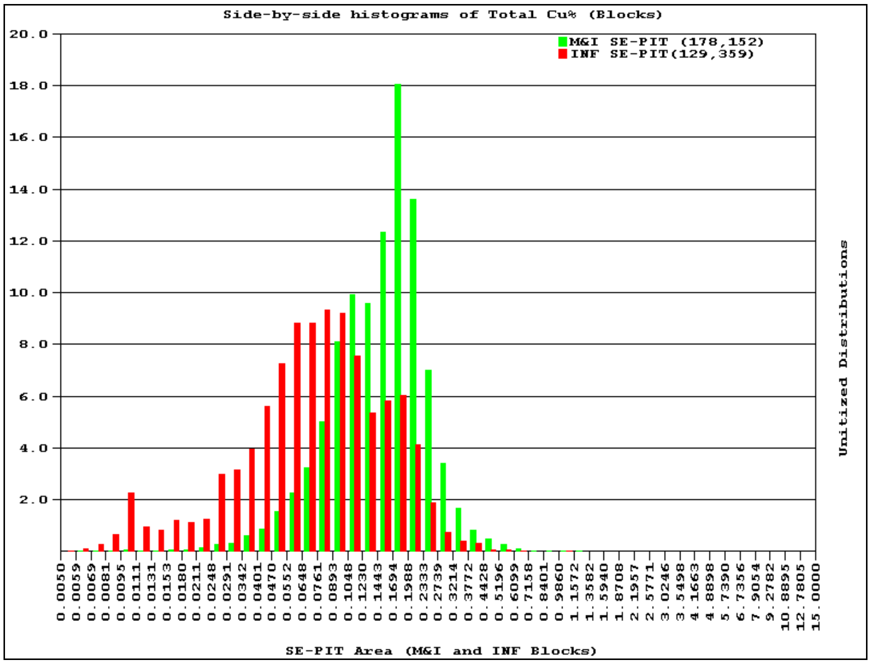


Table 14-15: NE Area Cu Block Statistics

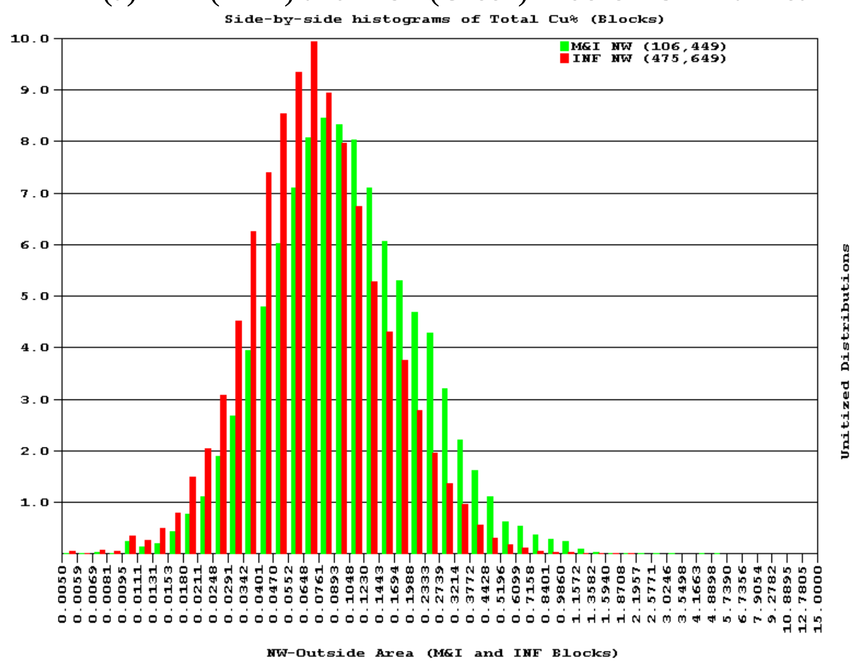
POLYGON: NW_Area
CURRENT LABEL : (G101) Kriged Grade kCu%
MINIMUM CUT-OFF ENTERED = 0.005000 MAXIMUM CUT-OFF ENTERED = 15.000000

ROCK TYPE	BLOCK COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
0	6831514	0	0	0	0.	0.	0.	0.	0.	0.0000	0.0000	0.0000	0.0000	0.	0.0000
5	35161	0	0	14681	0.01053	0.66483	0.06292	0.00190	0.04356	0.6923	-2.9584	0.3647	0.6039	0.0623	0.6634
6	4	0	0	666	0.07295	0.43425	0.20023	0.00740	0.08604	0.4297	-1.7078	0.2080	0.4561	0.2011	0.4809
10	2628567	11	0	523117	0.00500	0.71106	0.06368	0.00170	0.04122	0.6472	-2.9417	0.3955	0.6289	0.0643	0.6965
11	484	0	0	30658	0.02949	1.3921	0.21535	0.00686	0.08282	0.3846	-1.5896	0.1011	0.3179	0.2146	0.3261
20	665929	0	0	368690	0.00500	2.0317	0.07904	0.00454	0.06734	0.8520	-2.8343	0.6072	0.7792	0.0796	0.9139
21	0	0	0	64000	0.04866	5.3773	0.28072	0.02822	0.16799	0.5984	-1.3896	0.2124	0.4609	0.2771	0.4865
30	12398884	59318	0	1200310	0.00500	1.9775	0.06078	0.00879	0.09375	1.5423	-3.2834	0.8422	0.9177	0.0571	1.1495
31	137	0	0	26936	0.01489	1.3064	0.24628	0.02642	0.16254	0.6600	-1.6039	0.4349	0.6595	0.2500	0.7381
105	130	0	0	859	0.01000	0.53346	0.07124	0.00236	0.04863	0.6826	-2.8315	0.3985	0.6312	0.0719	0.6997
106	0	0	0	66	0.03000	0.38161	0.11070	0.00616	0.07847	0.7089	-2.4019	0.3827	0.6187	0.1096	0.6829
110	8884	0	0	48868	0.00821	0.79600	0.07507	0.00272	0.05214	0.6946	-2.7898	0.4131	0.6427	0.0755	0.7152
111	4	0	0	3674	0.01100	0.88415	0.15869	0.01053	0.10262	0.6467	-2.0443	0.4488	0.6699	0.1621	0.7526
120	375	0	0	33548	0.00500	1.0800	0.09806	0.00699	0.08358	0.8523	-2.6881	0.8764	0.9362	0.1054	1.1841
121	0	0	0	9826	0.00500	1.9200	0.17645	0.02401	0.15495	0.8781	-2.0495	0.7078	0.8413	0.1835	1.0146
130	4673	3667	0	69263	0.00500	1.4077	0.06409	0.00548	0.07403	1.1551	-3.2069	0.8867	0.9416	0.0631	1.1946
131	0	0	0	1689	0.00800	0.75274	0.13646	0.01234	0.11107	0.8140	-2.3128	0.6579	0.8111	0.1375	0.9648
ALL	22574746	62996	0	2396851	0.00500	5.3773	0.07580	0.00897	0.09472	1.2495	-3.0168	0.8414	0.9173	0.0746	1.1487





(a) INF (RED) and M&I (Green) Blocks – SE-Pit Area



(b) INF (RED) and M&I (Green) Blocks – NW-Out Area

Figure 14-6: Side-by-Side Histograms M&I vs INF for (a) SE and (b) NW-Out

14.8 KRIGING ERROR AND RESOURCE CLASSIFICATION

Tt used a two-part approach to classify the total copper resources. This approach takes into account the spatial distribution of the drilling, the distance to the nearest data points used to estimate a block, and finally the relative kriging error generated by the estimate. Tt has found this approach to be very robust and provide highly reproducible results. The following points detail this approach.

- A measured block requires 16 samples, with a maximum of five samples per sector in a 6 sector search pattern and a maximum of 2 composites coming from a single drill hole. This implies that in most cases, for a block to be classified as measured there must be a least 8 drill holes in four cardinal directions
- The constraints for an indicated block are not as stringent as those for a measured block. An indicated block requires a minimum of 6 samples, with a maximum of 4 samples per sector in a sector search pattern and a maximum number of 2 samples coming from a single drill hole. This implies that for most cases an indicated block must have at least 3 drill holes in three of the four cardinal directions.
- Relaxing the constraints even more, an inferred block requires a minimum of 2 samples, with a minimum of 2 samples per sector and a maximum of 2 composites from a single drill hole. This implies that an inferred block must have a least one drill hole from one of the four cardinal directions.

In addition to the search parameters, kriging error comes into play when determining if a block falls into a particular class. Tt has found that by plotting the kriging error as a log-probability plot, there is a natural break in the distribution which signifies when the error is too great to allow a block to be classified as measured or indicated (Figure 14-7). In the case of the MacArthur deposit, any block with a kriging error of 0.75 or larger was classified as inferred.

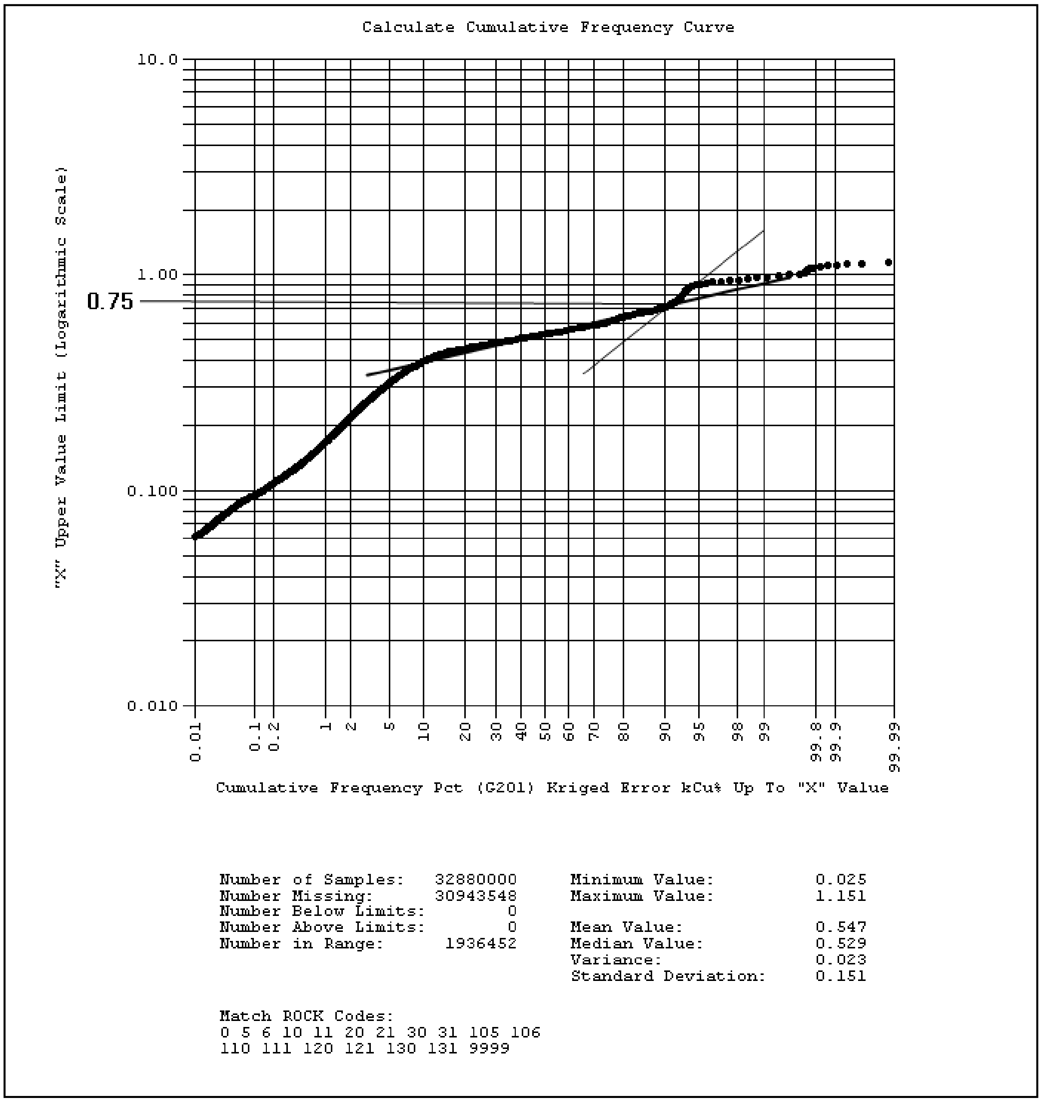
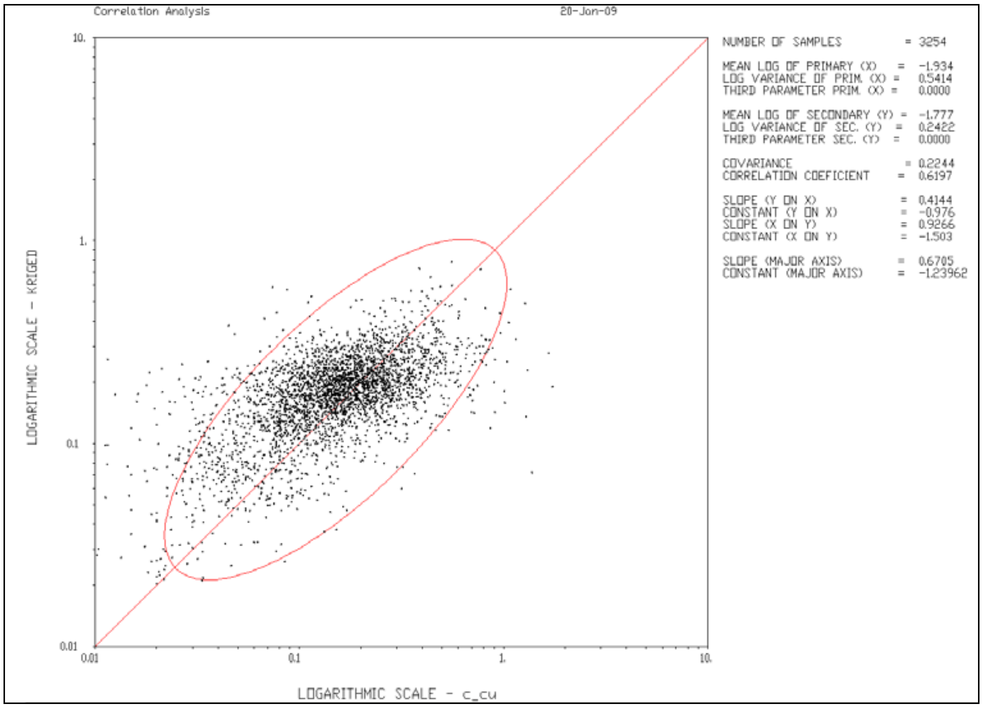
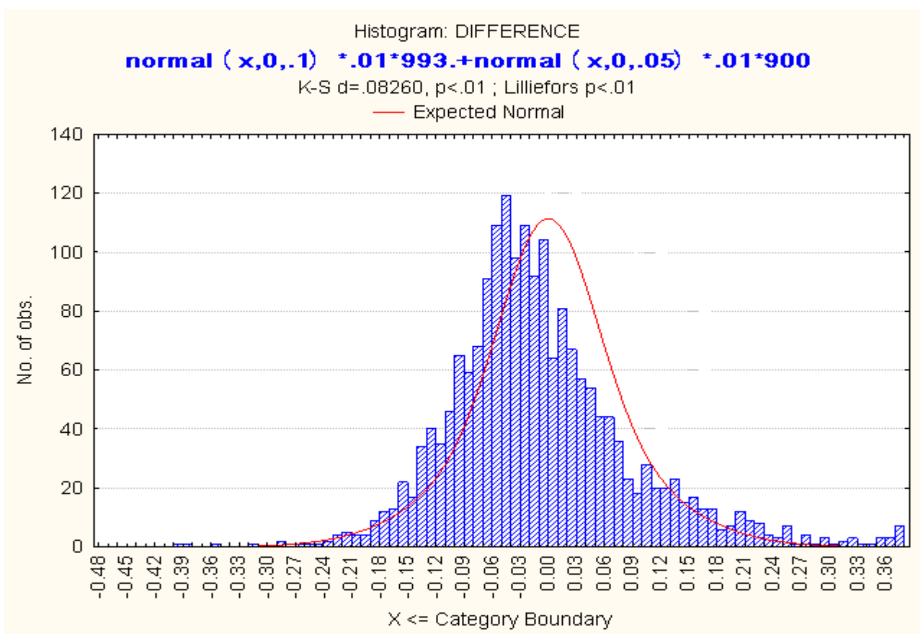


Figure 14-7: Probability Plot of Kriging Error



(a) Scatter plot of the Jackknife estimate and target grades



(b) Histogram of the Jackknife estimate and target difference.

Figure 14-8: Jackknife Method of Model Validation

The use of a model validation technique called "jackknifing" has been used to help validate the chosen search parameters. The technique removes, in sequence, a target value and uses the chosen estimation method to predict its value. The target and estimate are then compared. Figure 14-8(a) shows a scatter plot of the plotted target and estimated grades. A perfect estimation would produce a 45-degree slope of points and a correlation of 1.0. This plot has a correlation of 0.62 with approximately 80% of the points falling within the plotted ellipse. Figure 14-8(b) show the histogram of the difference between the target and estimate grades. An unbiased estimation would show a difference of zero. A precise estimate should have a small spread in the differences. Figure 14-9 show a jackknife scatterplot with the results of the three passes for MinZones 10 and 11. Correlation for pass 1 is 0.7. Pass 2 has a correlation of 0.5 and Pass 3 a correlation of 0.3. The nested ellipses capture approximately 80% of the plotted points for each test.

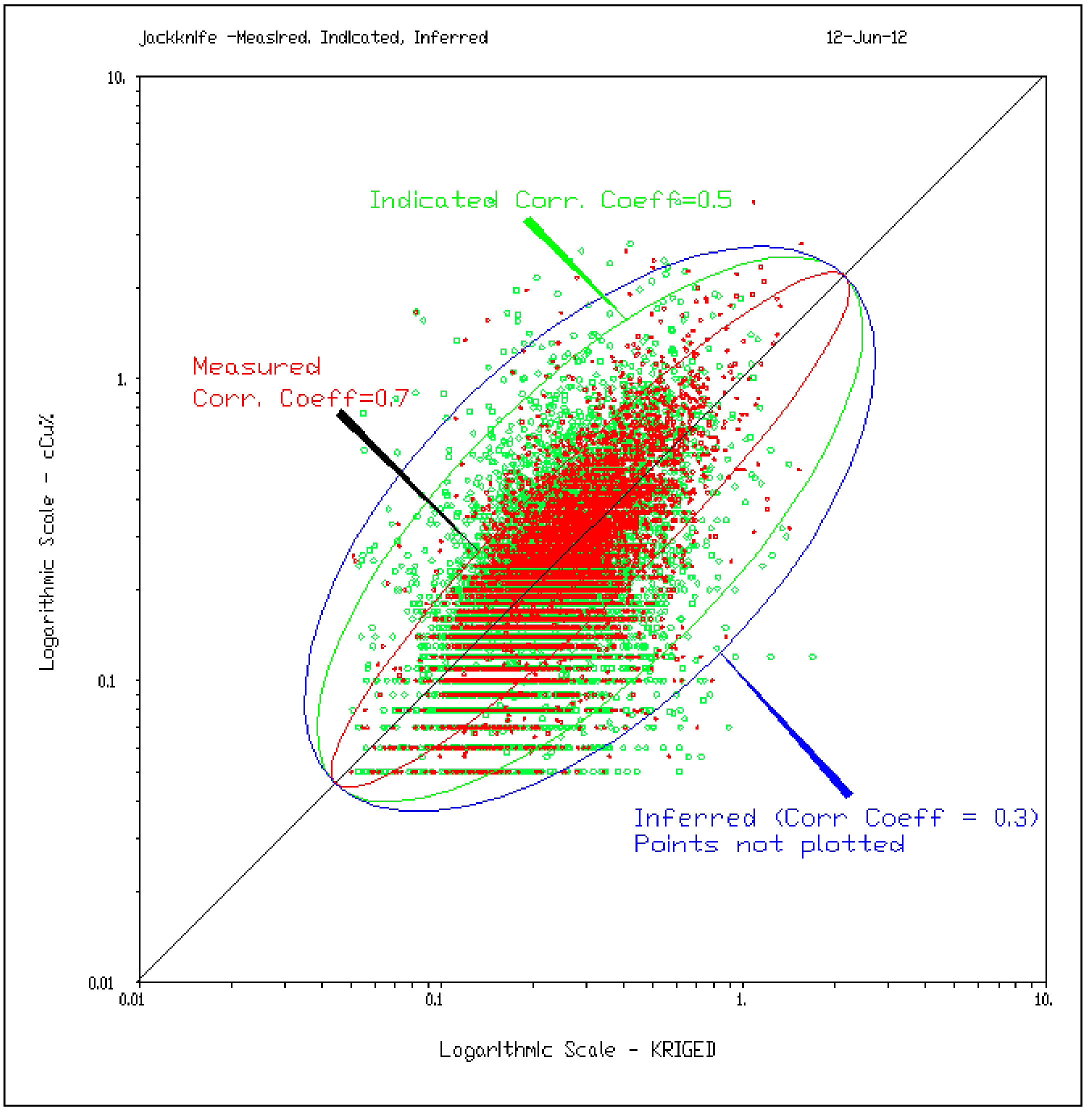


Figure 14-9: Jackknife validation of kriging model (SE Area, MinZones 10 and 11)

14.9 VALIDATION OF BLOCK MODEL: VISUAL AND STATISTICAL CHECKS

The resultant block model was validated both with visual checking of the block model grades against drill hole assays and composites using DataMine®. Figure 14-10 shows a graphic of one of the statistical comparisons. Side-by-side histograms of samples assays, composite grades and block grades shows that the three histograms are well centered. The histogram of the assays shows the grade spikes at low grades, which may be a function of laboratory detection limits.

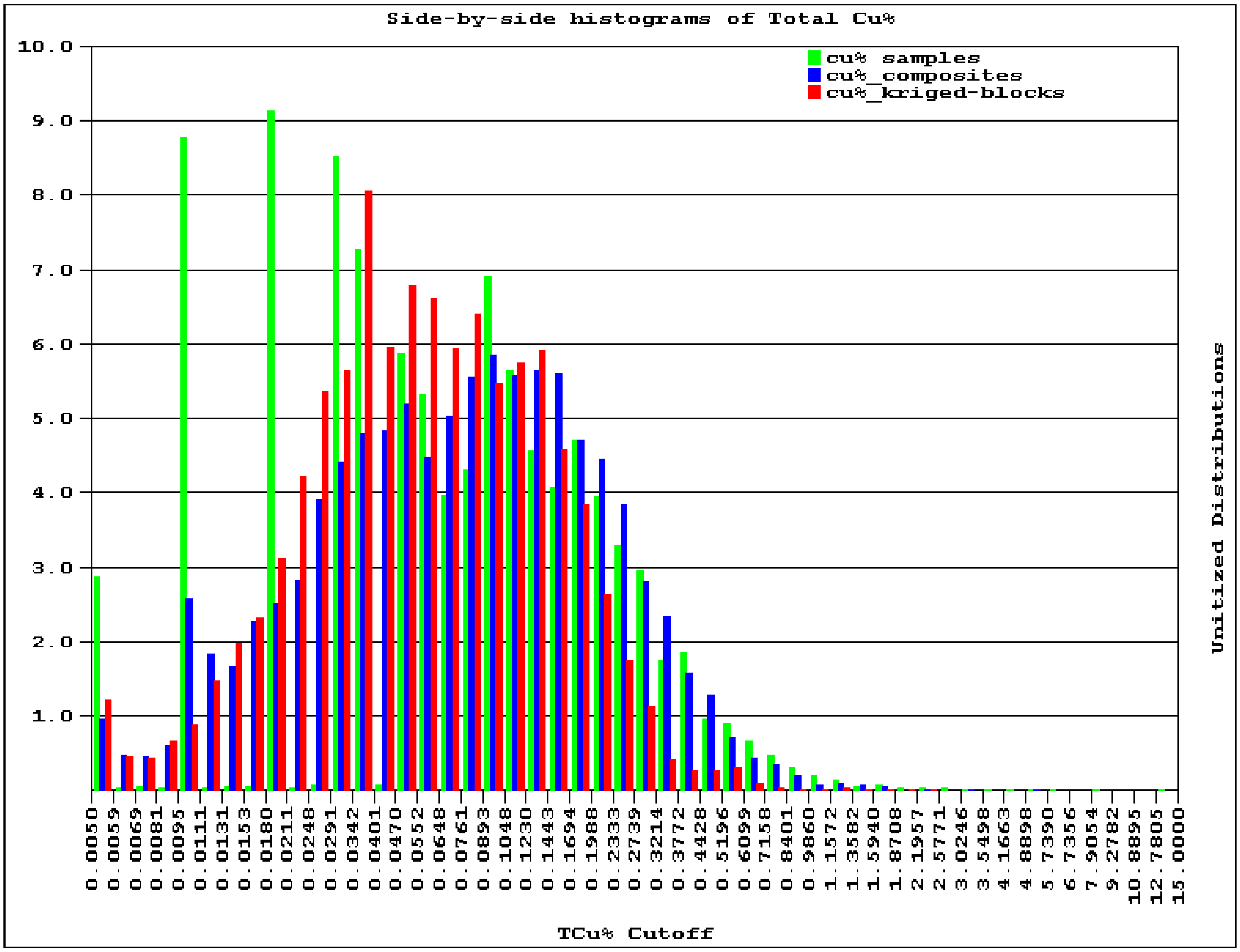


Figure 14-10: Side-by-Side Samples, Composites and Blocks

Figure 14-11 and Figure 14-12 show an east-west cross section used for visual check of the copper grades and resource classes and Figure 14-13 and Figure 14-14 similarly show a north-south section used for visual inspection. Surfaces shown below the current topography are the bottom of oxide /top of chalcocite, and the bottom of chalcocite-mix / top of primary sulfides.

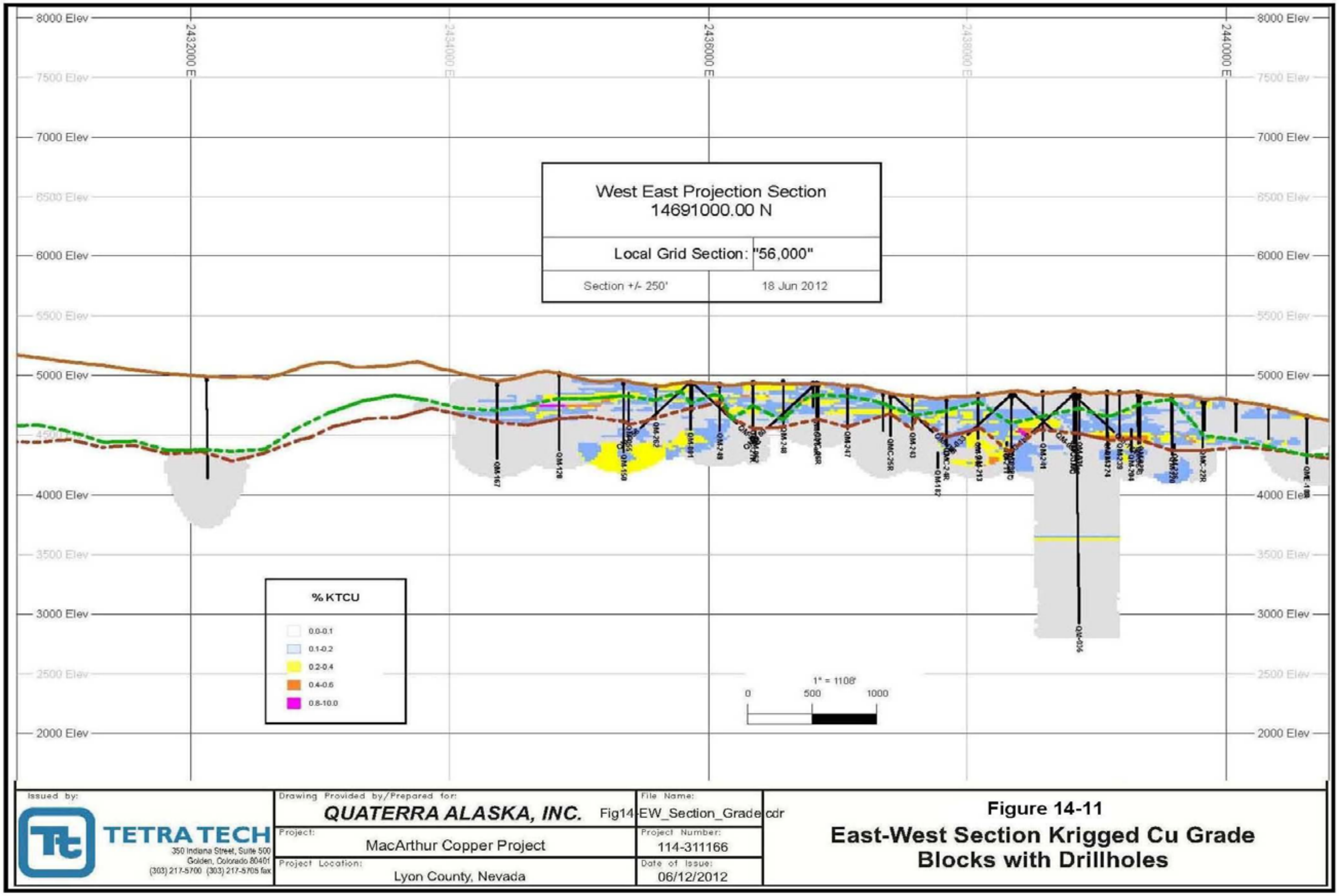


Figure 14-11: East West Cross Section Looking North (Cu blocks)

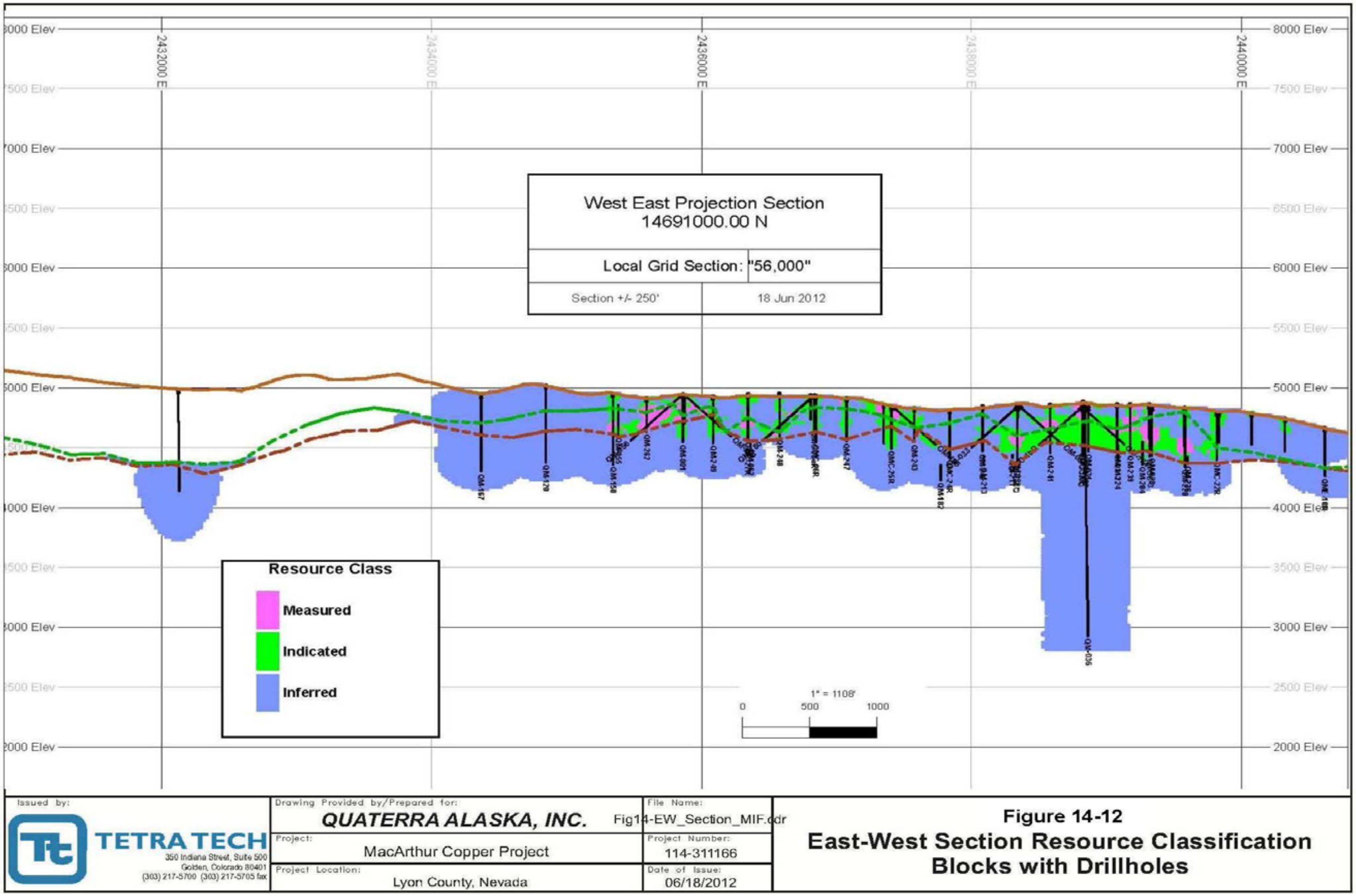


Figure 14-12: East-West Cross Section Looking North (Resource Class)

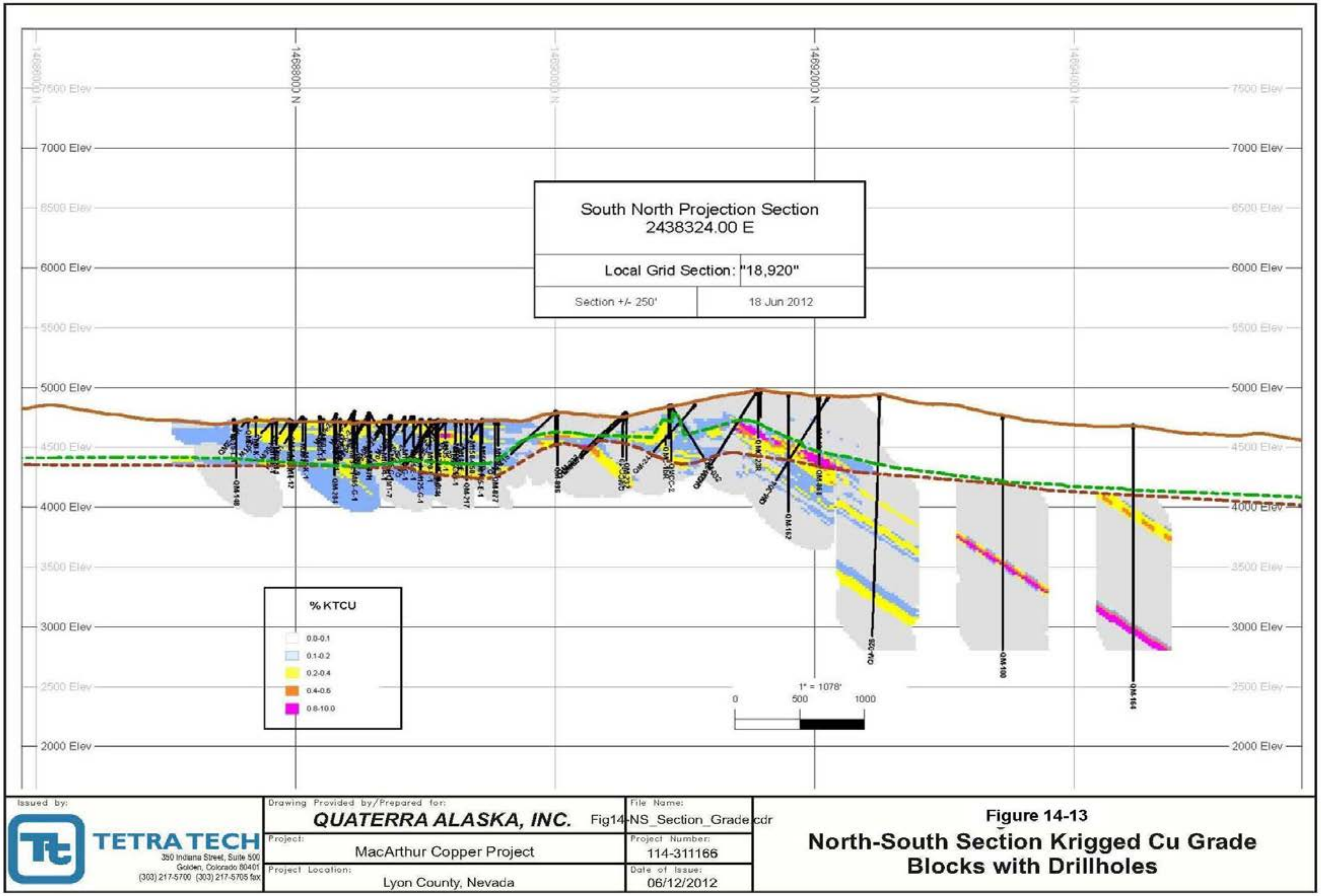


Figure 14-13: North-South Cross Section Looking West (Cu Blocks)

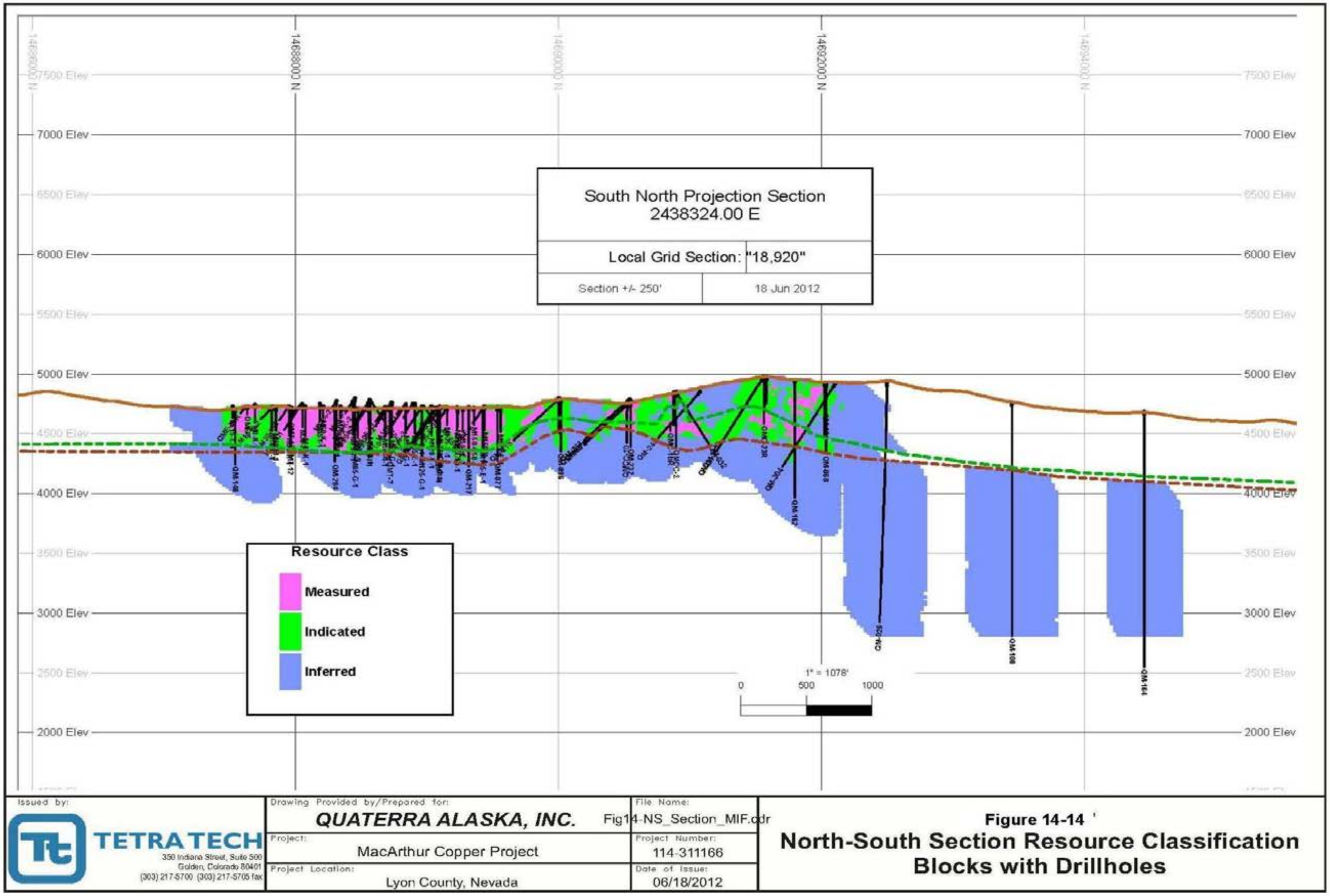


Figure 14-14: North South Cross Section Looking West (Resource Class)

14.10 MINERAL RESOURCE ESTIMATE

A summary of the measured copper resource is shown in Table 14-16. A summary of the indicated copper resources is shown in Table 14-17. The combined Measured and Indicated Copper Resources are shown in Table 14-18, and a summary of the inferred copper resources by deposit area is shown in Table 14-19. The base case cutoff grade for the leachable resources is 0.12% Cu. The base case cutoff grade for the primary sulfide resources is 0.15% Cu. Both of these values are representative of actual operating cutoff grades in use as of the date of this report. It is Tt's opinion that the MacArthur Mineral Resources meet the current CIM definitions for classified resources. The CIM definitions are as follows:

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

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

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

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

An "Inferred Mineral Resource" is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling

and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

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

Source: CIM DEFINITION STANDARDS** - For Mineral Resources and Mineral Reserves **CIM Standing Committee on Reserve Definitions, November 27, 2010

Table 14-16: Measured Copper Resources

MEASURED RESOURCES MACARTHUR COPPER PROJECT –YERINGTON, NEVADA May 2012				
	Cutoff Grade %TCu	**Tons (x1000)**	**Average Grade %TCu**	**Contained Copper (lbs x 1000)**
Oxide and Chalcocite Material (MinZone 10 and 20)	0.5	1,444	0.675	19,491
	0.4	3,196	0.548	35,041
	0.35	5,074	0.483	49,025
	0.3	8,633	0.417	71,930
	0.25	15,929	0.35	111,599
	0.2	33,472	0.283	189,518
	0.18	43,753	0.261	228,566
	0.15	58,388	0.237	276,993
	0.12	**71,829**	**0.218**	**313,174**
Primary Material (MinZone 30)	0.5			
	0.4			
	0.35			
	0.3			
	0.25			
	0.2			
	0.18			
	0.15	N/A	N/A	N/A

Table 14-17: Indicated Copper Resources

INDICATED COPPER RESOURCES MACARTHUR COPPER PROJECT –YERINGTON, NEVADA May 2012				
	Cutoff Grade %TCu	**Tons (x1000)**	**Average Grade %TCu**	**Contained Copper (lbs x 1000)**
Oxide and Chalcocite Material (MinZone 10 and 20)	0.5	1,957	0.753	29,484
	0.4	3,533	0.615	43,442
	0.35	5,018	0.543	54,516
	0.3	7,618	0.468	71,259
	0.25	13,930	0.379	105,478
	0.2	31,949	0.29	185,049
	0.18	45,554	0.26	236,607
	0.15	67,271	0.229	308,639
	0.12	**87,264**	**0.208**	**362,320**
Primary Material (MinZone 30)	0.5	98	0.72	1,411
	0.4	193	0.586	2,263
	0.35	273	0.523	2,857
	0.3	354	0.478	3,382
	0.25	507	0.416	4,216
	0.2	670	0.369	4,938
	0.18	796	0.34	5,414
	0.15	**1,098**	**0.292**	**6,408**

Table 14-18: Measured + Indicated Copper Resources

MEASURED + INDICATED RESOURCES MACARTHUR COPPER PROJECT –YERINGTON, NEVADA May 2012				
	Cutoff Grade %TCu	**Tons (x1000)**	**Average Grade %TCu**	**Contained Copper (lbs x 1000)**
Oxide and Chalcocite Material (MinZone 10 and 20)	0.5	3,401	0.72	48,974
	0.4	6,730	0.583	78,485
	0.35	10,092	0.513	103,544
	0.3	16,251	0.441	143,171
	0.25	29,859	0.364	217,075
	0.2	65,421	0.286	374,601
	0.18	89,306	0.26	465,106
	0.15	125,659	0.233	585,822
	0.12	**159,094**	**0.212**	**675,513**
Primary Material (MinZone 30)	0.5	98	0.72	1,411
	0.4	193	0.586	2,263
	0.35	273	0.523	2,857
	0.3	354	0.478	3,382
	0.25	507	0.416	4,216
	0.2	670	0.369	4,938
	0.18	796	0.34	5,414
	0.15	**1,098**	**0.292**	**6,408**

Table 14-19: Inferred Copper Resources

INFERRED COPPER RESOURCES MACARTHUR COPPER PROJECT –YERINGTON, NEVADA May 2012				
	Cutoff Grade %TCu	**Tons (x1000)**	**Average Grade %TCu**	**Contained Copper (lbs x 1000)**
Oxide and Chalcocite Material (MinZone 10 and 20)	0.5	4,294	0.657	56,423
	0.4	9,656	0.538	103,899
	0.35	15,357	0.477	146,444
	0.3	25,851	0.414	213,788
	0.25	43,695	0.356	311,108
	0.2	82,610	0.293	483,929
	0.18	109,920	0.267	587,412
	0.15	166,930	0.232	774,889
	0.12	**243,417**	**0.201**	**979,510**
Primary Material (MinZone 30)	0.5	10,644	0.819	174,413
	0.4	18,442	0.653	240,742
	0.35	23,316	0.594	277,181
	0.3	33,831	0.511	345,415
	0.25	53,060	0.423	449,312
	0.2	89,350	0.341	609,188
	0.18	101,375	0.323	654,680
	0.15	**134,900**	**0.283**	**764,074**

15 MINERAL RESERVE ESTIMATES

At this time, the MacArthur Copper Property does not have any CIM definable mineral reserves.

16 MINING METHODS

Mining of the MacArthur deposit will be done by open pit methods utilizing a traditional drill, blast, load and haul sequence. Leachable material will be delivered to the run of mine heap leach facility and waste rock will be deposited in the waste dumps located to the north and south of the proposed pits along with pit backfilling of the pits later in the mine life. The pit design is based on a 20 foot bench height to match the resource model bench height. The mine plan calls for the delivery of 15 million short tons (tons) per year (approximately 41,000 tons per day, tpd) of material to the heap leach. During peak production, about 96,000 tpd of total material (heap material plus waste) will be mined. The mine equipment fleet requirements are estimated so as to mine and deliver the heap and waste tonnages to the appropriate locations. The major mining equipment fleet will include 9 inch blast hole drills, 26 cubic yard (cu yd) hydraulic shovels, a 17 cu yd loader, and multiple 150 ton trucks. From the estimate of the mine fleet requirements, an estimate of capital and operating costs was developed.

The open pit resource tonnages included in this section are a sub-set of the mineral resource presented in Section 14. The open pit resource is contained within three pits. The mine schedule includes a brief pre-production period followed by 18 years of material delivery to the heap leach pad totaling 271 million tons averaging 0.21% total copper. The life of mine average waste to heap material ratio is 0.90. The heap leach material is the sum of oxide overburden, oxide rock and transition rock. Metallurgical test work estimates recovery of 70% for oxide material in the main pit, 65% for oxide in the north area and Gallagher and 60% recovery for all transition material from their respective total copper grades. No sulfide material is included in the open pit resource used for the mine production schedule.

16.1 GEOTECHNICAL PARAMETERS

No geotechnical investigations for pit slope angles have been completed for this PEA. An overall slope angle of 42 degrees was used for the pit definition floating cone runs and an inter-ramp slope angle of 45 degrees was used for the final pit and phase designs for the east, south and west pit walls. An inter-ramp slope angle of 46 degrees was used on the north wall because it is cutting the north dipping bedding.

16.2 DILUTION MODELING AND FACTORS

The resource model is described in Section 14. At this time, no additional dilution factors or mining losses have been applied to the grade model.

16.3 OPEN PIT MINING

The PEA open pit design is based on a floating cone geometry using the available process recoveries, cost data and copper prices which range from $2.00 to $2.85/lb copper. Table 16-1 summarizes inputs to the floating cone algorithm used for pit definition and internal phases. The process costs and recoveries are provided by M3 Engineering & Technology (M3). IMC provided the mining costs based on recent, similar size projects. Process cost for the heap leaching of copper is estimated on per pound of refined copper and the G&A costs are per ton of heap material. Mining costs are estimated using a base cost of $1.25 per ton of material moved with an addition haul cost from benches below the 4670 elevation of $0.02/t per 20 foot bench. The floating cones were run with a discount rate of 0.5% per bench of depth.

Final pits for the PEA are designed from the floating cone geometry with the inclusion of haul ramps and the smoothing of the pit walls. The inter-ramp slope angle is 45 degrees on all but the north walls where it is 46 degrees. Ramps have a maximum grade of 10% and are 112 feet wide (including an allowance for berms and ditches). There are three final pits (Main, North and Gallagher).The Main pit is sub-divided into three mining phases. The North pit is divided into two phases with Gallagher a single phase. The mining phases and the copper price cone geometry used for each of the mining phases are in Table 16-2. The pits and phases are tabulated on Table 16-3 and illustrated as Figure 16-1 (final pits) and Figure 16-2 through Figure 16-7 show the mining phase development. The pit exits are on the east side of the pits because the heap leach area is located to the east of the pits. On the figures, the bold lines show the mining faces of the current phase and the gray lines represent mining advances from previous mining phases.

The tabulations on Table 16-3 show the heap leach tonnage split into material types and class (measured (26%), indicated (29%) and inferred (45%) categories, all of which will be used for developing the mine production schedule). The mine production schedule is preliminary in nature, includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The term 'heap material' or 'heap tonnage' are used to describe the tonnage being delivered to the heap for the extraction of copper, but does not imply that there is a mineral reserve at MacArthur. No pre-feasibility or feasibility study has been completed which is required to declare a mineral reserve. The tonnages of the heap leach material are tabulated using the 0.12% total copper cutoff grade. The mine schedule presented later in Table 16-4 uses an elevated cutoff grade in some years to improve the head grade, thus all of the tonnage shown on Table 16-3 will not be delivered to the heap for leaching.

Table 16-1: Pit Definition Inputs

Copper Recovery:	
Oxide in Main MacArthur Pit	70% of TCu
Oxide in North Area and Gallagher	65% of TCu
Transition material type	60% of TCu
Costs:	
Process (Heap and SXEW)	$1.02/lb recovered copper
G&A Cost	$0.50/ton of heap material
Heap Material Haul and Heap dozing	$0.25/ton of heap material
Mining (heap and waste), base cost	$1.50/ton
Mining Lift Charge	$0.02/ton per 20 ft bench below 4760
Floating Cone Discount Rate	0.5% per 20 ft bench

Table 16-2: Floating Cone Geometries Used for Pit Designs

Mining Phase	Location	Active Mining Years	Floating Cone Used to Design Phase
1	Main Pit	PP – 5	$2.00/lb Cu cone
2	Main Pit	3 – 10	Increment between $2.00/lb & $2.25/lb cone
3	North Area	3 – 12	$2.50/lb cone
4	North Area	8 – 15	Increment between $2.50/lb & $2.75/lb cone
5	Gallagher	12 – 17	$2.60/lb cone
6	Main Pit	12 - 19	Increment between $2.25/lb & $2.85/lb cone

Table 16-3: Phase Tonnage and Grade Available for Mine Production Schedule (0.12% Tcu Cutoff)

Phase	Class	Overburden		Oxide		Transition		Total Heap Material		Waste	Total	Waste/Heap
		ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	ktons	Ratio
	Measured	426	0.19	39,656	0.22	755	0.23	40,837	0.22			
	Indicated	283	0.18	15,206	0.20	2,154	0.24	17,643	0.20			
1	Meas&Indic.	709	0.19	54,862	0.21	2,909	0.24	58,480	0.22			
	Inferred	26	0.19	186	0.18	2,020	0.22	2,232	0.22			
	Total	735	0.19	55,048	0.21	4,929	0.23	60,712	0.22	9,905	70,617	0.16
	Measured	94	0.17	17,293	0.20	14	0.17	17,401	0.20			
	Indicated	362	0.16	22,652	0.18	120	0.18	23,134	0.18			
2	Meas&Indic.	456	0.16	39,945	0.19	134	0.18	40,535	0.19			
	Inferred	333	0.17	6,387	0.19	2,537	0.19	9,257	0.19			
	Total	789	0.17	46,332	0.19	2,671	0.19	49,792	0.19	9,574	59,366	0.19
	Measured	0		409	0.20	647	0.22	1,056	0.21			
	Indicated	56	0.20	3,967	0.19	3,550	0.23	7,573	0.21			
3	Meas&Indic.	56	0.20	4,376	0.19	4,197	0.23	8,629	0.21			
	Inferred	604	0.19	21,229	0.19	16,678	0.24	38,511	0.21			
	Total	660	0.20	25,605	0.19	20,875	0.24	47,140	0.21	55,932	103,072	1.19
	Measured	0		173	0.21	4,520	0.31	4,693	0.31			
	Indicated	4	0.14	1,179	0.18	13,371	0.28	14,554	0.27			
4	Meas&Indic.	4	0.14	1,352	0.18	17,891	0.29	19,247	0.28			
	Inferred	112	0.15	4,414	0.17	9,958	0.25	14,484	0.22			
	Total	116	0.15	5,766	0.17	27,849	0.27	33,731	0.26	62,063	95,794	1.84
	Measured	2	0.14	565	0.23	0		567	0.23			
	Indicated	22	0.18	3,250	0.21	348	0.26	3,620	0.21			
5	Meas&Indic.	24	0.18	3,815	0.21	348	0.26	4,187	0.22			
	Inferred	340	0.17	18,851	0.19	12,444	0.21	31,635	0.20			
	Total	364	0.17	22,666	0.20	12,792	0.22	35,822	0.20	25,104	60,926	0.70
	Measured	4	0.16	4,688	0.18	2,017	0.21	6,709	0.19			
	Indicated	135	0.14	11,166	0.17	4,232	0.21	15,533	0.18			
6	Meas&Indic.	139	0.14	15,854	0.17	6,249	0.21	22,242	0.18			
	Inferred	755	0.16	17,933	0.18	10,347	0.26	29,035	0.21			
	Total	894	0.16	33,787	0.18	16,596	0.24	51,277	0.20	74,778	126,055	1.46
	Measured	526	0.19	62,784	0.21	7,953	0.27	71,263	0.22			
Total Pits	Indicated	862	0.17	57,420	0.19	23,775	0.26	82,057	0.21			
	Meas&Indic.	1,388	0.17	120,204	0.20	31,728	0.26	153,320	0.21			
	Inferred	2,170	0.17	69,000	0.19	53,984	0.24	125,154	0.21			
	Total	3,558	0.17	189,204	0.19	85,712	0.24	278,474	0.21	237,356	515,830	0.85

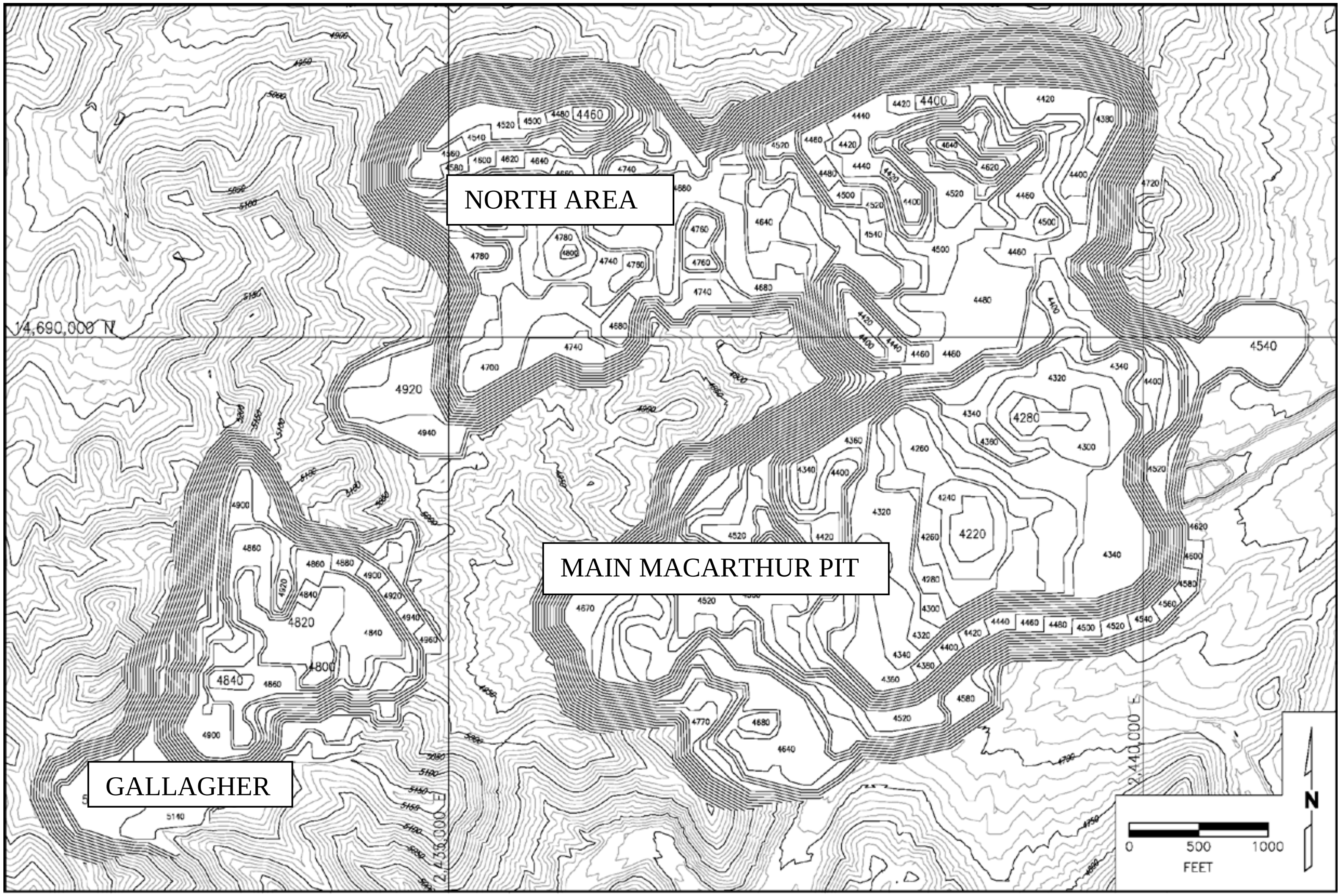


Figure 16-1: Final Pits

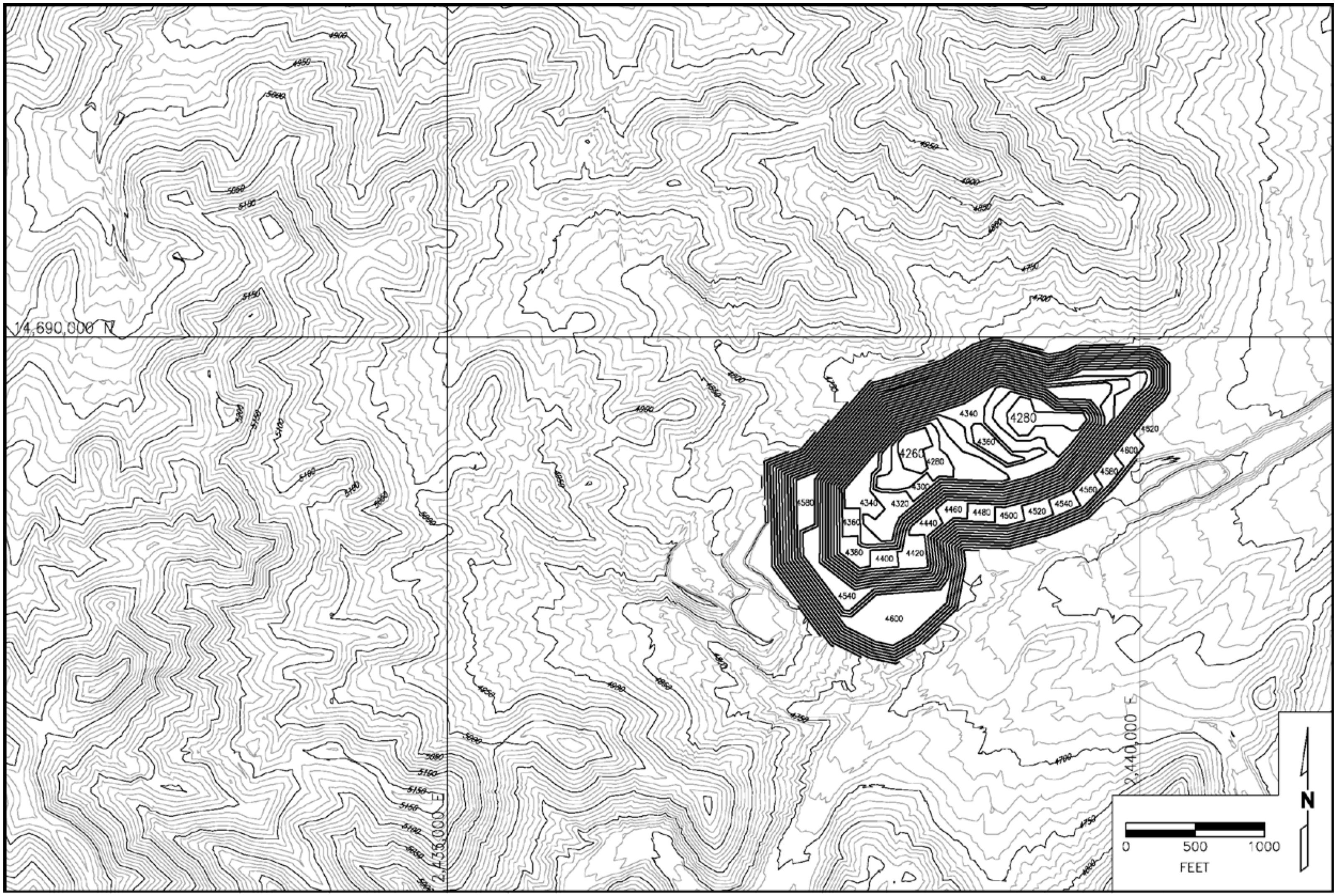


Figure 16-2: Mining Phase 1 in MacArthur Pit

Figure 16-3: Mining Phase 2 in MacArthur Pit

Figure 16-4: Mining Phase 3 in North Pit Area

Figure 16-5: Mining Phase 4 in North Pit Area

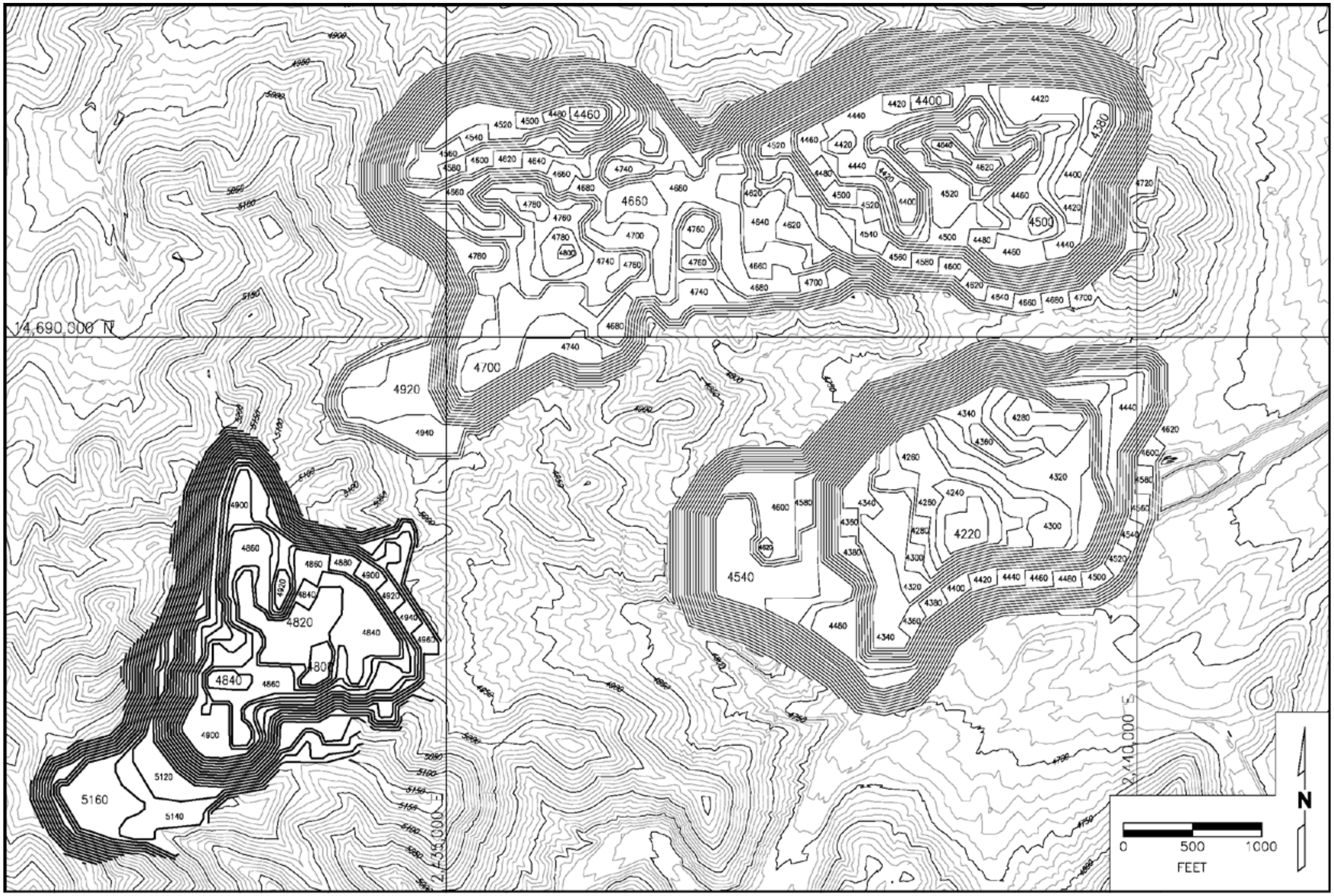


Figure 16-6: Mining Phase 5 (Gallagher Pit)

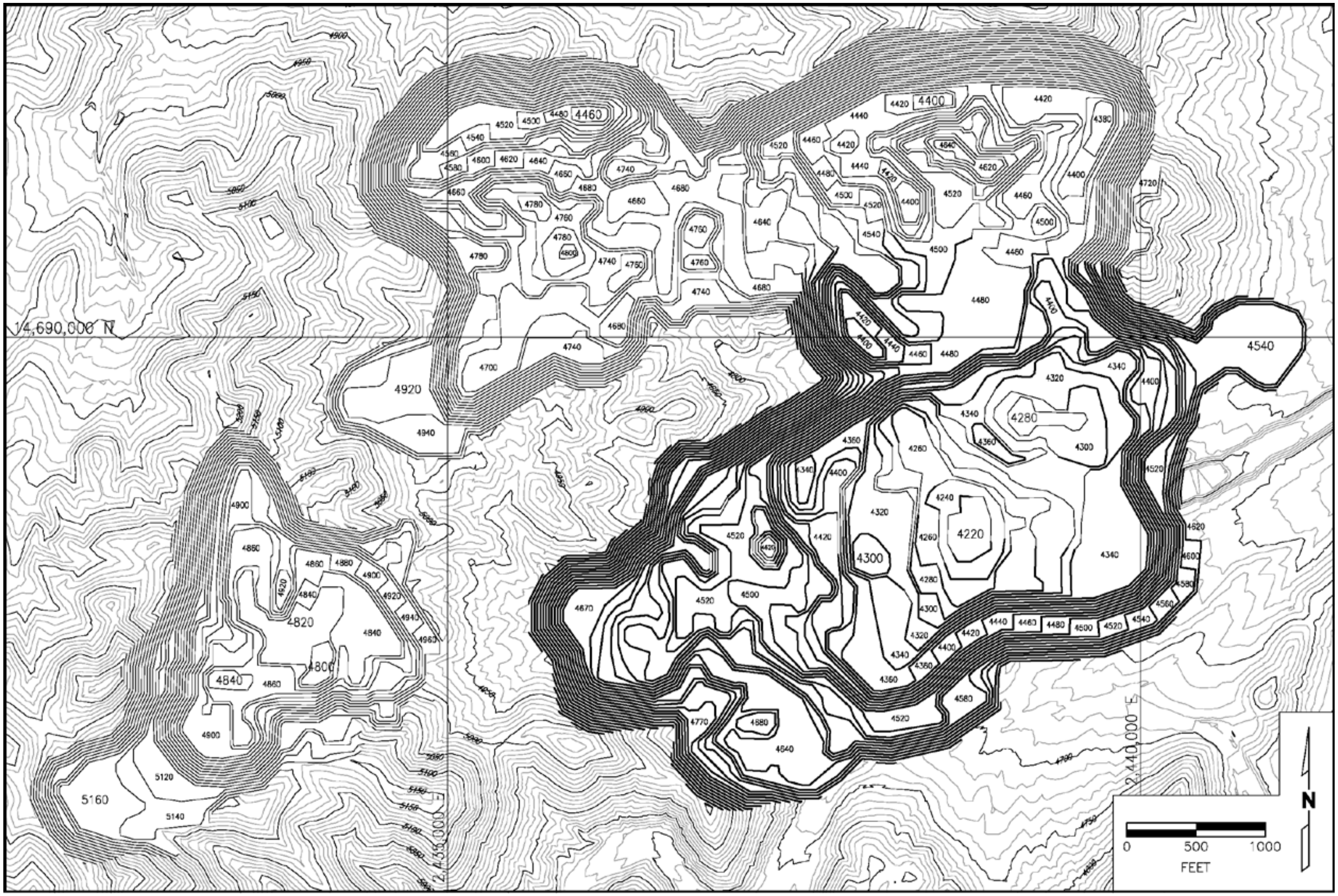


Figure 16-7: Mining Phase 6 in MacArthur Pit

16.4 MINING SCHEDULE

A mining schedule to deliver 15 million tpy to the heap was developed from the mining phases previously described. Mining starts in the Main pit (phases 1 and 2), progresses to the North pit (phases 3 and 4), then to Gallagher (phase 5) and returns to the outer limits of the Main pit during phase 6. The mine production schedule is presented in Table 16-4. The tonnage of leach feed mined by mining phase by year is summarized in Table 16-5. During years 3, 4, 6 and 7, the cutoff grade for tonnage going to the heap is raised above the 0.12% total copper cutoff in order to improve the head grade to the heap. Maps showing mine advance in the pits are included in Section 16.5.

The percent of the heap leach tonnages in the measured plus indicated and the inferred categories are shown on Table 16-6. During years 1 through 4, over 90% of the heap leach tonnage is in the measured plus indicated categories. In years 5 and 6, this percentage drops to 77% and 72%.

Table 16-4: Production Schedule

Year	Cutoff Grade %Tcu	Heap Tonnage & Grade			Waste	Total	Pounds of Copper Contained	Pounds of Copper Recoverable	Heap Tonnage by Type: Main Pit Area Oxide (ox + ovb)		Other Areas Oxide (ox + ovb)		All Areas Mixed	
		ktons	%Tcu	Avg recCu	ktons	ktons	x 1000	x 1000	kt	%Tcu	kt	%Tcu	kt	%Tcu
Pre-Prod	0.12	399	0.24	0.17	101	500	1,920	1,344	399	0.24	0		0	
1	0.12	15,000	0.21	0.15	4,042	19,042	62,851	43,996	15,000	0.21	0		0	
2	0.12	15,000	0.24	0.17	2,174	17,174	71,590	50,113	15,000	0.24	0		0	
3	0.14	15,000	0.21	0.15	5,000	20,000	62,445	43,572	14,204	0.21	790	0.17	6	0.24
4	0.14	15,000	0.21	0.15	5,000	20,000	62,785	43,745	14,228	0.21	538	0.19	234	0.22
5	0.12	15,000	0.21	0.14	5,000	20,000	62,358	41,313	9,306	0.20	1040	0.19	4,654	0.23
6	0.14	15,000	0.19	0.13	15,000	30,000	58,368	40,166	11,794	0.19	3149	0.21	57	0.19
7	0.13	15,000	0.19	0.13	20,000	35,000	56,740	38,134	8,181	0.18	5723	0.20	1,096	0.20
8	0.12	15,000	0.20	0.13	20,000	35,000	60,395	39,018	4,369	0.18	6360	0.19	4,271	0.24
9	0.12	15,000	0.20	0.13	20,000	35,000	60,655	38,923	4,405	0.19	4771	0.18	5,824	0.23
10	0.12	15,000	0.21	0.13	20,000	35,000	63,881	39,475	1,190	0.19	4337	0.16	9,473	0.24
11	0.12	15,000	0.23	0.14	20,000	35,000	69,140	41,913	0		2383	0.18	12,617	0.24
12	0.12	15,000	0.25	0.15	17,403	32,403	75,051	46,021	1,202	0.16	2841	0.21	10,957	0.27
13	0.12	15,000	0.24	0.15	18,156	33,156	71,036	45,714	3,440	0.21	7862	0.21	3,698	0.32
14	0.12	15,000	0.22	0.14	20,000	35,000	65,071	41,281	3,670	0.18	5124	0.18	6,206	0.27
15	0.12	15,000	0.21	0.13	17,261	32,261	63,113	39,970	4,400	0.16	4093	0.17	6,507	0.27
16	0.12	15,000	0.20	0.12	12,749	27,749	58,730	37,360	4,639	0.17	3108	0.18	7,253	0.22
17	0.12	15,000	0.20	0.13	15,256	30,256	59,232	38,748	8,809	0.18	418	0.14	5,773	0.23
18	0.12	15,000	0.18	0.12	7,612	22,612	54,907	35,830	8,489	0.17	0		6,511	0.2
19	0.12	482	0.18	0.11	194	676	1,735	1,052	31	0.18	0		451	0.18
Total		270,881	0.21	0.14	244,948	515,829	1,142,003	747,688	132,756	0.20	52,537	0.19	85,588	0.24
					0.90									

Note: Heap tonnage is a combination of oxide, overburden and mixed heap material types

Table 16-5: Heap Material Production Schedule by Mining Phase

Summary				**Sum of Oxide and Transition Heap Material Types**															
	Cutoff			Main Pit								North Area Pit						Gallager Pit	
	Grade	Heap Material		Phase 1		Phase 2		Phase 6		Total - Main Pit		Phase 3		Phase 4		Total - North Area		Gallager, Phase 5	
Year	%Tcu																		
		ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu
Pre-Prod	0.12	399	0.24	399	0.24					399	0.24								
1	0.12	15,000	0.21	15,000	0.21					15,000	0.21								
2	0.12	15,000	0.24	15,000	0.24					15,000	0.24								
3	0.14	15,000	0.21	12,764	0.21	1,446	0.22			14,210	0.21	790	0.17			790	0.17		
4	0.14	15,000	0.21	6,920	0.20	7,543	0.22			14,463	0.21	537	0.19			537	0.19		
5	0.12	15,000	0.21	8,727	0.22	5,233	0.19			13,960	0.21	1,040	0.18			1,040	0.18		
6	0.14	15,000	0.19			11,796	0.19			11,796	0.19	3,204	0.21			3,204	0.21		
7	0.13	15,000	0.19			8,182	0.18			8,182	0.18	6,818	0.20			6,818	0.20		
8	0.12	15,000	0.20			4,398	0.18			4,398	0.18	10,394	0.21	207	0.14	10,601	0.21		
9	0.12	15,000	0.20			4,963	0.19			4,963	0.19	9,418	0.21	618	0.16	10,036	0.21		
10	0.12	15,000	0.21			3,268	0.19			3,268	0.19	8,057	0.24	3,676	0.19	11,733	0.22		
11	0.12	15,000	0.23							0		4,006	0.23	10,994	0.23	15,000	0.23		
12	0.12	15,000	0.25					1,203	0.16	1,203	0.16	148	0.19	11,214	0.27	11,362	0.27	2,436	0.21
13	0.12	15,000	0.24					3,533	0.21	3,533	0.21			3,614	0.32	3,614	0.32	7,853	0.21
14	0.12	15,000	0.22					5,289	0.22	5,289	0.22			2,589	0.33	2,589	0.33	7,122	0.18
15	0.12	15,000	0.21					6,435	0.21	6,435	0.21			818	0.25	818	0.25	7,747	0.21
16	0.12	15,000	0.20					6,514	0.20	6,514	0.20							8,486	0.20
17	0.12	15,000	0.20					12,821	0.19	12,821	0.19							2,179	0.20
18	0.12	15,000	0.18					15,000	0.18	15,000	0.18								
19	0.12	482	0.18					482	0.18	482	0.18								
Total		270,881	0.211	58,810	0.218	46,829	0.193	51,277	0.195	156,916	0.203	44,412	0.214	33,730	0.255	78,142	0.232	35,823	0.201

Table 16-6: Heap Material Production Schedule by Mining Phase and Resource Classification

Summary				Sum of Oxide and Transition Heap Material Types (Measured + Indicated)																		
	Cutoff			Main Pit								North Area Pit						Gallager Pit		TOTAL PITS		
	Grade	Heap Material		Phase 1		Phase 2		Phase 6		Total - Main Pit		Phase 3		Phase 4		Total - North Area		Gallager, Phase 5				% of
Year	%Tcu																					Total
		ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	
Pre-Prod	0.12	399	0.24	396	0.24					396	0.24									396	0.24	99.2%
1	0.12	15,000	0.21	14,891	0.21					14,891	0.21									14,891	0.21	99.3%
2	0.12	15,000	0.24	15,000	0.23					15,000	0.23									15,000	0.23	100.0%
3	0.14	15,000	0.21	12,761	0.21	1,186	0.23			13,947	0.21	1	0.15			1	0.15			13,948	0.21	93.0%
4	0.14	15,000	0.21	6,747	0.20	7,129	0.22			13,876	0.21	8	0.29			8	0.29			13,884	0.21	92.6%
5	0.12	15,000	0.21	6,815	0.22	4,665	0.19			11,480	0.21	42	0.23			42	0.23			11,522	0.21	76.8%
6	0.14	15,000	0.19			10,029	0.19			10,029	0.19	791	0.22			791	0.22			10,820	0.19	72.1%
7	0.13	15,000	0.19			7,425	0.18			7,425	0.18	1,419	0.19			1,419	0.19			8,844	0.18	59.0%
8	0.12	15,000	0.20			4,061	0.18			4,061	0.18	1,868	0.20	22	0.14	1,890	0.20			5,951	0.19	39.7%
9	0.12	15,000	0.20			2,895	0.19			2,895	0.19	1,647	0.20	100	0.21	1,747	0.20			4,642	0.19	30.9%
10	0.12	15,000	0.21			384	0.17			384	0.17	1,677	0.22	931	0.20	2,608	0.22			2,992	0.21	19.9%
11	0.12	15,000	0.23							0		715	0.23	6,168	0.24	6,883	0.24			6,883	0.24	45.9%
12	0.12	15,000	0.25					38	0.15	38	0.15	7	0.19	7,823	0.29	7,830	0.29	273	0.21	8,141	0.28	54.3%
13	0.12	15,000	0.24					923	0.20	923	0.20			2,529	0.33	2,529	0.33	1,426	0.23	4,878	0.27	32.5%
14	0.12	15,000	0.22					1,794	0.18	1,794	0.18			1,474	0.36	1,474	0.36	927	0.18	4,195	0.24	28.0%
15	0.12	15,000	0.21					2,332	0.17	2,332	0.17			197	0.27	197	0.27	771	0.22	3,300	0.19	22.0%
16	0.12	15,000	0.20					2,713	0.17	2,713	0.17							714	0.22	3,427	0.18	22.8%
17	0.12	15,000	0.20					7,307	0.19	7,307	0.19							76	0.21	7,383	0.19	49.2%
18	0.12	15,000	0.18					7,117	0.18	7,117	0.18									7,117	0.18	47.4%
19	0.12	482	0.18					17	0.20	17	0.20									17	0.20	3.5%
Total		270,881	0.211	56,610	0.216	37,774	0.194	22,241	0.182	116,625	0.203	8,175	0.209	19,244	0.279	27,419	0.258	4,187	0.214	148,231	0.213	54.7%

Heap Material Production Schedule by Mining Phase and Resource Classification (Continued)

Summary				Sum of Oxide and Transition Heap Material Types (Inferred)																		
	Cutoff			Main Pit								North Area Pit						Gallager Pit		**TOTAL PITS**		
	Grade	Heap Material		Phase 1		Phase 2		Phase 6		Total - Main Pit		Phase 3		Phase 4		Total - North Area		Gallager, Phase 5				% of
Year	%Tcu																					Total
		ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	ktons	%Tcu	
Pre-Prod	0.12	399	0.24	3	0.14					3	0.14									3	0.14	0.8%
1	0.12	15,000	0.21	109	0.19					109	0.19									109	0.19	0.7%
2	0.12	15,000	0.24	0						0										0		0.0%
3	0.14	15,000	0.21	2	0.28	259	0.20			261	0.20	789	0.18			789	0.18			1,050	0.18	7.0%
4	0.14	15,000	0.21	173	0.22	414	0.18			587	0.19	529	0.19			529	0.19			1,116	0.19	7.4%
5	0.12	15,000	0.21	1,911	0.22	568	0.17			2,479	0.21	998	0.18			998	0.18			3,477	0.20	23.2%
6	0.14	15,000	0.19			1,767	0.20			1,767	0.20	2,413	0.22			2,413	0.22			4,180	0.21	27.9%
7	0.13	15,000	0.19			757	0.20			757	0.20	5,398	0.20			5,398	0.20			6,155	0.20	41.0%
8	0.12	15,000	0.20			337	0.18			337	0.18	8,527	0.21	185	0.14	8,712	0.21			9,049	0.21	60.3%
9	0.12	15,000	0.20			2,069	0.19			2,069	0.19	7,771	0.21	518	0.15	8,289	0.21			10,358	0.21	69.1%
10	0.12	15,000	0.21			2,884	0.18			2,884	0.18	6,380	0.24	2,744	0.18	9,124	0.22			12,008	0.21	80.1%
11	0.12	15,000	0.23							0		3,291	0.22	4,826	0.21	8,117	0.21			8,117	0.21	54.1%
12	0.12	15,000	0.25					1,164	0.17	1,164	0.17	141	0.19	3,391	0.23	3,532	0.23	2,163	0.21	6,859	0.21	45.7%
13	0.12	15,000	0.24					2,610	0.22	2,610	0.22			1,086	0.31	1,086	0.31	6,428	0.21	10,124	0.22	67.5%
14	0.12	15,000	0.22					3,495	0.24	3,495	0.24			1,115	0.28	1,115	0.28	6,194	0.19	10,804	0.21	72.0%
15	0.12	15,000	0.21					4,103	0.24	4,103	0.24			621	0.24	621	0.24	6,976	0.21	11,700	0.22	78.0%
16	0.12	15,000	0.20					3,802	0.21	3,802	0.21							7,772	0.19	11,574	0.20	77.2%
17	0.12	15,000	0.20					5,514	0.19	5,514	0.19							2,103	0.21	7,617	0.20	50.8%
18	0.12	15,000	0.18					7,883	0.19	7,883	0.19									7,883	0.19	52.6%
19	0.12	482	0.18					465	0.18	465	0.18									465	0.18	96.5%
Total		270,881	0.211	2,198	0.217	9,055	0.190	29,036	0.207	40,289	0.203	36,237	0.215	14,486	0.222	50,723	0.217	31,636	0.198	122,648	0.208	45.3%

16.5 WASTE DUMPS

Two exterior dumps and one backfill pit have been designed to hold the 245 million tons of waste rock. The exterior dumps are located to the north and south of the pits and the pit backfill is primarily in the north pit area with some extending to the west of the north pit. The pit backfill has the potential of sterilizing further transition and sulfide resources and this will be further evaluated during the pre-feasibility (PFS) stage of the project. No extensive condemnation drilling has been done in the north and south waste dump areas. Further optimization of the mine plan including waste rock management will be completed as part of the PFS.

The dumps are designed using 20 foot contours with a setback between them so that the overall slope of the dump face is 2.5:1.0 (horizontal to vertical) to allow for either concurrent reclamation or reclamation of the dump faces at the end of mining. The overall slopes in areas with haul ramps for truck access to the upper lifts will be even flatter than 2.5:1.0. The dump locations relative to the final pit are shown on Figure 16-8.

The average density of the waste tonnage is 12.5 cuft/t in place, dry. A 30% swell factor has been applied for determining the waste volume required to hold the waste tonnage. The average density in the dump volume is 16.25 loose cuft/t, dry. The tonnage placed in the waste dumps by year is shown on Table 16-7. Figure 16-9 through Figure 16-13 show the pit and dump advances through the first 10 years of mining.

Table 16-7: Waste Tonnage by Source and Destination

Year	Source Phases - Waste ktons							Waste Destinations			
	1	2	3	4	5	6	Total	South Dump	North Dump	North Pit Backfill SW area	Expand
PP	101						101	101			
1	4,042						4,042	4042			
2	2,174						2,174	2174			
3	2,866	350	1,784				5,000	3,216	1,784		
4	1,308	3,013	679				5,000	4,321	679		
5	1,316	2,236	1,448				5,000	3,552	1,448		
6		4,513	10,487				15,000	4,513	10,487		
7		993	19,007				20,000	993	19,007		
8		188	12,365	7,447			20,000	188	19,812		
9		387	7,127	12,486			20,000	387	19,613		
10		858	3,820	15,322			20,000	858		19,142	
11			1,780	17,968	251		19,999		12,402	7,597	
12			161	6,503	4,162	6,577	17,403	6,577		10,826	
13				911	6,143	10,891	17,945	5,446		7,054	5,446
14				1,031	6,777	12,402	20,210	6,000		7,808	6,402
15				394	4,002	12,865	17,261			4,396	12,865
16					2,670	10,079	12,749				12,749
17					1,099	14,158	15,257				15,257
18						7,612	7,612				7,612
19						194	194				194
Total	11,807	12,538	58,658	62,062	25,104	74,778	244,947	42,368	85,232	56,823	60,525

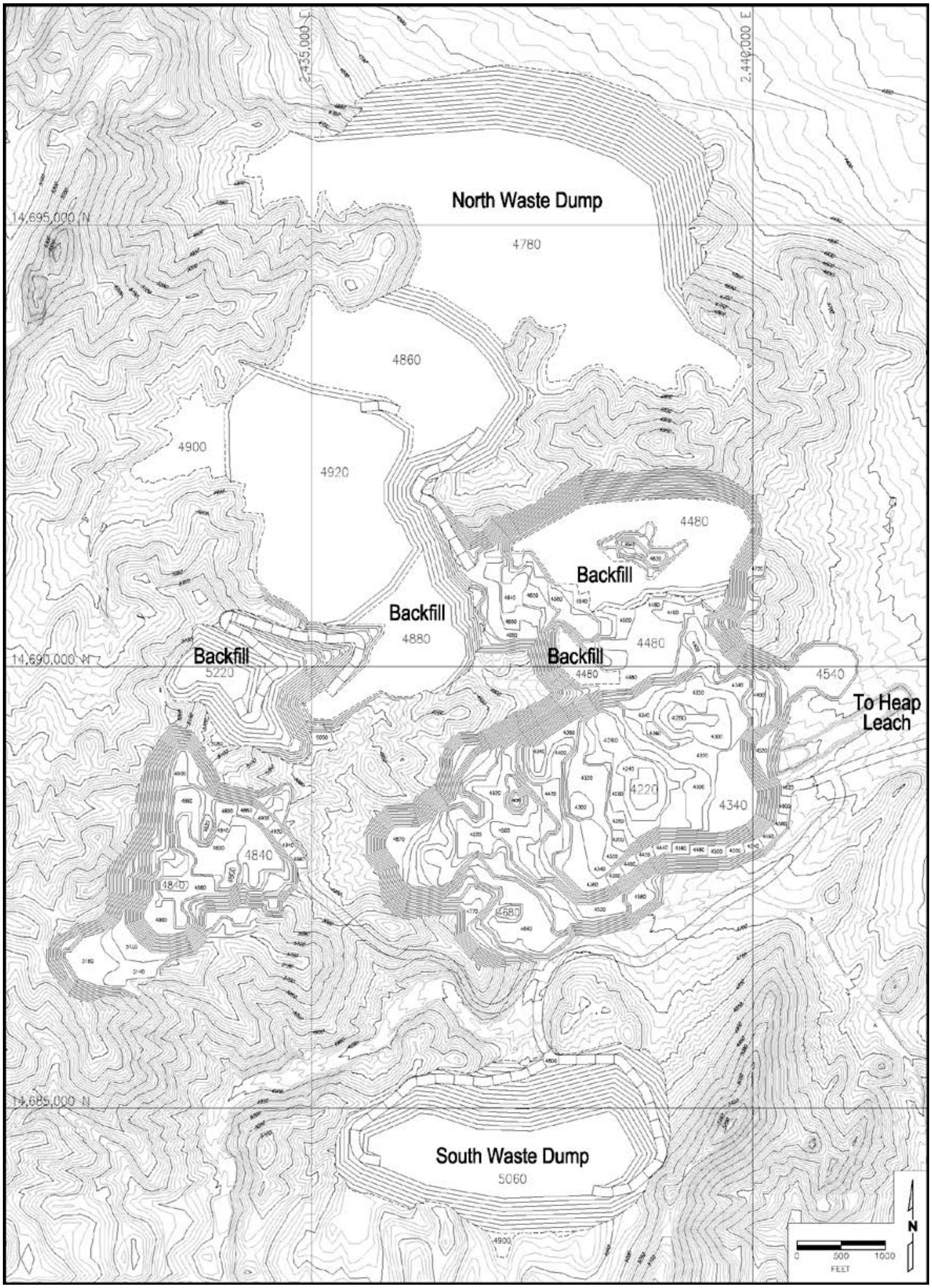


Figure 16-8: Final Pit and Dumps (including pit backfill)

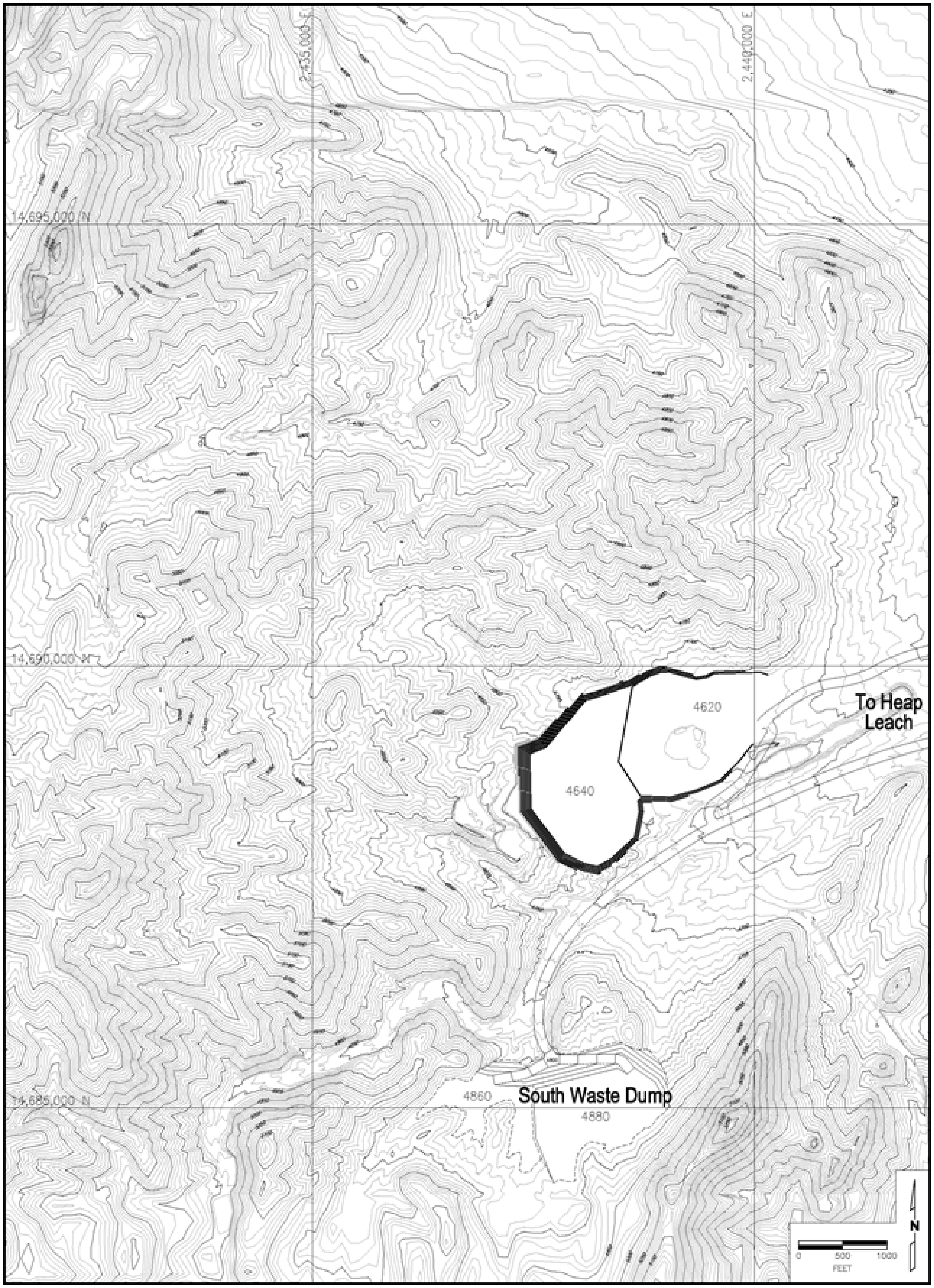


Figure 16-9: End of Year 1

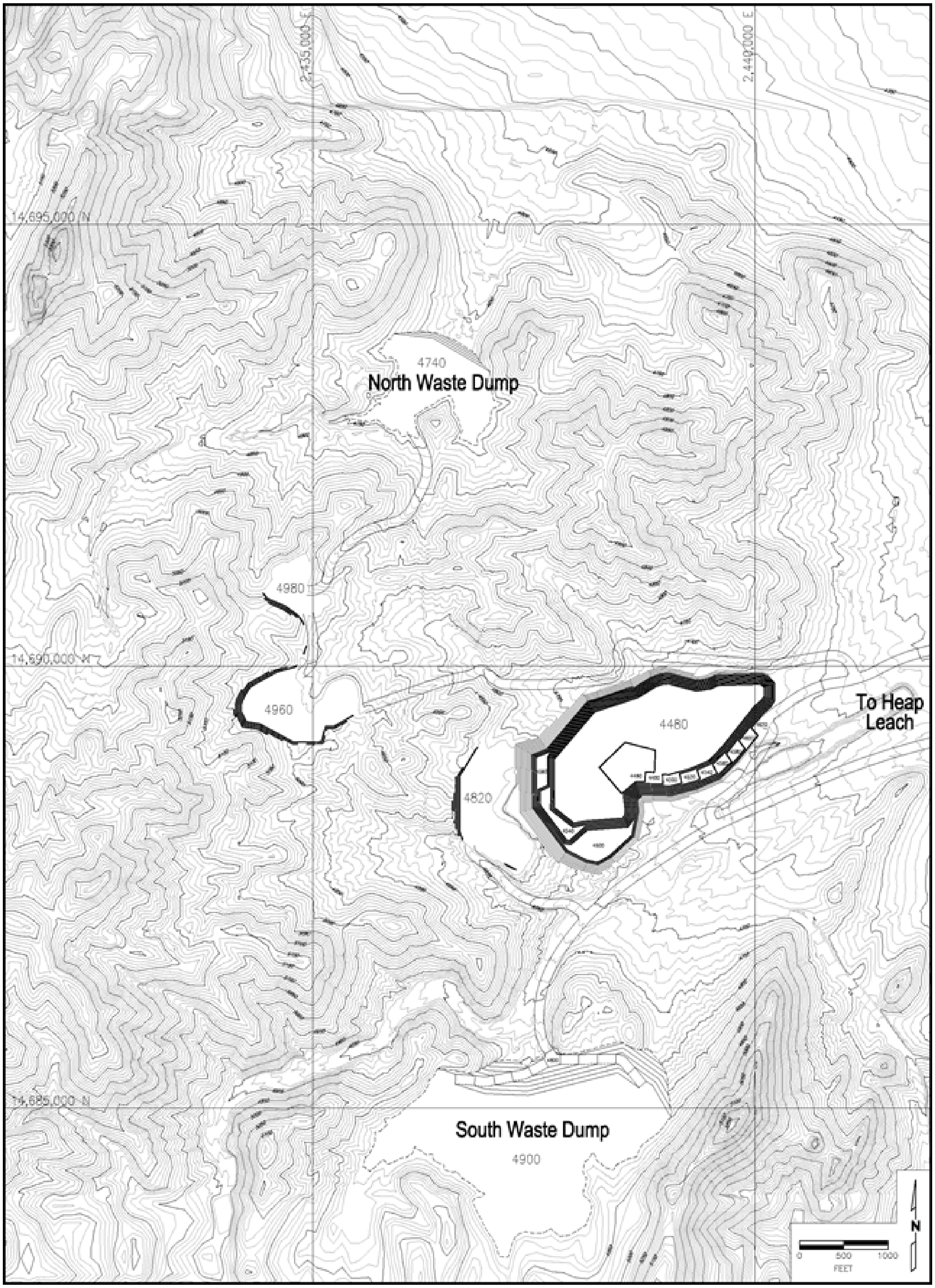


Figure 16-10: End of Year 3

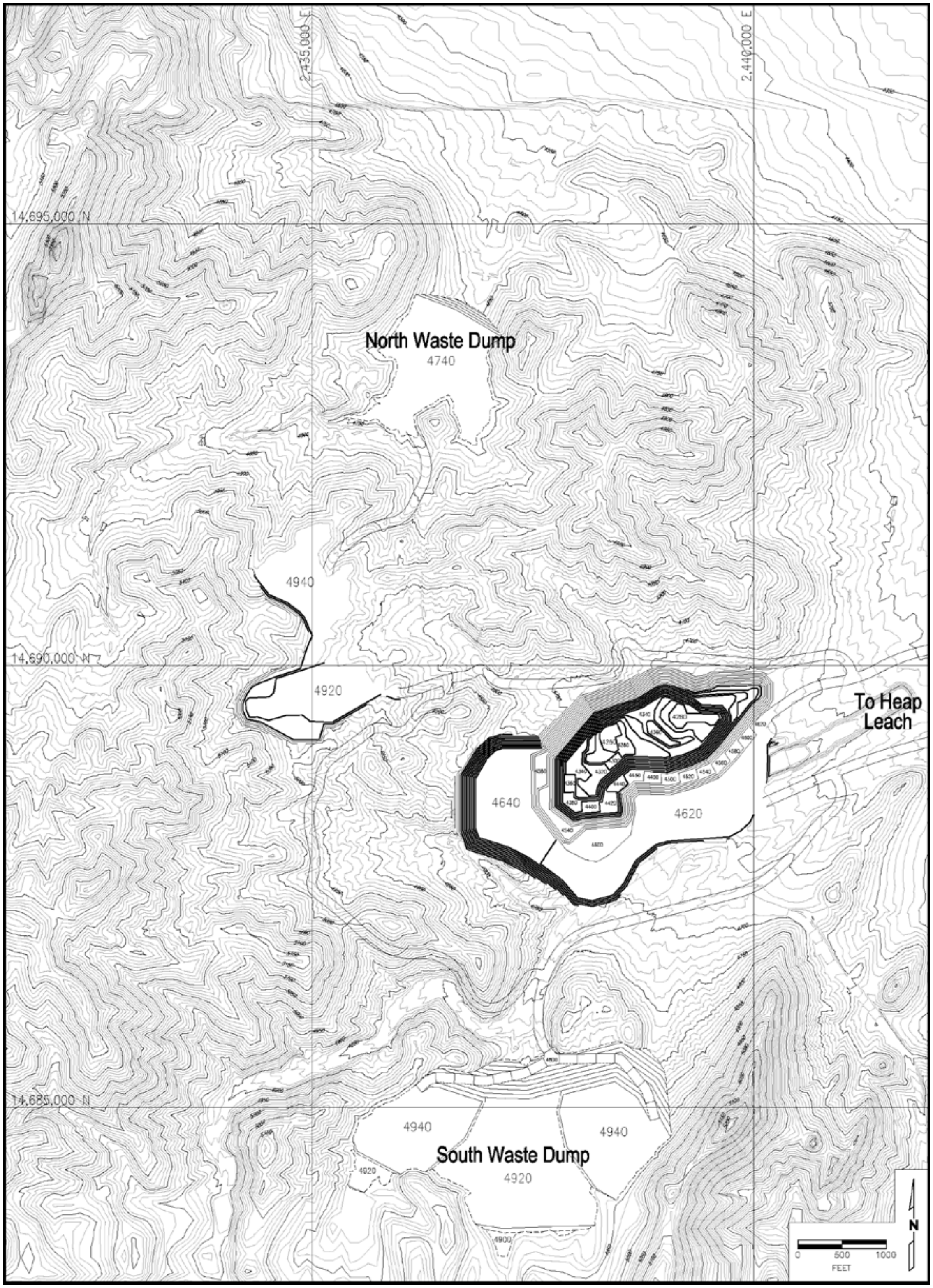


Figure 16-11: End of Year 5

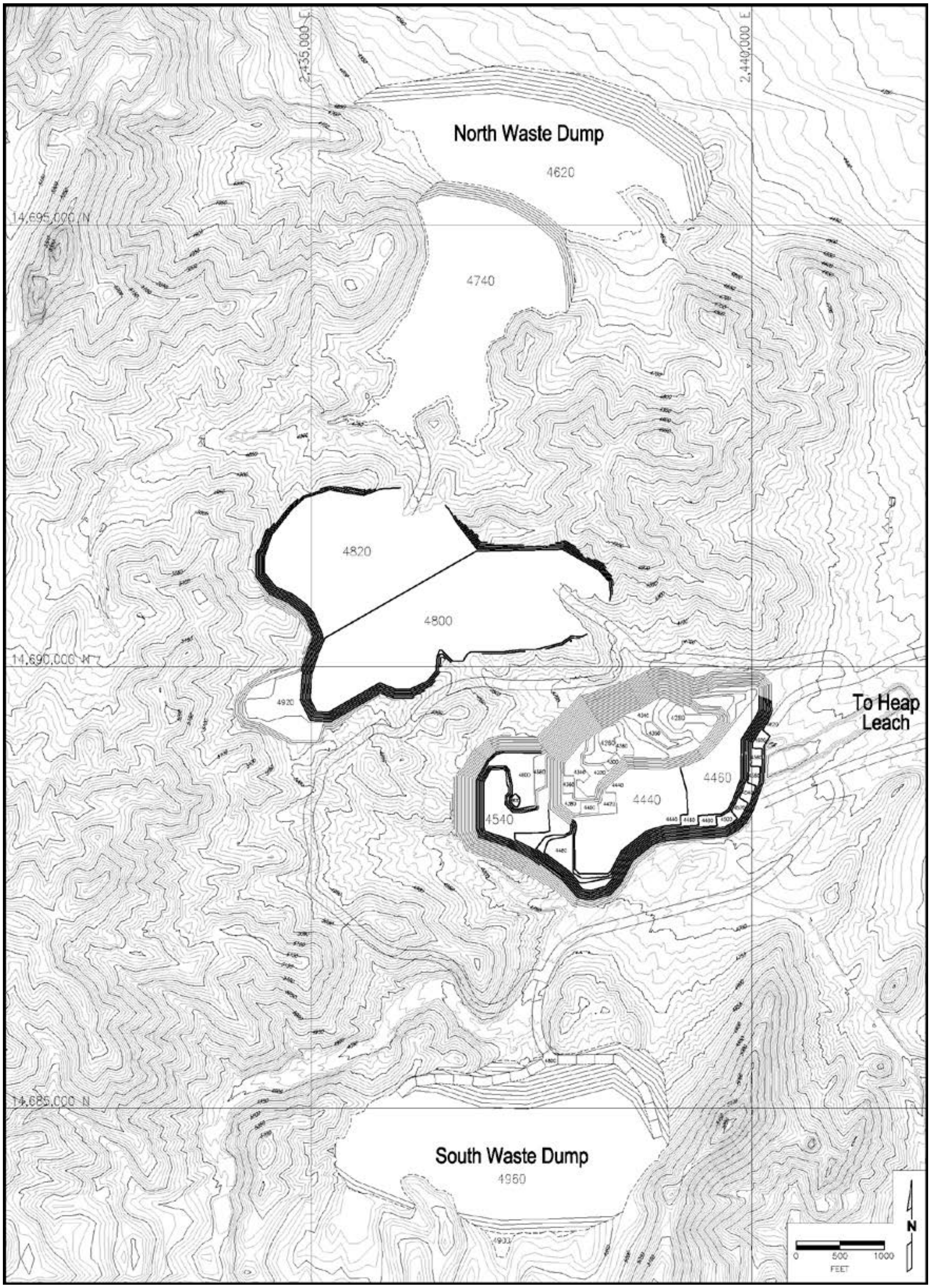


Figure 16-12: End of Year 7

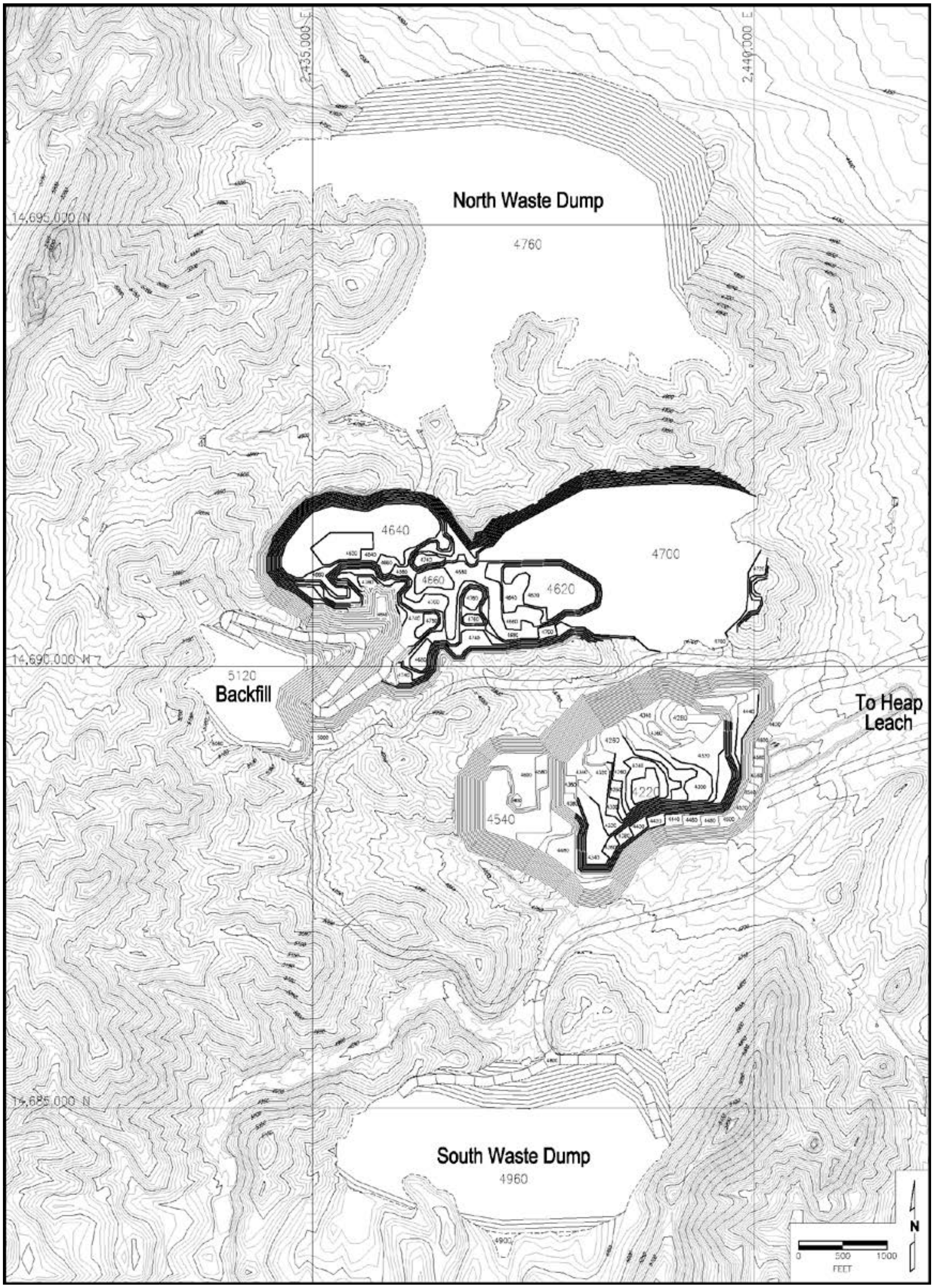


Figure 16-13: End of Year 10

16.6 MINING EQUIPMENT

Mine equipment requirements were calculated based on the annual mine production schedule, the mine work schedule, and equipment shift production estimates. The size and type of mining equipment is consistent with the size of the project, i.e. peak run-of-mine material movements of 35 million tons per year.

A summary of the total mine fleet by year for the major mine equipment is shown in Table 16-8. There is sufficient equipment to perform the following duties:

- Construct additional roads, after preproduction, as needed to support mining activity, including pioneering work necessary for mine and dump expansion.
- Strip topsoil in advance of mining and dumping.
- Mine and transport the mineralized material to the heap leach pad. Mine and transport the waste material from the pit areas to the waste storage areas.
- Maintain all the mine work areas, in-pit haul roads, waste storage areas, and external haul roads.
- Build and maintain in pit and on dump drainage structures as required.

Mine equipment requirements were not estimated for the following activities:

- Construction of any major surface water diversion channels and settlement ponds and dams, other than the ditching and sedimentation ponds for the waste storage areas.
- Construction of the shop area and plant area.
- Preproduction road construction outside of the immediate mine area.
- Contouring or reclamation of dumps at the end of the project.
- Mine dewatering for slope stability.

The mine equipment fleet calculations are based on two 12 hour shifts for 355 days per year (710 operating shifts). The number of pieces of equipment is based on equipment productivity for projects of similar tonnage movements. Detailed equipment requirement calculations on a year by year basis have not been completed for the PEA.

The truck haul routes and profiles were measured for mineralized material to the heap and waste to the waste dumps or pit backfill areas. Truck cycles were simulated to determine the cycle times and tons hauled per truck shift. From this, the number of operating trucks required was determined. The reference to specific equipment vendors is intended only to associate these vendors with the size of the equipment included for this PEA and is not intended to be a recommendation of a particular equipment vendor.

The major mine equipment consists of 9 inch blast hole drills, 26 cubic yard hydraulic shovels, a 17 cubic yard loader, and multiple 150 ton trucks plus major and minor support equipment.

Years 7 through 11 have the peak tonnage movement at 35 million tons per year; during year 6 the mine is ramping up to this capacity. One additional drill, shovel and 3 more trucks are added to the fleet in year 5 to handle the increased tonnage.

Table 16-8: Mine Equipment

Equipment	Initial Fleet Yr. -1 & 1	Peak Fleet Start Yr. 6
Mine Major Equipment:		
9 inch Blast Hole Drill	3	4
26 cu yd Shovel	1	2
17 cu yd Front End Loader	1	1
150 t Haul Truck	8	11
D10T Track Dozer	2	2
16m Motor Grader	2	2
777F Water Truck	2	2
Mine Major Support Eqpt.:		
988HH Wheel Loader	1	1
385C Excavator	1	1
D8T Dozer	1	1
735 ATD Haul Truck	2	2
CM 785 Rock Drill	1	1
1 cum Backhoe Loader	1	1
Support Equipment:		
Cable reeler, fuel & lube trucks, cranes, flatbed trucks, tire handler, forklifts light plants, etc.	1 lot	1 lot
Mine communications & radios, dispatch system, survey equipment, safety equipment, engineering & geology supplies	1 lot	1 lot

16.7 MINE LABOR

Mine personnel includes all salaried supervisory and staff people working in mine operations, maintenance, and engineering/geology departments, and the hourly people required to operate and maintain the drilling, blasting, loading, hauling, and mine support activities. In general mining activities end once the heap material is delivered to the heap leach facility.

The mine operating and maintenance labor will operate on a three crew rotation. The estimates of personnel are based on similar size projects. The salaried staff includes supervision labor in operations and maintenance and the personnel in the engineering and geology departments. This staff is between 35 and 40 people and includes the shift supervisors in both operations and maintenance. The number of hourly personnel in mine operations could range from 90 to 110 people during peak of operations. The number of mine maintenance personnel will range between 50 to 70 people depending on the maintenance philosophy adopted by the management (how much component replacement programs along with equipment dealer maintenance support is used).

The Yerington District has a strong mining history and the state of Nevada has many active mines thus a skilled labor force is available for this operation.

16.8 MINE CAPITAL COSTS

The mine initial and sustaining costs are based on similar projects, estimated equipment requirements and file quotations for major equipment. Major ancillary equipment (dozers, graders, etc.), are shown as lot purchases as is support equipment (blasting truck, fuel trucks, pickups, cranes, etc.) and the engineering, geology and safety equipment (listed as other). The replacement of equipment is based on years of service versus hours of operation, which could change the replacement schedule. No mine capital purchases are made after year 11 and the last major equipment replacement is in year 8 when the initial fleet of 6 trucks, one drill and the loader are replaced. Table 16-9 shows the purchases of the mine capital equipment. Cost for the mine shops, warehouse and allowance for explosives storage are carried elsewhere in the overall capital estimate.

Table 16-9: Mine Capital Estimate

Purchased Units		Pre-Prod	1	2	3	4	5	6	7	8	9	10	11	Total
Drills		1	2				1			1	1			
Hyd Shovel		1					1						1	
Loader		1								1				
Trucks		6		1	1		3		1	6			1	
Major Aux Equip		1							1			0.5		
Mine Support Equip		1						1						
Other		1												
Capital Cost	Unit Price	Capital Purchases												
	$ x 1000													
Drills	1,091	1,091	2,182	0	0	0	1,091	0	0	1,091	1,091	0	0	
Hyd Shovel	6,204	6,204	0	0	0	0	6,204	0	0	0	0	0	6,204	
Loader	2,900	2,900	0	0	0	0	0	0	0	2,900	0	0	0	
Trucks	2,564	15,384	0	2,564	2,564	0	7,692	0	2,564	15,384	0	0	2,564	
Major Aux Equip	16,200	16,200	0	0	0	0	0	0	16,200	0	0	8,100	0	
Mine Support Equip	5,000	5,000	0	0	0	0	0	5,000	0	0	0	0	0	
Other	1,000	1,000	0	0	0	0	0	0	0	0	0	0	0	
Total		47,779	2,182	2,564	2,564	0	14,987	5,000	18,764	19,375	1,091	8,100	8,768	131,174
rounded		48,000	2,200	2,600	2,600		15,000	5,000	18,800	19,400	1,100	8,100	8,800	131,600

16.9 MINE OPERATING COSTS

An estimate of mine operating cost includes costs for drilling, blasting, loading and hauling plus ancillary activities (dump maintenance, road development and maintenance, pit clean up, etc.), plus the mine services, mine maintenance and mine G&A departments. The general mine (mine services), general maintenance (maintenance supervision and maintenance of the smaller vehicles) and the mine G&A are estimated on a total cost per year basis with an increase in year 6 as the mine begins the ramp up to the maximum material rate. The direct mining activities of drill ($0.15/t), blast ($0.20/t), load ($0.18/t), haul (range from $0.25/t to $0.47/t) and ancillary services ($0.22/t) are estimated on a cost per ton basis using information from similar size operations. The haul costs are escalated by year assuming longer hauls as the pits deepen and the waste dumps and heap leach facilities gain height. Table 16-10 is a summary of the mine operating costs by year.

Table 16-10: Mine Operating Costs

Year	Total ktons	Drill 0.15	Blast 0.20	Load 0.18	Haul cost/ton	Haul $ x 1000	Auxiliary 0.22	General Mine $ x 1000	General Maint. $ x 1000	G&A $ x 1000	Total $ $ x 1000	Total $/t
Pre-Prod	500	75	100	90	0.25	125	1,000	500	500	1,000	3,390	6.78
1	19,042	2,856	3,808	3,428	0.25	4,761	4,189	2,000	2,000	4,000	27,042	1.42
2	17,174	2,576	3,435	3,091	0.27	4,637	3,778	2,000	2,000	4,000	25,517	1.49
3	20,000	3,000	4,000	3,600	0.30	6,000	4,400	2,000	2,000	4,000	29,000	1.45
4	20,000	3,000	4,000	3,600	0.32	6,400	4,400	2,000	2,000	4,000	29,400	1.47
5	20,000	3,000	4,000	3,600	0.31	6,200	4,400	2,000	2,000	4,000	29,200	1.46
6	30,000	4,500	6,000	5,400	0.32	9,600	6,600	2,500	2,500	4,000	41,100	1.37
7	35,000	5,250	7,000	6,300	0.35	12,250	7,700	2,500	2,500	5,000	48,500	1.39
8	35,000	5,250	7,000	6,300	0.37	12,950	7,700	2,500	2,500	5,000	49,200	1.41
9	35,000	5,250	7,000	6,300	0.39	13,650	7,700	2,500	2,500	5,000	49,900	1.43
10	35,000	5,250	7,000	6,300	0.33	11,550	7,700	2,500	2,500	5,000	47,800	1.37
11	35,000	5,250	7,000	6,300	0.33	11,550	7,700	2,500	2,500	5,000	47,800	1.37
12	32,403	4,860	6,481	5,833	0.35	11,341	7,129	2,500	2,500	5,000	45,644	1.41
13	33,156	4,973	6,631	5,968	0.36	11,936	7,294	2,500	2,500	5,000	46,802	1.41
14	35,000	5,250	7,000	6,300	0.37	12,950	7,700	2,500	2,500	5,000	49,200	1.41
15	32,261	4,839	6,452	5,807	0.38	12,259	7,097	2,500	2,500	5,000	46,454	1.44
16	27,749	4,162	5,550	4,995	0.40	11,100	6,105	2,500	2,500	5,000	41,912	1.51
17	30,256	4,538	6,051	5,446	0.43	13,010	6,656	2,500	2,500	5,000	45,701	1.51
18	22,612	3,392	4,522	4,070	0.45	10,175	4,975	2,500	2,500	5,000	37,134	1.64
19	676	101	135	122	0.47	318	149	500	500	500	2,325	3.44
Total	515,829	77,372	103,165	92,850	0.354	182,762	114,372	43,500	43,500	85,500	743,021	1.44

17 RECOVERY METHODS

17.1 OVERVIEW OF PLANNED FACILITIES

The process facilities planned for the MacArthur Copper Project include a ROM heap leach facility to recover copper in a leach solution, and a solvent extraction and electrowinning (SX/EW) facility to recover the copper from the leach solution and produce a cathode quality copper for sale. Also included is a sulfur burning sulfuric acid plant with a power plant to generate electrical power from the waste heat produced from the combustion of sulfur. Other facilities include solution ponds, water and power distribution, and infrastructure to support the facilities. An overall flow sheet for the heap leach and SX/EW facilities is shown in Figure 17-1 at the end of this section.

17.2 HEAP LEACH PAD

MacArthur mineralized material will be mined from open benches, loaded into mine haul trucks, transported directly to the heap leach pad and stacked. The mineralized material will be dumped on the leach pad and irrigated with an acidified leach solution (raffinate). Raffinate will be pumped from the raffinate pond through a pipeline and distribution network to drip emitters which will distribute the leach solution to the surface of the mineralized material pile on the leach pad to minimize evaporation losses. Some sprays may be used on side slopes or to increase evaporation if required to maintain the process water balance. The leach solution will percolate through the mineralized material pile and dissolve soluble copper from the mineralized material before being directed along the impermeable leach pad liner system to the solution collection system.

The heap leach pad design will conform to the Nevada Division of Environmental Protection (NDEP) requirements and will consist of, from bottom to top, a compacted soil base, covered by clay or a geosynthetic clay liner (GCL), an HDPE liner, and HDPE perforated piping network of collection pipes with 24 inches of crushed over-liner to protect the collection piping.

Copper bearing leach solution, called pregnant leach solution (PLS), will flow by gravity from the leach pad collection system to a lined collection pond (pregnant pond). The PLS will be pumped at a rate of 10,400 gallons per minute with 1 gpl copper to the solvent extraction mixer-settlers. The pump discharge pipes will be combined in a single pipeline to the solvent extraction circuit.

While one lot of material is being leached, the next will be mined and placed in another section of the leach pad. When a lot of mineralized material has completed the primary leach cycle, solution application will be transferred to the next lot of mineralized material. When all the mineralized material on a lift (or layer) has been leached, additional lifts will be placed on top of the previous lift and leaching will continue. The process of layering and leaching the mineralized material will be repeated to a maximum acceptable number of lifts on the leach pad. If required, one inner-lift liner may be used at one-half the pad height.

A storm water pond will be installed to handle excess water that might occur during a large precipitation event. The PLS collection pond will be designed to overflow to the storm water pond which is sized to accommodate a 100 year storm event. Water that may accumulate in the storm water pond will be periodically pumped to the raffinate solution pond.

17.3 SOLVENT EXTRACTION

Copper contained in the aqueous phase PLS will be extracted by contact with organic reagents carried in an organic solution (organic phase) in the solvent extraction circuit. Copper transferred to the organic phase will be stripped from the organic solution by contact with an acidic electrolyte solution (lean electrolyte) that will have circulated through the electrowinning cells. This transfer of copper enriches the electrolyte solution to form the rich electrolyte. The rich electrolyte will be pumped to the electrowinning cells for copper electrowinning onto stainless steel cathode blanks. Copper loaded on the stainless steel blanks will be harvested from the electrowinning cells on a weekly schedule. Copper will be removed from the stainless steel blanks by a stripping machine. Copper plates produced by this process, LME Grade A, will be weighed and bundled into 2 to 3 ton packages for shipment to market.

The solvent extraction plant will consist of one train of mixer-settlers. The train will have three stages of extraction arranged in a series parallel configuration and one stage of stripping. Aqueous and organic streams will flow counter-currently in extraction.

The PLS will be divided into two streams. One stream enters the two stage extraction pumper mixers, operated in series, and is contacted with stripped organic. After the two phases, organic and aqueous, have been mixed by flowing through mix tanks in series, the resulting mixture will be discharged into a single extraction settler to allow the two phases to disengage. The organic solution will float on top of the aqueous solution (raffinate) allowing the two phases to be separated by a weir system at the discharge end of the settler. The partially loaded organic then passes on to two other mixer-settlers, operated in series, where it counter-currently contacts the other half of the remaining leach solution to produce another raffinate stream and loaded organic. The resulting copper depleted aqueous solution, or raffinate, will flow by gravity to the raffinate pond.

The stripping mixer-settler will process loaded organic solution to remove copper extracted from the aqueous solution. Loaded organic solution from the first stage of extraction will flow by gravity to a loaded organic tank. Loaded organic solution will be pumped to the stripping stage pumper mixer and will be mixed with lean electrolyte solution from electrowinning. The same flow pattern occurs in the stripping circuit as in each stage of the extraction circuit. Organic solution will enter the strip stage mix tank and will be mixed with lean electrolyte solution. After the two phases, organic and aqueous will have been mixed by flowing through mix tanks in series, the resulting mixture will be discharged into settlers to allow the two phases to disengage. The organic solution will float on top of the aqueous solution allowing the two phases to be separated by a weir system at the discharge end of the settler. The stripped organic solution will flow to the second stage of extraction. The aqueous enriched electrolyte solution will be split with a portion advancing to electrowinning and the balance recycled within the organic strip stage.

17.4 ELECTROWINNING

The rich electrolyte solution from solvent extraction will flow by gravity to the electrolyte filter feed tank and will be pumped through two electrolyte filters operating in parallel. The filters will be backwashed periodically with lean electrolyte solution and air from a scour air blower. Filter backwash solution will be returned to the extraction settlers.

Filtered electrolyte solution will be pumped from the filtered electrolyte storage tank to the electrolyte heating circuit. The filtered electrolyte will flow through two heat exchangers operating in series. In the first heat exchanger, electrolyte will be warmed by lean electrolyte returning to solvent extraction from electrowinning. In the second heat exchanger, electrolyte will be heated, if necessary, with supplemental heat to final temperature for electrowinning. A hot water heating system will be installed to provide supplemental heat.

A diesel fired steam boiler will heat water in a hot water tank through a steam loop from the boiler. Hot water will be circulated through the heat exchanger when additional heat is required to heat the electrolyte solution, as during start-ups.

The electrowinning circuit tank house will contain 54 electrowinning cells. Heated electrolyte solution will enter the electrowinning cell circuit by flowing to an electrolyte recirculation tank where it will be mixed with electrolyte solution returning from the other electrowinning cells. The electrolyte recirculation tank will be a two compartment tank. The lean electrolyte from electrowinning will return to the smaller compartment that contains a pump connection for returning the electrolyte to the stripping circuit. The excess solution will overflow a baffle and be mixed with rich electrolyte and will be pumped to electrowinning cells. Lean electrolyte will be pumped from the recirculation tank through the heat exchanger and to the strip stage in the solvent extraction circuit.

Copper will be plated onto stainless steel cathode blanks. A cathode stripping machine will be used to remove the copper plates from the stainless steel blanks. The cathode stripping machine will perform several steps in sequence; cathode washing, hammering and flexing the blanks to loosen the copper plates, stripping the plates from the blanks, stacking and banding the copper plates, and stainless steel blank preparation for return to the electrowinning cells.

The tank house will have an overhead bridge type crane for transporting cathodes and anodes to and from the cells.

A filter system will be installed to process solvent extraction "crud" (the material that forms from an accumulation of solids and organic and aqueous solution at the organic/aqueous interface in the settlers) to separate organic solution that will be reused in solvent extraction. Crud will be drained or decanted from the settlers to a crud holding tank. When sufficient material has been collected it will be pumped to a crud decant tank. The crud will be diluted with diluent and alternatively mixed and allowed to settle in the decant tank. The organic layer that will form in the decant tank will be pumped from the tank to the loaded organic tank. Sediment from the crud holding tank will be pumped to a mix tank where it will be mixed with diatomaceous earth filter media. The mixture will be pumped to a plate and frame filter. Filtrate will be collected in a tank. Filtrate will be periodically transferred, depending on the phase content of the filtrate, to the solvent extraction aqueous system or to the solvent extraction organic system.

17.5 SULFURIC ACID PLANT

Molten sulfur will be received at a rail siding offsite in rail tank cars of approximately 100 ton capacity. The rail cars must be heated by steam to liquefy the sulfur since heat loss in the car during transit will solidify some of the sulfur. When re-heated, the molten sulfur is discharged to a receiving pit and pumped into a heated storage tank. The molten sulfur will be transferred from the heated storage tank at the rail siding by truck and tanker to heated storage tanks at the sulfuric acid plant.

Molten sulfur is pumped from the storage tanks at the acid plant to the sulfur furnace where it is mixed with high pressure air to atomize the sulfur and air to combust the sulfur. A bleed stream of sulfur recirculates back to the sulfur storage tanks to ensure a consistent feed of sulfur to the sulfur burners. Excess air is provided at the burners to ensure complete combustion and sufficient excess oxygen in the off gas for the conversion of SO_2 to SO_3 in the acid plant. The combustion process in the sulfur burner produces an off-gas at about 11% SO_2.

The combustion air for the sulfur furnace is first dried to remove any moisture in the air prior to combustion. This is to prevent corrosion in the rest of the downstream equipment. Ambient air is drawn into an air inlet filter and silencer ahead of the main acid plant blower and then delivered to the bottom of a packed drying tower by the main acid plant blower. In the drying tower, the ambient air flows through a packed section of ceramic saddles in countercurrent flow with 96% H_2SO_4. The air leaves the top of the drying tower and is delivered to the sulfur furnace for combustion. The circulating acid, at 96% H_2SO_4, will absorb the water in the air, which will tend to reduce the acid strength in the drying tower. This will be offset by a cross bleed of higher strength acid from the absorption towers downstream. Excess 96% H_2SO_4 generated in the drying tower is advanced to the absorption towers.

Off gas from the combustion process in the sulfur furnace will pass through a fire tube waste heat boiler and then to the converter. Steam is generated in the waste heat boiler which can be used to generate electrical power, discussed later. The converter is a four bed converter with vanadium pentoxide catalyst in each bed. As the gas passes through the catalyst beds, SO_2 is converted to SO_3. The reaction is exothermic and increases the temperature of the gas. After each pass through a converter bed, the gas is cooled through gas-to-gas heat exchangers to cool the gas for the next pass. After the second or third pass, approximately 85% of the SO_2 has been converted to SO_3 and the gas then passes to an intermediate absorption tower before returning to the final one or two passes. At the outlet of the fourth pass, the gas then passes to the final absorption tower.

The intermediate and final absorption towers are similar in design as the drying tower. The gas stream from the converter enters the absorption towers at the bottom and passes through a section of ceramic packing saddles where the SO_3 comes in contact with, and absorbed by, the circulating 98.5% H_2SO_4. The gas leaves the absorption tower at the top and returns to the converter for further SO_2 conversion (in the case of the intermediate absorption tower) or is discharged to atmosphere through a stack (in the case of the final absorption tower). As the SO_3 is absorbed into the 98.5% circulating acid, the acid will tend to gain in strength. A cross bleed of the higher strength acid from the absorption towers is directed to the drying tower to maintain 96% acid strength in the drying tower. Excess 96% acid in the drying tower advances to the intermediate and final absorption towers. Water is also added to the absorption towers to maintain a constant acid strength of 98.5% acid. Excess 98.5% acid in the absorption towers is pumped to storage as the final product from the acid plant.

The double-contact double-absorption sulfuric acid plant conforms to the Best Available Demonstrated Control Technology (BADCT) for controlling sulfur dioxide emissions.

17.6 POWER PLANT

The power plant will be located adjacent to the sulfuric acid plant. Steam from the waste heat boiler will drive a steam turbine generator to generate electrical power. The turbine exhaust will be directed to a shell and tube steam condenser operating under vacuum. The condensate from the steam condenser is collected and pumped through a water treatment system to maintain boiler quality water. A dump condenser will also be provided to condense steam in the event the turbine generator cannot accept the steam from the waste heat boiler. Boiler blow down will be cooled and directed to the raffinate pond.

The power plant turbine generator will be connected to the main electrical substation buss for distribution to the SX/EW and acid plant facilities.

Fresh water make-up to the steam system will be treated by filtration to remove particulates, cation exchange water softening to remove scale producing ions, chemical treatment of the softened water with a dispersant and anti-scalant, and Reverse-Osmosis (RO) filter to achieve the desired water quality. The circulating condensate water will be treated through a condensate polishing system as a precautionary measure to ensure boiler quality water is maintained.

Cooling towers will be required to provide cooling water for the sulfuric acid plant acid coolers and the dump condensers at the power plant. The cooling towers will have chemical water treatment for the fresh water make-up. Fresh water will be required for make-up to the system to account for the evaporation loss and blow down. Blow down from the cooling towers is required to maintain a proper level of dissolved solids in the cooling towers. The blow down will be directed to the raffinate pond.

17.7 ANCILLARY FACILITIES

Ancillary facilities to support the MacArthur process facilities include fuel storage and distribution systems for heavy equipment and light vehicles, an electrical substation, a guardhouse and truck scale at the entrance to the property. Existing buildings at the Yerington site will be refurbished and used for administration, an analytical lab, a mine truck shop, warehouse, and maintenance facilities.

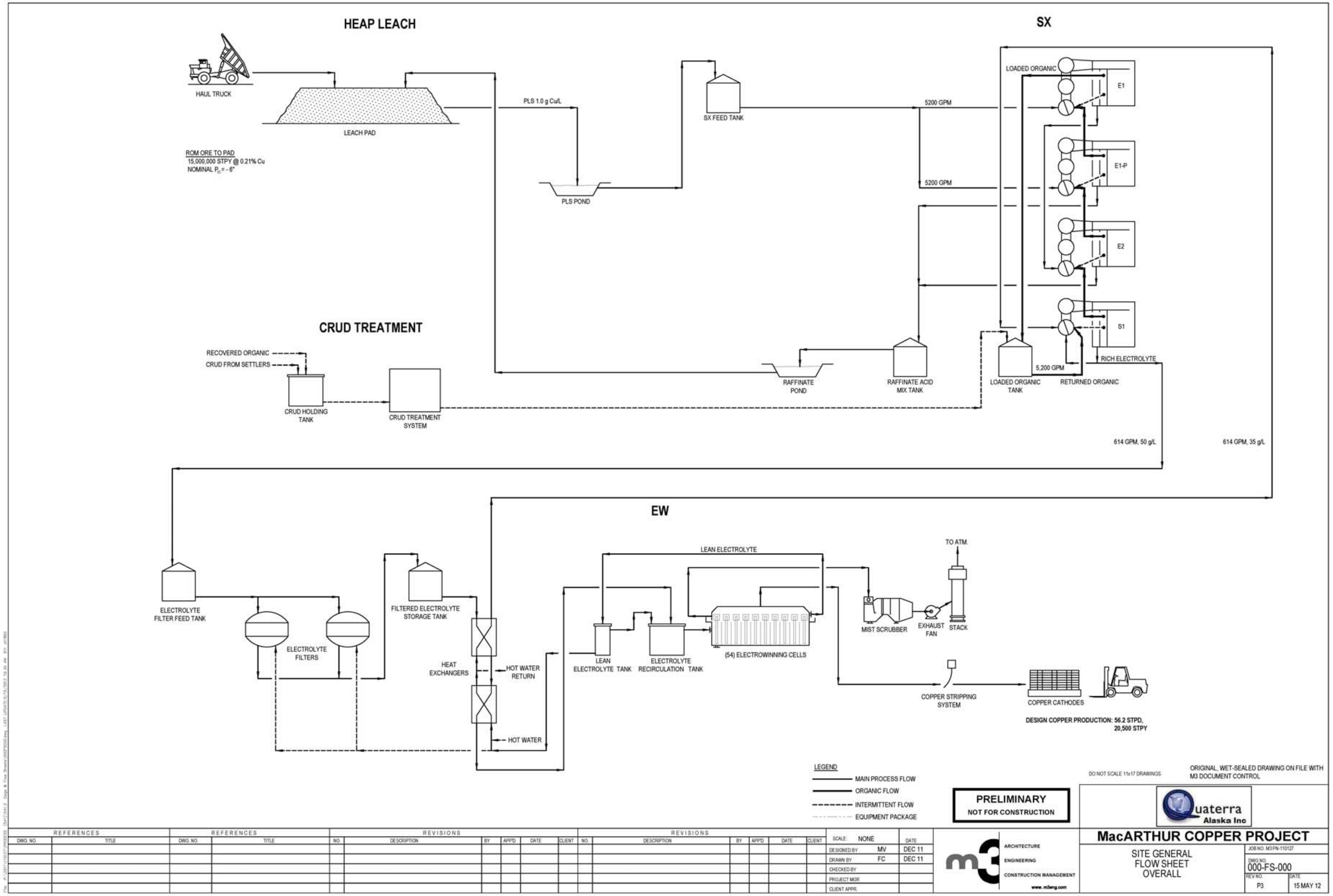


Figure 17-1: Overall Process Flowsheet

18 PROJECT INFRASTRUCTURE

18.1 SITE LOCATION

The MacArthur Copper Project is located near the geographic center of Lyon County, Nevada, along the northeastern flank of the Singatse Range, approximately seven miles northwest of the town of Yerington, Nevada. The property is accessible from Yerington by approximately five miles of paved roads and two miles of Lyon-county maintained gravel road. The nearest major city is Reno, Nevada, approximately 75 miles to the northwest.

The process facilities are located east of the mineralized material body and west of Alternate Highway 95. The heap leach pad is directly east of the MacArthur pit with the process facilities located south of the heap leach pad. The area covered by the heap leach pad is approximately 390 acres and the process facilities occupy another 85 acres. The heap leach pad and process facilities are shown in Figure 18-1 at the end of this section.

18.2 PROCESS BUILDINGS

The process facility generally consists of solvent extraction settlers, a tank farm, and an electrowinning building. The solvent extraction settlers are four covered tanks approximately 60 feet by 115 feet and 4 foot deep. The tank farm is located below the solvent extraction facilities and contains all the circulation tanks, pumps, heat exchangers, and filters that service the solvent extraction and electrowinning facilities. The electrowinning building is a pre-engineered steel building with corrugated metal roofing and siding. The main cell area is approximately 150 feet long and 70 feet wide holding two rows of 27 electrowinning cells each. A building extension, approximately 98 feet by 180 feet, is located at one end of the electrowinning cells to house the automatic stripping machine and the cathode handling equipment. An overhead crane in the building services the electrowinning cells and stripping machine. An electrical equipment room and control room is located on one side of the cathode handling section which overlooks the cathode stripping operation. Cathode handling, weighing and banding is performed at the other side of the cathode handling section. An asphalt paved laydown area is provided outside the cathode handling area to allow cathode storage and loading of cathodes onto flatbed trailers for shipment to market. The building is provided with ventilation fans and scrubbers to ventilate the cell area.

The control room will have space for offices, a laboratory, and restrooms. The laboratory will be for analyzing routine shift samples. An existing building in the Yerington operation will be refurbished for a main analytical laboratory to supplement the laboratory at in the tank house. The grey and black water from the restrooms will report to a dedicated septic system.

18.3 ANCILLARY BUILDINGS

Ancillary buildings necessary to support the MacArthur Copper Project include an administration building, a warehouse / maintenance building, an analytical laboratory, mine truck shop, a change house, fuel storage and dispensing facilities, and the main gatehouse with truck scale. Some ancillary buildings at the existing Yerington facility will be refurbished and used to support the MacArthur Copper Project.

18.3.1 Administration Building

Quaterra occupies offices in the town of Yerington that will continue to be used for some of the administrative functions. A smaller building at the Yerington mine site will be re-furbished to provide additional offices for onsite supervisors. An allowance was provided to include approximately 1,200 square feet of office space at the mine site.

18.3.2 Warehouse / Plant Maintenance Building

An existing industrial building at the Yerington mine site will be converted to a maintenance building and warehouse. The building is approximately 12,000 square feet and will be partitioned in the center to provide 6,000 square feet for warehouse space and 6,000 square feet for maintenance space. The building is of steel construction and corrugated roofing and siding. Metal shelving will be provided for the warehouse and the maintenance side will have offices and restrooms and will house the plant maintenance facilities. A fenced area will be provided outside the warehouse for secure outdoor storage.

18.3.3 Analytical Laboratory

An analytical laboratory will be provided at the Yerington mine site to supplement the SX/EW laboratory. An existing building, approximately 1,500 square feet, will be re-furbished and provisioned with sample preparation equipment, laboratory equipment, ventilation systems and offices. Mine samples will be processed at this laboratory.

18.3.4 Mine Truck Shop

An existing mine truck shop exists at the Yerington mine site and can continue to be used. The building is approximately 9,000 square feet and is equipped with an overhead crane. An allowance was provided for re-furbishing of the building and an allowance for new equipment.

18.3.5 Change House

A new change house building will be provided at the MacArthur SX/EW facility. The change house is a pre-engineered steel building with corrugated roofing and siding. The building is approximately 1,000 square feet.

18.3.6 Main Gatehouse

A new modular building will be provided at the main gate to control access to the SX/EW plant. The building is 44 feet by 14 feet with 10 foot eaves. A truck scale will be provided at the main plant entrance to weigh all receipts of reagents and consumables as well as cathode copper production leaving the plant.

18.3.7 Fuel Storage and Dispensing

Fuel storage and dispensing facilities will be provided for mine trucks and in-plant vehicles. Two 50,000 gallon diesel storage tanks are provided to service the mine trucks and mining equipment. Two 5,000 gallon storage tanks will also be provided for small vehicles and equipment. One tank will be for diesel and the second tank for gasoline. Fuel will be received by tank trucks from Reno, Nevada or other nearby source and dispensed at site.

18.4 ACCESS ROADS

Access to site is from Alternate Highway 95, west on Luzier Lane, and north on Mason Pass Road. Entrance to the SX/EW facility is off of Mason Pass Road. Once on site existing haul roads connect to the mine site and the existing Yerington facilities. No new access roads will be required.

18.5 RAILROAD FACILITIES

A railroad and siding exists at Wabuska, Nevada, approximately 10 miles north of Yerington on Alternate Highway 95. The rail line connects from Hawthorn, Nevada to Salt Lake City, Utah, generally following Interstate Highway 80 east. The existing siding will be used to receive molten sulfur by rail, with the sulfur transferred from rail cars to truck and tankers at Wabuska for the final transport to site. Molten sulfur will be received in 100 ton rail cars at the rate of approximately 14 rail cars per week.

18.6 POWER SUPPLY & DISTRIBUTION

Power for the facility will be taken from an existing 69 kV power line feeding from the existing Fort Churchill generating facility to the town of Yerington, a distance of approximately 10.5 miles. The 69 kV power line approaches the SX/EW plant site along the eastern project boundary. A tap will be taken from the existing power line and a short, 800-foot long, power line will be constructed to connect to the SX/EW main electrical substation. The condition of the existing 69 kV power line from Fort Churchill has not been assessed at this time and may require upgrades in order to service the MacArthur Project site.

At the SX/EW main substation, power will be transformed to 34.5 kV or 13.8 kV for distribution throughout the plant. Additional transformers will be provided in the various process areas to provide medium voltage (4160 V) and low voltage (480 V) to feed the end users. Electrical equipment rooms and motor control centers will be located at solvent extraction, tank farm, electrowinning, acid storage, and the solution ponds. Step down transformers will be provided to serve the change house, guard house and fuel storage facilities.

18.7 WATER SUPPLY & DISTRIBUTION

The total water consumption for the project is estimated to be approximately 1,600 gpm, or 2,590 ac-ft per year. Quaterra and its subsidiary companies own approximately 8,600 ac-ft of water rights at the Yerington Mine Site. Fresh water for the project will be taken from wells on or near the MacArthur property and pumped to a 380,000 gallon fresh water/fire water storage tank. The lower 120,000 gallons in the storage tank will be reserved for fire water. Fresh water for plant use will be taken from the storage tank above this reserve level for fire suppression. An additional 10,000 gallon potable water tank will be provided to service the potable water system. The fresh water will be filtered and chlorinated before stored in the potable water tank and distributed throughout the plant. Potable water will be used for offices, labs, restrooms, and eye wash stations. Fresh water will be used for fire suppression, wash down, and process water make-up. The facility will be designed as a zero discharge facility.

18.8 WASTE MANAGEMENT

It is assumed a private landfill will be provided on the property for non-hazardous solid waste. This facility will not accept any off site wastes and will be used primarily for construction debris, non-putrescible materials and waste from maintenance and operations meeting the definition of inert or non-hazardous materials; such as air filters, gloves, boxes, non-recyclable packaging material, hoses, piping, etc.

Recyclable materials that are non-hazardous, such as scrap metal, paper, used oil, batteries, wood products, etc., will be collected in suitable containers and disposed of through recyclers.

Hazardous materials such as contaminated greases, chemicals, paint, reagents, etc. will be collected and shipped off-site for destruction or disposal. Some hazardous materials, such as lead flakes and anodes, may also be recycled through appropriate recyclers.

18.9 SURFACE WATER CONTROL

Storm water run-off will be diverted around the plant facilities as much as possible. The natural gradient of the heap leach area generally slopes to the northeast. The SX/EW facilities slope to the south east. A diversion channel is provided to direct non-impacted run-off water around or through the plant without contacting impacted areas. Impacted run-off water from within the plant will flow to the tank farm area and collected in the tank farm sump. The impacted water will be pumped from the tank farm sump to the raffinate pond. A storm water pond is located adjacent to the PLS pond to accept any overflow from the PLS pond during storm events. The overflow will then be pumped back to the process or the raffinate pond. Annual precipitation ranges from 5 to 8 inches per year, more in the higher elevations.

18.10 TRANSPORTATION & SHIPPING

All materials coming into the plant will be by truck, including the molten sulfur transferred from the rail facilities at Wabuska. The facility will also be able to receive sulfuric acid by truck in emergencies, if needed, with the acid unloaded into the same sulfuric acid storage tanks. Incoming materials include reagents, extractant, kerosene, gasoline, diesel, warehouse stock and spare parts.

The primary product leaving the plant is cathode copper, which will be by flatbed tractor trailers. Recycle materials leaving the plant will also be by truck. Scales to weigh full loads and empty loads into and out of the plant are provided at the main gate for the highway trucks. The approximate quantity of major products and consumables entering and leaving the plant are shown in Table 18-1 below.

Table 18-1: Products & Consumables

Material	Quantity per Year	Units	Quantity per day	Units
Extractant	38,700	lbs. / yr.	106	lbs. / day
Diluent (Kerosene)	516,500	lbs. / yr.	1,415	lbs. / day
Cobalt Sulfate	59,500	lbs. / yr.	163	lbs. / day
Guar	107,700	lbs. / yr.	295	lbs. / day
Sulfur for Sulfuric Acid	77,400	tons / yr.	212	tons / day
Cathode Copper (Production)	20,700	ton / yr.	57	tons / day

18.11 COMMUNICATIONS

The connection to telephone and internet services for the project has not been confirmed at this time; however, telephone service is available at the town of Yerington and the existing Yerington mine site. It is assumed that the telecommunication system will be integrated with the onsite data network system utilizing a voice over I/P (VoIP) phone system. A dedicated server will be provided for setup and maintenance of the VoIP system and for accounting of all long distance phone calls. Handsets will plug into any network connection in the system for telecommunications. The office Ethernet network will support accounting, payroll, maintenance and other servers as well as individual user computers. High bandwidth routers and switches will be used to logically segment the system and provide the ability to monitor and control traffic over the network.

A process control system Ethernet network will support the screen, historian and alarm servers connected to the control room computers as well as Programmable Logic Controllers (PLC). This system will incorporate redundancy and a gateway between the office system and control system to allow business accounting systems to retrieve production data from the control system. No phone or user computer will be connected to this system.

The internal communications within the plant will utilize the same VoIP phone system, which will provide direct dial to other phones throughout the plant site. Mobile radios and cell phones will also be used by operating and maintenance personnel for daily communications while outside the office.

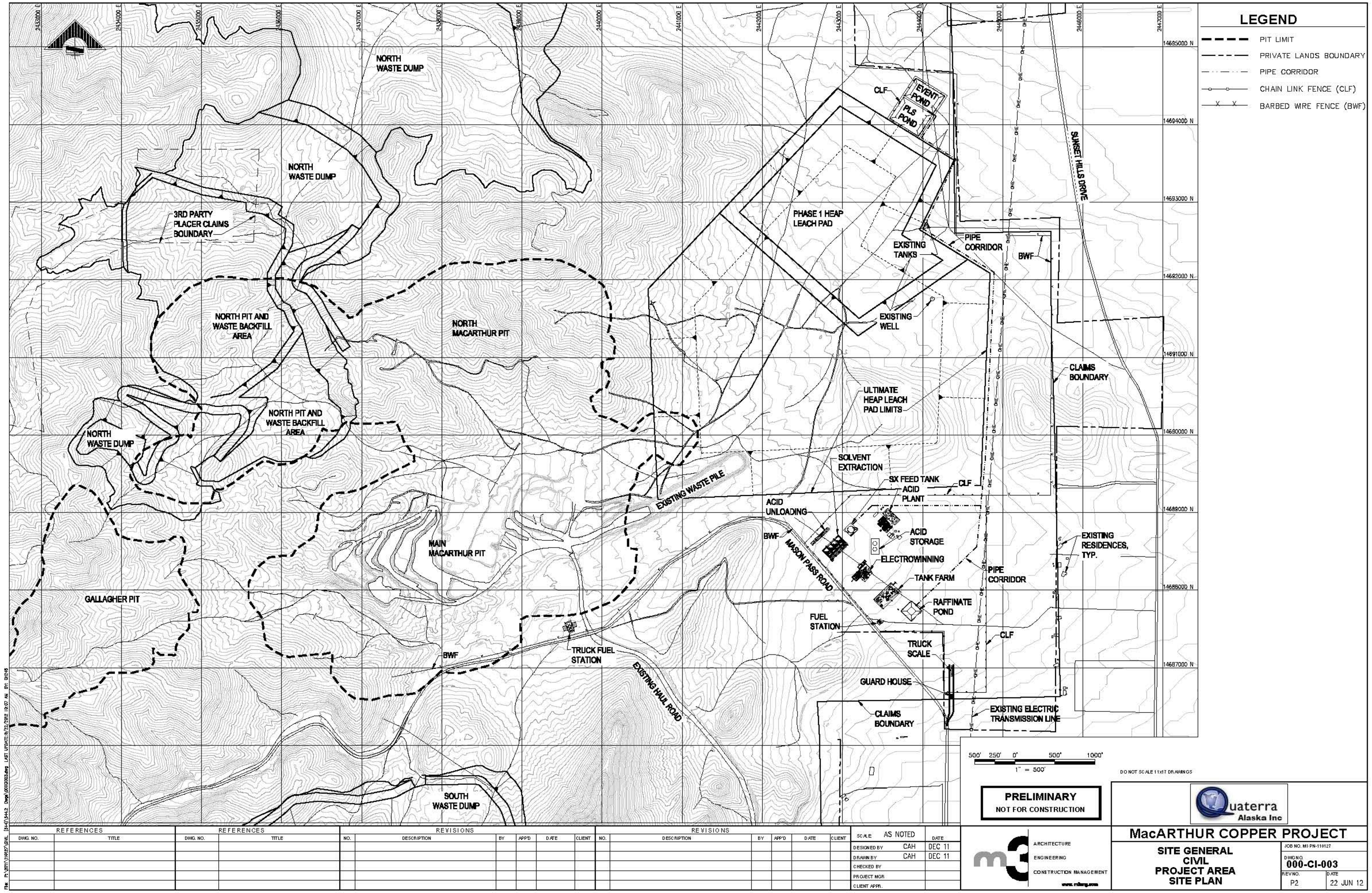


Figure 18-1: MacArthur Heap Leach and Process Facilities

19 MARKET STUDIES AND CONTRACTS

Copper is an international traded commodity with the price governed by the worldwide balance of supply and demand. The copper price is determined by the major metals exchanges; consisting of the New York Mercantile Exchange (COMEX), the London Metals Exchange (LME), and the Shanghai Future Exchange (SHFE). Recent historical copper prices are shown in Figure 19-1 below.

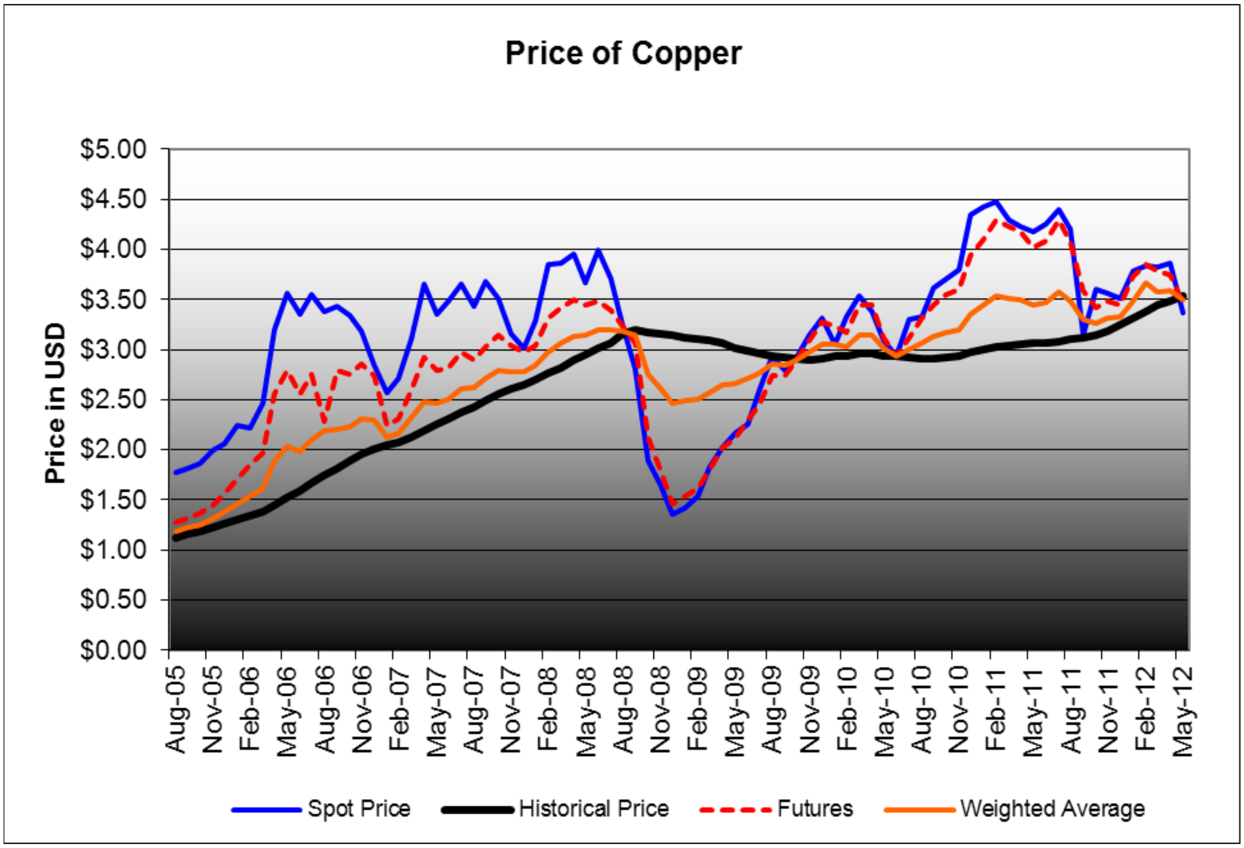


Figure 19-1: Historic Copper Price

The final product from the MacArthur facilities will be high purity (99.9%) electrolytic cathode copper in sheets of about 100 pounds each; bundled into approximately 55 cathode sheets or 5,500 pounds per bundle. Approximately 90% of copper cathode production in the United States goes to wire rod mills and eventual wire production and to brass mills producing various copper and copper alloy shapes. North America is a net importer of refined copper with a projected consumption of refined copper for 2012 of 2.5 million tons and a production of 2.0 million tons. The 0.5 million ton shortfall is made up of imports primarily from Chile, Canada, Peru, and Mexico. It is expected that the production from the MacArthur facilities would be absorbed into the North American copper market for refined copper, displacing the import copper.

Typical terms related to cathode copper shipping include FCA (Free Carrier) at the refinery; that is the buyer arranges and pays for cathode transportation from the refinery and the seller loads the cathode onto buyer's trucks. The price is based on the average COMEX price during the Quotation Period plus a premium for ASTM Grade 1 quality material, with negotiated discounts for lesser quality material. The net premium is the quoted premium, less freight charges and a margin allowed to the merchant buyer. The Quotation Period is the month of shipment or the month following the month of shipment. Payment is typically 2 days after the date of shipment. Northeastern Texas is the major regional market for western US cathodes.

Transportation is by flatbed truck for shorter distances or by rail in box cars for the longer distances.

A formal market study has not been conducted in this phase of the project and there are no established contracts for the sale of copper cathode in place at this time.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL LIABILITIES

Previous mining at the MacArthur site was conducted by Arimetco in the late 1990's, and included the construction of open pits, a waste rock dump, and access roads. The waste rock dump was reclaimed by the U.S. Bureau of Land Management following Arimetco's bankruptcy in 1997 and abandonment of the site in 2000. Numerous historic adits and underground workings are located throughout the project area, many of which have been secured by the Nevada Division of Minerals (in coordination with Quaterra) to prevent unauthorized access.

Extensive exploration has also occurred throughout the project area since the 1970's. Exploration related disturbance, including both historic disturbance and new disturbance created by Quaterra, consists of historic drill sites, trenches, and numerous drill roads. Quaterra has a reclamation bond that covers exploration and is responsible only for the exploration disturbance created during their tenure at the site since 2007.

The material from the previous MacArthur Pit was processed through heap leaching at the Yerington Mine facility which is located approximately five miles from the project area. These materials are not associated with the current operation at MacArthur and are being evaluated as a potential resource for reprocessing at the Yerington site (see Section 24).

Because the site is basically at or near elevation, no pit lake formed following the cessation of mining by Arimetco. Although uncertain, it is unlikely that a pit lake would form as a result of Quaterra's proposed mining operations at MacArthur. However, additional hydrogeological investigations will be necessary before a final determination can be made in this regard.

There are no known liabilities to which the MacArthur property is subject.

20.2 PERMITS

The mineral resources at MacArthur are located on public (unpatented) mining claims administered by the U.S. Department of the Interior, Bureau of Land Management, Carson City District, Sierra Front Field Office (BLM). Mine development will require participation of the BLM as the primary land manager. Various departments within the State of Nevada will be cooperating agencies in permitting mining development and process facilities at the site. Based on the current mine plan, and proposed facilities, the following Table 20-1 lists the principal permits necessary to commence mining operations. To date, none of these permits have been acquired for mining operations, although permitting for exploration activities is complete.

Table 20-1: Summary of Major Permits for Future Mining

Regulatory Agency	Permit Name
Federal Permits	
Bureau of Land Management	• Approved Plan of Operations/Decision Record • Roads and utility Rights-of-Way
Bureau of Alcohol, Tobacco, Firearms, and Explosives	• Authorization to purchase, transport, or store explosives
Mine Safety and Health Administration	• Notification of Commencement of Operation • Employee and Facility Health and Safety
Environmental Protection Agency	• Hazardous Waste ID No. (small quantity generator)
State Permits	
Nevada Division of Environmental Protection	
Bureau of Mining Regulation and Reclamation	• Water Pollution Control Permit • Reclamation Permit
Bureau of Air Pollution Control	• Class I (PSD) or Class II Permits to Construct and Operate • Mercury Permit
Bureau of Water Pollution Control	• Stormwater NPDES General Permit • Septic Permit
Bureau of Waste Management	• Approval to Operate a Solid Waste System (if necessary) • Hazardous Waste Management Permit
Bureau of Safe Drinking Water	• Potable Water Permit
Nevada Division of Water Resources	
	• Permit to Appropriate Water • Permit to Construct a Dam • Mineral Exploration Hole Plugging
Nevada Department of Wildlife	
	• Industrial Artificial Pond Permit
State Fire Marshall	
	• Hazardous Materials Permit
Local Permits	
Lyon County	
	• Special Use Permit • Building Permit • Business License

20.2.1 Federal Permitting

A mine plan of operations (PoO) will be prepared to describe the construction, operation, and reclamation of each facility along with a cost estimate that presents the reclamation and closure costs if the BLM were required to take over reclamation of the mine site. Information required for the PoO includes: pit location(s) and lateral and vertical extent of disturbances; heap leach pad conceptual designs; location of haul roads, mineralized material stockpiles, waste rock dumps, growth media stockpiles, office/laboratory, shops, diesel/lubricant storage and distribution system, well and associated piping; power line locations; generators; schedule of construction and operation; mining schedule; and equipment list. A reclamation plan is an important part of the PoO, which describes the activities that will take place to estimate the reclamation cost for bonding. The PoO will also function as the Reclamation Permit application for the State of Nevada (BMRR) (see below).

The PoO provides sufficient detail to identify and disclose potential environmental issues during the National Environmental Policy Act (NEPA) review, including an environmental impact statement (EIS) or Environmental Assessment (EA). The BLM will likely require an EIS for a project of this type. As a general rule, the PoO/EIS process for a mining/mineral beneficiation project is a minimum 36 months process (including 12 months for baseline data collection and PoO development plus a minimum of 24 months on review and EIS preparation). However, internal BLM situations could occur beyond the control of the project proponent, and a number of potential external events (public or cooperating agency opposition) could lengthen the overall EIS schedule. It is not uncommon for a mining PoO/EIS process to be three to five years before a Record of Decision (ROD) is issued.

The BLM has recently implemented new procedures requiring that at least one year of baseline data be submitted with the PoO in accordance with the state-wide Instruction Memorandum No. NV-2011-004 (dated November 5, 2010). The purpose of this guidance is to “improve the efficiency and effectiveness of processing mine Plans of Operation.” To that end, the BLM front loaded the permitting process for the collection of baseline data and environmental studies before the PoO is submitted for BLM review and NEPA analysis. BLM believes this should reduce the review period and overall NEPA process.

The requirements of the BLM PoO document are fairly well defined. However, baseline data necessary for the impact assessment phase of the project will need to be collected, analyzed, and interpreted in conjunction with the BLM to ensure the information collected meets the Data Quality Objectives (DQOs) of the program. Longer-lead items to be considered include:

- Groundwater sampling (hydrogeology) in the project area for depth and quality (for use in both the NEPA analysis and the State’s Water Pollution Control Permit application); and
- Geochemical characterization of waste rock, mineralized material, and spent leach material including acid-base accounting (ABA), meteoric water mobility procedures (MWMP) testing, and humidity cell (HCT) testing. The geochemical characterization program must be approved in advance by the BLM and the NDEP, and be in accordance

with BLM *Instruction Memorandum (IM) No. NV-2010-014 Nevada Bureau of Land Management Rock Characterization Resources and Water Analysis Guidance for Mining Activities* (January 8, 2010).

The collection of environmental baseline data necessary for development of the mine operations PoO and EIS review process was initiated with an expanded vegetation monitoring program in May 2012. Vegetation, including special status species, is a time-critical environmental element with limited windows for data collection. It is generally the first element initiated as part of a baseline data collection program. Other “elements of the environment” (BLM National Environmental Policy Act Handbook H-1790-1, 2008) to be considered during the NEPA process include:

- Air Quality,
- Areas of Critical Environmental Concern,
- Cultural Resources,
- Environmental Justice,
- Floodplains,
- Grazing Management,
- Land Use Authorization,
- Migratory Birds,
- Minerals,
- Native American Religious Concerns,
- Noxious Weeds, Invasive and Non-Native Species,
- Paleontological Resources,
- Recreation,
- Social and Economic Values,
- Soils,
- Special Status Species (plants and animals),
- Threatened and Endangered Species (plants and animals),
- Vegetation,
- Visual Resources,
- Wastes (solid and hazardous),
- Water quality (surface and ground),
- Wetlands/Riparian Zones,
- Wild Horses and Burros,
- Wilderness and wilderness characteristics, and
- Wildlife.

Table 20-2 presents a list of elements or studies that generally require more detailed investigations and may need to be undertaken during the mine planning phase in advance of the NEPA process. These studies will also be used to support the acquisition of various other operating permits. Many of these studies were performed within the project site as part of the PoO/EA for Exploration and may have to be updated for the EIS (BLM, 2009).

Table 20-2: Future Baseline Studies

	Permit/Authorization	Investigations/Studies
Water	• NEPA Analysis • Water pollution control permit • Stormwater control	Monitor surface waters in project vicinity on a seasonal basis for quality and quantity Monitor groundwater for level and water quality especially in the pit, dump, and heap areas to collect baseline quality data
Geology and Geochemistry	• NEPA Analysis • Water pollution control permit • Waste rock dump design • Dump and heap closure • Closure planning for dumps, heaps, and tailings	Collect representative samples of waste rock, mineralized material, and spent heap mineralized material for geochemical characterization (ABA, MWMP, and HCT) Condemnation drilling in proposed locations of facilities
Cultural Resources	• NEPA Analysis	Conduct a Class III survey in previously unsurveyed or as directed by the BLM Mitigate sites that cannot be avoided
Biological Resources	• NEPA Analysis	Determine presence or absence of threatened, endangered, or special status plant and animal species including golden eagles in previously unsurveyed areas Determine presence or absence of game species

Other federal permits that may be required include a hazardous waste identification number from the U.S. Environmental Protection Agency and an explosives use permit from the Bureau of Alcohol, Tobacco, Firearms, and Explosives.

20.2.2 State Permitting

The State of Nevada requires permits for all mineral exploration and mining operations regardless of the land status of the project. The two most important operational permits include the Water Pollution Control Permit (WPCP) and the Reclamation Permit; both issued by the Department of Conservation and Natural Resources, Division of Environmental Protection, Bureau of Mining Regulation and Reclamation (BMRR). The BMRR is composed of three distinct technical branches; Regulation, Closure, and Reclamation, and its mission is to ensure that Nevada's waters are not degraded by mining operations and that the lands disturbed by mining operations are reclaimed to safe and stable conditions to ensure a productive post-mining land use.

The Regulation Branch of the BMRR issues a WPCP to a mine operator prior to the construction of mining, milling or other beneficiation processes. Facilities utilizing chemicals for processing mineralized material are generally required to meet a zero discharge performance standard to protect of surface waters, which standard requires containment of all process fluids. The WPCP covers mine facility components including buildings, structures, facilities or other installations from which there is or may be a discharge of pollutants. In the case of the MacArthur Copper Project, this includes, to the following facilities:

- Acid leach pad;
- Process solution ponds;
- SX/EW plant and reagent tank farm;
- Sulfuric acid plant and storage facilities;
- Water treatment facilities
- Fuel storage and dispensing facilities; and
- Waste rock dump(s)

Due to processing timeframes, a WPCP application should be submitted at least 180 days prior to the planned construction date of any component of a mining operation or the planned start of mining. This time frame includes the public notice and a 30-day public review and comment period. A WPCP is valid for 5 years, provided the operator is in compliance with the regulations.

The Reclamation Branch of the BMRR issues a Reclamation Permit to an operator prior to construction of an exploration, mining, milling or other beneficiation process activity that proposes to disturb over five acres or remove more than 36,500 tons of material. As noted above, the Reclamation Permit is issued in coordination with the BLM PoO.

Air quality permits are issued by the Bureau of Air Pollution Control (BAPC), while water-related issues (e.g., storm water discharges, sanitary septic systems, and underground injection control) are generally regulated by the Bureau of Water Pollution Control (BWPC). As part of the air permitting process, the project's potential to emit (PTE) is reviewed to determine whether it constitutes a major stationary source. A major stationary source is defined as either one of the sources identified in 40 CFR § 52.21 (including hydrofluoric, sulfuric or nitric acid plants) and which has a PTE of 100 tons or more per year of any regulated pollutant, or any other stationary source which has the PTE of 250 tons or more per year of a regulated pollutant. Based on these thresholds, the MacArthur Copper Project (with its sulfuric acid plant) will likely be classified as a major source of air pollutants that would require a Prevention of Significant Deterioration (PSD) and Class I air quality permit. This permit generally requires enough time to collect ambient air quality data and conduct detailed modeling, and will run concurrent with the development of the Plan of Operations and NEPA process, but would not likely be the critical path for the overall permitting program.

Water appropriations, which will be important to the MacArthur Copper Project given the hydrologic groundwater basin in which the operations area will be located (No. 108 – Mason Valley) which has been “designated” with preferred uses of commercial, industrial, stock water, and mining, are handled through the Nevada Division of Water Resources (NDWR) and the State Engineer’s Office. Quaterra controls approximately 8,700 acre-feet per year (2.8 billion gallons per year) of appropriated water rights for mineral extraction and processing in the Yerington District. Some of these rights date back to the 1950’s when Anaconda operated the Yerington Mine. Preliminary estimates that approximately 40 percent of this water right will be required for the MacArthur Project.

20.2.3 Local Permitting

A Special Use Permit must be acquired from Lyon County; typically a copy of the Plan of Operations is sufficient information for the county to review and issue this permit, although some additional studies may be requested, (e.g., traffic study, noise and lighting studies). However, these would also be addressed in the EIS.

In addition, under county code, Title 10 – Land Use Regulations, Chapter 13 – Lyon County Interim Plan for Federally-Managed Public Lands, Lyon County "*recognizes that the development of its abundant mineral resources is desirable and necessary to the state and the nation. Therefore, it is the policy of Lyon County to encourage mineral exploration and development consistent with custom and culture and to eliminate unreasonable barriers to such exploration and development, except for those that arise naturally from a recognition of secured private property rights and free market conditions.*"

20.3 ENVIRONMENTAL STUDIES

In 2009, Quaterra expanded its Notice-level (NVN-83324) mineral exploration activities on the MacArthur site to include additional drilling as well as bulk sampling and up to 200 acres of additional surface disturbance. An exploration Plan of Operations and Reclamation Permit application was submitted to the BLM and NDEP, respectively, which required analysis under NEPA. A Plan authorization and Permit for Reclamation (Record Number NVN 085212/Reclamation Permit No. 0294) was received in August 2009. As part of this process, a number of environmental baseline studies were performed to characterize the existing conditions within the project boundary. Much of the existing MacArthur Project site includes previously disturbed lands that were part of the Arimetco operations. As such, the baseline updates were focused on undisturbed areas.

Vegetation, sensitive plant, weed inventories and a Class III cultural resources inventory were conducted in 2009. The findings were submitted to the BLM as independent baseline reports. An EA (DOI-BLM-NV-C020-2010-0001-EA) disclosing the potential environmental impacts associated with the expanded MacArthur exploration program was also published in October 2009. A supplemental inventory was carried out in May 2012 for the areas identified for mine facilities in this PEA; only the sand cholla (*Grusonia pulchella*), a BLM special status species, was found within the identified project area. Mitigation of impacts to this species may include simply relocating the individual cacti to other locations. No federally-listed (Threatened and Endangered) wildlife or plant species are known to occur in the project area.

There are no perennial surface water sources within the project area; therefore, foraging or incidental use for BLM sensitive bat species would be limited. Mule deer and pronghorn antelope distribution exist within the project area. There are no known distributions of bighorn sheep (*Ovis Canadensis*), a BLM special status species within the project area.

To date, there have been no hydrogeological investigations or geochemical characterization programs for the mineralized material, waste rock and spent leach materials, performed for the MacArthur Copper Project. These will be important studies to both the Plan of Operations and Water Pollution Control Permit, and are planned to be initiated by Quaterra as soon as practicable. Arimetco installed a supply-water well in 1993 at the eastern end of the property located approximately 4,000 ft east of the proposed mine open pits. In 2011, Quaterra rehabilitated this well and installed a new pump for use at the mine and in the upcoming hydrogeological investigations.

In summary, at this time, there are no known environmental issues that would be expected to materially impact Quaterra's ability to construct or operate the MacArthur Copper Project.

20.4 WASTE AND TAILINGS DISPOSAL

As part of both the State Water Pollution Control Permit and the BLM Plan of Operations (PoO), Quaterra will submit a detailed monitoring plan for monitoring to demonstrate compliance with the approved PoO and other federal or state environmental regulations, to provide early detection of potential problems, and to assist in directing potential corrective actions, should they become necessary. Areas of likely monitoring (particularly water monitoring) in the MacArthur Copper Project include: all process solutions; groundwater upgradient and downgradient of the process facilities (acid leach pad, solution ponds, acid plant, SX/EW); liner leak detection on the process ponds; and cooling tower blowdown.

The site-wide monitoring plan will include a discussion on area water quality; monitoring locations, analytical profiles (NDEP Profiles I, II, or III), and sampling/reporting frequency. Typical monitoring programs include surface- and groundwater quality and quantity, air quality, revegetation, stability, noise levels, and wildlife mortality.

The State of Nevada, through the Bureau of Mining Regulation and Reclamation (BMRR) will require a process fluid management plan as part of the Water Pollution Control Permit. This plan will describe the management of process fluids including the heap leach pad, process ponds, acid plant, and SX/EW plant. The plan will also provide a description of the means to evaluate the conditions in the fluid management system, so as to be able to quantify the available storage capacity for meteoric waters.

The management of non-process (non-contact) stormwater around and between process facilities is a necessary part of the *Nevada General Permit for Stormwater Discharges Associated with Industrial Activity from Metals Mining Activities* (NVR300000), and is typically part of the site-wide Stormwater Pollution Prevention Plan (SWPPP).

20.5 PROJECT PERMITTING REQUIREMENTS

A detailed discussion of the project permitting requirements is provided under Section 20.2 of this report (above). Because of the land position of the project, both state and federal approvals will be required for the MacArthur Copper Project. No mine permits have thus far been acquired, though the appropriate permits for have been obtained under an exploration Plan of Operations. Bonding requirements for the operations are provided under Section 20.7 (below).

20.6 SOCIAL OR COMMUNITY RELATED REQUIREMENTS

Both the BLM NEPA EIS and the Lyon County SUP consider the socioeconomic impacts of a project prior to authorization. The MacArthur Copper Project workforce (including shorter-term construction contractors) will reside mainly in the town of Yerington and the surrounding communities in Lyon County, and possibly Storey, Douglas and Mineral counties as well. The project proponent will coordinate closely with local government and businesses to ensure that the needs of both the community and the workforce are being met. According to the Nevada State Demographer, the population of Lyon County was 51,980 in 2010, up from 34,501 in 2000. This population growth has been slow, but steady, mainly because of an increase in agriculture and mining activity in the area.

An important part of the income of predominantly rural counties in Nevada, like Lyon County, is produced by sales tax and the net proceeds tax on mining activity within the county. Sales tax revenues are collected by the county in which delivery of the goods are taken. For the MacArthur Copper Project, this would be Lyon County. The median household income in the county rose from $40,699 in 1999 to $47,518 in 2009, but is 7% below the current Nevada median income. In 2010, there were less than 500 persons employed in agriculture, forestry, fishing and hunting, and mining in Lyon County.

20.7 MINE CLOSURE REQUIREMENTS

Both the BLM's 43 CFR 3809 and State of Nevada's mine reclamation regulations require closure and reclamation for the MacArthur Project. In addition, any operator who conducts mining operations under an approved BLM PoO or State Reclamation Permit shall furnish a financial surety (bond) in an amount sufficient for stabilizing and reclaiming all areas disturbed by the operations.

In general, buildings and facilities not identified for a post-mining use will be removed from the site during the salvage and site demolition phase. Above-ground concrete will be demolished and removed from site or buried on site. Below-ground concrete will remain and be covered. Residual solution remaining in heap leach pad and process circuit will be recirculated until the rate of flow from these facilities can be passively managed through evaporation from the ponds or a combination of evaporation and infiltration. The heap leach pad and mine waste dumps will be re-contoured to a 3:1 slope, covered with available growth media, and revegetated. Reclamation and closure activities will be conducted concurrently, to the extent practical, to reduce the overall reclamation and closure costs, minimize environmental liabilities, and limit bond exposure.

The revegetation release criteria for reclaimed areas are presented in the "Guidelines for Successful Revegetation for the Nevada Division of Environmental Protection, the Bureau of Land Management, and the U.S.D.A. Forest Service." The revegetation goal is to achieve the permitted plant cover as soon as possible.

Conceptual reclamation and closure methods were used to evaluate the various components of the project to estimate reclamation costs. Quantities were estimated based on the physical layout, geometry and dimensions of the proposed project components of the site plan and facilities layout. These included current conceptual designs for the main project components including the open pit, infrastructure, waste rock facilities, acid leach pad, and process ponds. Equipment and labor costs were also estimated based on current industry rates. A 20-percent contingency was applied to this estimate.

Because the closure activities for the PEA are based on preliminary designs and conceptual approaches, the overall closure cost estimate is considered to be conservative. The closure cost associated with the MacArthur Copper Project is currently estimated to be $92 million ($82 million after salvage). This total is an undiscounted internal cost to reclaim and close the facilities associated with the mining and processing project.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST

21.1.1 Mine Capital Cost

The mine capital cost estimate was provided by Independent Mining Consultants (IMC) and is estimated to be $48 million for the initial capital and $83.6 million in sustaining capital. The initial and sustaining capital consists of the initial fleet of mining and support equipment, with additions to the fleet as necessary. Replacement of some of the equipment fleet is also included in the sustaining capital.

21.1.2 SX/EW Capital Cost

The total installed capital cost for the SX/EW and ancillary facilities is estimated to be $114.3 million and is summarized by process area in Table 21-1 below.

Table 21-1: SX/EW Capital Cost

	Direct Field Cost	$ 000
000	Plant General	$1,039
300	Heap Leach Pad	$17,368
350	Solutin Ponds	$7,767
400	Solvent Extraction	$10,918
500	Tank Farm	$10,359
600	Electrowinning	$16,263
650	Water Systems	$1,309
700	Main Substation	$2,499
750	Transmission Line	$48
800	Reagents	$1,160
900	Ancillary Facilities	$3,963
		$72,693
	Indirect Cost	
	Mobilization	$727
	Lyon County Sales Tax	$3,740
	Freight	$4,743
	EPCM	$12,932
	Vendor Supervision & Commissioning	$423
	Contingency (20%)	$19,052
	Total Direct and Indirect Capital Cost	$114,310

The initial capital cost is based on recent M3 Engineering & Technology in-house data and previous estimates for SX/EW facilities of similar size. The construction labor was adjusted to Davis-Bacon March 2012 prevailing shop wages in Lyon County, Nevada and construction materials and equipment were factored as required based on PLS flow rates or total copper production to arrive at a total direct capital cost. Indirect capital costs were developed from the direct field cost based on in-house factors. Indirect field mobilization is 1.0% of the direct field cost; Lyon County sales tax is 7.1% of direct field cost less labor; freight is 10% of total equipment and materials; engineering, procurement and construction management (EPCM) is 16.7% of the direct field cost plus the indirect costs listed above; commissioning, commissioning spares, and vendor pre-commissioning and supervision is 3.1% of the plant equipment cost; and a contingency of 20% was applied. The accuracy range of the estimate is -20% to +25%, suitable to support a Preliminary Economic Assessment.

Sustaining capital for the SX/EW and heap leach pad is summarized in Table 21-2 below and includes expansions of the heap leach pad and replacement of the mobile process equipment. Normal maintenance and repair of process equipment is included as part of the operating cost.

Table 21-2: SX/EW Sustaining Capital

Year	SX/EW Mobile Equipment $000	Heap Leach Phases $000	Total $000
3		$5,383	$5,383
5	$50	$31,439	$31,489
6	$50		$50
7	$51	$8,689	$8,740
8	$151		$151
9	$135		$135
10	$316	$5,346	$5,662
11	$250		$250
12	$76	$3,344	$3,420
13	$76	$3,749	$3,825
14	$50	$4,812	$4,862
	$1,205	$62,762	$63,967

The process areas making up the initial capital cost estimate for the MacArthur SX/EW facility are defined below.

Site General (Area 000)

The Site General Area consists of systems or facilities that cross multiple areas of the plant. This area consists of the overall site grading, internal access roads, perimeter fencing, and instrumentation software, licenses and programming. Since the project is adjacent to existing roads and infrastructure, there are no costs required for main access roads to the property.

Heap Leach Pad (Area 300)

This area consists of the site grading for the initial phase heap leach pad, a GCL liner, and a 60 mil LLDPE liner with an anchor trench around the perimeter to anchor the lining. Also included is a perforated HDPE piping network to collect the leach solution and the crushing and placement of over-liner material to protect the collection piping system. Raffinate distribution piping on top of the heap leach pad is also included. The initial heap leach pad covers approximately 123 acres. The development of the remaining area of the heap leach pad (~264 acres) is included in the sustaining capital.

Solution Ponds (Area 350)

This area consists of the pregnant leach solution (PLS) pond, raffinate pond and a storm water collection pond. The PLS and raffinate ponds are double lined with HDPE liners and a leak detection system between liners. The storm water event pond is lined with a single HDPE liner. After a storm event, any solution flowing into the event pond will be pumped to the raffinate pond. All solution ponds are fenced. The solution piping between the heap leach pad and the solvent extraction facility is also included in this area.

Solvent Extraction (Area 400)

This area consists of four extraction settlers and one stripping settler, including primary and secondary mix tanks and agitators. An SX feed tank is included to provide a consistent gravity feed to the extraction settlers. PLS is pumped from the PLS pond to the SX feed tank and then by gravity to the extraction settlers. The copper depleted raffinate will flow by gravity from the extraction settlers to the raffinate pond. A foam fire suppression system is provided in this area for fire suppression.

Tank Farm (Area 500)

This area contains the circulation tanks, pumps, heat exchangers, and filters that support the solvent extraction and electrowinning facilities. Included in this area are the loaded organic tanks, electrolyte circulating tanks, electrolyte filters, electrolyte heat exchangers, and a diluent storage tank. A crud holding tank and crud recovery equipment is also included.

Electrowinning Facility (Area 600)

This area includes the electrowinning tank house with 54 electrolytic cells, cathode and anode electrodes, a transformer / rectifier, a semi-automatic cathode stripping machine and a boiler to provide hot water for cathode washing and for maintaining heat in the circulating electrolyte system. Also included is a tank house ventilation system and scrubber for tank house mist control.

Fresh Water System (Area 650)

This area consists of fresh water wells located on site; a combined fresh water and fire water storage tank; a potable water treatment, storage, and distribution system; and the fire water pumps and distribution system. Fresh water will be pumped from onsite wells to the fresh water storage tank and distributed to all areas of the plant.

Main Electrical Substation (Area 700)

This area consists of the main electrical substation, switchgear and transformers to transform electrical power from the 69 kV main line to intermediate voltages for distribution throughout the plant site. The cost for motor control centers in various areas of the plant, along with the low voltage distribution to the end users, is included in the process area cost.

Power Transmission Line (Area 750)

This area includes a dead end structure and tap at an existing 69 kV main power line and a short overhead line to the plant main electrical substation. The main power line feeds from the Fort Churchill power plant, owned by Nevada Energy, to the town of Yerington and runs adjacent to the SX/EW location. An allowance is provided for minimum upgrades to the main power line in the area of the connection.

Reagents (Area 800)

The reagents area consists of receiving, storage and distribution of reagents used in the SX/EW process. Reagents include the SX extractant, diluent (kerosene) for the organic, cobalt sulfate and guar in the tank house, and mist suppressor (FC-1100) to suppress acid mist in the tank house. Sulfuric acid for the leaching process will be supplied by an onsite sulfuric acid plant discussed in Section 2.1.3.

Ancillary Facilities (Area 900)

Ancillary facilities provided for the project include a change house, a modular guard house and truck scale at the plant entrance, and fuel storage and dispensing facilities for diesel fuel and gasoline. Allowances have also been provided to upgrade existing buildings at the Yerington property to be used for an administration building, warehouse, analytical laboratory, maintenance building, and mine truck shop. Powder and detonator magazines are assumed to be provided by the explosives supplier in exchange for a long term supply contract.

21.1.3 Sulfuric Acid Plant Capital Cost

The base case for the MacArthur Project considers an onsite sulfuric acid plant sized for 640 tons per day of sulfuric acid, in accordance with the expected acid consumption from the leaching operation. The total installed capital cost for the sulfuric acid plant is estimated to be $65.4 million and is summarized in Table 21-3 below.

Table 21-3: Sulfuric Acid Plant Capital Cost

Area		$000
	Direct Cost	
810	Sulfur Unloading	$5,563
820	Acid Plant	$28,065
830	Acid Storage	$1,638
840	Power Plant	$6,155
850	Water Treatment	$1,344
860	Cooling Towers	$921
	Total Direct Cost	**$43,687**
	Indirect Cost	
	Mobilization	$430
	Lyon County Sales Tax	$2,184
	EPCM	$7,732
	Vendor Supervision & Commissioning	$500
	Contingency (20%)	$10,906
	Total Direct and Indirect Cost	**$65,439**

The initial capital cost is based on recent M3 Engineering & Technology in-house data on previous sulfuric acid plant facilities. Construction labor was adjusted for January 2012 Davis-Bacon prevailing shop wages in Lyon County, Nevada. The direct field costs were factored based on the acid plant capacities. As with the other facilities, indirect capital costs were developed from the direct field cost based on in-house factors. Indirect field mobilization is 1.0% of the direct field cost; Lyon County sales tax is 7.1% off direct field cost less labor; freight is 10% of total equipment and materials; engineering, procurement and construction management (EPCM) is 16.7% of the direct field cost plus the indirect costs listed above; commissioning, commissioning spares, and vendor pre-commissioning and supervision is 3.1% of the plant equipment cost; a contingency of 20% was applied. The accuracy range of the estimate is -20% to +25%, suitable to support a Preliminary Economic Assessment.

The capital cost estimate for the sulfuric acid plant and associated facilities is an additional cost to the estimate for the SX/EW facilities. Common facilities already included in the SX/EW estimate are not included in the sulfuric acid plant estimate.

Sustaining capital for the sulfuric acid plant and associated facilities was not estimated; however, an accrual is included in the operating and maintenance cost for major repairs required at intervals of 1.5 to 2 years.

The process areas that make up the direct capital cost for the sulfuric acid plant are defined below:

Sulfur Handling and Unloading (Area 810)

This area consists of facilities to receive molten sulfur by rail tank cars, unloading to receiving pits, and pumping to heated storage. Also included is a direct fired boiler used to generate steam for heating the rail cars for unloading and maintaining heat in the sulfur tanks and pipelines.

These facilities are to be located at an existing rail siding near Wabuska, Nevada, approximately eight miles from the project site. The molten sulfur will be transferred by truck from the unloading location to the molten sulfur storage tanks at the acid plant. Trucks are provided in the capital cost estimate for this transfer.

Sulfuric Acid Plant (Area 820)

The sulfuric acid plant is a double-absorption double-contact plant and consists of the sulfur burning furnace, a waste heat boiler to cool the combustion gases and generate steam, a main gas blower to provide dry combustion air to the sulfur furnace and deliver the combustion gas through the sulfuric acid plant, and a converter with associated heat exchangers to convert SO_2 to SO_3 in the combustion gas. Also included are a drying tower, intermediate absorption tower and final absorption tower with associated acid pump tanks and acid coolers. Final waste gas from the final absorption tower is vented to atmosphere through a final tail gas stack.

Sulfuric Acid Storage (Area 830)

This area consists of sulfuric acid storage tanks for the product acid from the acid plant. Sulfuric acid will be pumped or gravity fed to the raffinate pond and SX plant for use in the leach operation.

Power Plant (Area 840)

The power plant includes the steam turbine generator, main condenser, dump condenser, and steam separator. Cooling water for the steam condensers will be provided by the cooling towers in Area 860. The power generated by the steam turbine will be connected with the main buss at the SX/EW main substation for distribution to all areas of the plant. The breakers and synchronizing equipment to connect to the main buss is included in this area.

Water Treatment (Area 850)

This area includes the water treatment facilities to produce boiler quality water. The treatment facility includes fresh water filters, water softeners, Reverse osmosis (RO) filters, oxygen scavengers, boiler feed water pumps and tanks. Chemical water treatment for the cooling towers is included with the cooling towers in Area 860.

Cooling Towers (Area 860)

This area consists of the cooling towers, fans, circulating water pumps, and a chemical water treatment system. The cooling tower serves the sulfuric acid plant and the power plant. The cooling towers are located adjacent to the sulfuric acid plant.

21.1.4 Exclusions

Owner's costs have been excluded from the capital cost estimate; however, an allowance of $5 million has been included in the financial analysis for typical Owner's costs such as first fills of reagents and lubricants, office equipment, and Owner's pre-production staffing.

The Owner’s costs noted below are not included in the $5.0 million allowance.

a) Environmental permits,
b) Performance bond,
c) Builder’s risk insurance,
d) Land Acquisition,
e) Water rights acquisition,
f) Sunk costs prior to the estimate, and
g) Escalation.

21.2 RECLAMATION COST ESTIMATE

A capital cost estimate was prepared for reclamation of the SX/EW and acid plant site, as well as the leach pad and mine waste dumps. The reclamation cost includes dismantling all buildings and equipment and removing from the site. Above ground concrete will be demolished and removed from site or buried on site. Below ground concrete will remain and be covered. Solution ponds will be drained and the top lining removed to inspect the bottom lining for leaks. If there is evidence of leaks; the bottom lining will be removed, the soil at the leak tested for contamination, and any required remediation performed before the pond can be covered. If no evidence of leaks is found, the top lining can be folded over in place and the pond covered. The ponds will be filled by the push down of the heap leach pad as part of the reclamation cost for the leach pad. A mound is provided over the pond area to prevent storm water from collecting over the pond and migrating into the pond. The plant site will be graded to approximate original contours. Roads will be left in place; however, any asphalt will be removed. The leach pad and mine waste dumps will be re-contoured to a 3.5: 1 slope, with setbacks, covered with reclaimed soil, and hydro-seeded for plant growth. The area of the SX/EW and sulfuric acid plant will also be hydro-seeded for plant growth.

The indirect cost for the reclamation estimate includes field mobilization at 1% of the direct field cost, Lyon County sales tax at 7.1% of direct field cost less labor, and a contingency of 20%. It is assumed the management of the reclamation effort will be by the Owner’s team already on site; therefore, no allowance is provided for EPCM.

The total cost for reclamation of the site is estimated to be $92.2 million and is summarized by process area in Table 21-4 below. The cost to re-contour the mine waste dumps is included in Area 300 with the cost to re-contour the heap leach pad.

Table 21-4: Reclamation Cost Estimate

Direct Field Cost		$000
000	Plant General	$53
300	Heap leach	$58,067
350	Solution Ponds	$868
400	Solvent Extraction	$1,772
500	Tank Farm	$1,655
600	Electrowinning	$2,339
650	Water Systems	$137
700	Main Substation	$237
750	Overhead Transmission line	$8
800	Reagents & Acid Plant	$8,840
900	Ancillaries	$115
	Total Direct Field Cost	**$74,091**
Indirect Costs		
	Mobilization	$741
	Lyon County Sales Tax	$1,968
	Contingency (20%)	$15,360
	Total Indirect Cost	**$92,159**

An allowance of $9.2 million was provided for equipment and materials salvage to offset the reclamation cost. Total reclamation cost with the salvage deduct is $82.96 million, which occurs in years 19 through 22.

21.3 OPERATING COST

The overall annual average operating cost for the MacArthur Copper operation is $1.89 per pound of copper and is summarized in Table 21-5 below. The costs include the mine operations, SX/EW facility, the sulfuric acid plant, general administrative expenses and the cost of transportation to ship the product cathode to market. The sulfuric acid operating cost represents the cost of acid in the table below.

Table 21-5: MacArthur SX/EW and Mine Operating Cost

	$ / lb. Cu
Mine	$0.99
SX/EW	$0.38
Cost of Acid	$0.35
General & Administrative	$0.12
Transportation	$0.05
Sub-Total	**$1.89**

21.3.1 Mine Operating Cost

The mine operating costs were provided by Independent Mining Consultants (IMC) based on a selected fleet of mine and support equipment for the 18 year life of the MacArthur mine. The average life of mine operating cost is $1.44 per ton of material mined or $2.74 per ton of mineralized material mined. The mine operating cost, as a cost per pound of copper recovered, is $0.99 per pound of copper produced. These costs include drilling, blasting, loading, hauling, and road and dump maintenance.

21.3.2 SX/EW Operating Cost

The operating cost estimate for the MacArthur SX/EW facilities is estimated to be $0.38 per pound of copper produced and include labor, reagents, electrical power, maintenance parts and services and operating supplies and services. The costs are summarized in Table 21-6 below.

Table 21-6: SX/EW Operating Cost

	$ / lb. Cu
Labor	$0.08
Reagents	$0.10
Electrical Power	$0.14
Maintenance Parts & Services	$0.03
Operating Supplies & Services	$0.03
Sub-Total	**$0.38**

The average annual labor cost for the SX/EW area is $3.5 million based on a staffing plan of 43 operating and maintenance personnel. The average wage rate for the SX/EW staff is $58,600 per year plus 40% for fringe benefits. The labor staffing consists of four supervisory personnel, twenty-two operating personnel and seventeen maintenance personnel.

The average annual cost of reagents for this area is $4.15 million and includes extractant, diluent, cobalt sulfate, and guar. The cost of acid is noted separately and is based on the operating costs of the onsite sulfuric acid plant discussed in Section 21.3.3 below. The annual consumption of SX/EW reagents and cost is shown in Table 21-7 below.

Table 21-7: Reagent Cost

	Consumption	$ / lb. Cu
Extractant	106 gallons / day	$0.032
Diluent	1,415 gallons / day	$0.053
Cobalt Sulfate	163 pounds /day	$0.007
Guar	29.5 pounds / day	$0.005
Total		**$0.097**

The annual power cost for the SX/EW facility is $5.7 million, or $0.14 per pound of copper recovered, and is based on a power consumption of approximately 2.1 kWh per lb. of cathode copper and a cost of power of $0.065 /kWh.

The annual cost for maintenance parts and services is approximately $1.4 million and is based on 7% of the SX/EW equipment cost. The annual cost of operating supplies and services is $1.2 million and is based on $0.03 / lb. of cathode copper produced.

21.3.3 Sulfuric Acid Plant Operating Cost

The annual operating cost for the sulfuric acid plant, power plant and associated facilities is $14.4 million or $62.04 per ton of acid and $0.35 per pound of copper produced. The estimated cost for sulfuric acid delivered to site by rail from the west coast is estimated to be $140 per ton of acid compared to the cost to manufacture on site at $62 per ton of acid. The acid plant operating costs are summarized in Table 21-8 below.

Table 21-8: Sulfuric Acid Plant Operating Cost

	Annual Cost Cost $000	**Cost / ton Acid**	**Cost / lb. Copper**
Labor	$1,401	$6.03	$0.03
Reagents (Sulfur)	$9,512	$40.95	$0.23
Fuels (Propane)	$634	$2.73	$0.02
Power (Credit)	($2,624)	($11.30)	($0.06)
Maintenance	$3,846	$16.56	$0.09
Operating Supplies	$1,643	$7.07	$0.04
	$14,412	$62.04	$0.35

The labor cost is based on a staffing plan of 10 operators and 7 maintenance personnel. The operating crew consists of a general foreman and technician on day shift, 5 days per week and a control room operator and field operator each shift seven days per week. The average annual wage rate for acid plant personnel is $58,800 plus 40% fringe benefits. The wage rate is slightly higher in the acid plant than the SX/EW facility due to the higher mix of higher pay positions in the acid plant.

Reagents needed in the sulfuric acid plant includes elemental sulfur (molten) for acid production and water treatment chemicals for the cooling tower and boiler feed water systems. One ton of sulfur will produce a little over 3 tons of sulfuric acid. Based on an annual requirement of 232,300 tons of sulfuric acid, approximately 77,400 tons of elemental sulfur will be required. The cost of sulfur used in the estimate is $125 per ton delivered molten to site and is based on the average cost for U. S. West Coast sulfur over the last five years of available published information with freight allowed to the project site. An allowance of $30,000 per year was used for the water treatment chemicals.

Propane is assumed as the fuel to fire the steam boiler at the sulfur unloading area and is based on a boiler sized for 5 million BTU/hr and a heat value for Propane of 92,500 BTU/gallon. It is assumed that the boiler would operate 16 hours per day. The cost of Propane was set at $2.00 per gallon, the average of current wholesale and residential cost.

The power requirement to produce sulfuric acid was estimated to be 2,300 kW or $1.3 million annually at the cost of power of $0.065 per kWh. The turbine generator is expected to produce approximately 6,900 kW of power at a value of $3.9 million annually at the same cost of power. The excess power can be used to displace purchased power in the SX/EW facility or sold back to the power company. The net power credit is $2.6 million annually. The power consumption and power produced were factored from existing in-hours data on similar sulfur burning acid plants.

Annual maintenance cost for the sulfuric acid plant was estimated at 4% of the installed cost of the acid plant, or $2.6 million. The annual maintenance for the power plant was estimated to be $0.02 / kWh or $1.2 million. The total annual maintenance cost for the acid plant and power plant is $3.8 million. The maintenance cost includes an accrual for major repairs that will occur at intervals of 1.5 to 2 years.

Operating supplies and services was estimated at 2.5% if the total installed cost of the acid plant and power plant or $1.6 million annually.

21.3.4 General and Administrative Costs

The total annual general and administrative cost for the facility is $5.0 million, or $0.12 per pound of cathode copper produced. The G&A labor is the largest component at $2.2 million per year, based on a staffing of 23 employees. Allowances were made for non-labor components for G&A expenses, which includes office supplies, fuels, communications, small vehicle maintenance, claims assessments, legal and auditing, insurance, travel, meals and expenses, community relations, recruiting and relocation expenses, and janitorial services. The breakdown of G&A cost and labor detail is shown in Table 21-9 General & Administrative Cost Summary and Table 21-10 General & Administrative Labor Cost Summary.

Table 21-9: General & Administrative Cost Summary

Cathode Produced (lbs.)	**41,500,000**	
	Total	
Cost Item	**Annual Cost - $**	**$/lb. Cathode**
Labor & Fringes	$2,186,800	$0.053
Accounting (excluding labor)	$25,000	$0.001
Safety & Environmental (excluding labor)	$25,000	$0.001
Human Resources (excluding labor)	$25,000	$0.001
Security (excluding labor)	$25,000	$0.001
Office Operating Supplies and Postage	$40,000	$0.001
Fuel/ Propane	$25,000	$0.001
Communications	$70,000	$0.002
Small Vehicles	$125,000	$0.003
Claims Assessment	$10,000	$0.000
Legal & Audit	$300,000	$0.007
Consultants	$250,000	$0.006
Janitorial Services	$50,000	$0.001
Insurances	$1,000,000	$0.024
Taxes - property	$500,000	$0.012
Subs, Dues, PR, and Donations	$60,000	$0.001
Travel, Lodging, and Meals	$150,000	$0.004
Recruiting/Relocation	$125,000	$0.003
Total General & Administrative Cost	**$4,991,800**	**$0.120**

Table 21-10: General & Administrative Labor Cost Summary

		Per Person		
Department and Position	**Number Of Personnel**	**Total Annual Direct Salary**	**Total Annual Benefits 40%**	**Total Annual Salary**
General & Administrative				
General Manager	1	$200,000	$80,000	$280,000
Administrative Assistant	1	$34,000	$13,600	$47,600
			$0	
Controller	1	$120,000	$48,000	$168,000
Accountant	1	$60,000	$24,000	$84,000
Accounts Payable	1	$34,000	$13,600	$47,600
Purchasing Manager	1	$120,000	$48,000	$168,000
Purchasing Agent	1	$55,000	$22,000	$77,000
Warehouseman	2	$40,000	$16,000	$112,000
IT Technician	2	$60,000	$24,000	$168,000
HR Manager	1	$120,000	$48,000	$168,000
HR Specialist	1	$40,000	$16,000	$56,000
HR Administrative Assistant	1	$34,000	$13,600	$47,600
Safety Manager	1	$120,000	$48,000	$168,000
Environmental Manager	1	$120,000	$48,000	$168,000
Safety Specialist	2	$55,000	$22,000	$154,000
Hydrological Engineer	0	$60,000	$24,000	$0
Environmental Technician	1	$55,000	$22,000	$77,000
Security Guard	4	$35,000	$14,000	$196,000
Total General and Administrative Labor Cost	**23**			**$2,186,800**

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

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

This PEA is preliminary in nature and includes discussion of mineral resources including inferred mineral resources that are too speculative geologically to have economic considerations applied to them. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

22.2 MINE PRODUCTION STATISTICS

Mine production is reported as mineralized material and overburden from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report. The life of mine mineralized material quantities and mineralized material grades are presented in Table 22-1 below.

Table 22-1: Life of Mine Mineralized Material, Waste Quantities, and Mineralized Material Grade

	Tons (kt)	**Copper %**
Oxide Mineralized Material – Main Pit	132,756	0.20
Oxide Mineralized Material – Other Areas	52,537	0.19
Mixed Mineralized Material	85,588	0.24
Total Mineralized Material	**270,881**	**0.21**
Waste	244,948	

22.3 HEAP LEACH PAD AND SX/EW PRODUCTION STATISTICS

The mineralized material will be processed using heap leaching and SX/EW plant recovery technology to produce copper cathode. Below are the recoveries assigned to each of the mineralized material types:

Oxide Mineralized Material – Main Pit	70.0%
Oxide Mineralized Material – Other Areas	65.0%
Mixed Mineralized Material	60.0%

The estimated life of mine metal production is estimated to be 747.7 million pounds.

22.3.1 Cathode Shipping

The cost for cathode shipping of $0.05 per pound of copper is included in cash operating costs.

22.4 CAPITAL EXPENDITURE

22.4.1 Initial Capital

The financial indicators have been determined with 100% equity financing of the initial capital. Any acquisition cost or expenditures prior to start of mine pre-development have been treated as "sunk" cost and have not been included in the analysis.

The total initial capital carried in the financial model for new construction is expended over a 3 year period. The initial capital includes Owner's costs and contingency. The cash flow will be expended in the years before production and a small amount carried over into the first production year.

The initial capital is presented in Table 22-2 below.

Table 22-2: Initial Capital

	$ in millions
Mining	$48.0
SXEW Plant	$114.3
Sulfuric Acid Plant	$65.4
Owner's Cost	$5.0
Total	**$232.7**

22.4.2 Sustaining Capital

A schedule of capital cost expenditures during the production period was estimated and included in the financial analysis under the category of sustaining capital. The total life of mine sustaining capital is estimated to be $147.6 million. This capital will be expended during a 14 year period during the 18 year mine life.

22.4.3 Working Capital

A 15 day delay of receipt of revenue from sales is used for accounts receivables. A delay of payment for accounts payable of 30 days is also incorporated into the financial model. In addition, working capital allowance of $1.1 million for plant consumable inventory is estimated in year -1 and year 1. All the working capital is recaptured at the end of the mine life and the final value of these accounts is $0.

22.4.4 Salvage Value

An allowance based on 10% of the total capital equipment cost, including mine equipment for salvage value at the end of the mine life has been included and is estimated at $9.2 million.

22.5 REVENUE

Annual revenue is determined by applying estimated copper price to the annual payable metal estimated for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The revenue is the gross value of payable metals sold before treatment charges and transportation charges. The copper sales price used in the evaluation is based on the three year historical price as of May 1, 2012, which is consistent with Securities and Exchange Commission (SEC) guidelines.

Copper $3.48/pound

22.6 OPERATING COST

The average Cash Operating Cost over the life of the mine is estimated to be $1.89 per pound of copper, excluding the cost of the capitalized pre-stripping. Cash Operating Cost includes mine operations, process plant operations, general administrative cost, and shipping charges. Table 22-3 below shows the estimated operating cost by area per pound of copper.

Table 22-3: Life of Mine Operating Cost

Operating Cost	$/lb
Mine	$0.99
SXEW Plant	$0.38
Sulfuric Acid Cost	$0.35
General Administration	$0.12
Transportation	$0.05
Total Operating Cost	**$1.89**

22.7 TOTAL CASH COST

The average Total Cash Cost over the life of the mine is estimated to be $2.04 per pound of copper. Total Cash Cost is the Operating Cost plus royalty, salvage value, reclamation and closure costs.

22.7.1 Royalty

The royalty charges for the life of the mine are estimated at $31.3 million. There are two royalties and they are based on a % of net smelter return. The net smelter return is calculated as gross revenues, less SX/EW cost (excluding sulfuric acid cost) and transportation cost. The royalties are defined as follows:

- Arimetco royalty which is based on 2% of the net smelter return and has a cap of $7.5 million.
- A 3rd Party royalty which is based on a payment of $1.0 million at the start of production plus 1% of the net smelter return for life of mine.

22.7.2 Reclamation and Closure

An estimate for reclamation and closure was included in the cash flow of $92.2 million.

22.8 DEPRECIATION AND DEPLETION

22.8.1 Depreciation

Depreciation is calculated using the MACRS method starting with first year of production. The initial capital and sustaining capital used a 7 year life. The last year of production is the catch-up year if the assets are not fully depreciated by that time.

22.8.2 Depletion

The percentage depletion method was used in the evaluation. It is determined as a percentage of gross income from the property, not to exceed 50% of taxable income before the depletion deduction. The gross income from the property is defined as metal revenues minus downstream costs from the mining property (smelting, refining and transportation). Taxable income is defined as gross income minus operating expenses, overhead expenses, and depreciation and state taxes. The estimated depletion deduction for income tax use is $346.2 million for the life of the mine.

22.9 TAXATION

22.9.1 Income Tax and Mineral Tax

Taxable income for income tax purposes is defined as metal revenues minus operating expenses, royalty, property and severance taxes, reclamation and closure expense, depreciation and depletion. The Federal income tax rate is 35% in accordance with IRS Publication 542.

Federal income taxes were calculated on the taxable income described above using the federal tax rate. Nevada does not have a state corporate income tax.

Federal income taxes paid are estimated to be $151.7 million.

In addition to the Federal income tax, a Nevada mineral tax has also been estimated at $27.4 million. The Nevada Net Proceeds of Minerals Tax is an ad valorem property tax assessed on minerals mined or produced in Nevada when they are sold or removed from the state.

22.10 PROJECT FINANCING

The project was evaluated on an unleveraged and un-inflated basis.

22.11 NET INCOME AFTER TAX

The operating margin before tax is $840.87 million, including depreciation, and the net income after tax amounts to $514.2 million.

22.12 NPV AND IRR

The economic analysis indicates that the project has an NPV of $201.6 million at a discount rate of 8%, an Internal Rate of Return (IRR) of 24.2% with a payback period of 3.1 years after taxes.

Table 22-4: Economic Indicators

	Before Taxes $000	After Taxes $000
NPV @ 8%	284,138	201,576
IRR %	29.3%	24.2%
Payback, years	2.7	3.1

22.13 SENSITIVITIES

Table 22-5 compares the base case project after tax financial indicators with the financial indicators when different variables are applied. By comparing the results it can be seen that the copper price has the most impact on the project followed by the operating cost and then by the initial capital cost. This data is represented in graph form in Figure 22-1. The discounted cash flow model for the project is shown in Table 22-6 at the end of this section.

Table 22-5: Sensitivity Analysis

		NPV @ 8%	IRR	Payback
Base Case		$201,576	24.2%	3.1
Copper Price	+20%	$377,172	35.2%	2.3
	+10%	$290,768	29.9%	2.7
	-10%	$107,566	17.4%	3.7
	-20%	$9,797	9.0%	8.4
Capital Cost	+20%	$167,445	19.4%	3.6
	+10%	$184,561	21.6%	3.4
	-10%	$218,571	27.3%	2.8
	-20%	$234,567	31.0%	2.5
Operating Cost	+20%	$107,289	17.8%	3.5
	+10%	$156,080	21.3%	3.3
	-10%	$245,478	26.8%	2.9
	-20%	$286,955	29.1%	2.8

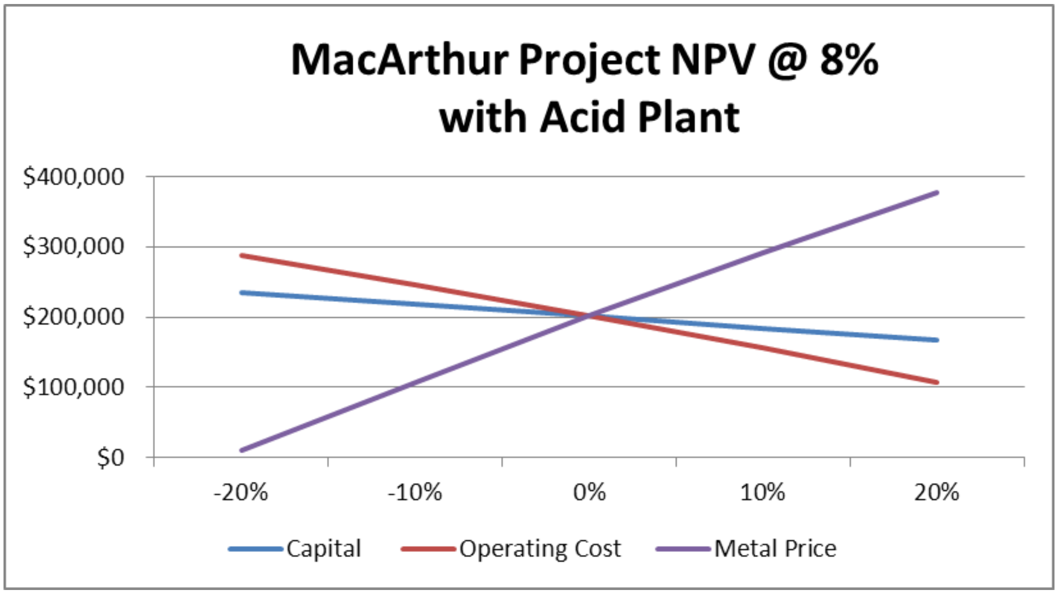


Figure 22-1: MacArthur Project NPV Sensitivities

Table 22-6: Discounted Cash Flow Model

Base Case with Acid Plant	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Mine Operations																										
Oxide Material - Main Pit (kt)	132,756	-	399	15,000	15,000	14,204	14,228	9,306	11,794	8,181	4,369	4,405	1,190	-	1,202	3,440	3,670	4,400	4,639	8,809	8,489	31	-	-	-	-
Copper Grade %	0.198%	0.000%	0.241%	0.210%	0.239%	0.210%	0.210%	0.199%	0.190%	0.180%	0.180%	0.190%	0.190%	0.000%	0.163%	0.210%	0.180%	0.160%	0.168%	0.179%	0.170%	0.180%	0.000%	0.000%	0.000%	0.000%
Contained Copper (klbs)	525,765	-	1,920	62,851	71,590	59,695	59,721	37,013	44,751	29,452	15,728	16,739	4,522	-	3,926	14,482	13,198	14,071	15,628	31,505	28,863	112	-	-	-	-
Mixed Material (kt)	85,588	-	-	-	-	6	234	4,654	57	1,096	4,271	5,824	9,473	12,617	10,957	3,698	6,206	6,507	7,253	5,773	6,511	451	-	-	-	-
Copper Grade %	0.244%	0.000%	0.000%	0.000%	0.000%	0.240%	0.220%	0.230%	0.190%	0.200%	0.240%	0.230%	0.240%	0.240%	0.270%	0.320%	0.270%	0.270%	0.220%	0.230%	0.200%	0.180%	0.000%	0.000%	0.000%	0.000%
Contained Copper (klbs)	418,012	-	-	-	-	29	1,030	21,408	217	4,384	20,501	26,790	45,470	60,562	59,168	23,667	33,512	35,138	31,913	26,556	26,044	1,624	-	-	-	-
Oxide Material - Other Areas (kt)	52,537	-	-	-	-	790	538	1,040	3,149	5,723	6,360	4,771	4,337	2,383	2,841	7,862	5,124	4,093	3,108	418	-	-	-	-	-	-
Copper Grade %	0.189%	0.000%	0.000%	0.000%	0.000%	0.172%	0.189%	0.189%	0.213%	0.200%	0.190%	0.179%	0.160%	0.180%	0.210%	0.209%	0.179%	0.170%	0.180%	0.140%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Contained Copper (klbs)	198,224	-	-	-	-	2,721	2,034	3,936	13,400	22,904	24,166	17,126	13,888	8,579	11,957	32,888	18,361	13,904	11,189	1,170	-	-	-	-	-	-
Waste (kt)	244,948	-	101	4,042	2,174	5,000	5,000	5,000	15,000	20,000	20,000	20,000	20,000	20,000	17,403	18,156	20,000	17,261	12,749	15,256	7,612	194	-	-	-	-
Total Material Mined (kt)	515,829	-	500	19,042	17,174	20,000	20,000	20,000	30,000	35,000	35,000	35,000	35,000	35,000	32,403	33,156	35,000	32,261	27,749	30,256	22,612	676	-	-	-	-
SXEW Operations																										
Oxide Material - Main Pit (kt)	132,756	-	-	15,399	15,000	14,204	14,228	9,306	11,794	8,181	4,369	4,405	1,190	-	1,202	3,440	3,670	4,400	4,639	8,809	8,489	31	-	-	-	-
Copper Grade	0.198%	0.000%	0.000%	0.210%	0.239%	0.210%	0.210%	0.199%	0.190%	0.180%	0.180%	0.190%	0.190%	0.000%	0.163%	0.210%	0.180%	0.160%	0.168%	0.179%	0.170%	0.180%	0.000%	0.000%	0.000%	0.000%
Contained Copper (klbs)	525,765	-	-	64,771	71,590	59,695	59,721	37,013	44,751	29,452	15,728	16,739	4,522	-	3,926	14,482	13,198	14,071	15,628	31,505	28,863	112	-	-	-	-
Recovery	70.0%	0.0%	0.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	0.0%	0.0%	0.0%	0.0%
Recovered Copper (klbs)	368,036	-	-	45,340	50,113	41,786	41,805	25,909	31,326	20,616	11,010	11,717	3,165	-	2,748	10,137	9,239	9,850	10,939	22,054	20,204	78	-	-	-	-
Mixed Material (kt)	85,588	-	-	-	-	6	234	4,654	57	1,096	4,271	5,824	9,473	12,617	10,957	3,698	6,206	6,507	7,253	5,773	6,511	451	-	-	-	-
Copper Grade	0.244%	0.000%	0.000%	0.000%	0.000%	0.240%	0.220%	0.230%	0.190%	0.200%	0.240%	0.230%	0.240%	0.240%	0.270%	0.320%	0.270%	0.270%	0.220%	0.230%	0.200%	0.180%	0.000%	0.000%	0.000%	0.000%
Contained Copper (klbs)	418,012	-	-	-	-	29	1,030	21,408	217	4,384	20,501	26,790	45,470	60,562	59,168	23,667	33,512	35,138	31,913	26,556	26,044	1,624	-	-	-	-
Recovery	60.0%	0.0%	0.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	0.0%	0.0%	0.0%	0.0%
Recovered Copper (klbs)	250,807	-	-	-	-	17	618	12,845	130	2,630	12,300	16,074	27,282	36,337	35,501	14,200	20,107	21,083	19,148	15,933	15,626	974	-	-	-	-
Oxide Material - Other Areas (kt)	52,537	-	-	-	-	790	538	1,040	3,149	5,723	6,360	4,771	4,337	2,383	2,841	7,862	5,124	4,093	3,108	418	-	-	-	-	-	-
Copper Grade	0.189%	0.000%	0.000%	0.000%	0.000%	0.172%	0.189%	0.189%	0.213%	0.200%	0.190%	0.179%	0.160%	0.180%	0.210%	0.209%	0.179%	0.170%	0.180%	0.140%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Contained Copper (klbs)	198,224	-	-	-	-	2,721	2,034	3,936	13,400	22,904	24,166	17,126	13,888	8,579	11,957	32,888	18,361	13,904	11,189	1,170	-	-	-	-	-	-
Recovery	65.0%	0.0%	0.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	0.0%	0.0%	0.0%	0.0%
Recovered Copper (klbs)	128,846	-	-	-	-	1,769	1,322	2,559	8,710	14,888	15,708	11,132	9,027	5,576	7,772	21,377	11,934	9,038	7,273	761	-	-	-	-	-	-
Payable Metals																										
Payable Copper (klbs)	747,689	-	-	45,340	50,113	43,572	43,745	41,313	40,166	38,134	39,018	38,923	39,475	41,913	46,021	45,714	41,281	39,970	37,360	38,748	35,830	1,052	-	-	-	-
Income Statement ($000)																										
Metal Prices																										
Copper ($/lb.)	$3			$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$3	$0	$0	$0	$0
Revenues																										
Copper Revenue ($ 000)	$2,601,957			$157,782	$174,392	$151,632	$152,232	$143,768	$139,776	$132,708	$135,783	$135,454	$137,373	$145,858	$160,153	$159,086	$143,657	$139,096	$130,013	$134,843	$124,689	$3,662	$0	$0	$0	$0
Total Revenues	$2,601,957	$0	$0	$157,782	$174,392	$151,632	$152,232	$143,768	$139,776	$132,708	$135,783	$135,454	$137,373	$145,858	$160,153	$159,086	$143,657	$139,096	$130,013	$134,843	$124,689	$3,662	$0	$0	$0	$0
Operating Cost																										
Mine Operations	$743,021	$0	$3,390	$27,042	$25,517	$29,000	$29,400	$29,200	$41,100	$48,500	$49,200	$49,900	$47,800	$47,800	$45,644	$46,802	$49,200	$46,454	$41,912	$45,701	$37,134	$2,325	$0	$0	$0	$0
SXEW Plant	$282,315	$0	$0	$16,871	$18,127	$16,406	$16,451	$15,811	$15,509	$14,974	$15,207	$13,419	$15,327	$15,969	$14,581	$16,969	$15,802	$15,458	$14,771	$15,136	$14,368	$1,159	$0	$0	$0	$0
Acid Cost	$260,230	$0	$0	$14,330	$13,959	$14,082	$14,042	$14,120	$14,447	$14,847	$14,945	$14,699	$14,632	$14,329	$14,400	$15,178	$14,754	$14,594	$14,441	$14,024	$13,959	$449	$0	$0	$0	$0
General Administration	$91,100	$0	$0	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$4,992	$1,248	$0	$0	$0	$0
Transportation	$37,384	$0	$0	$2,267	$2,506	$2,179	$2,187	$2,066	$2,008	$1,907	$1,951	$1,946	$1,974	$2,096	$2,301	$2,286	$2,064	$1,999	$1,868	$1,937	$1,792	$53	$0	$0	$0	$0
Total Operating Cost	$1,414,051	$0	$3,390	$65,502	$65,100	$66,658	$67,072	$66,189	$78,057	$85,220	$86,295	$84,956	$84,724	$85,185	$81,918	$86,227	$86,812	$83,496	$77,983	$81,790	$72,244	$5,233	$0	$0	$0	$0
Property Tax	$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
Royalty - Arimetco	$7,500	$0	$0	$2,773	$3,075	$1,652	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Royalty - 3rd Party	$23,823	$0	$0	$2,386	$1,538	$1,330	$1,336	$1,259	$1,223	$1,158	$1,186	$1,201	$1,201	$1,278	$1,433	$1,398	$1,258	$1,216	$1,134	$1,178	$1,085	$25	$0	$0	$0	$0
Salvage Value	($9,196)				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	($9,196)	$0
Reclamation & Closure	$92,159	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$23,040	$23,040	$23,040	$23,040	$0
Total Production Cost	$1,528,336	$0	$3,390	$70,661	$69,713	$69,640	$68,408	$67,448	$79,279	$86,378	$87,481	$86,157	$85,925	$86,463	$83,350	$87,626	$88,070	$84,712	$79,117	$82,968	$73,330	$28,297	$23,040	$23,040	$13,844	$0
Operating Income	$1,073,620	$0	($3,390)	$87,121	$104,679	$81,992	$83,823	$76,321	$60,497	$46,330	$48,301	$49,297	$51,448	$59,395	$76,802	$71,460	$55,587	$54,384	$50,896	$51,875	$51,360	($24,635)	($23,040)	($23,040)	($13,844)	$0
Initial Capital Depreciation	$232,749			$33,260	$57,000	$40,708	$29,070	$20,784	$20,761	$20,784	$10,381	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Mine Development	$0			$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital Depreciation	$147,568			$314	$910	$2,162	$2,685	$8,561	$13,532	$14,444	$17,271	$15,392	$14,082	$14,383	$11,993	$9,225	$7,673	$5,305	$3,725	$2,676	$1,484	$1,749	$0	$0	$0	$0
Total Depreciation	$380,317	$0	$0	$33,574	$57,911	$42,870	$31,755	$29,345	$34,294	$35,229	$27,651	$15,392	$14,082	$14,383	$11,993	$9,225	$7,673	$5,305	$3,725	$2,676	$1,484	$1,749	$0	$0	$0	$0
Net Income After Depreciation	$693,304	$0	($3,390)	$53,546	$46,768	$39,122	$52,068	$46,975	$26,204	$11,101	$20,650	$33,904	$37,366	$45,012	$64,809	$62,236	$47,914	$49,079	$47,171	$49,200	$49,875	($26,385)	($23,040)	($23,040)	($13,844)	$0
Income Taxes & Minerals Tax	$179,087		$0	$11,145	$9,821	$8,216	$12,054	$10,538	$5,503	$2,331	$4,337	$7,120	$7,847	$9,672	$16,543	$15,609	$10,905	$11,593	$11,335	$11,863	$12,656	$0	$0	$0	$0	$0
Net Income After Taxes	$514,217	$0	($3,390)	$42,401	$36,947	$30,906	$40,014	$36,438	$20,701	$8,770	$16,314	$26,785	$29,519	$35,340	$48,266	$46,627	$37,009	$37,486	$35,836	$37,337	$37,219	($26,385)	($23,040)	($23,040)	($13,844)	$0

Base Case with Acid Plant	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Cash Flow																										
Operating Income	$1,073,620	$0	($3,390)	$87,121	$104,679	$81,992	$83,823	$76,321	$60,497	$46,330	$48,301	$49,297	$51,448	$59,395	$76,802	$71,460	$55,587	$54,384	$50,896	$51,875	$51,360	($24,635)	($23,040)	($23,040)	($13,844)	$0
Working Capital																										
Accounts Receivable (15 days)	$0	$0	$0	($6,484)	($683)	$935	($25)	$348	$164	$290	($126)	$14	($79)	($349)	($587)	$44	$634	$187	$373	($198)	$417	$4,974	$150	$0	$0	$0
Accounts Payable (30 days)	$0	$0	$279	$5,105	($33)	$128	$34	($73)	$975	$589	$88	($110)	($19)	$38	($269)	$354	$48	($273)	($453)	$313	($785)	($5,508)	($430)	$0	$0	$0
Inventory - Parts, Supplies	$0	$0	($440)	($660)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$1,100	$0	$0	$0	$0
Total Working Capital	$0	$0	($161)	($2,039)	($716)	$1,063	$9	$275	$1,139	$879	($38)	($97)	($98)	($311)	($856)	$398	$682	($85)	($80)	$114	($367)	$566	($280)	$0	$0	$0
Capital Expenditures																										
Initial Capital																										
Mine	$48,000	$2,400	$43,200	$2,400	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
SXEW Plant	$114,310	$5,716	$102,879	$5,716	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Acid Plant	$65,439	$3,272	$58,895	$3,272	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owners Cost	$5,000	$250	$4,500	$250	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Land Acquisition	$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
Mine Development	$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
Sustaining Capital																										
Mine	$83,600	$0	$0	$2,200	$2,600	$2,600	$0	$15,000	$5,000	$18,800	$19,400	$1,100	$8,100	$8,800	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
SXEW Plant	$63,968	$0	$0	$0	$0	$5,383	$0	$31,489	$50	$8,740	$151	$135	$5,662	$250	$3,420	$3,825	$4,862	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Capital Expenditures	$380,317	$11,637	$209,474	$13,837	$2,600	$7,983	$0	$46,489	$5,050	$27,540	$19,551	$1,235	$13,762	$9,050	$3,420	$3,825	$4,862	$0	$0	$0	$0	$0	$0	$0	$0	$0
Cash Flow before Taxes	$693,304	($11,637)	($213,026)	$71,244	$101,363	$75,072	$83,833	$30,107	$56,587	$19,669	$28,712	$47,965	$37,588	$50,034	$72,526	$68,034	$51,407	$54,299	$50,816	$51,990	$50,992	($24,069)	($23,319)	($23,040)	($13,844)	$0
Cumulative Cash Flow before Taxes		($11,637)	($224,663)	($153,419)	($52,056)	$23,017	$106,849	$136,956	$193,543	$213,212	$241,924	$289,889	$327,478	$377,512	$450,038	$518,072	$569,478	$623,777	$674,594	$726,583	$777,576	$753,506	$730,187	$707,147	$693,304	$693,304
				$1	$1	$1	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0									
Taxes																										
Income Taxes	$179,087	$0	$0	$11,145	$9,821	$8,216	$12,054	$10,538	$5,503	$2,331	$4,337	$7,120	$7,847	$9,672	$16,543	$15,609	$10,905	$11,593	$11,335	$11,863	$12,656	$0	$0	$0	$0	$0
Cash Flow after Taxes	$514,217	($11,637)	($213,026)	$60,099	$91,542	$66,857	$71,778	$19,569	$51,084	$17,337	$24,376	$40,845	$29,741	$40,362	$55,982	$52,425	$40,502	$42,706	$39,481	$40,127	$38,337	($24,069)	($23,319)	($23,040)	($13,844)	$0
Cumulative Cash Flow after Taxes		($11,637)	($224,663)	($164,564)	($73,022)	($6,165)	$65,613	$85,182	$136,266	$153,604	$177,979	$218,825	$248,566	$288,928	$344,911	$397,336	$437,838	$480,544	$520,025	$560,152	$598,489	$574,419	$551,100	$528,060	$514,217	$514,217
				$1	$1	$1	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Economic Indicators before Taxes																										
NPV @ 0%	0%	$693,304																								
NPV @ 5%	5%	$395,451																								
NPV @ 8%	8%	$284,138																								
IRR		29.3%																								
Payback	Years	2.7																								
Economic Indicators after Taxes																										
NPV @ 0%	0%	$514,217																								
NPV @ 5%	5%	$288,066																								
NPV @ 8%	8%	$201,576																								
IRR		24.2%																								
Payback	Years	3.1																								

23 ADJACENT PROPERTIES

The only adjacent property according to CIM definitions is the Ann Mason Deposit. The data below is taken from an NI-43-101 Technical Report, March 2012. The authors of this Technical Report have not verified the information presented below. Entrée Gold holds the rights to the Ann Mason Deposit that is adjacent to the southwest of Quaterra's MacArthur Deposit. Entrée has been exploring Ann Mason since acquiring the property in 2010 and has completed 97,274.31 feet of drilling in 27 drill holes. The drilling has outlined an area of copper and molybdenum mineralization.

Table 23-1 lists Entrée Gold's mineral estimates for the Ann Mason deposit, dated March 26, 2012. The results are reported as in the Technical Report. The QP responsible for this section has been unable to verify this information; it is not necessarily indicative of the mineralization on the property that is the subject of this technical report.

Table 23-1: Ann Mason Deposit Resource Estimate (Entrée Gold, March 2012)

Cut-off	**Indicated Resources**					
Cu (%)	Tonnes (million)	Cu (%)	Mo (%)	lb Cu (billion)	lb Mo (billion)	CuEq* (%)
0.2	1,115	0.33	0.007	8.04	0.16	0.35
0.25	904	0.35	0.007	6.98	0.13	0.38
0.3	**640**	**0.38**	**0.007**	**5.38**	**0.10**	**0.41**
0.35	391	0.42	0.007	3.60	0.06	0.45
0.4	201	0.46	0.008	2.04	0.03	0.49

Cut-off	**Inferred Resources**					
Cu (%)	Tonnes (million)	Cu (%)	Mo (%)	lb Cu (billion)	lb Mo (billion)	CuEq* (%)
0.2	1,131	0.29	0.004	7.31	0.11	0.31
0.25	780	0.32	0.004	5.56	0.07	0.34
0.3	**444**	**0.36**	**0.004**	**3.54**	**0.04**	**0.38**
0.35	215	0.40	0.004	1.90	0.02	0.42
0.4	82	0.45	0.005	0.82	0.01	0.47

Figure 23-1 shows a map of the area that includes adjacent properties. See Sections 24.1 and 24.2 for information on nearby properties in which Quaterra holds an interest.

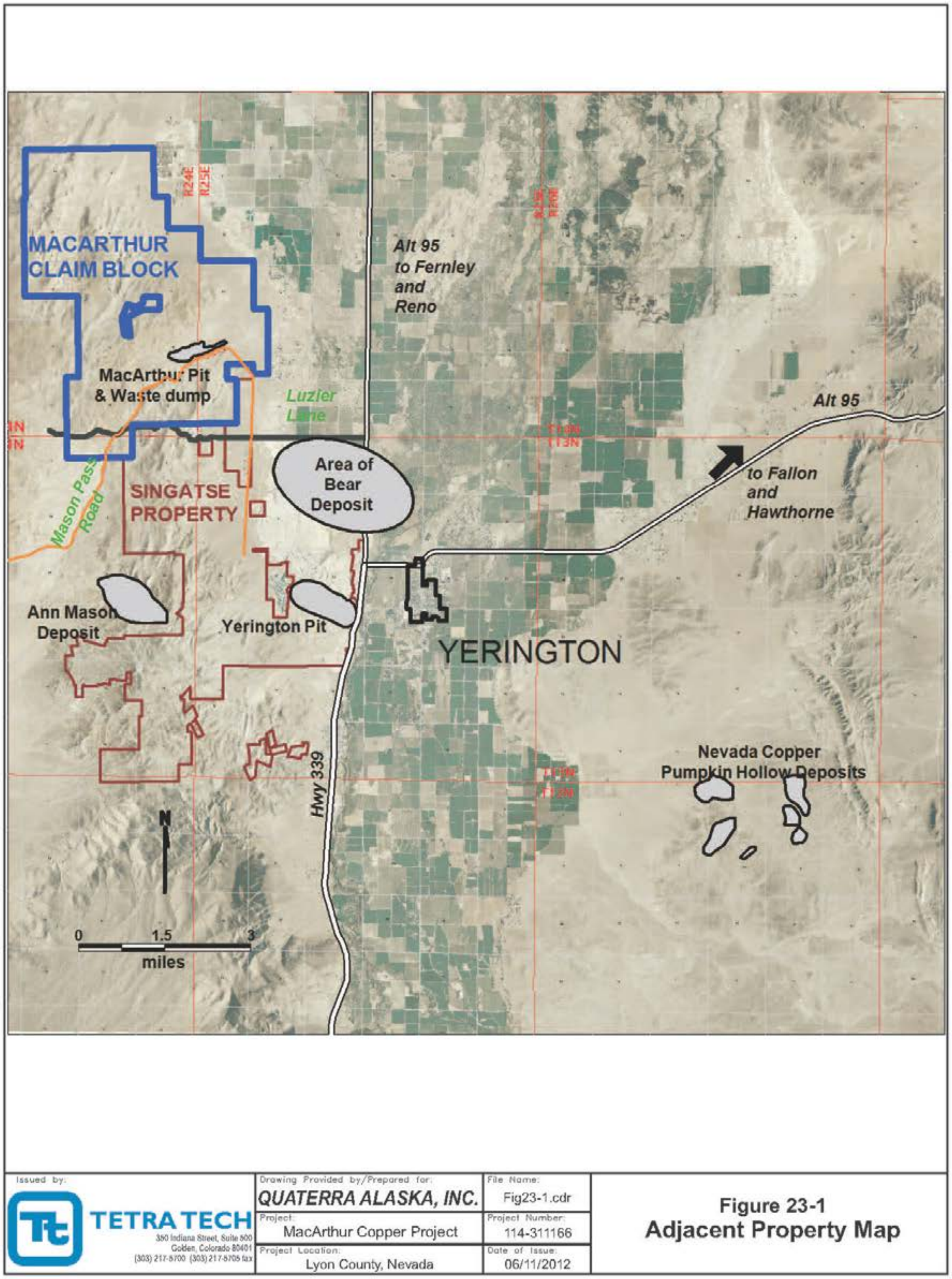


Figure 23-1: Adjacent Properties

24 OTHER RELEVANT DATA AND INFORMATION

Quaterra holds an interest in two nearby properties, including Singatse Peak Services Yerington Mine Copper Property, described in Section 24.1 and the Bear MacArthur-Lagomarsino Deposit, described in Section 24.2. Section 24.3 describes the potential for reprocessing residual material from the historic Yerington site as part of the Project.

24.1 SINGATSE PEAK SERVICES PROPERTIES (YERINGTON)

During April 2011, Singatse Peak Services, LLC (SPS), a wholly owned subsidiary of Quaterra Alaska Inc., purchased the historic Yerington Mine Copper Property comprising over 12,000 acres of private lands and unpatented lode mining claims south of and contiguous with Quaterra Alaska's MacArthur property Figure 24-1. The Yerington Mine Copper Property was operated from 1952 to 1977 by The Anaconda Company and from 1977 to 1979 by Atlantic Richfield Corporation.

During February 2012, SPS published a NI 43-101 compliant independent resource estimate at the Yerington Mine Copper Property. As cited in their Technical Report, resources at a 0.2% Cu cutoff are shown in Table 24-1.

Table 24-1: Singatse Peak Services, LLC- Yerington Mine Resources, Feb. 2012

Resource Category	Cutoff Grade	Tons	Average Grade	Contained Copper
Measured Copper Resource	**% TCu**	**(x 1000)**	**% TCu**	**(lbs x 1000)**
Oxide and Chalcocite Material	0.20	2,853	0.32	18,122
Primary Material	0.20	25,866	0.33	169,629
Indicated Copper Resource				
Oxide and Chalcocite Material	0.20	6,592	0.3	39,117
Primary Material	0.20	45,914	0.28	260,332
Measured + Indicated Copper Resource				
Oxide and Chalcocite Material	0.20	9,445	0.3	52,237
Primary Material	0.20	71,781	0.3	429,968
Inferred Copper Resource				
Oxide and Chalcocite Material	0.20	8,596	0.28	47,347
Primary Material	0.20	63,918	0.25	322,530

SPS's assets on the Yerington Mine Copper Property also include over 120 million tons of resource piles ("Residuals") representing sub-grade stripping material from the Yerington mine vat leach tailings (VLT) representing the oxide tailings from Anaconda's copper oxide vat leaching, and historic heap leach pads previously mined by Arimetco. Approximately 44 million tons consisting primarily of sub-grade material (Stockpiles W-3 and S-23 in Table 24-2) and VLT were heaped and leached (including an estimated 6 million tons of oxide material from the MacArthur mine) by Arimetco International Inc. during 1989-1998.

No copper extraction from the Arimetco heaps or mining has occurred since Arimetco closure in 1999, leaving an estimated (non-compliant NI43-101) over 300 million pounds of contained copper in the residuals as summarized in Table 24-2. References 2 through 5 shown on the table refer to documents published by the USEPA (EPA), as listed in Section 27, References. Work is ongoing by SPS to further characterize the Yerington residuals, including drilling and metallurgical testing to assess the viability of reprocessing some or all of these materials. Depending on the results of the ongoing work, some of these materials may be integrated with the MacArthur Oxide project in the future. The residuals are further discussed in Section 24.3 of this TR.

Table 24-2: Yerington Mine Residual Copper Resources, SRK March, 2012 (Non NI 43-101 Compliant)

Material Type	Material (tons)	Volume (cu. yds.)	Total Cu Grade (%)	Contained Cu (k lb)	Expected Leach Recovery
Oxide Tails (VLT)[1,3,5]	57,571,505	35,545,074	0.130	149,686	75%
Oxide Low-Grade W-3 [1,4]	19,643,073	12,127,779	0.200	78,572	60%
Sulfide Low-Grade [4,]	2,316,440	1,430,187	0.200	9,266	85%
Phase 1/2 HLP [1,2]	2,104,570	1,362,710	0.099	4,159	50%
Phase 3 HLP 4 [1,2]	8,547,269	5,147,407	0.120	20,513	50%
Phase 3 HLP S [1,2]	10,117,573	5,836,837	0.083	16,714	50%
Phase 4 Slot HLP [1,2]	12,927,862	8,793,567	0.091	23,399	50%
Phase 4 HLP [1,2]	11,556,016	6,539,352	0.075	17,242	50%
Total	124,784,308	76,782,913	0.128	319,551	

Notes:
1 Volume based on: SRK 2010 digitization and volume calculations using MineSight 3D Software.
2 Density based on: Draft Supplemental RI Report_OCT_2010 - Page 47.
3 Grade based on: AnacondaArimetco_RI_Report.pdf - Page 170-172.
4 Grade based on: VLT XRF DSR July 2010 - Page 99.
5 Grade based on: HistoricalSummaryReport-YeringtonMine-2010-10.pdf - Page 19.

24.2 BEAR-MACARTHUR LAGOMARSINO DEPOSIT

The Bear deposit is located near the MacArthur area and is a large porphyry copper system that was discovered and partially delineated by Anaconda in the 1960s and by Phelps Dodge in the 1960's and 1970's. The deposit is open in several directions and has never been consolidated under a single owner. Quaterra has data from 49 drill holes totaling 126,400 feet that defines a system covering an area of at least 2 square miles. The portion of the deposit controlled by Anaconda in the 1960s covered approximately 25% of this area. Dilles and Proffett (1995) estimated that the Anaconda drilling program defined more than 500MT of mineralized material with an average copper grade of 0.40%. The resource estimate is shown in Table 24-3.

The Bear-MacArthur-Lagomarsino resources referenced (which were obtained from MineMarket.com in 2004 are shown in Table 24-3) have not been classified according to current

CIM standards. A portion of the Bear-MacArthur-Lagomarsino prospect underlies the Yerington Site.

Table 24-3: Bear-MacArthur-Lagomarsino Resource Estimate (May 2012)

	Material Tons (kTons)	Average Grade (% Cu)	Contained Cu (kTons)	Contained Cu (000s lbs)
Bear-MacArthur-Lagomarsino Deposit	500,000	0.40	2,000	4,000,000

24.3 RE-PROCESSING OF YERINGTON RESIDUALS

24.3.1 Introduction

A scoping study was prepared by SRK Consulting (U.S.), Inc. (SRK) for Singatse Peak Services LLC (SPS), a wholly-owned subsidiary of Quaterra Resources Inc., to evaluate the technical and economic feasibility for re-processing residual spent mineralized material, waste rock, and oxide leach tailings at the now closed Yerington Copper Mine located near the town of Yerington in Lyon County, Nevada. The Yerington Copper Mine is owned by Singatse Peak Services (SPS), a subsidiary company of Quaterra Resources, and is located within four miles of the MacArthur Pit with possibility for sharing infrastructure facilities.

The scoping study, dated March 12, 2012, was intended for SPS internal use only and does not conform to CIM NI 43-101 standards for public disclosure. Results of the scoping study are presented here for the purpose of highlighting potential synergies between the MacArthur Copper Project and the re-processing of residual material stockpiles and tailings at the Yerington Copper Mine and to get an early look at the potential economic benefit of a combined project.

24.3.2 Residual Copper Resources

SRK quantified four material types at Yerington as potential resources for re-processing and extracting residual copper at Yerington. Three of the four mineral types are oxide in nature and considered suitable for combining with the MacArthur process facilities. The fourth material is a low grade sulfide more amenable to a mill / flotation circuit. The three oxide material types considered for a combined project with MacArthur are noted below.

a) Crushed vat leach tailings (VLT) from the former Anaconda processing of oxide ore.

b) A low grade run-of-mine (ROM) oxide stockpile (W-3) from the Yerington pit that was below Anaconda’s cutoff grade of 0.3% copper for copper ore.

c) Five heap leach pads (HLPs) built and operated by Arimetco containing mostly W-3 oxide, VLT, and small additions of copper oxide from the MacArthur mine.

24.3.2.1 Vat Leach Tailings (VLT)

The VLT stockpile area covers approximately 500 acres primarily on private land owned by SPS with an average height of approximately 100 feet. The tops surfaces are composed of multiple benches and VLT mounds channeled to prevent storm water runoff. The VLT resource was estimated to be approximately 57.6 million tons of mineralized material at an average grade of 0.13% Cu. The resource estimate was based on volumetric calculations and density measurements from historical data and recent test work. The grade determination was based on averages of samples reported in a recent VLT characterization study by Atlantic Richfield (ARC, 2010). The average grade of the VLT material from this report was 0.13% Cu. Subsequently, METCON (2011) conducted head assays for materials used in six column leach tests, which averaged 0.18% Cu. SRK believed that the column test average grade was biased high by one sample with a head assay of 0.356% Cu; therefore SRK used the 0.13 % Cu assay from the Atlantic Richfield study.

SPS contracted METCON Research of Tucson, Arizona to conduct column leach tests on six samples of VLT stockpile material with head grades ranging from 0.35% down to 0.06%. The column tests were run in locked cycle for 93 days followed by an eight day rinse and drain down. Testing indicated that copper recovery exceeded 70% on all but two samples with low head grades and low (~50%) acid soluble Cu to total Cu ratios. Test work also indicated a gangue acid consumption of approximately 30 lb. acid per ton of mineralized material in 90 day column leach tests. Gangue acid consumption plots show that consumption continues at a constant rate over the 93 day leach cycle indicating that a shorter leach cycle should produce high copper recoveries at a reduced acid consumption. The design basis for the VLT material was 18 lbs. acid per ton of mineralized material based on 20 day leach results from METCON (METCON, 2011).

24.3.2.2 W-3 Low Grade Oxide

Anaconda originally stockpiled low grade oxide that was below their operating cutoff of 0.3% Cu, but above their 0.2% threshold. The stockpile is north of the Yerington open pit and is primarily on land controlled by BLM. The current W-3 low grade oxide stockpile covers approximately 80 acres, with a maximum height of 210 feet, and averaging about 160 feet. Side slopes are generally 1.4H: 1V.

The W-3 resource is estimated to be approximately 19.6 million tons at an average grade of 0.2% Cu. The resource estimate was based on volumetric calculations and density measurements from historical data and recent test work. The acid consumption for the W-3 material was set at 35 lbs. acid per ton of mineralized material.

24.3.2.3 Heap Leach Pads (Arimetco)

Arimetco constructed five distinct heap leach pads (HLPs), built in four phases, covering nearly 250 acres during their operation between 1990 and 1999 (see Figure 24-1). Phase 1 is located immediately north of the Yerington open pit and southeast of the original SX/EW facility. Phase II is contiguous with phase I, extending it to the northwest. Phase III consists of two separate lined heap leach pads, with Phase III South and Phase III 4X both located north of the access road and west of the historic process areas. Phase IV also consists of two separate HLPs; 1) the Slot bordering the eastern property boundary and including portions of Anaconda's W-3 stockpile; and 2) the VLT heap located northeast of the VLT footprint. It should be noted that much of the Phase III 4X and Phase IV Slot HLPs reside on BLM administered land.

The Arimetco heap leach pads consist of mostly minus 6-inch material sourced from the W-3 oxide stockpile, with some MacArthur mineralized material, stacked approximately 100 to 120 feet high in nominal 20-foot high lifts. The exception is the Phase IV VLT, which is comprised of primarily VLT material. The estimated resource for all the heap leach pads is approximately 45.3 million tons at an average grade of approximately 0.1% Cu, representing the post-leaching residual grade of the mineralized material that carried between 0.2% and 0.3% Cu when originally stacked. The grades were discounted based on Arimetco production reports and recent drilling results (CH2M HILL, 2008). Acid consumption should be similar to the VLT material, even after partial leaching, and has been set at 30 lbs. acid per ton of mineralized material for the SRK scoping study.

The total resource identified in the scoping study is shown in Table 24-4 below. For the scoping study, SRK used 75% recovery for the VLT, 60% recovery for the W-3 and 50% for the Arimetco HLPs. The recovery estimates were based on historic records of production from the Anaconda vat leach operation (70%), and the Arimetco recovery records from the initial leach of the HLP materials in phase I to IV (53%) (Sawyer, 1999). Recent column leach test work on both VLT and MacArthur (METCON, 2011) established a basis for slightly higher forecasts for the VLT.

Table 24-4: Yerington Residual Oxide Copper Resources, SRK March 2012

Material Type	Mineralized Material (tons)	Average Total Cu Grade	Contained Cu (lb.)	Expected Leach Recovery	Extracted Cu (lb.)
Oxide Tails (VLT)	57,572,000	0.130%	149,686,000	75%	112,265,000
Oxide Low Grade (W-3)	19,643,000	0.200%	78,572,000	60%	47,143,000
Phase I and II HLP	2,105,000	0.099%	4,159,000	50%	2,080,000
Phase III HLP 4X	8,547,000	0.120%	20,513,000	50%	10,257,000
Phase III HLP S	10,118,000	0.083%	16,714,000	50%	8,357,000
Phase IV Slot HLP	12,928,000	0.091%	23,399,000	50%	11,700,000
Phase IV HLP	11,556,000	0.075%	17,242,000	50%	8,621,000
Totals	122,469,000	0.127%	310,285,000	65%	200,423,000

24.3.3 Mining Methods

24.3.3.1 Vat Leach Tailings (VLT)

SRK proposed an on/off leach pad for the VLT material in order to reduce acid consumption. The VLT mineralized material would be mined from the existing stockpile, agglomerated, and conveyed to one of four leach cells comprising the on/off leach pad. Mineralized material will be stacked to a height of 24 feet for leaching. After the leach cycle, the pad would be rinsed and drained before removing the spent leached material from the pad and stored in a lined VLT storage facility. Removal of the leached material would be by conventional mining equipment, including rubber tired loaders and 100 ton off-highway trucks. The liner system for the on/off pad would tie to the existing VLT stockpile liner and the new spent VLT lined storage facility. The entire operation, therefore, would be carried out on containment.

The mining rate for the VLT material was set at 7.9 million tons per year with a life of mine of 7.25 years. The daily mining rate was estimated to be approximately 21,700 tons per day. The leach solution flow from the VLT leach pad was set at 3,000 gpm based on an irrigation rate of 0.04 gpm / $ft.^2$. The solution grade was estimated to be 1.18 gpl Cu.

24.3.3.2 W-3 Low Grade Oxide

The W-3 material will be leached by a conventional heap leach pad. The leach pad would be constructed as a continuation of the "Slot Pad" concept initiated by Arimetco in the late 1990's. The remaining material in the existing slot (Slot 1) would be removed with loader and trucks and slot 1 will be lined. W-3 mineralized material from the next slot (Slot 2) would be mined and placed in the new Slot 1 pad and Slot 2 would be lined. The sequence would continue until the entire W-3 stockpile has been placed on a lined leach pad and leached. It is anticipated that four phases of slot expansions would be lined to create a total contiguous lined area of 2.70 million square feet.

The mining rate for the W-3 material was set at 2.7 million tons per year with the same life of mine of 7.25 years. The daily mining rate is estimated to be approximately 7,400 tons per day. The leach solution flow from the W-3 leach pad also was set at 3,000 gpm with an expected grade of 0.5 gpl Cu.

24.3.3.3 Heap Leach Pads (HLP)

For the scoping study, it was assumed that the existing heap leach pads from the Arimetco operation would be leached again in place. The horizontal top surfaces of these pads would be ripped with dozers to enhance the permeability at the surface. Some repairs are expected to be required to the existing perimeter ditches and ponds prior to re-leaching. In some cases the existing ponds may need to be replaced. The existing heap leach pads would be leached until the cost of power and reagents is greater than the revenue generated from the copper production. The pads would then drain down, be re-graded to a 3H: 1V slope and capped.

For the purpose of the scoping study, SRK assumed the time of leaching would coincide with the completion of the VLT and W-3 leaching, or 7.25 years. The heap leach pads would be leached in sequence to minimize the capital expenditures for pumps, piping and the SX/EW plant. A maximum of two heap leach pads would be under leach at any one time. The PLS flow from the HLPs was set at 3,000 gpm with an expected grade of 0.4 gpl Cu.

For the purposes of the scoping study, it was assumed that the HPL leach solution would be staged in series with the W-3 leach pads to produce a combined PLS flow of 3,000 gpm at a grade of 0.9 gpl Cu. The total PLS flow from the Yerington leach operation is expected to be 6,000 gpm at a grade of 1.05 gpl. This total PLS flow would be combined with the MacArthur leach solution (10,400 gpm at 1.0 gpl Cu) providing a total PLS flow to the MacArthur SX/EW plant of 16,400 gpm at about 1.0 gpl.

24.3.4 Capital Cost Summary

SRK developed capital and operating costs for three major case scenarios. The base case (Case 1) included leaching the VLT, W-3 stockpile and re-leaching the Arimetco heap leach pads. Case 2 assumed that the VLT and W-3 stockpile would be processed. Case 3 assumed only the VLT material would be leached. Capital cost estimates were modified in each case to reflect the change in assumptions. A second set of cases (Case 1A, Case 2A, and Case 3A) were also run without the SX/EW facilities, assuming the leach solution from the residual copper leach operation at Yerington would be combined and treated in the MacArthur SX/EW facility.

M3 updated the capital cost and operating cost for the SX/EW and sulfuric acid plant in order to accommodate the increased solution flow and increased sulfuric acid consumption in the MacArthur facilities to accommodate the SRK Case 1A. The combined capital cost for the Yerington residual leach operation and the increased MacArthur operation are summarized in Table 24-5 below.

Table 24-5: Combined Yerington Oxide Residuals / MacArthur Mine Capital & Sustaining Costs

	Initial Capital	Sustaining Capital
MacArthur Copper Operation		
Mine Equipment	$48,000,000	$83,600,000
EX/EW	$138,737,000	$63,968,000
Sulfuric Acid Plant	$100,105,000	$0
Owner's Cost	$5,000,000	
Sub Total	**$291,842,000**	**$147,568,000**
Yerington Residual Leach Operation		
Mine Equipment	$28,694,000	$1,306,000
Process & Leach Pads	$29,724,000	$18,462,000
Infrastructure	$1,300,000	$0
Owner's Cost	$408,000	$2,792,000
Sub Total	**$60,126,000**	**$22,560,000**
Total Combined Yerington/MacArthur	**$351,968,000**	**$170,128,000**

24.3.5 Operating Costs

Operating costs for the combined Yerington leach operation and MacArthur leach operation is summarized in Table 24-6 below. The SKR operating cost was adjusted to account for the cost of sulfuric acid from an onsite sulfuric acid plant instead of purchased acid. The MacArthur sulfuric acid cost will also see a reduction because of the economy of scale with a larger acid plant.

Table 24-6: Combined Yerington Oxide Residuals / MacArthur Mine Operating Costs

	Annual Cost	$ / lb. Cu
MacArthur Copper Production, Lbs.	41,538,000	
MacArthur Copper Operation		
Mine	$41,279,000	$1.00
SX/EW	$19,903,000	$0.48
Acid	$12,633,000	$0.31
G&A	$5,082,000	$0.12
Transportation	$2,077,000	$0.05
Sub-Total	**$80,974,000**	**$1.96**
Yerington Copper Production, Lbs.	27,666,000	
Yerington Residual Leach Operation		
Mine	$11,204,000	$0.40
Heap Leach	$12,925,000	$0.47
Solution Pumping	$7,416,000	$0.27
G&A	$291,000	$0.01
Transportation	$1,383,000	$0.05
Sub-Total	**$33,219,000**	**$1.20**
Combined Yerington / MacArthur	**$114,193,000**	**$1.65**

SRK based their capital cost estimate on their in-house experience with similar projects, scaled to the size of this project. Costs for many of the equipment items used were from recent vendor quotes. Where recent cost data was not available, commercially available mining cost services, such as InfoMine, was used. Mine equipment was selected with a life cycle of 30,000 to 40,000 hours, which equates to a 7 to 8 year mining operation. SRK considered using the same equipment for haulage at both the VLT off-loads and to move the W-3 stockpile. SRK assumed that the existing buildings on site would be used for office and warehousing. A new truck shop was provided to support the proposed fleet of 100 ton trucks. SRK applied a contingency of 30% on all their estimates.

M3 factored the cost of the SX/EW facility and sulfuric acid plant from the PEA capital cost estimate to the new capacities. The capital cost of the sulfuric acid plant was factored from 640 tons per day plant to 1,220 tons per day. The solvent extraction plant, originally sized for MacArthur, can be used in combination with the Yerington residual PLS by changing the configuration of the 3 extraction settlers from a 2-series, 1-parallel configuration to 3 parallel settlers. The capital cost of the tank farm, electrowinning, main substation, and reagent areas were factored based on the increase in copper production.

24.3.6 Economic Analysis

A financial analysis was prepared for the combined MacArthur and Yerington operation using the same parameters as for the stand-alone MacArthur Project and the preliminary Yerington oxide residuals case without a SX/EW facility. The combined life of mine recovered copper is

948,266,000 pounds with combined revenues of \$3.3 billion at the \$3.48 per pound price of copper. The Net Present Value (NPV), Internal Rate of Return (IRR) and payback period before and after taxes are shown in Table 24-7 below.

Table 24-7: Combined Yerington Oxide Residuals / MacArthur Mine Economic Indicators

Economic Indicators	Before Taxes \$000	After Taxes \$000
NPV at 8% Discount Rate	\$435,681	\$308,307
IRR, %	32.9%	26.5%
Payback, years	2.5	2.9

The preliminary economic evaluation for the combined MacArthur heap leach operation and the Yerington residual re-processing operation is estimated to add over \$100 million to the stand-alone MacArthur NPV at 8% discount rate, increase the IRR by over 2 percentage points, and reduce the payback period by approximately 2 months.

Although the Yerington resource determination and recoveries are not sufficiently defined to comply with NI 43-101 reporting standards, there is sufficient justification for further investigation of the combined MacArthur heap leach operation and re-processing of Yerington residual materials. This work is ongoing and would be included with the pre-feasibility work for a combined oxide project. See Figure 24-1.

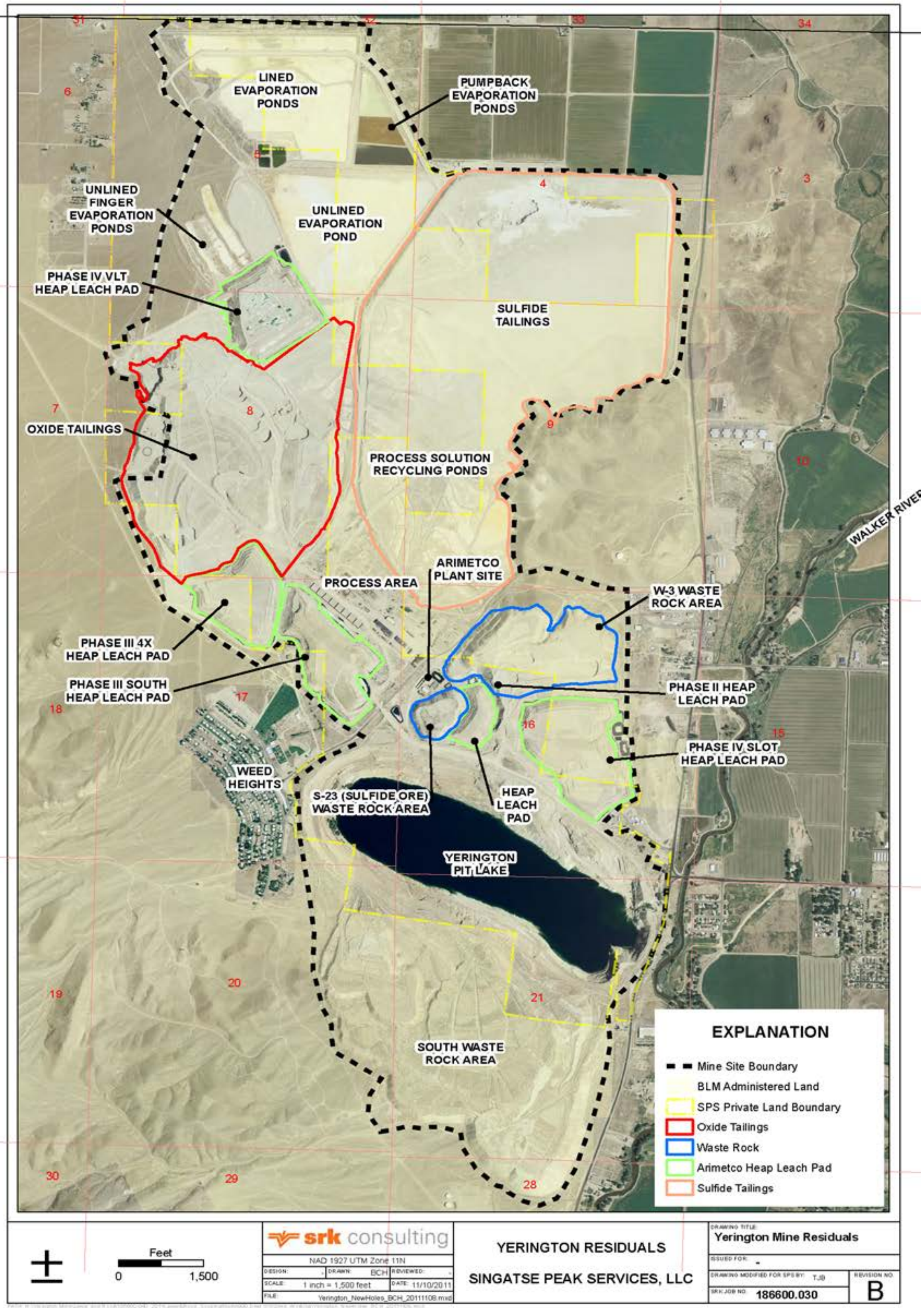


Figure 24-1: Yerington Mine Residuals

25 INTERPRETATION AND CONCLUSIONS

The intent of this report is to incorporate previous resource information and technical reports prepared earlier with additional resource information and the Preliminary Economic Assessment (PEA). The results of this PEA suggest that the project may be technically feasible utilizing ROM heap leaching and solvent extraction / electrowinning technology and may be economically viable based on the resources, grade, and recovery information presented to date. There is potential to further enhance the project economics by integrating the MacArthur Copper Project with other copper oxide materials on property controlled by Quaterra Resources, Inc. and Singatse Peak Services (SPS) in the Yerington District. Further work will be necessary, however, to quantify the other resources grade and recoveries to comply with NI 43-101 standards of disclosure for mineral projects.

A PEA should not be considered to be a pre-feasibility study or feasibility study, as the technical and economic viability of the project has not been demonstrated at this time. This PEA is preliminary in nature and includes Inferred Mineral Resources that are considered to be too geologically speculative at this time to have the economic considerations applied to them to be categorized as Mineral Reserves.

25.1 RESOURCES

The measured and indicated oxide and chalcocite resource for the project, at a cutoff grade of 0.12% total copper is 159 million tons containing 675.5 million pounds of copper. The inferred oxide and chalcocite resource is 243.4 million tons at a cutoff grade of 0.12% containing 979.5 million pounds of copper.

The primary sulfide measured and indicated resource at a 0.15% total copper cutoff is 1.1 million tons containing 6.4 million pounds of copper. The inferred sulfide resource at the 0.15% cutoff is 134.9 million tons containing 764 million pounds of copper.

Further exploration drilling may enhance the project resources and reduce project risk.

25.2 MINING METHODS

Mining at the MacArthur pit will be by open pit at an approximate rate of 41,000 tons per day over an 18 year life of mine. Approximately 271 million tons of mineralized material will be mined over the life of mine with an average waste to mineralized material ratio of 0.90. There are three final pits consisting of the main MacArthur pit, the North pit and Gallagher pit. Mining will start in the main MacArthur pit and progress to the North pit, then to Gallagher pit and ending at the outer limits of the main MacArthur pit in the last phase.

Waste dumps are located to the north and to the south of the pits with some pit backfill in the North pit area extending to the west of the North pit.

There were no geotechnical studies for the pit slope angles performed for this PEA. No extensive condemnation drilling has been done in the north and south waste dump areas. Further optimization of the mine plan, including waste rock management, will be completed in the next phase of the project.

25.3 METALLURGY

During the PEA review of historic and recent metallurgical test work for the MacArthur Project, several issues were identified that require additional test work to improve understanding of these issues and what impact they may have on the project. This work will be undertaken as part of the PFS which will include a comprehensive Phase II metallurgical test program.

25.3.1 Run-of-Mine Heap Leaching

The PEA was based on ROM mineralized material truck dumping to a permanent heap leach pad. Based on the project mineralized material grade of 0.211% total copper and high projected copper extraction, this approach provides simplicity of mineralized material processing while optimizing acid consumption.

Historical test work provides limited ROM data for copper extraction and acid consumption. However, MacArthur ROM mineralized material was successfully processed by Arimetco with good copper extraction and acid consumption, supporting the ROM leaching approach. The proposed Phase II PFS metallurgical program will address particle size vs. copper extraction and further evaluate acid consumption.

During the PFS, the Phase II test work program will be performed to provide data from which to make a final determination on optimization of leach particle size. During the Phase II test work program, a number of sites will be selected from the present bench faces in the MacArthur Pit. Using a portable screen and loader, this material will be screened to provide data on ROM top particle size distribution.

25.3.2 Spatial Variability of In-Situ Size Distribution

The 2011 column leach study by METCON Research included 32 column tests on PQ size core from 32 separate drill holes which spatially provided preliminary representivity of the block model. This study showed significant variation in particle size distribution from hole to hole. In heap leaching, particle size distribution is extremely important, particularly in particle sizes less than 100 mesh, and more importantly, less than 200 mesh. During the PFS, this issue will be well defined.

25.3.3 Chemical Degradation of the Mineralized Material during Leaching

The 2011 METCON Research column study also identified chemical degradation in some columns during leaching, impacting post leach size distribution. It is likely that the mineralized material was over acidified during leaching resulting in un-necessary chemical degradation. However, this is likely not to be the only cause of variability in size distribution from drill hole to drill hole. This issue is extremely important to understand and will be studied during pre-feasibility metallurgical testing.

25.3.4 Permeability and Agglomeration

Considering both the in-situ size distribution variability and chemical degradation during leaching, permeability of leached mineralized material must be better understood as part of future design efforts. To accommodate uncertainty at this stage of project definition, the PEA design includes one inter-lift liner to be placed at approximately half the final pad elevation.

Pre-feasibility test work will address spatial variations in size distribution and chemical degradation using size distribution measurements of column head and tails and analytical results. Column tails will be subjected to permeability/consolidation testing to define the ultimate permeability of multi-lift permanent leach pad processing.

25.3.5 Spatial Variability of Copper Extraction and Acid Consumption

The METCON column study also identified spatial variability in copper extraction and acid consumption. Moving away from the existing MacArthur Pit to the periphery zones of the block model tends to show reducing copper extraction and increasing acid consumption.

The relationship of copper extraction and acid consumption versus depth in the deposit is also not well understood from existing test work and must be better quantified.

The mine plan as provided by Independent Mining Consultants, Inc. (IMC) in this PEA has defined the materials to be extracted during the LOM. The PFS will aim to determine optimum processing techniques addressing both the spatial variation in copper extraction and acid consumption. Crushing and/or agglomeration, if justified, would distribute acid more efficiently through the mineralized material resulting in reduced acid consumption and increased copper extraction, but at higher operating cost. This trade off will be further evaluated in future design studies.

25.3.6 Relationship of Total Iron Mineralization to Acid Consumption

One relationship that is clear in leaching MacArthur Project mineralized material is the relationship between iron mineralization and acid consumption. Acid consumption in bottle roll and column testing consistently shows a linear increase in acid consumption versus time. Acid consumption tends to mirror iron extraction. This relationship will require optimal control of available free acid during leaching to maximize copper extraction while minimizing iron extraction.

Again, the pre-feasibility Phase II metallurgical program will be designed to study this relationship including mineralogical studies. XRD, XRF, or QEMSCAN studies will provide total iron content and distribution of the iron species to determine which iron minerals are largely responsible for acid consumption.

25.4 ECONOMIC ASSESSMENT

Based on the historical three year average price of copper of $3.48 per pound, M3 engineering and Technology Corporation concluded that the after tax preliminary economic assessment of the project would provide a net present value, at an 8% discount rate, of approximately $201.5 million with an internal rate of return of 24.2 % and a payback period of 3.1 years. The preliminary economic assessment was based on an onsite sulfuric acid plant with a byproduct electrical power credit, an initial capital investment of $232.7 million and sustaining capital of $230.5 million over an 18 year life of mine. The overall average life of mine operating costs was calculated to be approximately $1.89 per pound of recovered copper.

This PEA is preliminary in nature and includes discussion of mineral resources including inferred mineral resources that are too speculative geologically to have economic considerations applied to them. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

M3 concluded that the economic indicators for the stand alone MacArthur Copper Project has a potential for development as a large-scale copper oxide heap leach operation. There are also opportunities for enhanced economic indicators by combining the MacArthur Copper Project with the re-processing of residual vat leach tails, waste rock and historic leach pads at the existing Yerington site. The residual material at the Yerington site is not yet classified as resources according to CIM guidelines and will require further work to quantify.

25.5 RISKS

The project risks identified at this time are listed below. Using a staged approach to advance the project to full production will allow Quaterra Resources, Inc. to adequately assess the risks and associated costs and develop mitigation strategies before progressing to the next stage.

a) Further drilling may identify increases or decrease in resource tonnage and mineralized material grade.

b) Further metallurgical testing may demonstrate variable recoveries and acid consumption, resulting in changes to the economic indicators for the project.

c) The cost of sulfur delivered to site for the manufacture of sulfuric acid may be higher or lower than estimated in the PEA resulting in changes to operating costs for the project.

d) The capital cost estimates for initial capital and sustaining capital may be higher or lower than estimated in this PEA resulting in changes to the economic indicators for the project.

e) The future price of copper may fall below the level necessary to sustain a viable operation. Likewise, the copper price may increase above the base resulting in better economic indicators for the project.

f) The heap leach pad lift heights and overall height may impact the permeability of the leach solution through the heap, changing overall copper recovery.

g) More or less fines in the run-of-mine mineralized material may change the permeability of the leach solution, affecting overall copper recovery. Agglomeration of the mineralized material may be required if fines are excessive.

h) The allowances for refurbishing existing buildings at the existing Yerington site may not be sufficient, increasing the capital cost for the Project.

26 RECOMMENDATIONS

This section summarizes the recommendations made for Quaterra Resources, Inc. consideration as the project progresses to the next phase.

26.1 METALLURGY TEST PROGRAM

Additional drilling and metallurgical test work is required to complete a PFS or FS. Based on the mine plan prepared for the PEA, LOM tonnage and grade has been generated along with tonnage and grade mined per year. This plan also includes annual mined material segregated by MacArthur oxide material, oxide material from areas other than MacArthur Pit, and tonnage and grade of mixed oxide/secondary sulfide material.

With the mine plan and phasing complete, a PFS or FS metallurgical drilling program can be defined. Geology and mine planning personnel in consultation with metallurgical personnel will produce a drill program defining drill site location and depth so as to provide core representative of at least 70 percent of the total mined tonnage. The geological team developing the resource model will be required to establish the drill density and location necessary to achieve the representivity required for the PFS metallurgical test work program. It is probable that at least 30 to 40 holes will need to be drilled which, in elevation and by depth, consider geology, boundaries, lithology, grade, mineralogy etc. With the 32 existing METCON drill holes, metallurgical results will be available covering 62 to 72 holes.

Due to the size of this metallurgical program, it is expected that testing will contain elements of all of the following stages of metallurgical test work:

- Stage I-Sample preparation
- Stage II-Bottle roll and acid characterization testing
- Stage III-Small column testing
- Stage IV-Large column testing
- Stage V-Final study preparation and recommendations for a Final Feasibility Study

Note that this program may be revised prior to the PFS based on project needs and professional judgment.

26.1.1 Stage I- Sample Preparation

Once core is logged, the core will be composited into 50 foot interval composites versus depth in each hole. Each 50 foot composite will be sent to a metallurgical laboratory where the composites will be dried, screened, and the screen fractions retained separately. Each screen size will be assayed for total copper, oxide copper and cyanide sequential analysis, total iron and ICP.

26.1.2 Stage II- Acid Bottle Roll and Acid Characterization Testing

Each 50 foot composite will have a standard acid bottle roll test performed. With leaching at 100 g/l sulfuric acid, variation in acid consumption is magnified. This variation can be evaluated spatially and by depth to allow an increased understanding of global acid consumption in the deposit.

Selected acid characterization testing will be performed. Static leach testing evaluates copper extraction and acid consumption versus crush sizes versus acid concentration over time. Crush sizes of one inch, 2 inches 4 inches and ROM will be leached at 3, 5, 10, 20 and 100 grams per liter for 40 days, maintaining the acid concentration within 90% of the designated acid concentration in each test. The PLS will be analyzed periodically over time to provide leach kinetics for copper extraction and acid consumption and the same for the gangue elements. Analytical testing of each acid characterization test provides an analysis of gangue dissolution occurring over specific time intervals by ICP analysis for selenium, uranium, aluminum, potassium, iron and troublesome anions of nitrates, chlorides and fluorides, all of which may be problematic and need to be addressed in SX/EW operations.

26.1.3 Stage III- Small Column Leach Tests

Selected 50 foot composites with head grades above the mine cut-off grade, but providing variable copper head grades, will be selected for small column testing by spatial location (horizontally and at depth). These tests will define metallurgical performance for copper extraction and acid consumption and variations spatially and by depth in the deposit

26.1.4 Stage IV- Large Column Leach Tests

Based on the data generated by the acid bottle roll, acid characterization testing, and the small column testing, master composites derived from the 50 foot individual composites will be prepared for large column testing to simulate ROM leaching. Some large columns will be run at a 20 foot height to evaluate the planned 20 foot leach pad lift in practice.

26.1.5 Stage V- Study Preparation and Recommendations for a Final Feasibility

Once the metallurgical study is completed, all data will be evaluated relative to copper extraction and acid consumption spatially in the deposit. Sufficient data will be available to resolve the questions raised in Section 26. Additional questions or observations generated from this work will be defined for evaluation during the feasibility study.

From the metallurgical and block model data base generated by this study, extensive modeling of all elements of the project will be performed in an effort to select operational procedures in practice to optimize project performance.

26.2 BUDGET AND SCHEDULE

A budget and schedule has been developed for the follow on additional test work. The budget is in two phases, with the second phase dependent on positive results from the first phase. The first phase will consist of additional drilling to better define the resource, particularly in the area north of the MacArthur pit and at depth, and additional metallurgical drilling to obtain core samples for the metallurgical testing. This phase will also include an updated resource model and the nine month metallurgical test program as outlined above.

The second phase will consist of incorporating the additional resource information and metallurgical test results into a pre-feasibility study NI 43-101 Technical Report with an updated capital and operating cost estimate and an updated economic assessment of the project. This phase will develop additional preliminary engineering to establish the criteria for heap leach pad overall height, evaluation of the need for agglomeration, and market studies for the cost of sulfur delivered to the site.

The budget and schedule for the two phase program is summarized in Table 26-1 below.

Table 26-1: Budget for MacArthur Follow on Test Work

		Item	Schedule	Cost
Phase 1	Additional Resource Drilling	100 holes	4 Months	$1,100,000
	Additional Metallurgical Drilling	30 to 40 holes	4 Months	$1,500,000
	Additional Metallurgical Test Work		9 Months	$1,000,000
	Subtotal Phase 1		**9 Months**	**$3,600,000**
Phase 2	Pre-feasibility Study & Technical Report		9 months	$800,000
	Total Both Phases		**18 months**	**$4,400,000**

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APPENDIX A: CERTIFICATE OF QUALIFIED PERSON (“QP”) AND CONSENT OF AUTHORS

APPENDIX A
CERTIFICATE OF QUALIFIED PERSON (“QP”) AND CONSENT OF AUTHORS
MACARTHUR COPPER PROJECT
JANUARY 2014

CERTIFICATE OF QUALIFIED PERSON

I, Myron R. Henderson, P.E., do hereby certify that:

1. I am currently employed as a Project Manager by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, St 101
   Tucson, Arizona 85704
   U.S.A.

2. I graduated with a Bachelor of Science degree in Mechanical Engineering from the University of Arizona in 1966 and a Master of Business Administration (MBA) from the University of Phoenix in 1989.

3. I am a:

   - Registered Professional Engineer in good standing in the State of California in Manufacturing (No.4133)

4. I have worked as an engineer for a total of 46 years in the mining industry since my graduation from the University of Arizona. My work experience includes 15 years in supervisory positions in operating plants and 31 years in process engineering, project engineering and project management positions with design engineering, procurement and construction management firms.

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 principal author for the preparation of the technical report titled "MacArthur Copper Project, Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Lyon County, Nevada, USA" dated effective 23 May 2012 and amended on 17 January 2014 (the "Technical Report"); and am responsible for the preparation of Sections 1 through 3, Sections 17 through 19, Sections 21 and 22, and Sections 24 through 27.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I visited the project site November 30, 2011 and December 1, 2011.

9. As of the date of this certificate, to the best of my knowledge, information and belief, those portions of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of Quaterra Resources applying all of the tests in section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 17th day of January, 2014.

(signed) (sealed) "*Myron R. Henderson*"
Signature of Qualified Person

Myron R. Henderson, P.E.
Print name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, Rex C. Bryan, Ph.D., do hereby certify that:

1. I am currently employed as a Senior Geostatistician by Tetra Tech, Inc. with a business address of:

   Tetra Tech, Inc.
   350 Indiana Street, Suite 500
   Golden, Colorado 80401
   U.S.A.

2. I graduated with a degree in Engineering (BS with honors) in 1971 and a MBA degree in 1973 from the Michigan State University, East Lansing. In addition, I graduated from Brown University, Providence, Rhode Island with a MS degree in Geology in 1977, and The Colorado School of Mines, Golden, Colorado, with a graduate degree in Mineral Economics (Ph.D.) in 1980.

3. I am a Registered Member (#411340) of the Society for Mining, Metallurgy, and Exploration, Inc.

4. I have worked as a resource estimator and geostatistician for a total of thirty-one years since my graduation from university; as an employee of a leading geostatistical consulting company (Geostat Systems, Inc. USA), with large engineering companies such as Dames and Moore, URS, and Tetra Tech and as a consultant for more than 30 years.

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

6. I am responsible for the preparation of Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 15, and 23 of the Technical Report titled "MacArthur Copper Project, Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Lyon County, Nevada, USA" dated effective 23 May 2012 and amended on 17 January 2014 (the "Technical Report").

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have visited and inspected the subject property September 9 and 10, 2011.

9. As of the date of this certificate, to the best of my knowledge, information and belief, those portions of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of Quaterra Resources applying all of the tests in section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 17th day of January, 2014.

"signed"
Signature of Qualified Person

Rex C. Bryan
Print name of Qualified Person

CERTIFICATE OF AUTHOR - HERBERT E. WELHENER

I, Herbert E. Welhener of Tucson, Arizona, do hereby certify that as the author of the Technical Report entitled "MacArthur Copper Project, Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Lyon County, Nevada, USA" (Technical Report) dated May 23, 2012 and amended on January 17, 2014; I hereby make the following statements:

1. I am currently employed by and carried out this assignment for Independent Mining Consultants, Inc. (IMC) located at 3560 E. Gas Road, Tucson, Arizona, USA, phone number (520) 294-9861.
2. This certificate applies to the Technical Report entitled "MacArthur Copper Project, Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Lyon County, Nevada, USA" (Technical Report) dated May 23, 2012 and amended on January 17, 2014 (the "Technical Report").
3. I graduated with the follow degree from the University of Arizona: Bachelors of Science – Geology, 1973.
4. I am a Qualified Professional Member (Mining and Ore Reserves) of the Mining and Metallurgical Society of America (#01307QP), a professional association as defined by NI 43-101. As well, I am a Registered Member of the Society of Mining, Metallurgy, and Exploration, Inc. (# 3434330RM).
5. I have worked as a mining engineer or geologist for 39 years since my graduation from the University of Arizona.
6. I am familiar with NI 43-101 and by reason of my education, experience and affiliation with a professional association (as defined in NI 43-101) and I am a Qualified Person (as defined in NI 43-101). I am a founding partner, Vice President and Principal Mining Engineer, of Independent Mining Consultants, Inc. since 1983.
7. I am responsible for Section 16 of the technical report entitled "MacArthur Copper Project, Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Lyon County, Nevada, USA" (Technical Report) dated May 23, 2012 and amended on January 17, 2014. I last visited the property on December 1, 2011.
8. I have had prior involvement with the property that is the subject of this Technical Report. The nature of my involvement is as a consultant to Quaterra Resources, Inc. in the preparation of previous internal studies on the MacArthur project.
9. I am independent of Quaterra Resources, Inc. as defined by Section 1.5 of NI 43-101.
10. That, as of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make this Technical Report not misleading.
11. I have read NI 43-101 and I certify that the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites assessable by the public.

Signed and dated 17th day of January, 2014 at Tucson, Arizona

(signed) "*Herbert E. Welhener*"

Herbert E. Welhener, MMSA-QPM

CERTIFICATE OF QUALIFIED PERSON

I, Richard W. Jolk, P.E., Ph.D., do hereby certify that:

1. I am currently employed as a Principal V Minerals Engineer by Tetra Tech, Inc. with a business address of:

   Tetra Tech, Inc.
   350 Indiana Street, Suite 500
   Golden, Colorado 80401
   U.S.A.

2. I am a graduate of the Colorado School of Mines, Bachelor of Science degree in Metallurgical Engineering, 1978 – Master of Science degree in Mine Engineering, 1986 – Master's degree in Environmental Engineering, 1993 – Doctorate in Mining Engineering specializing in Process Engineering Optimization, 2007.

3. I am a licensed Professional Engineer in good standing in the State of Colorado, license number 24448.

4. My relevant experience is that I have worked over 10 years with operators in mine and mineral processing operations, 11 years with engineering firms in project valuation, design, engineering, construction and commissioning, and 11 years as an independent minerals industry consultant.

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

6. I am responsible for the preparation of Section 13 of the Technical Report titled "MacArthur Copper Project, Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Lyon County, Nevada, USA" dated effective 23 May 2012 and amended on 17 January 2014 (the "Technical Report").

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have visited and inspected the MacArthur Copper Project site located in the Yerington Mining District on February 20, 2012, March 19, 2012, and April 17, 2012.

9. As of the date of this certificate, to the best of my knowledge, information and belief, those portions of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of Quaterra Resources applying all of the tests in section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 17th day of January, 2014.

"signed"
Signature of Qualified Person

Richard W. Jolk
Print name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Mark Willow, SME-RM 4104492

I, Mark Willow, do hereby certify that:

1. I am Practice Leader of:

   Steffen Robertson & Kirsten (U.S.), Inc.
   d.b.a. SRK Consulting (U.S.), Inc.
   5250 Neil Road, Suite 300
   Reno, Nevada 89502-6568

2. I graduated with Bachelor's degree in Fisheries and Wildlife Management from the University of Missouri in 1987 and a Master's degree in Environmental Science and Engineering from the Colorado School of Mines in 1995. I have worked as Biologist/Environmental Scientist for a total of 20 years since my graduation from university. My relevant experience includes environmental due diligence/competent persons evaluations of developmental phase and operational phase mines through the world, including small gold mining projects in Panama, Senegal, Peru and Colombia; open pit and underground coal mines in Russia; several large copper mines and processing facilities in Mexico; and a mine/coking operation in China. My Project Manager experience includes several site characterization and mine closure projects. I work closely with the U.S. Forest Service and U.S. Bureau of Land Management on several permitting and mine closure projects to develop uniquely successful and cost effective closure alternatives for the abandoned mining operations. Finally, I draw upon this diverse background for knowledge and experience as a human health and ecological risk assessor with respect to potential environmental impacts associated with operating and closing mining properties, and have experienced in the development of Preliminary Remediation Goals and hazard/risk calculations for site remedial action plans under CERCLA activities according to current U.S. EPA risk assessment guidance.

3. I am a Registered Member of the Society for Mining, Metallurgy and Exploration (SME) No. 4104492. In addition, I am a Certified Environmental Manager (C.E.M.) in the State of Nevada (#1832) in accordance with Nevada Administrative Code (NAC) 459.970 through 459.9729. Before any person consults for a fee in matters concerning: the management of hazardous waste; the investigation of a release or potential release of a hazardous substance; the sampling of any media to determine the release of a hazardous substance; the response to a release or cleanup of a hazardous substance; or the remediation soil or water contaminated with a hazardous substance, they must be certified by the Nevada Division of Environmental Protection, Bureau of Corrective Action;

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

5. I am responsible for the preparation of Section 20 of the technical Report titled "MacArthur Copper Project, Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Lyon County, Nevada, USA" dated effective 23 May 2012 and amended on 17 January 2014 (the "Technical Report"). I visited the MacArthur site on January 20, 2012 and on April 14, 2012.

6. I have not had prior involvement with the property that is the subject of the Technical Report.

7. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

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

9. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

10. As of May 23, 2012 (revised January 17, 2014) to the best of my knowledge, information and belief, Section 20 contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th Day of January 2014.

Mark A. Willow, M.Sc., C.E.M. #1832, SME-RM

SME
Society for
Mining, Metallurgy
& Exploration

Mark A. Willow
SME Registered Member No. 4104492
Signature
Date Signed
Expiration date 12/31/2014