Preliminary Economic Assessment, NI 43-101 Technical Report
Zonia Copper Project
Yavapai County, Arizona, USA

Prepared for:

Cardero Resource Corp.

Prepared by:

GRE Global Resource Engineering

Effective Date:
March 22, 2018
Issued Date:
April 17, 2018

Qualified Persons:

Terre Lane, MMSA 01407QP
Todd Harvey, SME-RM 4144120
Rex Bryan, SME-RM 411340

Zonia Copper Project

Date and Signature Page

This Technical Report on the Zonia Copper Project is submitted to Cardero Resource Corp. and is effective March 22, 2018.

The Qualified Persons and Responsible Report Sections follow:

Qualified Person	Responsible for Report Sections
Dr. Rex Bryan	Parts of 1, 2, 3, 14, 24, 25, 26, and 27 All of 7, 8, 9, 10, 11, 12, and 23
Terre Lane	Parts of 1, 2, 3, 14, 18, 19, 21, 22, 24, 25, 26, and 27 All of 4, 5, 6, 15, 16, and 20
Dr. Todd Harvey	Parts of 1, 2, 3, 18, 19, 21, 22, 24, 25, 26, and 27 All of 13 and 17

(Signed) "Rex Bryan" 3/22/2018

Signature Dr. Rex Bryan Date

(Signed) _"Terre Lane" 3/22/2018

Signature Terre Lane Date

(Signed) _"Todd Harvey" 3/22/2018

Signature Dr. Todd Harvey Date

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

LIST OF PHOTOS

APPENDICES

ABBREVIATIONS AND ACRONYMS

µm	micron
AA	atomic absorption
AAS	atomic absorption spectroscopy
ADEQ	Arizona Department of Environmental Quality
ADWR	Arizona Department of Water Resource
Ai	abrasion index
Amselco	American Selco Ltd.
Anp	porphyritic textured subvolcanic intrusions
APP	Aquifer Protection Permit
APP	Aquifer Protection Permit
Arimetco	Arimetco, Inc.
ASCu	acid soluble copper
AZPDES	Arizona Pollutant Discharge Elimination System
Bas	quaternary basalt
BLM	U.S. Bureau of Land Management
Boart Longyear	Boart Longyear Diamond Drilling
Cardero	Cardero Resource Corp.
Cg	Gila formation fanglomerate
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm/sec	centimeters per second
CNCu	cyanide soluble copper
Constellation	Constellation Copper Corporation

Copper Mesa	Copper Mesa Mining Corporation
Cs	chlorite schist
CSA	Canadian Securities Administrators
CuRes	residual copper assay (sulfide)
CuSol	total soluble copper (CNCu + ASCu)
CV	coefficient of variability
Dap	dacite porphyry
Dio	diorite
EA	Environmental Assessment
EIS	Environmental Impact Statement
EM	electromagnetic
EPA	U.S. Environmental Protection Agency
Equatorial	Equatorial Mining North America, Inc.
EW	electrowinning
Fel	felsite rocks
ft^3/ton	cubic feet per ton
gph/ft^2	gallons per foot squared per hour
g/Mt	grams per Metric ton
Gos	gossans
gpl	grams per liter
gpm	gallons per minute
gpm/ft^2	gallons per minute per square foot
GRE	Global Resource Engineering Ltd.
Grn	greenstone rocks
Hammon	Hammon Copper Company
Harris	Harris Exploration Drilling
HDPE	high density polyethylene
HLF	heap leach facility
Homestake	Homestake Mining Company
ICP	inductively coupled plasma
ILS	intermediate leach solution
IRR	Internal Rate of Return
kW	kilowatt
kw-hr/t	kilowatt-hour per ton
L	liter
lb	pound
LDRS	Leak Detection and Recovery System
LLDPE	linear low-density polyethylene
LME	London Metal Exchange
lph/m^2	liters per square meter per hour
m	meter
McAlester	McAlester Fuel Company
Metcon	Metcon Research, Inc.

Miami Copper	Miami Copper Company
ml	milliliter
mm	millimeter
MRA	Mine Reserve Associates, Inc.
MSRDI	Mountain States R & D International
Mss	metasiliceous sediments
MWh	megawatt-hour
Nerco	Nerco Minerals Company
NI 43-101	Canadian National Instrument 43-101
NPV	Net Present Value
Ob	overburden
OES	optical emission spectrometry
PCg	precambrian granitic rocks
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
Phelps Dodge	Phelps Dodge Corporation
PLS	pregnant leach solution
ppb	parts per billion
ppm	parts per million
Preston	Preston Drilling
QA/QC	Quality Assurance/Quality Control
Qep	quartzite-eye-porphyry
Qmp	quartz monzonite porphyry rocks
Qss	quartz sericite schist rocks
Qst	quartz sericite talc
Queenstake	Queenstake Resource LTD.
RC	reverse circulation
RCD	reverse circulation drilling
Redstone	Redstone Resources Corporation
ROM	run-of-mine
RQD	rock quality designation
SGV	Ste-Genevieve Resources Ltd.
Skyline	Skyline Assayers & Laboratories
Stg	sericite-talc granoblasts
SX	solvent extraction
SX/EW	solvent extraction/electro-winning
TCu	total copper
TE	trace element
TFe	total iron assay
Tetra Tech	Tetra Tech, Inc.
TMDL	total maximum daily load
tpd	tons per day
U.S.	United States

USA	United States of America
USBM	U.S. Bureau of Mines
USGS	U.S. Geological Survey
USSP NAD83	Arizona State Plane North American Datum 1983
VHMS	Volcanogenic Hosted Massive Sulfide
Wi	work index
WSE	Western States Engineering

1.0 SUMMARY

On August 27, 2015 Cardero Resource Corp. (Cardero) (TSX: CDU) entered into an Option Agreement with Redstone Resources Corporation (Redstone) under which Cardero has been granted an exclusive option to acquire a 100% interest in the Zonia Copper Project (the project), located in Yavapai County, Arizona, USA.

Global Resource Engineering was retained in December, 2017 to prepare a NI43-101 compliant Preliminary Economic Assessment (PEA) of the project using the October 2017 Amended Mineral Resource Estimate, prepared by Tetra Tech, as a basis for the economics.

1.1 Location, Property Description, and History

The Zonia property is located in Sections 11, 12, 13, and 14, T11N, R4W, Gila and Salt River Meridian, State of Arizona. Topographically, the project is located between French Gulch and Placerita Gulch approximately midway between Kirkland Junction and Walnut Grove, Arizona. The property is on the north end of the Weaver Mountains in the Walnut Grove Mining District, Yavapai County, Arizona. The geographic coordinates of the property are Latitude 34° 18’ 30” North and Longitude 112° 37’ 45” West. The nearest major city is Phoenix, Arizona, approximately 81 miles to the southeast. The Zonia Property can be reached from Phoenix, Arizona, by traveling 35 miles northwest on United States (U.S.) Highway (Hwy) 60 (paved) to Wickenburg, then 6 miles northwest on Hwy 93 (paved) to the junction with U.S. Hwy 89, then northeast on U.S. Hwy 89 (paved) 32 miles to Kirkland Junction, then east on the Walnut Grove gravel road for 3.5 miles to the Zonia road, and then south on the Zonia road 2.5 miles to the mine office. Kirkland Junction is 20 miles south of Prescott, Arizona on US Highway 89.

The Zonia Property consists of 261 patented (96) and unpatented (185) mineral claims and 566.85 acres of surface rights acquired from the State of Arizona; comprising 4,279.55 acres total. These claims include lode mining claims and millsite claims and are located in the Walnut Grove Mining District (Appendix A). Each mineral claim has a survey description and each patented claim was surveyed by a registered surveyor.

The Zonia property was discovered in the 1880s, most likely in the interest of developing gold-bearing veins. Prospecting for copper began in the 1890s, and a single stack smelter was built in 1900. From 1900 to present, the Zonia property has been extensively explored for copper by several holders and lessees. The property was mined by open pit from 1966 to March 1975 by McAlester Fuel Company (McAlester). Most recently, the property was explored by Copper Mesa Mining Corporation from 2008 to 2009, Redstone from 2009 to 2015, and currently Cardero from 2015.

1.2 Historical Drilling

Historical drilling, prior to Redstone and Copper Mesa Mining Corporation (Copper Mesa) involvement in 2008, totals some 553 drillholes on the property. Total historical drilling footage for the property is known to be greater than 139,000 feet; the drilled footage from 27 of the historical holes is unknown.

1.3 Geology and Mineralization

The Zonia property is in the southern part of the Basin and Range Transition province of the North American Cordillera, immediately south of the Colorado Plateau and north of the Basin and Range province. This section of the Basin and Range province in Arizona and New Mexico hosts a large number of base and precious metal mines and mineral occurrences. The Zonia deposit is hosted by the steeply dipping, northeast-trending, Precambrian Yavapai Series, which consists of schistose subvolcanic intrusions, volcanic flows and tuffs, and fine-grained sedimentary rocks. Portions of the area are covered by post-mineralization Quaternary basalt, fanglomerate, and alluvial material.

Rocks at the Zonia Property consist mainly of highly variably foliated rhyolitic to quartz monzonite, quartz-eye porphyry subvolcanic rocks, diorite, and minor diabase dikes, with highly schistose phyllites and chlorite schist along the southeast margin. Foliation strikes northeast and dips steeply to the northwest over most of the Zonia claims block, but changes to southeast dipping along the southeast margin in the Bragg Estate and Silver Queen claim block, and strikes east-west with northerly dips at the northeast end of the claims. This typical greenstone package is intruded and enclosed by younger Precambrian granitic batholiths which show only weak foliation at the margins.

The Zonia copper oxide deposit, as defined in this study, is the highly oxidized portion of a previously supergene-enriched metamorphosed porphyry deposit. Previously, it has been interpreted as a volcanogenic massive sulfide (VMS) deposit due to confusion of the protolith of the quartz-sericite schist along the northwest margin and the nature of the contact with adjacent, structurally overlying greenstone. The main mineralized unit is variably foliated quartz-feldspar porphyry (quartz-monzonite) and related sericite schist, with disseminated sulfides and stockwork quartz-sulfide veins that predominately pre-date the metamorphism.

Oxidation of the original chalcopyrite mineralization and younger secondary supergene chalcocite has been pervasive and deep, extending down over 250 meters (874 feet) in the central pit at the historical Cuprite shaft. Chrysocolla, malachite, azurite, tenorite, and cuprite are the most common copper minerals. Quartz and jasper accompany the copper minerals; oxides are ubiquitous in the mineralized zones. Higher copper grades are associated with contacts of the quartz-monzonite porphyry with acid-reactive mafic chlorite schist, which are zones of increased supergene deposition. Lower grades are associated with more massive enclosures of the quartz-monzonite porphyry, which were less permeable to supergene fluids.

1.4 2008-2010 Exploration Drilling Program

From 2008 to 2010, exploration drilling by Copper Mesa and Redstone totaled 54,211 feet. The drilling consisted of 131 drillholes, 77 HQ size core holes totaling 25,227 feet and 54 reverse circulation holes totaling 28,984 feet. Thirty-nine of the holes were designed to twin previous historical drilling for verification purposes. The remaining holes were drilled for exploration and resource development purposes. Drilling was completed by Boart Longyear (Boart) and Harris Exploration Drilling (Harris) under the supervision of Redstone corporate and contract geologic staff. Drillhole collars were surveyed by Mr. Gary Berg, a licensed Arizona surveyor. Downhole surveys were not performed on drillholes from the 2008 and 2009 programs or the reverse circulation drilling completed in 2010. Five diamond drillholes completed during the 2010 program were downhole surveyed, and no significant deviations were noted. A majority of the historical holes that were twinned in the 2008 and 2009 program were of vertical orientation and less than 350 feet in length.

For the 2017 NI 43-101 Technical Report of Mineral Resource Estimate (2017), Tetra Tech reviewed the geologic logging, sample selection, sample preparation, assaying, standards, duplicates, and blanks protocols and believed that the work is consistent with current standard practice and meets the requirements for calculating mineral resource estimations of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) standards and is compliant with NI 43-101.

1.5 Resource Estimation

For the 2017 Mineral Resource Estimate (2017), Tetra Tech completed an independent mineral resource and reserve estimate of the contained copper in the Zonia deposit. Geostatistics and resource estimation was done with MicroModel®. Additional statistical analysis was done with Statistica® and Excel®. Three-dimensional wireframes and model visualization was done with GemCom® software.

Table 1-1 shows the Tetra Tech estimated (2017) Zonia classified mineral resources at a base case cutoff of 0.2 %TCu. Mineral resources were reported within a shell generated using the Lerchs-Grossman algorithm. Mineral resources within an optimized shell are not mineral reserves and do not have demonstrated economic viability.

Table 1-1: Tetra Tech 2017 Zonia Classified Mineral Resources Base Case

Classification	Cutoff Grade Cu%	Tons M	Grade Cu%	Cu lbs M
Measured	0.2	15.4	0.42	129.3
Indicated	0.2	61.4	0.31	380.6
Measured + Indicated	**0.2**	**76.8**	**0.33**	**510.0**
Inferred	0.2	27.2	0.28	154.6

Notes:
(1) Resources are stated within a Lerchs-Grossman optimized shell using the following parameters: Mining (ore and waste) $1.5/ton, processing $3.4/ton, General and Administrative $0.45/ton, oxide recovery 73%, transition recovery 70%, and Cu price $2.50/lbs
(2) Columns may not total due to rounding, and
(3) One Ton is equal to 2,000 lbs or 0.9071847 Tonnes.
(4) Inferred Mineral Resources: It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

As part of this NI 43-101 Technical Report PEA, GRE used the 2017 Tetra Tech block model to generate new pit shells at metal prices from $0.50/lb to $5.00/lb Cu, in $0.25/lb increments. A description of the process is provided in Section 16. Preliminary analysis indicated that the $2.00/lb pit had the greatest potential for economic success. The pit shell for the $2.00/lb pit was imported into Geovia GEMS™ to design the ultimate pit layout using 45 degrees batter angle, 20-foot height, 12.7-foot bench width, 10% ramp grade, and ramp width of 100 feet for all but the lowest four benches, which were given a single-wide 50-foot ramp width. Table 1-2 shows the estimated classified mineral resources within the $2.00/lb pit at various cutoffs.

Table 1-2: Pit-Constrained Mineral Resources within the $2.00/lb Designed Pit

Category	Leachable Tons	Contained Copper lbs	Grade
0.12 Cutoff			
Measured	15.5	126.2	0.408
Indicated	65.1	362.3	0.278
Measured + Indicated	80.5	488.5	0.303
Inferred	26.4	131.1	0.248
0.16 Cutoff			
Measured	15.0	124.9	0.416
Indicated	58.2	342.9	0.295
Measured + Indicated	73.2	467.8	0.320
Inferred	22.2	119.1	0.268
0.20 Cutoff			
Measured	14.3	122.2	0.428
Indicated	48.3	306.9	0.318
Measured + Indicated	62.5	429.1	0.343
Inferred	17.0	100.1	0.295
0.22 Cutoff			
Measured	13.7	120.0	0.436
Indicated	42.0	280.4	0.334
Measured + Indicated	55.7	400.4	0.359
Inferred	14.4	89.6	0.310

Notes:

(1) Resources are stated within a floating cone optimized shell using the following parameters: Mining (ore and waste) $1.8/ton, processing $2.89/ton plus $0.12/lb copper SX/EW, General and Administrative $0.80/ton, oxide recovery 73%, transition recovery 70%, and Cu price $2.00/lbs

(2) Columns may not total due to rounding

(3) Inferred Mineral Resources: It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Cautionary statements regarding inferred mineral resource estimates:

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources will be converted into Mineral Reserves. Inferred resources are that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

1.6 Metallurgical Test Work

Several phases of metallurgical testing have been completed on the Zonia deposit. Initial studies were performed by Arimetco Incorporated (Arimetco) in 1995 followed by Constellation Copper Corporation (Constellation) in 2008. Redstone Resources Corporation (Redstone) conducted the most recent investigations on fresh core samples drilled in 2009 and 2010 along with trench samples taken from the deposit.

The test work that has been completed provides sufficient data to make informed assumptions on the potential copper leach recovery and acid balance for the deposit. The samples used for metallurgical test work were selected from across the Zonia deposit. Composite samples were prepared that represent varying feed grades, depths, and mineral types. The following table shows the copper deportment of the most recent samples.

Table 1-3: Zonia Sample Sequential Copper Assays (Redstone 2011)

Sample ID	Assays						
	TCu (%)	TFe (%)	ASCu (%)	CNCu (%)	CuRES (%)	Calc TCu (%)	Calc CuSOL (%)
High Secondary Copper	0.380	2.520	0.128	0.164	0.073	0.365	80.0
High Copper	0.499	3.540	0.350	0.010	0.120	0.480	75.0
Average Copper	0.292	2.330	0.199	0.006	0.088	0.293	70.0
Low Grade Copper	0.120	2.260	0.064	0.003	0.056	0.123	54.0
Intermediate Depth	0.349	3.060	0.237	0.013	0.093	0.343	73.0
Lower Depth	0.401	3.040	0.206	0.060	0.074	0.340	78.0
Run of Mine	0.585	3.320	0.466	0.011	0.155	0.592	81.0
Master Compostie	0.483	2.740	0.358	0.018	0.081	0.457	82.0

The summary results of the locked-cycle column leach tests on the above samples are shown below.

Table 1-4: Zonia Column Leach Results (Redstone 2011)

Sample	Crush Size (P80 mm)	Leach Cycle (days)	Cu Extraction (%)	Acid Cons	
				Net (kg/t)*	Net (kg/kg Cu)*
High Secondary Copper	25	107	69.5	7.7	2.7
High Copper	25	107	69.6	9.1	3.0
Average Copper	25	107	63.5	16.6	7.9
Lower Depth	25	107	54.0	17.9	9.8
Low Grade Copper	25	107	47.6	14.2	23.1
Intermediate Copper	25	107	58.8	14.5	7.1
Run of Mine	50	105	67.2	7.6	1.9
Master Composite	12	91	81.3	11.3	3.0
Master Composite	25	91	77.8	14.7	4.1
Master Composite	50	91	72.6	11.7	4.1

The master composite sample was developed from various drill locations and intercepts as a method to provide a reasonable representation of the complete Zonia deposit. Good copper extractions were achieved from the majority of the samples; ranging from 59% to 81% in a 91-day locked cycle column leach test (excluding the high sulfide and low grade samples). The copper extraction from the master composite sample with a nominal P_{80} size of 25 millimeters (mm) was 77.8%. The overall copper extraction based on the total copper assay (%TCu) for the deposit is estimated to be between 71% and 75%. For pit optimization copper recovery has been assigned based on mineral type; copper oxide minerals at 73%, secondary copper sulfides at 70% and primary copper sulfides at 0%.

1.7 Recovery Methods

The Zonia project would employ open pit mining with a conventional copper acid heap leach system. The mineralized material would be crushed in a three-stage crushing circuit to a nominal P_{80} size of 25mm.

The crushed material would be agglomerated with acid containing solutions using either raffinate or fresh sulfuric acid, and then be delivered to the heap via overland conveyor and grasshopper conveyors and stacked in 10-meter (m) lifts with a radial stacker operating in retreat mode. The heap is designed to contain up to 10 lifts for a maximum height of 100 m.

The heap leach pad and ponds are designed with a dual layer polyethylene liner system (LDPE and HDPE) with leak detection. Leach solution is transferred by gravity to either the pregnant solution (PLS) pond or the intermediate solution (ILS) pond. PLS is transferred to a conventional solvent extraction (SX) circuit for copper recovery from the solution. The depleted copper solution (raffinate) is transferred to the raffinate pond for reuse on the heap as the primary lixiviant. Solution is recycled to the heap via drip irrigation at a nominal rate of 12 liters per square meter per hour (12 lph/m^2). A storm water pond is provided that is designed to handle a 24 hour, 1 in a 100-year precipitation event.

The SX circuit consists of two extraction stages and one stripping stage (2+1) using a conventional mixer/settler arrangement. The loaded organic from the extraction stage is transferred to the stripper vessel producing a rich electrolyte solution for subsequent electrowinning. The copper depleted raffinate from the extraction circuit is recycled to the raffinate pond. Prior to electrowinning, the rich electrolyte is purified to remove entrained organic through column flotation and filtration.

The electrowinning (EW) circuit consists of two parallel banks of 50 polycement cells with 1m^2 cathodes. The plated copper cathodes are stripped using a mechanized striping system after being washed. Copper cathodes are then sampled and bundled for shipment.

1.8 Mining Methods and Economics

The selected base case uses the $2.00/lb pit with a grade cutoff of 0.17%. The mine life for this case is 8.6 years. Total material quantities for the base case are estimated to be 145 million total tons, 92.6 million leachable tons, 52.4 million waste tons, and 577.9 million pounds of contained copper. Of the contained copper pounds, 567.4 million are oxide material and 10.5 million are mixed material. The recovery rate for oxide material is projected to be 73% and for mixed material is projected to be 70%, resulting in 421.5 million pounds of recovered copper.

The Zonia project open pit would be mined using conventional open pit methods using off-highway trucks and loaders or shovels. Drilling, blasting, load, and haul would be used to remove overburden waste and leachable material. Waste would be hauled to disposal sites located as near as possible outside of the largest hypothetical pit rim. Ground pressure from stacking the waste is not expected to impact pit wall stability.

Leachable material would be hauled from the pit to the crusher. Crushed material would be transported via conveyors to the leach pad.

For the base case, contractor mining operations were selected.

To generate the economic model, GRE calculated revenues for the recovered copper using a copper price of $3.00/lb, refining charges of $0.032/lb, and transportation charges of $0.10/ton-mile and assuming an

80-mile transport to Phoenix, AZ. A ramp up was applied to year one, delaying 10% of the revenue that year and recovering it in year 9.

Operating costs were deducted from Net Revenue, yielding before-tax cash flow. Taxes were applied as follows:

- Depreciation of facilities capital costs was calculated on a straight-line, 10-year basis. Depletion allowance was calculated as 15% of revenues up to a maximum of 50% of before-tax income minus depreciation. Depreciation and depletion were deducted from before-tax cash flow to obtain taxable income.
- Federal tax at 21% was applied to the taxable income and Arizona severance tax at 7% was applied to the taxable income. The taxes were deducted from the taxable income, then the depreciation and depletion allowance were added back from taxable income to obtain after-tax cash flow.

Capital costs were deducted from the after-tax cash flow to obtain net cash flow after taxes. NPV at discount rates of 6%, 8%, and 10% and IRR were calculated from the net cash flow after taxes.

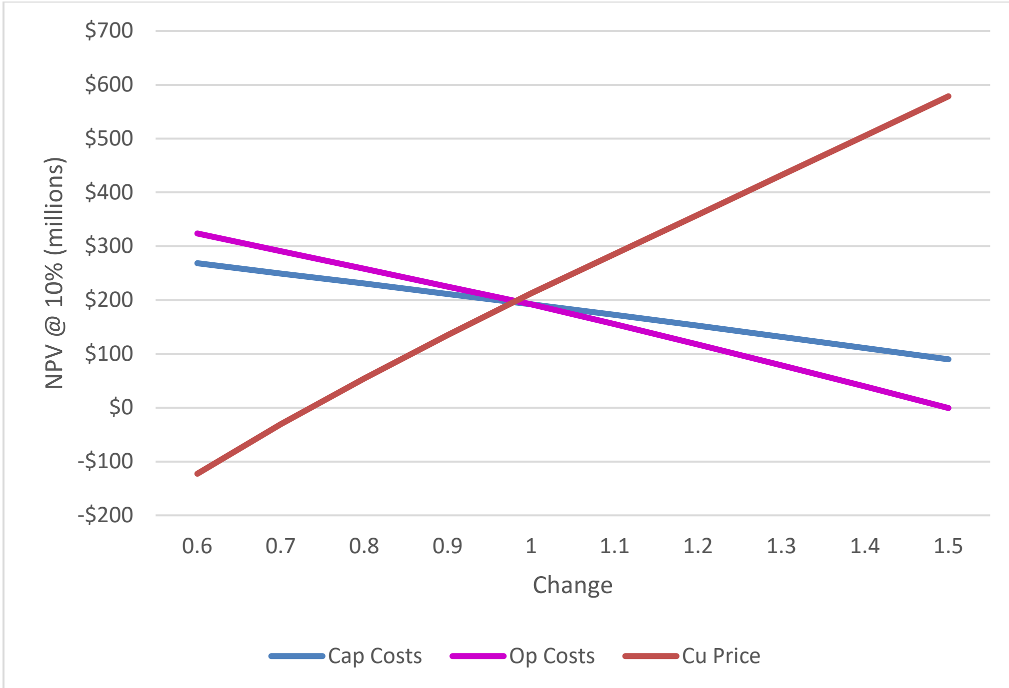
Table 1-5 shows the economic model results.

Table 1-5: Economic Model Results

Item	Result
NPV@6%	$225 million
NPV@8%	$192 million
NPV@10%	$163 million
IRR	29.0%
Initial Capital	$198 million
Cumulative Net Cash Flow After Taxes	$331 million
Payback Period	2.89 years
Op Cost/Lb	$1.46
All in Cost/Lb	$2.06

GRE evaluated the after-tax NPV@10% sensitivity to changes in copper price, capital costs, and operating costs. The results are shown in Figure 1-1.

Figure 1-1: NPV@10% Sensitivity to Changes in Copper Price, Capital Costs, and Operating Costs



The base case project scenario produces 92.6 million tons of leachable material over an 8.6-year mine life. The project is most sensitive to copper price, then operating costs, then capital costs.

At a copper price of $3.00/lb, the project shows an after-tax NPV@6% of $225 million, an NPV@8% of $192 million, an NPV@10% of $163 million, and an IRR of 29.0%. The project payback period is 2.89 years.

The project appears to be economically viable using open pit mining methods and heap leaching. The economic analysis suggests that the project should be further developed.

2.0 INTRODUCTION

Cardero Resource Corp. (Cardero) commissioned Global Resource Engineering Ltd. (GRE) to prepare a Preliminary Economic Assessment (PEA) Canadian National Instrument 43-101 (NI 43-101) technical report for the Zonia Copper Project in Yavapai County, Arizona, United States of America (USA). This report has been prepared in accordance with the Canadian Securities Administrators (CSA) NI 43-101, and the Resources have been classified in accordance with standards as defined by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "CIM Definition Standards – For Mineral Resources and Mineral Reserves," prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on December 17, 2010, as amended May 10, 2014.

The Qualified Persons for this report are Terre Lane and Dr. Todd Harvey of GRE and Dr. Rex Bryan of Tetra Tech. This technical report builds on the Resource Estimate Technical Report for the project prepared by Tetra Tech, Inc. (Tetra Tech) in 2017 (Tetra Tech, 2017).

2.1 Scope of Work

The scope of work undertaken by GRE is to prepare a Preliminary Economic Assessment (PEA) for the Zonia Copper Project and prepare recommendations on further work needed to advance the project to the Pre-Feasibility Study (PFS) stage.

This PEA is preliminary in nature and 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 under National Instrument 43-101. Readers are advised that there is no certainty that the results projected in this PEA will be realized.

2.2 Qualified Persons

The Qualified Persons responsible for this report are:

- Terre A. Lane, MMSA 01407QP, SME Registered Member 4053005, Principal Mining Engineer, GRE
- Todd Harvey, PhD, QP, Member SME Registered Member 4144120, Principal Process Engineer, GRE
- Rex C. Bryan, PhD, QP, SME Registered Member 411340, Senior Geostatistician, Tetra Tech

Practices consistent with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) (2010) were applied to the generation of this Resource Estimate.

Ms. Lane, Mr. Harvey, and Mr. Bryan are collectively referred to as the "Authors" of this PEA. Ms. Lane and Mr. Harvey visited the property on March 21 and 22, 2018. Mr. Bryan last visited the property on November 5, 2015. In addition to their own work, the Authors have made use of information from other sources and have listed these sources in this document under "References."

Table 2-1 identifies QP responsibility for each section of this report.

Table 2-1 List of Contributing Authors

Section	Section Name	Qualified Person
1	Summary	ALL
2	Introduction	ALL
3	Reliance on Other Experts	ALL
4	Property Description and Location	Rex Bryan
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Rex Bryan
6	History	Rex Bryan
7	Geological Setting and Mineralization	Rex Bryan
8	Deposit Types	Rex Bryan
9	Exploration	Rex Bryan
10	Drilling	Rex Bryan
11	Sample Preparation, Analyses and Security	Rex Bryan
12	Data Verification	Rex Bryan
13	Mineral Processing and Metallurgical Testing	Todd Harvey
14	Mineral Resource Estimates	R Bryan/T Lane
15	Mineral Reserve Estimates	Terre Lane
16	Mining Methods	Terre Lane
17	Recovery Methods	Todd Harvey
18	Project Infrastructure	T Lane /T Harvey
19	Market Studies and Contracts	T Lane /T Harvey
20	Environmental Studies, Permitting and Social or Community Impact	Terre Lane
21	Capital and Operating Costs	T Lane /T Harvey
22	Economic Analysis	T Lane /T Harvey
23	Adjacent Properties	Rex Bryan
24	Other Relevant Data and Information	ALL
25	Interpretation and Conclusions	ALL
26	Recommendations	ALL
27	References	ALL

Note: Where multiple authors are cited, refer to author certificate for specific responsibilities.

2.3 Sources of Information

The Resource Estimate portion of this Technical Report is taken verbatim from the 2017 Resource Estimate Update by Tetra Tech (Tetra Tech, 2017), which was based on data supplied by Cardero and Redstone with the use of historical data from the Shannon Copper Company, Anaconda Copper Company, Inspiration Consolidated Copper Company, Hammon Copper Company, U.S. Bureau of Mines and Gold Fields American Development Company, Miami Copper Company, McAlester Fuels, Homestake Mining, Equatorial Mining, and Arimetco International Inc. Drilling began in 1910 and continued intermittently with the most recent exploration drill program completed in 2011 and with evaluation of data continuing to present.

Information provided by Cardero and Redstone included:

- Legal Title Opinion
- Drillhole records
- Property history details

- Sampling protocol details
- Geological and mineralization setting
- Arizona Aquifer Protection Permit Application and supporting Appendices
- Data, reports, and opinions from prior owners and third-party entities
- Copper assays from original records and reports

This 2018 PEA relied on data provided to GRE by Cardero, notably, the 2017 block model generated by Tetra Tech.

2.4 Units

All measurements used in the Zonia Project are Imperial units. Tonnages are in short tons, and grade is reported as percent (%) unless otherwise noted. Costs and revenue are expressed in 2018 U.S. Dollars with no allowance for escalation, currency fluctuation, or interest.

3.0 RELIANCE ON OTHER EXPERTS

The authors relied on statements by Cardero concerning geological and exploration matters in Sections 7.0, 8.0 and 9.0, and Cardero and Redstone concerning legal and environmental matters included in Sections 4.0 and 5.0 of this report.

Dr. Bryan relied on statements and documents provided by Dana T. Jurika, Chief Financial Officer of Redstone and Mark I. Pfau, Consulting Geologist to Redstone regarding:

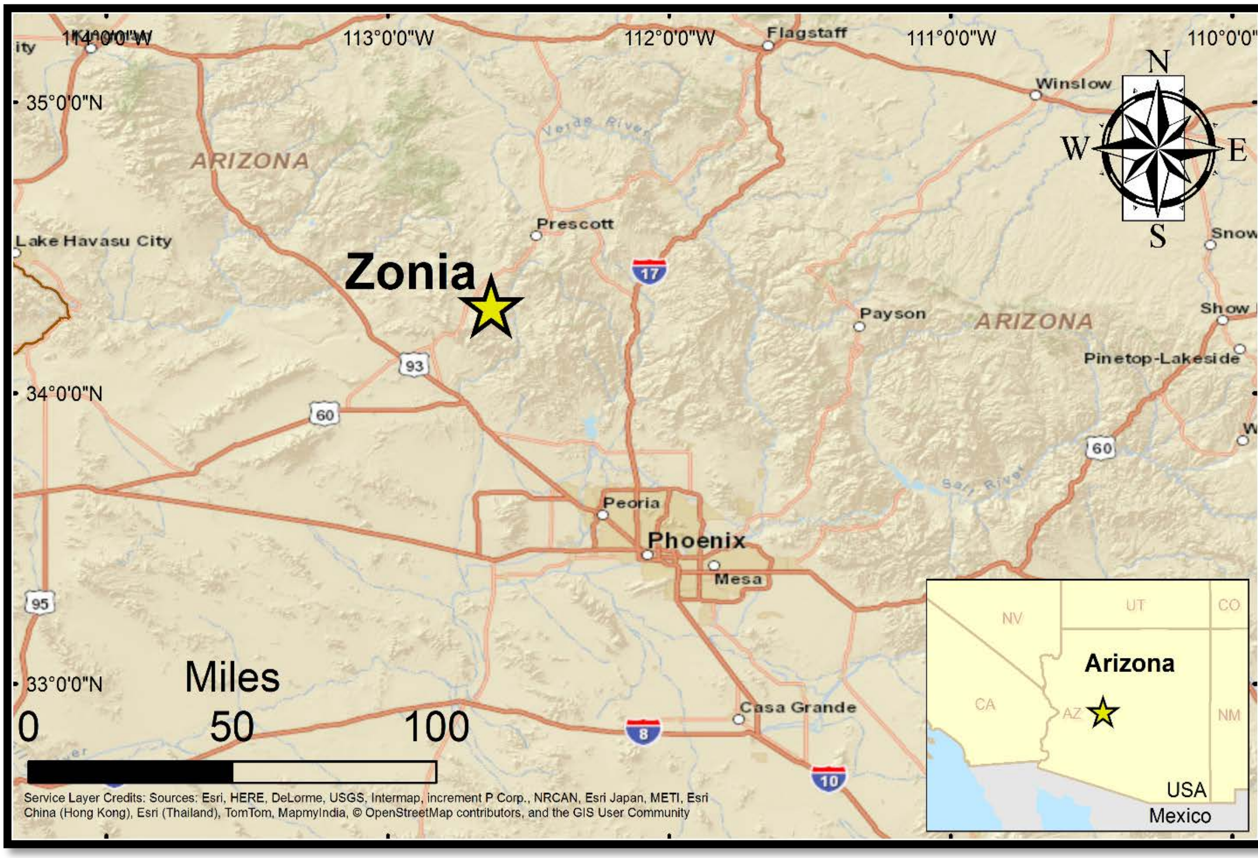
- Limitations of environmental liabilities associated with past operations
- Current status of environmental permitting and compliance
- Permitting requirements to initiate mining
- Location of the claims
- Claim and land ownership standing
- Surface access agreements.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Zonia Project is located in Yavapai county, Arizona, USA, as shown on Figure 4-1

Figure 4-1: Project Location Map



The Zonia property is located in Sections 11, 12, 13, and 14, T11N, R4W, Gila and Salt River Meridian, State of Arizona. Topographically, the project is located between French Gulch and Placerita Gulch, approximately midway between Kirkland Junction and Walnut Grove, Arizona. The property is on the north end of the Weaver Mountains in the Walnut Grove Mining District, Yavapai County, Arizona. The geographic coordinates of the property are Latitude 34° 18' 30" North and Longitude 112° 37' 30" West. The nearest major city is Phoenix, Arizona, approximately 81 miles to the southeast.

4.2 Area of the Property, Mineral Tenure, Title

On August 27, 2015, Cardero entered into an Option Agreement (as amended) with Redstone Resources Corporation (Redstone) under which Cardero has been granted an exclusive option to acquire a 100% interest in the Zonia copper project (Zonia or the Project), located in Yavapai County, Arizona, USA. To exercise the option, Cardero must pay Redstone $2,225,000 ($201,350 paid) and issue 16,500,000 common shares of Cardero Resource Corp. to Redstone (1,000,000 issued). Details of the option agreement are shown in Table 4-1.

Table 4-1: Option Agreement

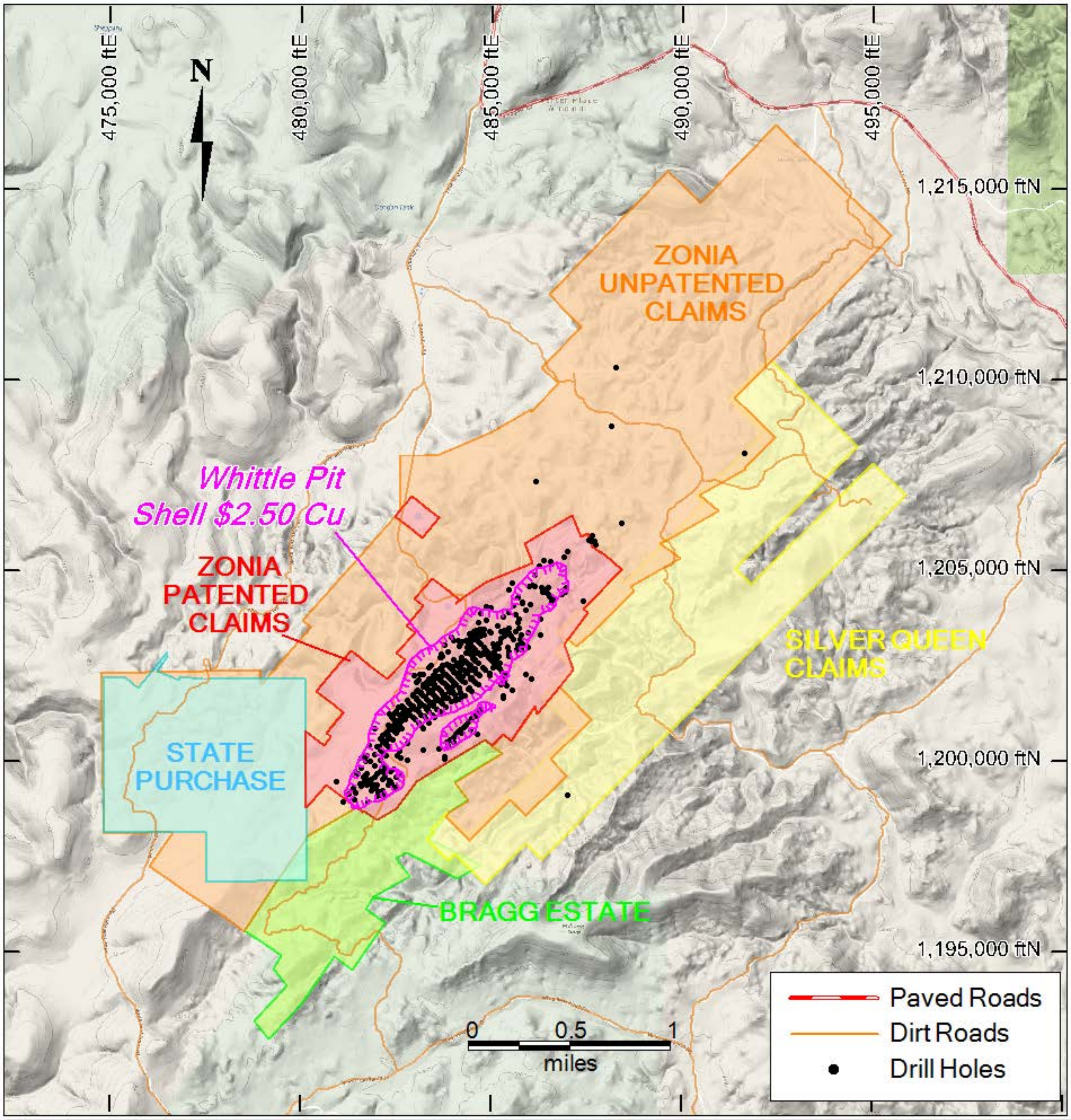
Date	Cash to Redstone ($)	Cardero Shares
Initial Payment	$25,000 (paid)	-
BLM Fees	$26,350 (paid)	-
October 15, 2015	$150,000 (paid)	1,000,000 (issued)
January 31, 2016	$75,000 (paid)	1,500,000 (issued)
July 31, 2016	$75,000 (paid)	-
January 31, 2017	$450,000 (paid)	2,500,000 (issued)
July 31, 2017	-	2,500,000 (issued)
January 8, 2018	$500,000 (paid)	4,000,000 (issued)
October 31, 2018	$923,650	5,000,000
Total	**$2,225,000**	**16,500,000**

The Zonia Property consists of 261 patented (96) and unpatented (185) mineral claims and 566.85 acres of surface rights acquired from the State of Arizona; comprising 4279.55 acres total (Figure 4-2). These claims include lode mining claims and millsite claims and are located in the Walnut Grove Mining District (Appendix A). Redstone reports that each mineral claim has a survey description and that each patented claim was surveyed by a registered surveyor.

Under United States law, title to unpatented mineral claims does not expire as long as a payment of an annual fee per mineral claim is made. Redstone reports that all fees for unpatented mineral claims are current. Ongoing obligations to maintain title are on the order of $25,000. Dr. Bryan has not performed any title searches to confirm land title for this report but has reviewed historical title opinions. Cardero has confirmed that all land title is in good standing as of the effective date of this report

Potential heap leach facilities could be located on adjacent surface acreage of 566.85 acres that were acquired by purchase from the State of Arizona in 2013. The Bureau of Land Management (BLM) continues to control the underlying mineral estate, but Cardero holds all mineral concessions underlying this acreage.

Figure 4-2: Zonia Property Map, Including Adjacent Silver Queen Claims



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The Zonia Property can be reached from Phoenix, Arizona, by traveling 35 miles northwest on United States (U.S.) Highway (Hwy) 6060 (paved) to Wickenburg, then 6 miles northwest on Hwy 93 (paved) to the junction with U.S. Hwy 89, then northeast on U.S. Hwy 89 (paved) 32 miles to Kirkland Junction, then east on the Walnut Grove gravel road for 3.5 miles to the Zonia road, and then south on the Zonia road 2.5 miles to the mine office. Kirkland Junction is 24 miles southwest of Prescott, Arizona, on U.S. Highway 89.

The nearest railroad access is the Burlington Northern railroad located 5 miles from Zonia, at Kirkland Junction. The Burlington Northern has a siding at the town of Kirkland, approximately 16 miles to the northwest of Zonia. Phoenix is the nearest major airport with regularly scheduled air service. There are local airports at Prescott and Wickenburg, and there is an old airstrip at Zonia.

5.2 Climate and Physiography

The climate in the Zonia area is semi-arid with an average rainfall of 18 inches per year. Winter temperatures are moderate with occasional light snowfall. The mean temperature during the summer months is usually less than 100° F. Mining operations could be carried out year-round.

Elevations on the property range from 4,800 feet above mean sea level (amsl) in the south to 4,100 feet in the north. Relief varies from gently rolling to moderately steep with local rugged canyon areas. Most of the vegetation is brush and grass and various types of cacti, manzanita, scrub oak, cat’s claw, and piñon. There are cottonwood trees along French Gulch and Zonia Creek. Zonia is located above the Sonoran desert zone.

5.3 Local Resources and Infrastructure

Sufficient water appears to be available on the Zonia property to conduct operations. Hydrogeological characterization activities at the Zonia mine (2010) included an assessment of existing wells at the site and the performance of hydraulic testing in selected wells. The results of the hydraulic testing indicate that the fractured rock aquifer has a moderate hydraulic conductivity (average value of approximately 5×10^{-4} centimeters per second [cm/sec]), suitable for water supply purposes. The alluvial material surrounding French Gulch is more permeable, with an estimated hydraulic conductivity of 1×10^{-3} cm/sec. Power is available at the mine site from the Arizona Public Service grid through a 33kV upgradeable power line. There are electrical substations at the mine. Local labor for mining is available, and residents in the area have previously worked at the mine.

6.0 HISTORY

6.1 Property History

Historical resources and reserves referenced in this section are not considered current and do not comply with the current standards of NI 43-101 and CIM definitions for reporting mineral resources and reserves, and are included for historical reference and context only. None of the reported numbers have been, or can be, verified and therefore should not be relied on.

The Zonia deposit was discovered in the 1880s, and remnants of old arrastras (rock crushers) indicate that the main interest was the mining of gold-bearing veins. The property was prospected for copper during the 1890s, and a single stack smelter was built in 1900. This smelter apparently did not operate successfully because visible copper is present in the old slag dumps.

In the early 1900s, the Shannon Copper Company of Clifton, Arizona, completed six churn drillholes on the property. Results, however, were not satisfactory, and work was stopped in 1911. From 1916 to 1920, a syndicate, reported to include the Anaconda Copper Company and Inspiration Consolidated Copper Company, explored the property and sank the Cuprite Shaft to a depth of 874 feet. This syndicate developed five levels and approximately 4,000 feet of laterals and crosscuts but did not outline a mineralized body that could be considered economic. The Cuprite Shaft was reported to produce about 150 gallons per minute (gpm) of water.

The property was then acquired by the Hammon Copper Company (Hammon) which, in 1927, rehabilitated part of the underground workings, explored adjacent gold bearing zones, and built a pilot leach plant. Hammon planned to put the property into production at the rate of 600 tons per day (tpd) as a copper leaching operation, but the plan was terminated during the depression.

In 1942, the U.S. Bureau of Mines (USBM) evaluated the property for strategic copper reserves and completed 2,035 feet of trenching and 2,960 feet of diamond drilling. USBM also carried out check sampling at the 210 underground level and conducted mill tests. This was done with the cooperation of Gold Fields American Development Co.

A Mr. Gottbehut of Los Angeles, California, is reported to have leased the property in 1947 and 1948 and shipped a small but insignificant amount of ore from the 210 level.

From around 1955 to 1957, Miami Copper Company (Miami Copper) conducted an exploration and evaluation program. This consisted of an aerial photogrammetric survey (black and white as well as color aerial photographs), topographic surveys, and geologic mapping and sampling programs. Miami Copper also completed 26 churn drillholes and 24 air rotary holes, but the results failed to meet their targets for tonnage and grade, and they terminated the program.

In 1964, McAlester Fuel Company (McAlester) obtained the Zonia property and carried out a program of airborne reconnaissance, surface geologic mapping, and an extensive drilling program of short drillholes. After delineation of the Zonia copper deposit and favorable results of pilot leaching studies, open pit mining and heap-leaching began in 1966. About 17.1 million tons were mined from the pit, of which 7.1 million tons were stacked on leach heaps and 10 million tons were reportedly dumped as waste.

The material mined by McAlester for leaching was placed on three asphalt-lined leach pads and continuously leached by sprinkling with diluted sulfuric acid on the pads. The copper minerals dissolved, and the pregnant solution was then passed to the launder, where copper was precipitated from solution in the form of cement copper on scrap iron or salvaged de-tinned cans. The waste solution was then treated with additional sulfuric acid and recycled to the leach areas. The sulfuric acid was largely produced at the property from native sulfur. From 1966 to March 1975, McAlester reportedly produced 33.2 million pounds of cement copper from the Zonia Mine by heap leaching of 7.1 million tons placed on heaps. However, there is uncertainty about the achieved copper recovery because the grade of material placed on the heaps was never properly evaluated (Scott Wilson RPA, 2006).

In addition to the heap leaching operation, two areas containing about 7.7 million tons of broken material were reportedly blasted and leached in situ by McAlester and the USBM. McAlester blasted material in the northern portion of the open pit by what was at the time reported as the world’s largest non-nuclear explosion. This area was then leached *in situ* from mid-1972 to March 1975, when the mine closed. In a 1979 report, McAlester reported that 2.7 million pounds of copper had been recovered from 7.7 million tons estimated to have been affected by *in situ* leaching. They also estimated 20,500,000 tons of material at an average grade of 0.3 % total copper (TCu) remained, exclusive of the *in situ* leached area.

In 1971, McAlester granted an option to Homestake Mining Company (Homestake) to explore and purchase the Zonia property. Homestake subsequently conducted a two-phase exploration program designed to identify potential economic sulfide targets below the oxide zones. Homestake, however, terminated their option over the property in about 1975.

In 1977, Phelps Dodge Corporation (Phelps Dodge) conducted an exploration program and reportedly got favorable results. However, it is reported that Phelps Dodge was not able to come to terms with McAlester.

In 1980, American Selco Ltd. (Amselco) acquired an option on the property and conducted an exploration program including reconnaissance mapping, geochemical sampling, and drilling. Results of this work defined two areas of interest with anomalous gold values in various samples. Drill results, however, were not encouraging, and Amselco returned the property to McAlester.

In about 1981, Nerco Minerals Company (Nerco) acquired the property and conducted an evaluation program on the south end. The program included drilling through the leach pads. In 1982, Nerco returned the property to McAlester when it decided to pursue other mining properties.

In about 1982, Queenstake Resources Ltd. (Queenstake) conducted an evaluation of the property with an emphasis on the gold potential as shown in the early Hammon data. This program was restricted to the area north of the pit.

In 1983, McAlester offered their interest in the property for sale which was purchased by Antioch Resources Ltd. (Antioch), who conducted exploration activities at the site with Queenstake as its joint venture partner who eventually transferred the title to the Zonia Company.

In October 1988, Zonia Company of Prescott, Arizona, acquired title to the property; they, in turn, leased it to Arimetco, Inc. (Arimetco) in late 1992, but Arimetco did not immediately take possession of the property. This was delayed until environmental liability issues were resolved, with Arimetco to be held harmless with regard to past operating practices.

In January 1993, Arimetco began working on the site pursuant to a water quality remediation plan on behalf of the Zonia Company. Concurrent with the remediation work, Arimetco conducted exploration on the site to determine the feasibility of reopening the mine and constructing a modern processing facility. In August 1993, an option to purchase the property was negotiated and signed with Zonia Company.

In 1995, Western States Engineering (WSE) prepared a feasibility study for Arimetco and concluded that the property appeared to be economically viable under market conditions of that time (Western States Engineering, 1995). Arimetco continued with plans to develop and build a new zero discharge, full-containment, mine and plant unit. The concept for extracting the copper from the deposit was to mine, crush, acid-cure, then leach and recover it by the electrowinning process rather than to mine, stack, and leach the material as done by McAlester. In preparing the feasibility study, WSE used a reserve report prepared by Mine Reserve Associates, Inc. (MRA) in 1994 (MRA, 1994), which estimated the mineable reserves to be 34.7 million tons at an average grade of 0.366 %TCu, at a cutoff grade of 0.19 %TCu. MRA estimated a strip ratio of 0.45 tons of waste to one ton of ore. In its reserve estimate, MRA reduced the grade of the samples in the database in the area blasted and leached in situ by McAlester.

In 1996, Arimetco went into liquidation proceedings due to issues unrelated to Zonia. In about 2000, Equatorial Mining North America, Inc. (Equatorial) optioned the property from the United States Bankruptcy Court for the District of Arizona, and from April 2000 to March 2001, completed 18,243 feet of reverse circulation drilling (RCD) in 39 drill holes. The assay certificates are available, and six drill logs have been located. The data from these holes were added to the MRA database. In 2001, Equatorial terminated its option on the property.

In July 2004, Ste-Genevieve Resources Ltd. (SGV) purchased the Zonia property from the United States Bankruptcy Court for the District of Arizona for US$350,000 as a result of Arimetco's liquidation proceedings. SGV conducted work to assess the economics of putting the Zonia Project back into production, including a NI 43-101 Technical Report by Scott Wilson RPA (2006).

In March 2008, Ascendant Copper Corporation, which legally changed its name to Copper Mesa Mining Corporation (Copper Mesa) in July of 2008, completed the acquisition of SGV, which included the Zonia property. In June 2008, Copper Mesa retained Tetra Tech to provide technical and engineering services with an objective of completing a Feasibility Study for the property in early 2009.

In support of these work efforts, in June 2008 Copper Mesa acquired a comprehensive metallurgical data package pertaining to Zonia. This metallurgical data and descriptive report, generated by Metcon Research, Inc. (Metcon) of Tucson, Arizona in 2007-2008, consists of information developed in column tests performed on several tons of material collected from four trenches cut within the existing Zonia open pit, and bottle roll tests on both the same material and numerous samples of previously collected drill cuttings from lower depths within the Zonia deposit. All tests were directed at determining the mineralized material's response to treatment by heap leaching and solvent extraction / electro-winning (SX/EW) recovery of copper. Indicated recoveries, based on various material sizes and leach times, were between 71% and 81%.

In August 2008, Copper Mesa commenced two separate drill programs. The first program consisted of 17 drill holes (approximately 1,800 feet) using sonic drilling to produce material both for subsequent assaying as well as for the testing of geotechnical characteristics of areas being considered as sites for construction of facilities for the proposed mining operation. As part of this program, Copper Mesa tested certain historically mined and processed material located on existing leach pads to determine if adequate copper values remain to warrant reprocessing. In addition, the drilling provided assay material from an area containing approximately 5 million tons of material, blasted by the USBM in the mid-1970s, which was the subject of an *in situ* leaching experiment.

A second drill program, comprised of 16 diamond drillholes (approximately 3,000 feet), was also conducted. The primary purpose of this program was to twin 16 historical drill holes for confirmation of historical tonnage and grades through check assays to facilitate the re-estimation and reclassification of the copper resource for the project. Core material from this program was saved to be used to provide additional confirmation of the metallurgy of the resource in the proposed mine plan area.

Upon completion of the drilling programs, work was suspended at the property due to lack of funding. In August 2009, private investors who had invested in Copper Mesa called in a loan of $1.7 million, which was secured with all shares of Redstone Resources, the subsidiary which held the Zonia project. Being unable to meet loan repayment requirements, Copper Mesa subsequently transferred all ownership of Redstone Resources Corporation to the lender in exchange for releasing the Company from all liabilities pertaining to Redstone.

6.2 Historical Drilling

Historical drilling on the property is identified, summarized, and updated in Table 6-1 with new information from Redstone. Historical drilling prior to Redstone and Copper Mesa's involvement in 2008 consisted of a total of 553 drillholes on the property. Total historical drilling footage for the property is known to be greater than 139,000 feet; the drilled footage from 27 of the historical holes is unknown. Drilling conducted on the property more recently by Redstone and Copper Mesa is discussed in Section 11.0.

Table 6-1: Summary of Historical Drilling

Year	Company	Type of Drilling	No. of Holes	Amount (feet)
1910-1911	Shannon Copper	Churn	6	Unknown
unknown	unknown	unknown	8 (D-Series)	1,100
unknown	unknown	unknown	1 (DH Series)	350
unknown	unknown	unknown	6 (T Series)	120
unknown	unknown	unknown	2 (UGDH-Series)	385
1942-1943	USBM	Core	11	2,690
1955-1956	Miami Copper	Rotary & Churn	24	6,912.5
1956	Miami Copper	Churn	25	10,062
1963-1964	Bunker Hill	Rotary & Churn	11	4,170
1964	McAlester	Air Rotary	357	85,500
1964	Cominco	unknown	4	Unknown
1965	McAlester	Air Rotary	1	295

Year	Company	Type of Drilling	No. of Holes	Amount (feet)
1974-1977	Homestake	Core	7	6,421
1979	Amselco	unknown	4	1,060
1981-1982	Newco Minerals	Auger	30	1,740
1994	Arimetco	unknown	12	unknown
2001	Equatorial Mining	Auger	5	unknown
2000-2001	Equatorial Mining	RC	39	18,243

MRA's 1994 work included compilation of drilling and sampling data from earlier programs into a digital database. The MRA database was updated in 2001 by Equatorial. Mintec used the updated 2001 database to prepare its estimates (Mintec, 2001); Scott Wilson RPA (2006) used the same database for their estimate. Tetra Tech further updated the electronic database with Copper Mesa and Redstone drilling. Figure 6-1 shows the locations of the historical drillholes.

6.3 Verification of Historical Data

In 1995, MRA compiled drilling and sampling data from earlier programs and placed them into a digital database. Scott Wilson RPA was able to obtain a copy of this database from Mintec. The MRA database was updated by Equatorial in 2001 and used by Mintec to prepare a resource and reserve estimate and a mine plan for Equatorial in 2001 (Mintec, 2001). The 2001 database used by Mintec was also made available to Scott Wilson RPA and is used for the resource estimate in this report (Scott Wilson RPA, 2006). Scott Wilson RPA was able to compare a number of files in the 1995 and 2001 databases to be assured that they are the same databases prior to the updates done by Equatorial.

Scott Wilson RPA reported a spot-check of 268 assays during a site visit to Zonia (Scott Wilson RPA, 2006). The assay certificates from the 2001 Equatorial drilling program were checked against the drillhole database obtained from Mintec, and no errors were found.

During their site visit on June 26, 2006, Scott Wilson RPA selected eleven bags of sample rejects from the 2000 to 2001 Equatorial drilling program. The rejects are stored on pallets in a locked building at the Zonia site. Eleven sample bags were chosen, opened, and sampled by Scott Wilson RPA. The samples were placed in plastic sample bags with blind tags and sent by courier to Scott Wilson RPA's Toronto, Ontario, office. The samples were then sent to SGS Laboratories in Don Mills, Ontario, for independent assays for total copper as well as acid-soluble copper. Scott Wilson RPA was satisfied that custody of the samples was maintained during their transfer from the Zonia sampling site to the assay lab in Toronto.

At SGS, the samples were crushed, ground, and assayed for %TCu, %AsCu, and gold. The %TCu content was determined by the Inductively Coupled Plasma (ICP) method, and the %AsCu content was determined by treating the sample with 5% sulfuric acid. The gold content was determined by the fire assay method. The sample preparation and assay procedures used were described by Scott Wilson RPA (2006).

Table 6-2 provides a comparison of the Equatorial assay results with the Scott Wilson RPA assay results for %TCu as well as %AsCu values. The %AsCu to %TCu assay ratios vary from 40% to 88%, and average 72%. The assay result ratios indicate that much of the copper is potentially soluble by heap leaching.

Figure 6-1: Historical Drillhole Location Map

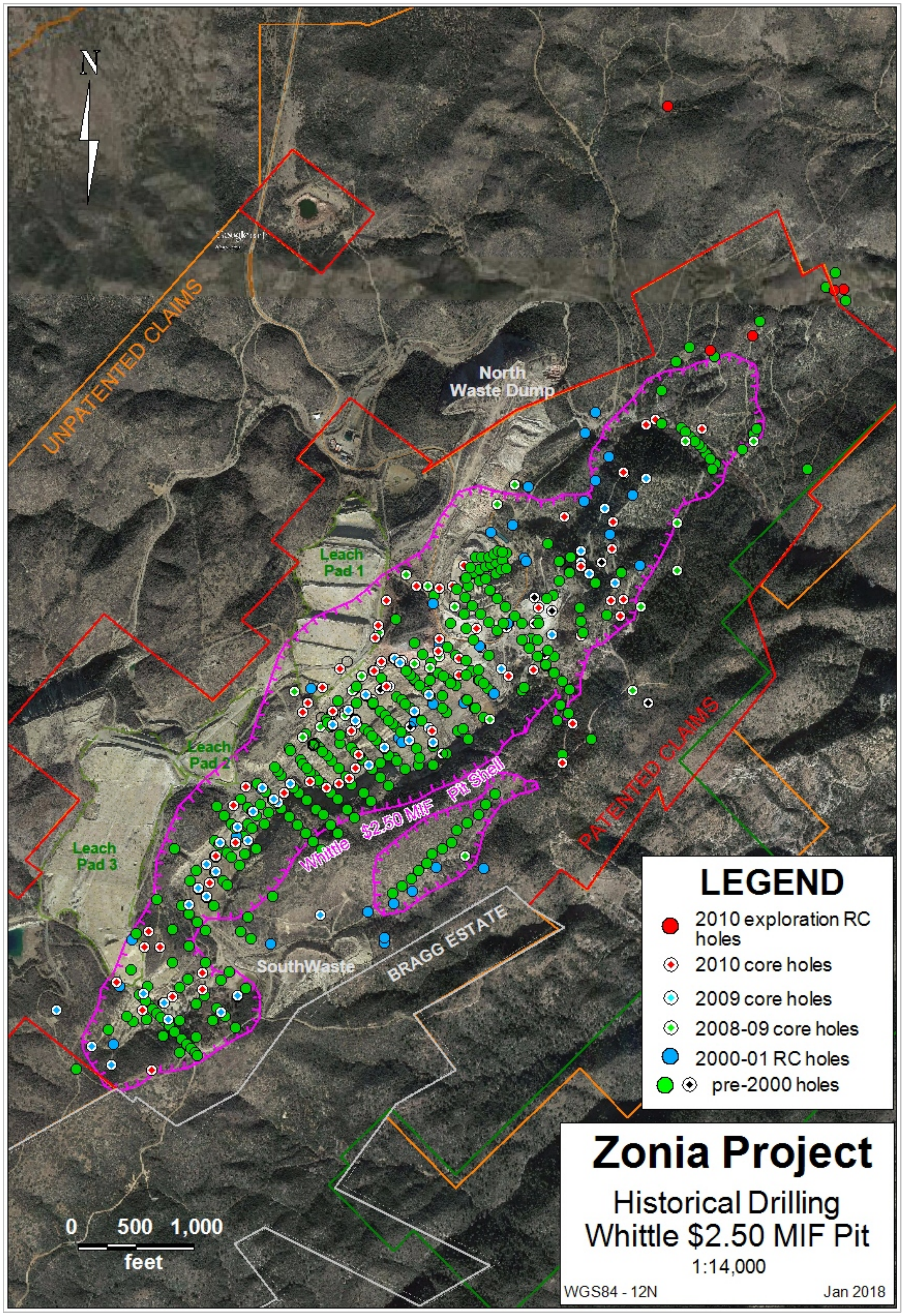


Table 6-2: SWRPA - Independent Sampling Results - June 2006

Scott Wilson RPA Sample No.	Equatorial Hole No.	From (ft.)	To (ft.)	Assay Value (%TCu)		Assay Value (% AsCu)	
				Equatorial	Scott Wilson RPA	Scott Wilson RPA	%AsCu to %TCu Ratio
71301	E-506	270	275	0.15	0.14	0.10	71%
71302	E-506	275	280	0.59	0.56	0.49	88%
71303	E-506	265	270	0.90	0.86	0.75	87%
71304	E-506	280	285	0.41	0.45	0.36	80%
71305	E-506	260	265	0.67	0.64	0.51	80%
71306	E-505	55	60	0.29	0.28	0.20	71%
71307	E-505	50	55	0.38	0.36	0.26	72%
71308	E-511	75	80	0.26	0.25	0.13	52%
71309	E-511	70	75	0.18	0.17	0.11	65%
71310	E-513A	190	195	0.23	0.23	0.19	83%
71311	E-513A	185	190	0.10	0.10	0.04	40%
Averages				**0.378**	**0.367**	**0.286**	**72%**

Based on the limited checks done on the database and the independent sampling, and on the fact that copper has been produced from the property, Scott Wilson RPA (2006) concluded that the drillhole data available were acceptable for estimation of mineral resources at the Zonia deposit. As noted in the Mineral Resources and Mineral Reserves Section, however, Scott Wilson RPA classified all of the mineral resource as inferred because of the lack of confirmation of the bulk of the drillhole database.

Through the work of Tetra Tech and other independent consultants, a total of 443 of the pre-2008 drillholes were able to be verified and were used in the development of the resource estimate in this and the 2017 Resource Estimate reports (Tetra Tech, 2017).

6.4 Historical Mining

Historical mine production numbers in this section are included for historical reference and context only and they are no longer relevant to the current project. None of the reported numbers have been, or can be, verified and therefore should not be relied on.

The Zonia property has historically had both underground and surface mining operations. The underground openings are no longer accessible and hence, cannot be confirmed; this information is presented for completeness.

The property was mined by open pit methods for a short time with a fairly small tonnage of material mined and copper produced. From 1966 to March 1975, McAlester produced 33.2 million pounds of cement copper from the Zonia Mine by heap leaching of 7.1 million tons placed on heaps. McAlester estimated a grade of 0.6 %TCu for the run-of-mine (ROM) material placed on the heaps, which indicates a recovery of 35%.

In addition to the heap leaching operation, two areas containing about 7.7 million tons of broken material were reportedly blasted and leached *in situ* by McAlester supported by the USBM. McAlester blasted material in the northern portion of the open pit by what was at the time reported as the world's largest non-nuclear explosion. This area was then leached *in situ* from mid-1972 to March 1975, when the mine closed. In a 1979 report, McAlester reported that 2.7 million pounds of copper had been recovered from 7.7 million tons estimated to have been affected by in situ leaching (McAlester, 1979).

6.5 Historical Resource and Reserve Estimates

Four historical resource and reserve estimates have been completed for the Zonia copper project.

6.5.1 Western States Engineering (1995)

In 1995, WSE prepared a Feasibility Study for Arimetco based on reserves estimated at 34.7 million tons at 0.366% total copper, using a 0.19% Cu cutoff grade (Western States Engineering, 1995).

This historical reserve is not considered current and does not comply with the current standards of NI 43-101 and CIM definitions for reporting mineral resources and reserves, and is included for historical reference and context only. The reported numbers have not been, nor can be, verified and therefore should not be relied on.

6.5.2 Mintec (2001)

In April 2001, Mintec prepared a preliminary resource and reserve estimate of the Zonia Project for Equatorial, based on the drill hole data provided by Equatorial (Mintec, 2001). Mintec, however, was not mandated to verify the database. In its report, Mintec estimated that the Zonia reserves included 73.6 million tons at an average grade of 0.325 %TCu of proven and probable reserves and 4.7 million tons at an average grade of 0.326 %TCu of possible reserves. Mintec used a cutoff grade of 0.18%TCu and a strip ratio of 0.35:1. Mintec also reduced the grades in the blasted and leached area.

These historical resources and reserves are not considered current and do not comply with the current standards of NI 43-101 and CIM definitions for reporting mineral resources and reserves, and are included for historical reference and context only. The reported numbers have not been, nor can be, verified and therefore should not be relied on.

6.5.3 Scott Wilson RPA (2006)

Scott Wilson RPA estimated the mineral resources of the Zonia copper deposit in 2006 using the results of the previous rotary and RCD holes (Scott Wilson RPA, 2006). As part of its estimate, Scott Wilson RPA carried out an interpretation of the mineralized zones and developed a block model of the copper deposit. Excluded from the mineral resource estimate were the previously mined open pit and the parts of the deposit where in-situ leaching was carried out in the past.

The Scott Wilson RPA historical resource estimate is not considered current, though the classification was in accordance with the 2006 CIM Definition Standards incorporated in NI 43-101. It is included here for historical reference and context only.

The mineral resource estimate is shown in Table 6-3 and totaled 63 million tons averaging 0.37 %TCu at the recommended cutoff grade of 0.25 %TCu. Scott Wilson RPA classified all of the Zonia mineral resource as inferred since only a limited amount of verification was carried out on the drillhole database.

Table 6-3: Historical Zonia Inferred Mineral Resource Estimate

Cutoff Grade (%TCu)	Short Tons (millions)	Average Grade (%TCu)	Contained Copper (millions of pounds)
0.30	42	0.41	343
0.25	63	0.37	460
0.20	90	0.32	584
0.15	102	0.31	627

Source: SWRPA NI 43-101 Technical Report October 16, 2006

The database used for the current mineral resource estimate includes data originally compiled by MRA in 1994. The 2000-2001 Equatorial drillholes were added in 2001. The current database consists of 113,297 feet of drilling in 447 holes.

A peculiarity of the Zonia database is that low-grade zones of several historic drillholes show assays for every second 5-foot interval. Inspection of the variability of assay values down these holes and assays in adjacent holes with complete assaying intervals indicates relatively low variability in the copper values along the drillholes. For this reason, the holes with incomplete assaying were kept in the database, and the non-assayed intervals were treated as unsampled intervals.

Based on the work of Scott Wilson RPA, MINTEC, and other independent third-party reviewers, Tetra Tech concluded that the data correctly represents the tenor of the mineralization and are useable in the estimation of mineral resources. Redstone completed a twin drillhole program with the express intent of proving this assumption, and the results of this work are presented in Section 14 of this report.

6.5.4 Tetra Tech Mineral Resource Estimate and PEA (2011)

In April 2011, Tetra Tech completed a PEA for Redstone (Tetra Tech, 2017). The report was not made publicly available because Redstone is a private company. Table 6-4 outlines the historical mineral resource estimated in 2011 at a 0.2 %TCu cutoff and unconstrained by a pit optimization.

Table 6-4: Historical 2011 Mineral Resource Estimate

Classification	Cutoff TCu%	Tons (000)	TCu% Kriged	Cu (lbs) (000)
Measured	0.2	15,414	0.42	129,693
Indicated	0.2	68,243	0.31	418,057
Measured + Indicated	**0.2**	**83,658**	**0.33**	**547,793**
Inferred	0.2	51,074	0.28	282,848

The historical resource estimate is not considered current, although the classification was in accordance with the 2006 CIM Definition Standards incorporated in NI 43-101. It is included here for historical reference and context only. The Tetra Tech PEA included measured, indicated, and inferred mineral resources; mineral resources are not mineral reserves and do not have demonstrated economic viability. Inferred mineral resources are too speculative for use in defining reserves, but have reasonable prospects for upgrading with continuing exploration.

6.6 Historical Metallurgy and Process Engineering

Historical mine production and grades referenced in this section are not considered current and are included for historical reference and context only; they are no longer relevant to the current project. None of the reported numbers have been, or can be, verified and therefore should not be relied on.

McAlester mined about 7.1 million tons from the open pit in the 1960s and 1970s and stacked it on three heaps. From 1966 to 1975, more than 33 million pounds of cement copper were recovered from this material by leaching with sulfuric acid. This represents a recovered grade of 0.23% copper. McAlester estimated a grade of 0.6 %TCu for the material placed on the heaps, which indicates a recovery of 35%. There is uncertainty in the grade of the material placed on the heaps and therefore in the recovery. Because on the uncertainty of the grade of the material placed on the heaps, it is uncertain how much copper remains in the heaps and how much of that copper is recoverable. Scott Wilson RPA considered that a program is warranted to test the three heaps for copper content and for copper recoverability by renewed leaching.

Approximately 7.7 million tons was blasted and leached *in situ*, from which some 2.7 million pounds of copper was recovered (McAlester, 1979), for a recovered grade of 0.02% copper. The recovery in the *in situ* leach area was reported to be about 7% based on a pre-leaching grade of 0.27 %TCu estimated by McAlester (1979).

Arimetco completed a series of column tests on some 10,500 pounds (lbs) of Zonia mine material in 1994 to 1995 at the Mountain States R&D International, Inc. laboratory. This work was supervised by Lerchs, Inc. The tests were designed to evaluate the process of crushing followed by an acid cure and leaching of the copper. The results indicated that about 75.4% of the copper could be recovered in column tests using this process. The tests were completed in 10-foot high, eight-inch diameter columns. Material was crushed to minus one inch. Leach, Inc.'s conclusion (1995)was as follows:

> Test Results suggest that if the Zonia ore is crushed to minus 1 inch, placed on a heap with lifts of 20 feet, cured (or pretreated) with a strong acid solution then leached with acidified raffinate at an irrigation rate of 0.045 [gallons per minute per square foot] gpm/ft^2 for 40 days it will be possible to achieve a recovery of 70 percent and a [pregnant leach solution] PLS grade averaging 2.8 [grams per liter] gpl copper. Net acid consumption will depend on the conditions of the cure step but can be expected to be about 6.5 pounds of acid per pound of copper leached.

The mining history and the testwork indicate that copper can be recovered by leaching Zonia copper mineralization. In Scott Wilson RPA's view, however, more metallurgical testwork is needed to determine the optimal process and the amount of copper that can be recovered by leaching. Scott Wilson RPA recommended 1,000 feet of drilling for metallurgical test samples.

6.7 Historical Feasibility Studies

In 1995, WSE prepared a feasibility study for Arimetco and concluded that the property was economically viable under market conditions of that time (Western States Engineering, 1995). The concept was to mine, crush, acid-cure, then leach and recover the copper by electrowinning processes rather than to mine, stack, and leach the ore as done by McAlester (Scott Wilson RPA, 2006).

WSE's feasibility study process plans included:

- Open-pit mining with 85-ton haulage trucks loaded with Caterpillar 992 front end loaders
- Mining to occur 5 days/week on a three shift basis
- Ore was to be crushed to minus 1-inch, agglomerated with sulfuric acid, and stacked on leach heap pads
- Leach pads irrigation with raffinate solution over approximately 644,000 square feet
- Leach cycles would average 40 days and were projected to produce 1,900 gpm of solution bearing copper at 2.8 grams per liter
- Copper was to be extracted by solvent extraction/electrowinning on a 7 day/week basis in a circuit consisting of a single train of two extraction and one strip mixer-settlers
- Copper in electrolyte solution processed into pure copper cathode sheets
- Daily production rate of 60,000 pounds copper

The economic assumptions of this study are considerably out-of-date and hence further conclusions and findings are not presented here.

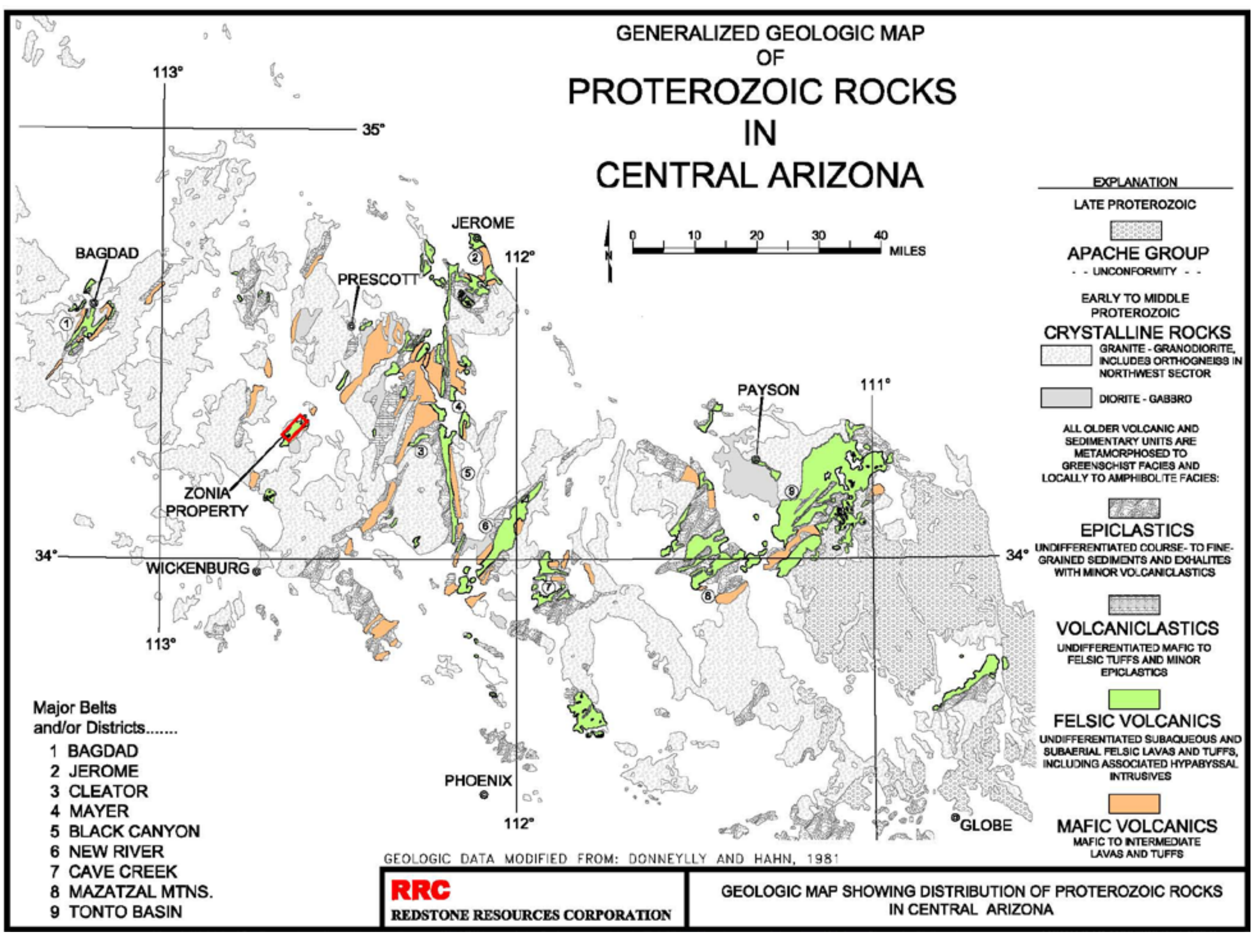
7.0 GEOLOGIC SETTING AND MINERALIZATION

In the physiographic context, the Zonia copper deposit is located in the Transition Zone between the Colorado Plateau to the north and the Basin and Range to the south. The Transition Zone is host to numerous base and precious metal deposits in Arizona and New Mexico. Within the Transition zone, Zonia lies within the Yavapai block, between the Mojave block to the northwest and the Mazatzal block to the southeast (Williams, 1991).

7.1 Regional Geologic Setting

The regional geology of central Arizona is dominated by Early Proterozoic greenstone complexes enclosed by Precambrian granitic batholiths, typical of Precambrian, continental shield rocks elsewhere in the world. Anderson (1989) defined the Yavapai Supergroup as all volcanic and related strata in the volcanic belts of central and northern Arizona. The Yavapai rocks appear to be Proterozoic island arc and rift basin rocks that accreted to the ancestral Wyoming craton during northwest directed subduction about 1750 Ma, and range in age from 1800 to 1740 Ma. The volcanic belts are generally older and more mafic in the northwest and younger and more felsic in the southeast. Figure 7-1 shows the distribution of Proterozoic age rocks in central Arizona, the location of major base metal mining districts hosted by the Proterozoic rocks, and the location of the Zonia project.

Figure 7-1: Generalized Geologic Map of Proterozoic Rocks in Central Arizona



Source: modified from Donnelly and Hahn (1991)

Rocks of the central Yavapai Supergroup are termed the Prescott Volcanic Belts, which is broken down (oldest to youngest) into the Bradshaw Mountains Group, the Mayer Group, and Black Canyon Creek Group (Anderson, 1989). Zonia lies within the Bradshaw Mountains Group, which comprises mostly deep-water oceanic units of tholeiitic basalt and related gabbro, tuff, greywacke, and chert.

The Yavapai metavolcanics are enclosed within large, older masses of gabbro to quartz monzonite intrusions, in which the oldest bodies are typically not the most deformed. Relative ages can only be assigned based on cross-cutting relationships. There was a gradual increase in alkali content with time, to the northwest, in the direction of overall younging. The porphyritic plutons, like that at Zonia, are grouped by Anderson (1989) as “Late Tectonic.”

The older units are intruded by Laramide age quartz monzonite. The Yavapai rocks are overlain by Tertiary to Quaternary fanglomerates (Gila Formation), Quaternary age basalts, and Quaternary gravels and sands as alluvium.

There are two distinct types of deformation affecting the Proterozoic volcanic rocks in central Arizona. The first, and most widespread, is vertical deformation caused by pure shear during the diapiric rise of the enclosing granitic batholiths (Anderson, 1989; Figure 7-2). This type of deformation is characterized by highly variable strain, mostly vertical stretching lineations, and a relationship between degree of strain and the stretching lineation (the greater the strain, the more vertical the lineation). The second type of deformation is more typical of Archean greenstone belts, and is folding and shearing related to the Yavapai orogeny, dated at 1700 Ma according to Karlstrom and Bowring (1991).

Karlstrom and Bowring (1991) divide the Yavapai into major structural blocks of which Zonia is situated at the boundary between the Green Gulch and Big Bug blocks, the two blocks of which are separated by the northeast trending Chaparral Shear zone shown in Figure 7-3. The Chaparral shear zone is not firmly delineated and mapped on the ground at Zonia, but such zones tend to be a kilometer or more in width. Much of the deformation noted on the southeast side of Zonia is believed to be related to the Chaparral shear zone.

Figure 7-2: How Vertical Deformation Affected the Yavapai Supergroup of Central Arizona

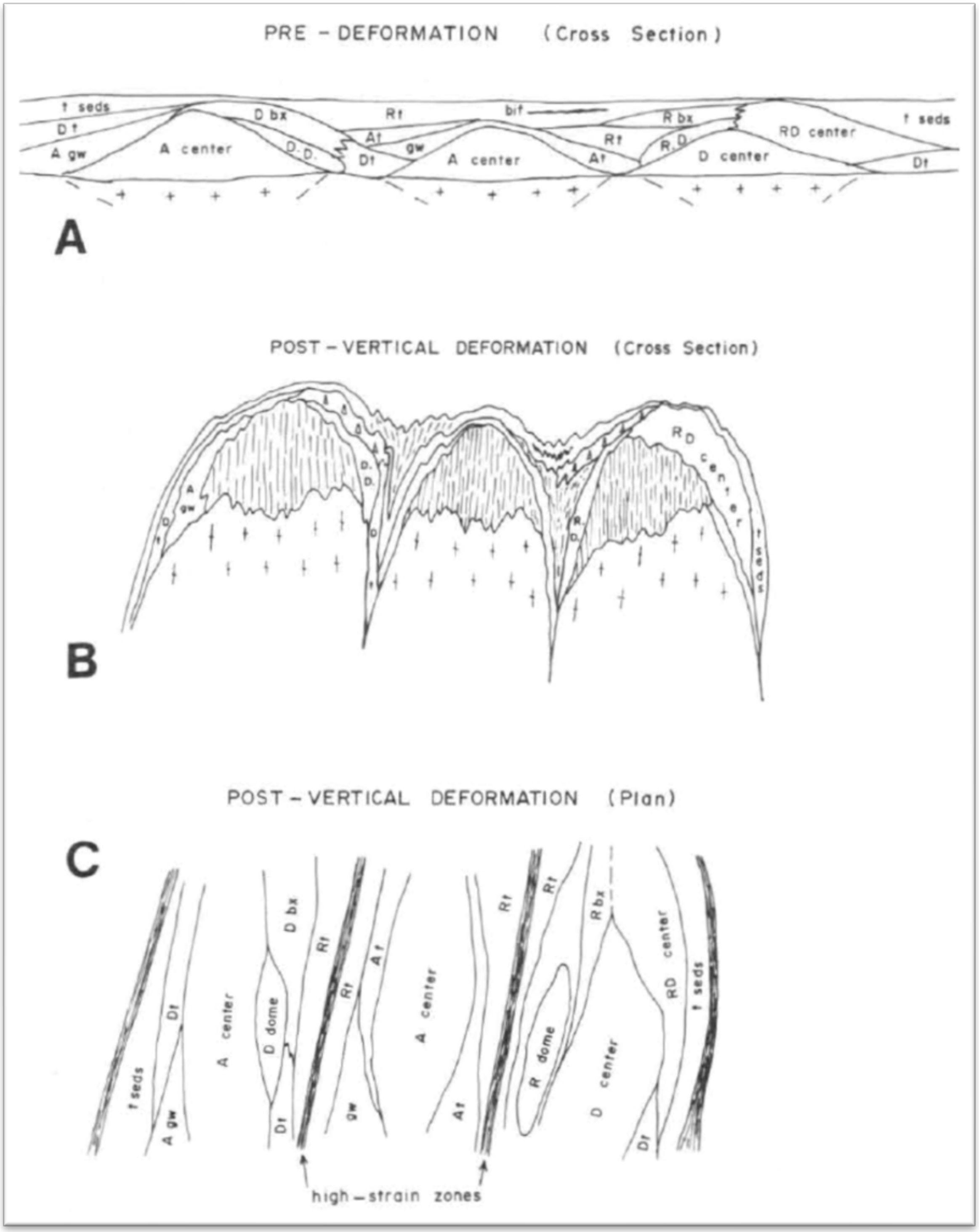
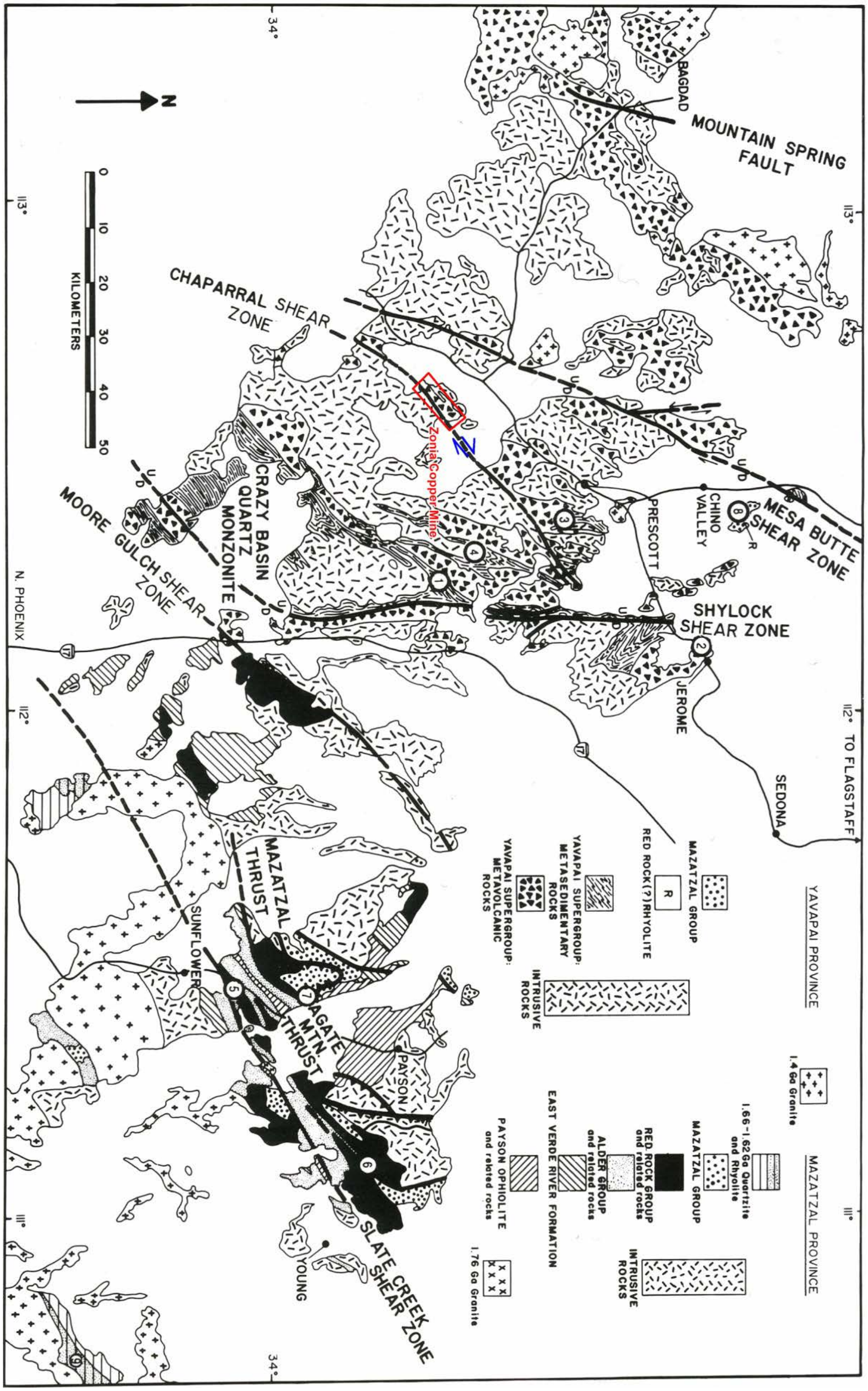


Figure 7-3: Major Structural Components of the Yavapai Block



7.2 Local and Property Scale Geology

The Zonia mine geology is taken from historical reports written by Homestake (1978), McAlester Fuel (1979), Queenstake Resources (1983), Arimetco (1995), and Davis (2007); and by internal studies conducted by Copper Mesa Mining, Redstone Resources, and Cardero personnel from 2015 to the present.

The Zonia copper deposit is situated in Proterozoic-age, greenschist-grade, metavolcanic and metasedimentary rocks of the Prescott Volcanic Belt, part of the Yavapai Supergroup. In the broadest sense, the geologic column is granitic rocks intruding basal greenstone and gabbro, subvolcanic quartz-monzonite porphyry (variably deformed to quartz sericite schist), and chloritic to calcareous phyllites/slates.

7.2.1 Geological Units

The general distribution of rock units within the mine area is described here from structurally highest to lowest (i.e. northwest to southeast), with abbreviations used in this text and on resource cross sections. Figure 7-4 consists of stratigraphic columns compiled from the three major areas of the Zonia deposit (Redstone, 2012). Note that many of the units are actually intrusions and therefore the layering is more structural than stratigraphic. Also, tops-up is not known as there are no preserved facing indicators. The columns are presented here to show the overall distribution of the various rock units, and therefore also the variable strain, across the deposit.

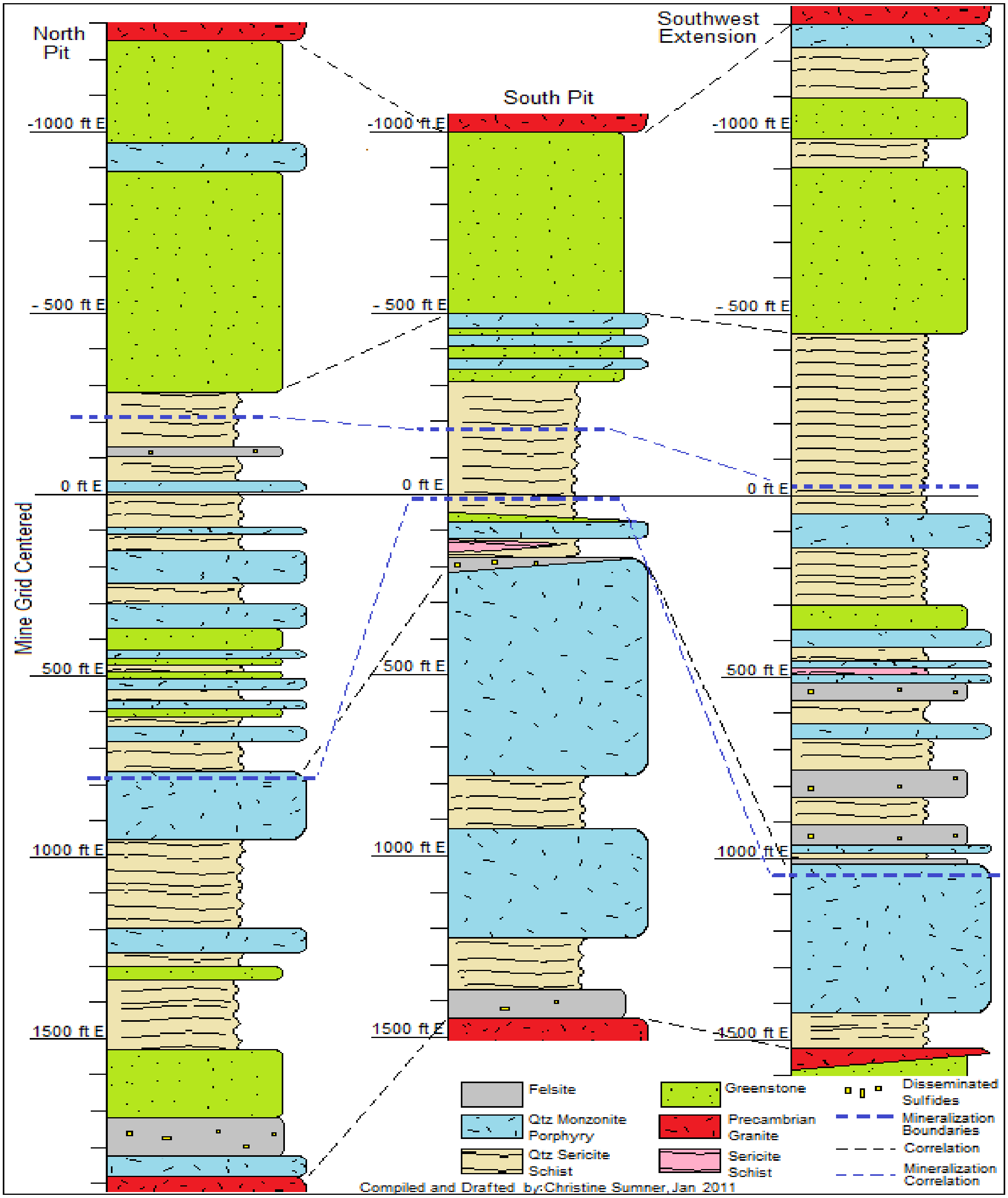
Ob Overburden, including Quaternary alluvial sands and gravels, Quaternary basalt (Bas), and Tertiary to Quaternary age Gila formation unconsolidated fanglomerate (Cg). The fanglomerate is a significant unit as it forms the northwest high wall in the North Pit and has been free standing at about 60 degrees for over 30 years. Some localities have basalt overlaying fanglomerate as at Red Hill (Figure 7-5).

PCg Precambrian granitic rocks, principally as monzogranite to granodiorite intruding older foliated units. This is a light brown, massive, leucocratic, holocrystalline rock. Contacts with Grn are frequently faulted but also crosscut by greenstone and felsite dikes. The PCg also contains numerous massive “bull” quartz veins and pegmatite dikes. The PCg shows little of the foliation common to the metavolcanics except near the contacts.

Grn Greenstone, massive to weakly foliated, dark green, chloritic meta-basalt and/or diabase, including some breccia and associated tuff, as well as porphyritic textured subvolcanic intrusions denoted as Anp. While only minor chalcopyrite occurs in the primary zone, within the oxide zone there are xenoliths of Grn that absorbed large quantities of mobilized copper, and were termed “ore pods” by early underground miners as grades exceeded 2% Cu (Cameron, 1975).

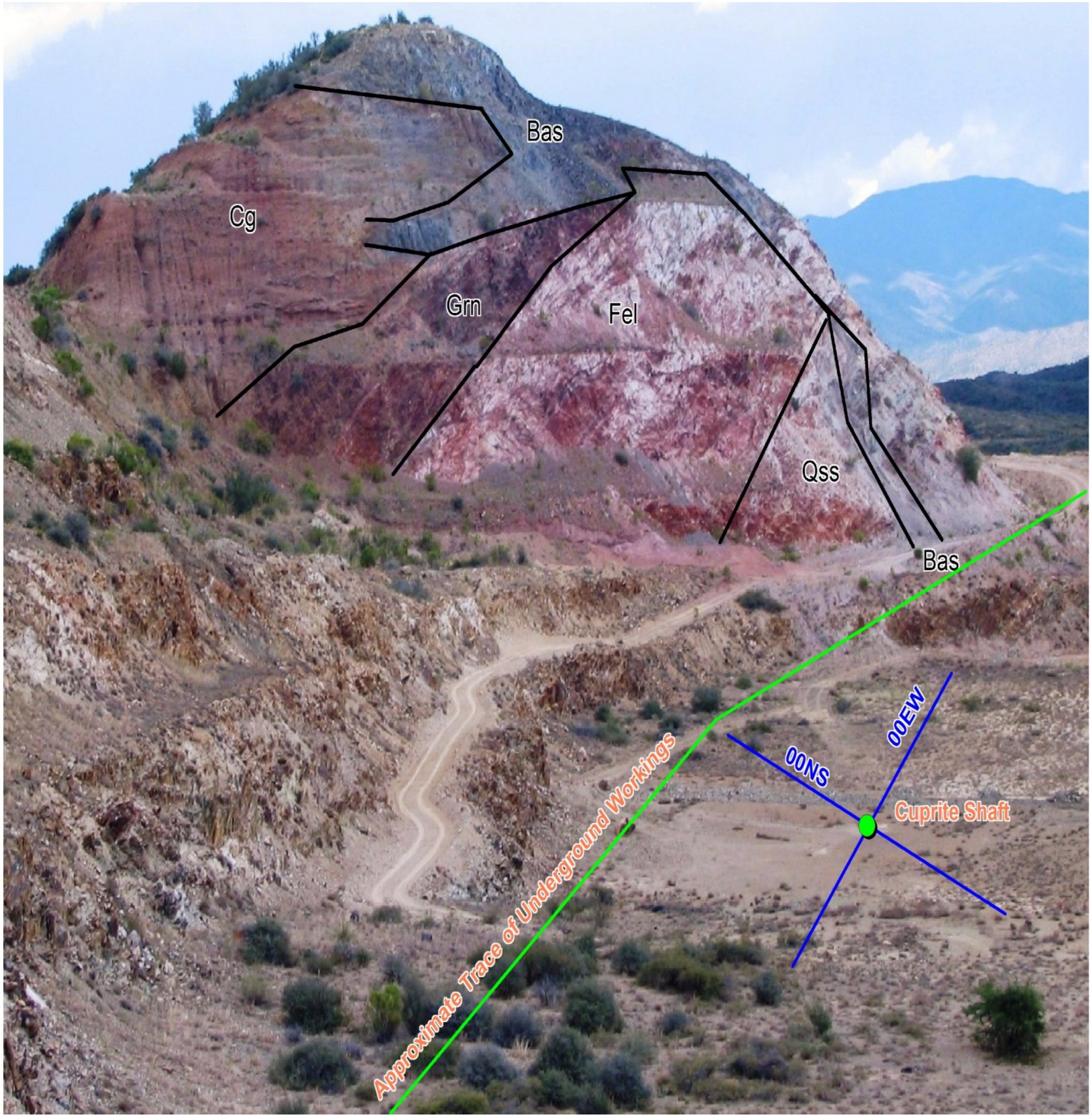
Qss Quartz sericite schist, the principal ore host at Zonia, and the schistose equivalent of altered unit Qmp. The unit is whitish brown in color with abundant limonite staining due to common sulfide clasts and is thoroughly foliated. The contact relationships between units Qss, Qmp, and Fel are ambiguous and prone to mapping/logging errors. This unit includes the fine grained Ss unit, which, while distinct, probably represents hydrothermal clay alteration of Qmp.

Figure 7-4: Stratigraphic Columns Compiled in the North Pit, South Pit, and Southwest Extension



Source: Redstone, 2012. Top of the columns corresponds with the NW side of the deposit.

Figure 7-5: Red Hill Looking Northeast in the North Pit



Qss also includes the Mss unit noted in some early core logging. Mss, logged as "metasiliceous sediments", and thought to represent exhalite horizons, is probably strained silicic alteration within unit Qmp. The numerous gossans (Gos) noted in the area historically and in core were thought to be derived from "exhalite horizons", but more closely resemble ferruginous supergene horizons produced during oxidation of the sulfide mineralization.

Qmp Quartz monzonite porphyry, a relatively light colored, quartz-feldspar porphyry subvolcanic intrusion that where massive forms the center "rib" through the deposit, Where variably strained and schistose, it interfingers mostly with Qss, but also with units Fel and Grn, and wth Cs progressively towards the east. The porphyry is not uniform in composition or texture, and varies from monzonite to diorite (Dio). The unit includes those rocks identified as Dap (dacite porphyry) which is now believed to be altered margin rocks to Qmp.

Qmp contains weak quartz stockwork vein development, minor disseminated sulfides, and weak potassic alteration in the form of orthoclase and secondary biotite. Its ability to fracture in a conjugate pattern has allowed descending oxidizing solutions carrying copper oxide minerals and silica to deposit that mineralization in the fractures. Foliation has developed along distinct planes that historically were mapped as Qss, but are actually distinct zones of weakness in the Qmp related to its rheology.

Fel Felsite rocks, the various units of which are undifferentiated. This includes light tan colored schist, interpreted once as tuffs, and flows, but now interpreted as a late-phase felsic intrusion, as some felsite is notable for a lack of mineralization, it is primarily a massive textured rock, and it includes quartz-eye-porphyry (unit Qep). Mineralization consists of bands and disseminations of sulfides.

Qst, Stg Quartz sericite talc and Sericite-talc granoblasts, which are minor talc bearing units in the footwall that are not well defined and might be related to discontinuous gabbro occurring in the same area.

Cs Chlorite schist, also referred to as phyllite or chloritic phyllite in the older reports. These olive green to black schists occur throughout the area, but are most prevalent in the southeast. Early maps of Homestake show calcareous horizons at the southeast edge of the property (Cameron, 1975).

Figure 7-5 shows Cg overlain by Quaternary basalt (Bas) as part of the post-mineralization overburden profile. Steep northwest dipping greenstone is structurally underlain by felsite, which is in turn structurally underlain by quartz sericite schist (Qss), the principal mineralized rock. The felsite intrudes the greenstone, as first noted by Cameron (1975), and changes laterally into Qss and then Qmp within the pit. All units are cut by Quaternary basalt feeder dikes (right side).

The origin of the mine grid at the Cuprite shaft is shown, approximately 90 feet below the original topographic surface, along with the surface trace of the underground workings. The highest grades of mineralization drilled at Zonia from 2008 to present were along the road in the center of Figure 7-5 and the next bench above to the left; and along the Fel-Qss contact. The maroon hematite of Red Hill is classic oxidized remnants of supergene chalcocite mineralization.

Lesser-mineralized Qmp occurs in the middle of the deposit and grades into Qss, with interleaved schistose margins and abundant manganese oxides (Figure 7-5). Qmp is more massive than its schistose Qss equivalent and overall less mineralized. Mineralization mostly occurs where downward migrating oxidized fluids that carried silica, iron, manganese, and copper entered into its well-developed, but relatively wide-spaced, fracture system.

Figure 7-6: North Pit Looking Toward the Northeast, With Red Hill to the Left

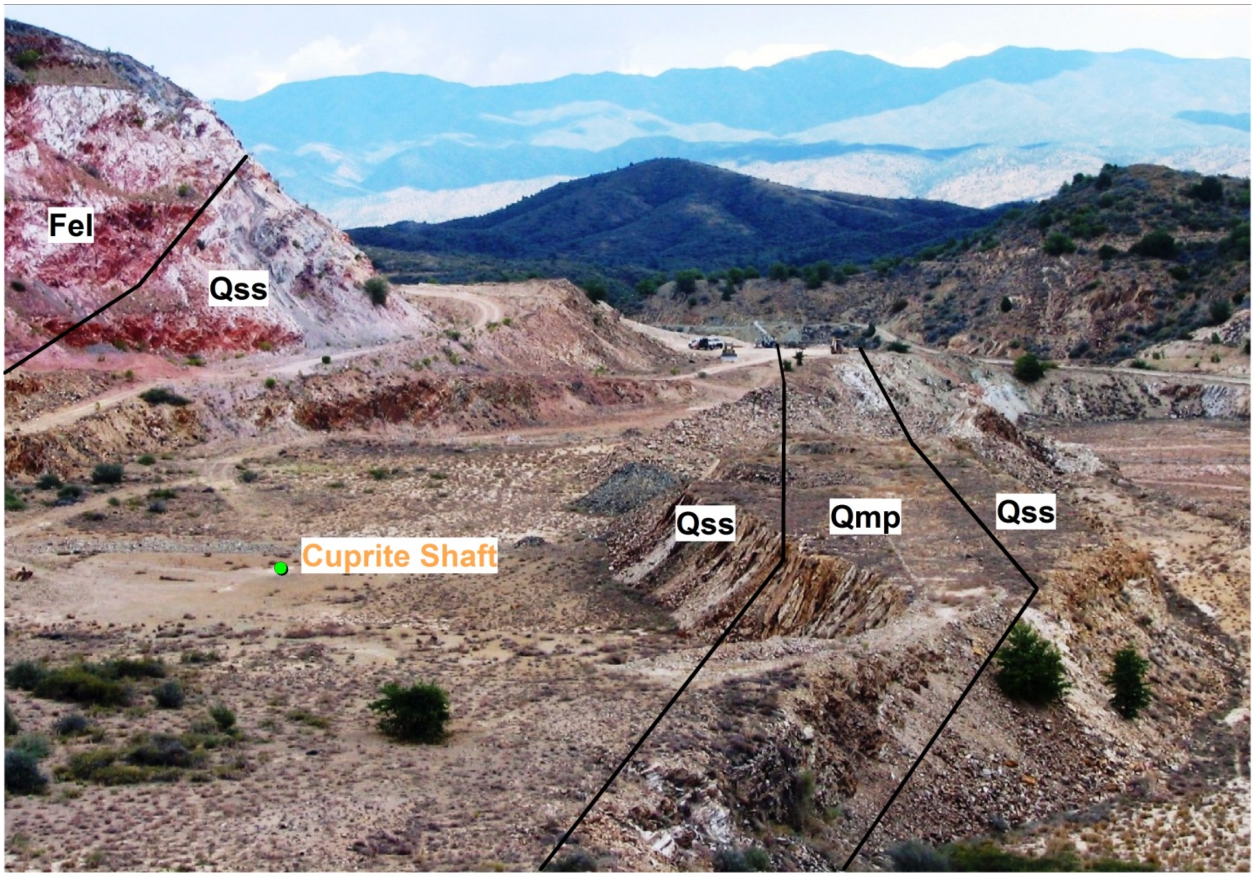
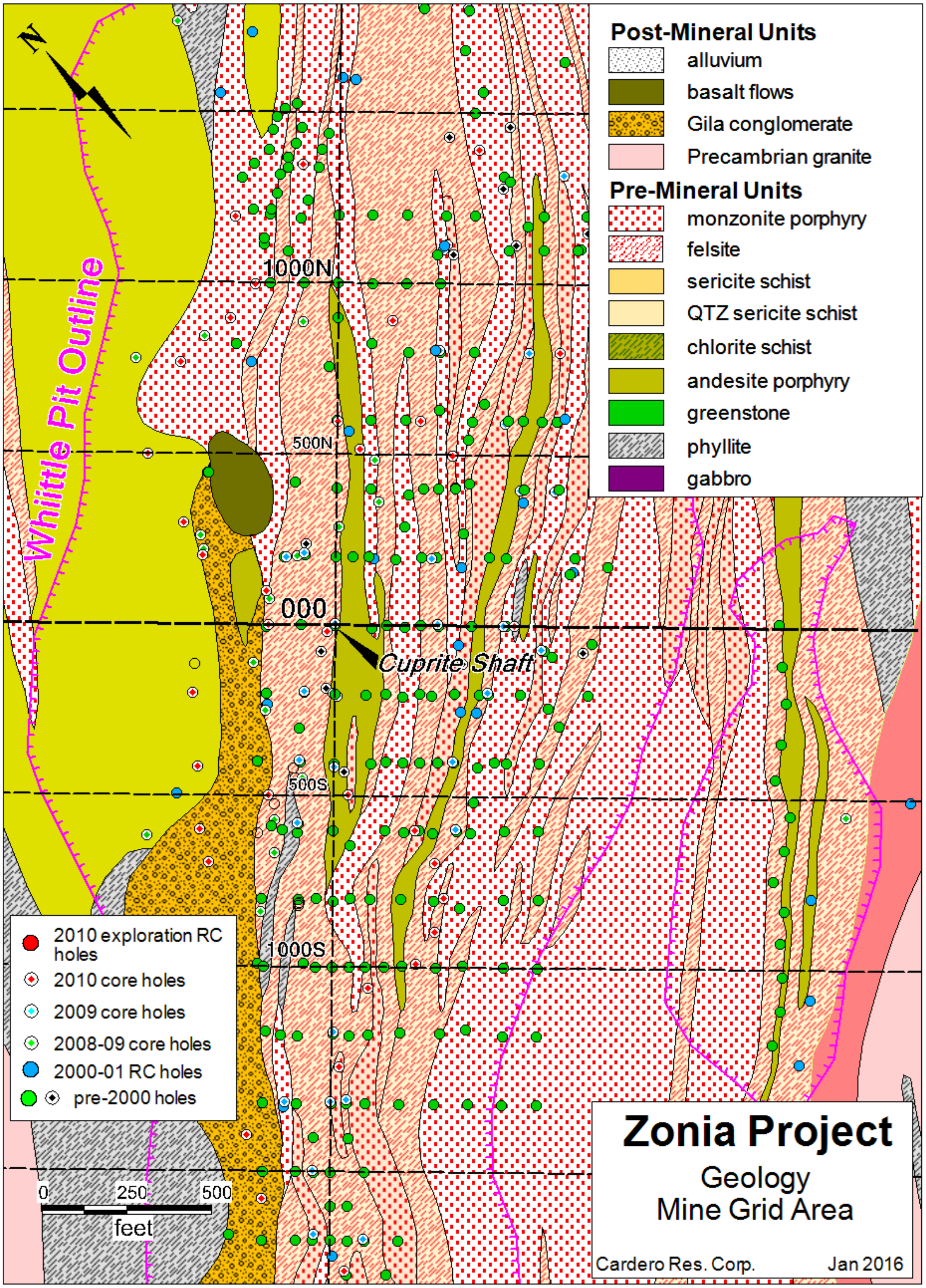


Figure 7-7 is a map of the detailed geology of the pit area, with color-coded core hole collars and the location of cross section N0000. Cross sections are on a 135°-315° orientation, orthogonal to foliation.

Figure 7-8 shows Cross Section N0000 through the North Pit, noting major lithologic units, drillholes, and faults discussed in text.

Figure 7-9 shows the South Pit looking north at bands of vertical standing, intense sericite-altered Qss separated by Grn, highlighting the strong vertical foliation throughout the outcrop.

Figure 7-7: Detailed Geology of Zonia with Drill Holes and $2.50 Whittle Pit Shell Outline



Previous Whittle pit outline shown above.

Figure 7-8: Cross-Section N0000 through the North Pit

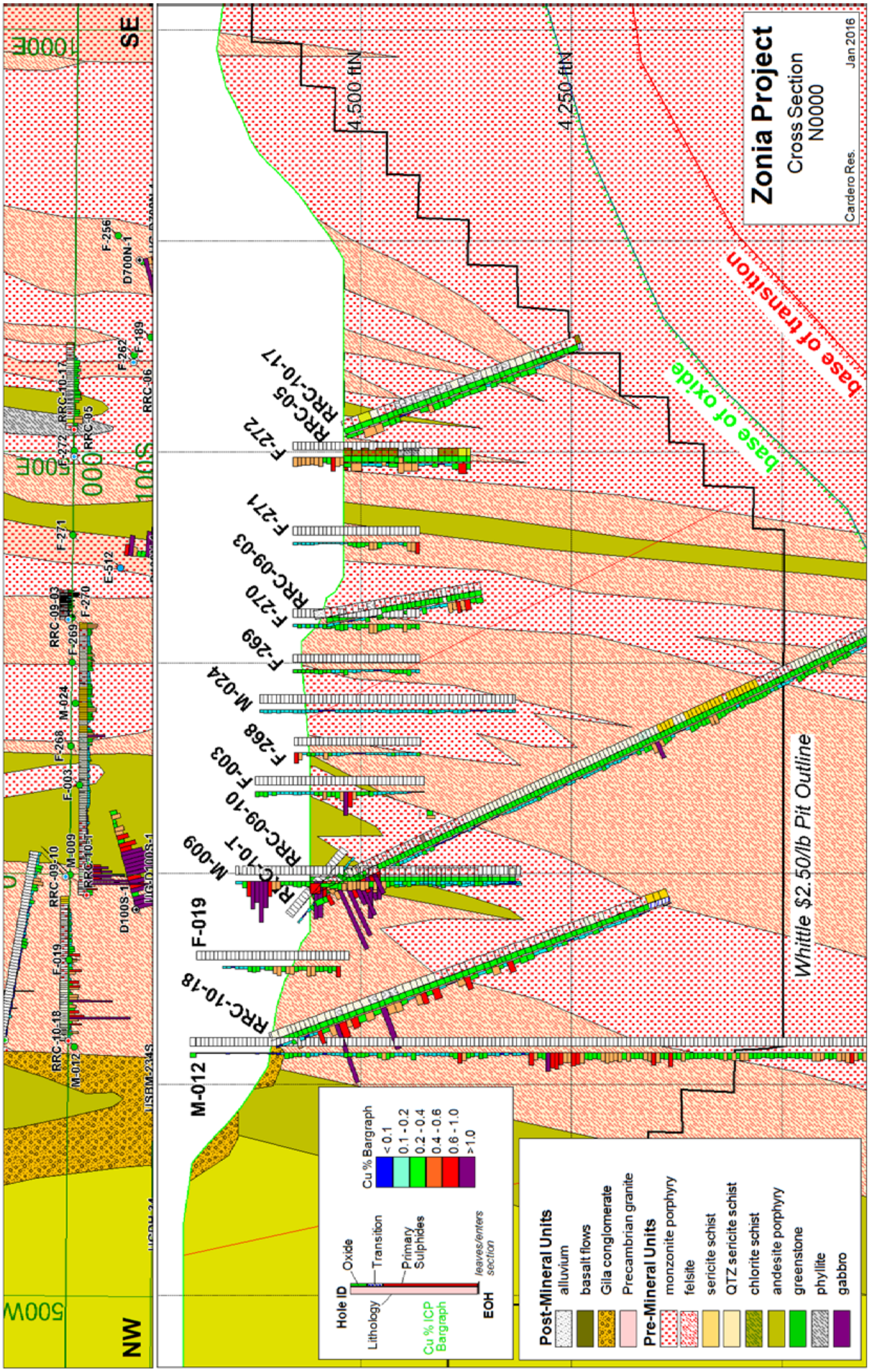


Figure 7-9: South Pit Looking Northeast

7.2.2 Structure

The regional structural setting of Zonia is typical of the Proterozoic pure shear, vertical deformation that affects the volcanic belts of central Arizona (Anderson 1989). The units at Zonia show highly variable foliation that changes on the scale of several inches. The foliation strikes principally northeast and dips steeply (70-80 degrees) to the northwest within the pit. Southeast of the pit, the foliation changes to steeply southeast dipping though it remains unclear whether this is an antiform or simply an over-steepening to the southeast.

The principal simple strain structure present at Zonia is the Chaparral shear zone, comprising regional-scale, anastomosing structures striking northeast with net right-lateral motion (Karlstrom, et al., 1991), as shown in Figure 7-3. It is visible in the pit as a zone of intense argillic alteration along the southeast wall.

North-trending structures cross-cut the earlier northeast structures fairly consistently. These cross-cutting structures show only minor offsets, and, in the North Pit area, appear to be down-dropped to the northeast based on core hole and cross-section correlations. The cross-cutting structures are easily recognized because they are conduits for oxidation and are strongly coated with limonite. In addition, some of these structures are host to late stage Quaternary basalt dikes.

Contact parallel structures are common and demonstrate a competency and rheological contrast between different rock units and how those units handle the regional strain. The thoroughly foliated Qss appears to absorb stress throughout the unit, while Qmp units show distinct zones of weakness that have slipped and produced a localized schist texture. Greenstone units appear to absorb stress along internal flow units and areas of weakness due to hydrothermal alteration.

Foliation is least developed in the PCg and Grn units. Mineral lineations are greatest in Qss and least recognized in Qmp.

A prominent group of east-northeast faults and quartz veins occur across the map area, from the south end of the pit to the Copper Crown mine. Most appear to be relatively short, on the order of 200-500 meters long, with little apparent movement, and dip to the north. Many host orogenic quartz veins with gold and copper mineralization and they appear to be simple dilation features related to the dextral movement along the Chaparral shear. Almost all of the historical workings northeast of the Zonia mine area are located along these structures.

Five important geotechnical considerations related to structure are present at Zonia and need to be addressed in detail at a future time:

- Foliation orientations, which vary from 50° west dipping in the North Pit to vertical in the South Pit areas, shown in Figure 7-7.
- Presence of a steep structure with offset in the west high-wall that will be part of any future layback.
- Cross cutting structures particularly in the North Pit that influence the depth of oxidation and therefore the depth of oxide mineralization.
- Zones of intense sericite alteration, particularly around the South Pit.
- Competency contrasts between ore bearing units, particularly Qss, Grn, and Qmp units.

7.3 Mineralization

Economic mineralization at Zonia is controlled by four dynamic events:

- Deposition of disseminated pyrite-chalcopyrite sulfides in a subvolcanic porphyry setting, slightly post-dating intrusion of unit Qmp, at approximately 1,750 Ma.
- Regional vertical deformation imposed by the voluminous intrusion of the granitic batholiths around the greenstone belts, with greenschist grade metamorphism related to the Yavapai orogeny from 1750 to 1690 Ma, followed by exhumation.
- Oxidation, mobilization, and supergene enrichment of the original copper sulfides along foliation and fracture plane controls, followed by burial.
- Second exhumation and oxidation of the supergene-enriched sulfides and remobilization of the copper oxide minerals into structural anomalies, resulting in *in situ* and transported copper oxides throughout the various units.

The original sulfide minerals were principally pyrite and chalcopyrite, with minor bornite, molybdenite, and sphalerite. The distribution of the minor sulfides has not been evaluated in detail, but recent grid sampling of the pit area indicates an overall mineral zonation of inner copper, molybdenum, and gold, zoning outwards to zinc and manganese.

Copper mineralization at Zonia occurs principally within the foliated quartz-sericite-schist (Qss) unit, which had argillic altered Qmp as a protolith. Mineralization also is concentrated along the contacts of various felsic units, as well as between mafic and felsic units. The latter is considered a late-stage effect of supergene, mobilized copper reacting with the more calcic mafic units.

Regional deformation during the Yavapai orogeny sheared the disseminated and blebby pyrite-chalcopyrite mineralized horizons into foliaform mineralization, parallel to schistosity, and ranging from dipping 45°NW to vertical. Subsequent oxidation-remobilization of the copper from chalcopyrite (35% Cu) followed the foliation down-dip to the water table, where copper then re-precipitated as enriched sulfide minerals, primarily secondary chalcocite (78% Cu). This chalcocite blanket was then itself oxidized during a second lowering of the water table and copper further mobilized into reactive units below. The early underground mining at Zonia exploited the high-grade chalcocite horizons preserved at depth.

7.3.1 Historical Mined Mineralization

Figure 7-10 and Figure 7-11 illustrate plan and long-section views, and Figure 7-12 is a cross-section view of the underground workings at Zonia, centered on the Cuprite shaft. Underground workings were developed on high grade chalcocite "veins" that developed along the supergene-enriched mineralized horizons. Photo 7-1 shows hand samples of oxidized, chalcocite rich stringer mineralization from the underground development.

Figure 7-10 and Figure 7-11 illustrate views of the developed mineralized blocks in the underground workings at Zonia. In plan, underground development extended 1400 feet to the northeast (right) to the

Figure 7-10: Underground Development Plan Drawing

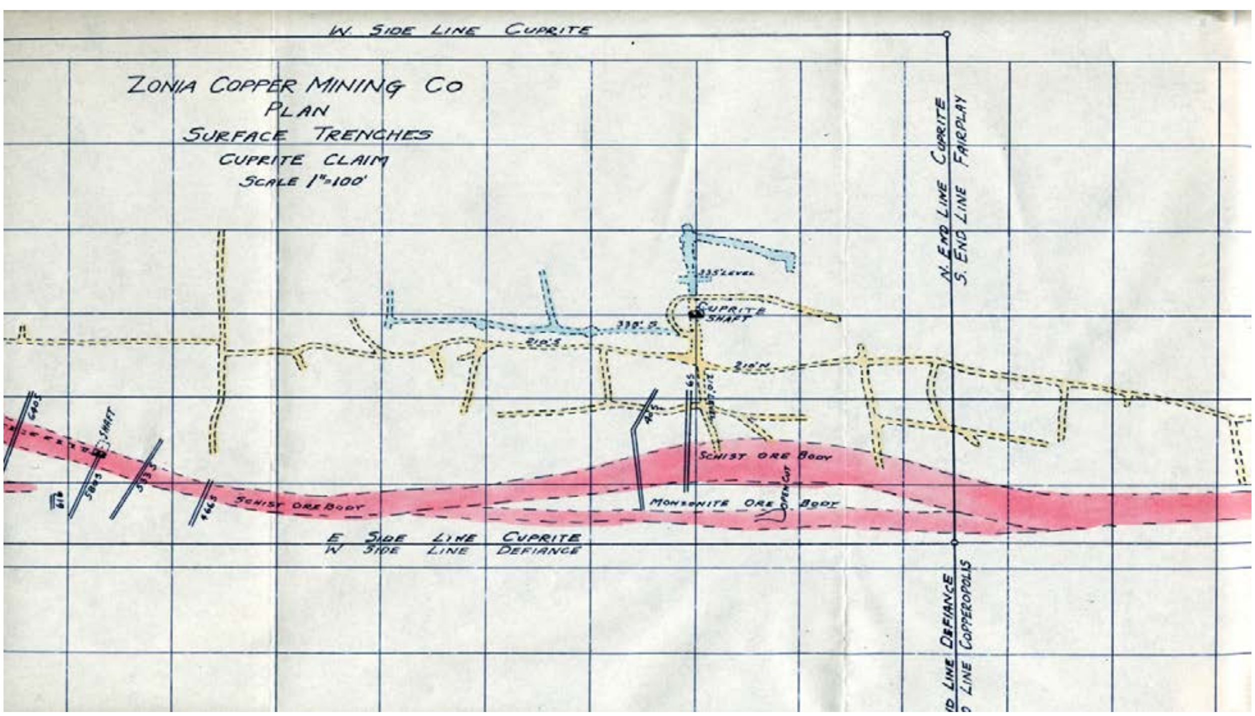


Figure 7-11: Underground Development Long-Section Drawing

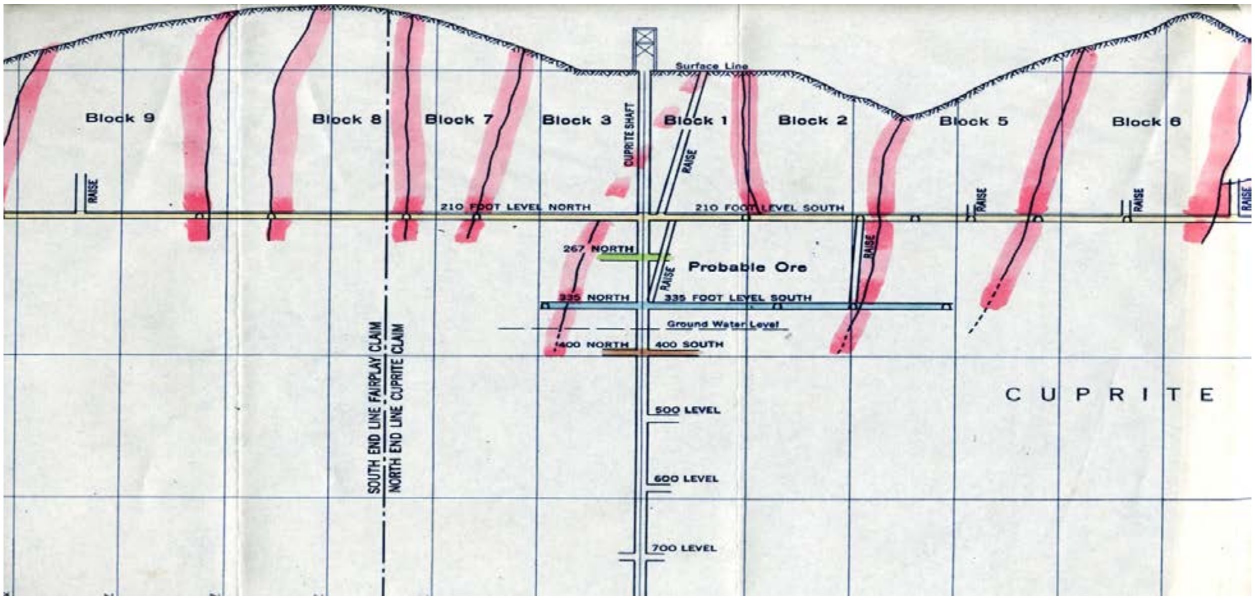


Figure 7-12: Underground Development Cross-Section

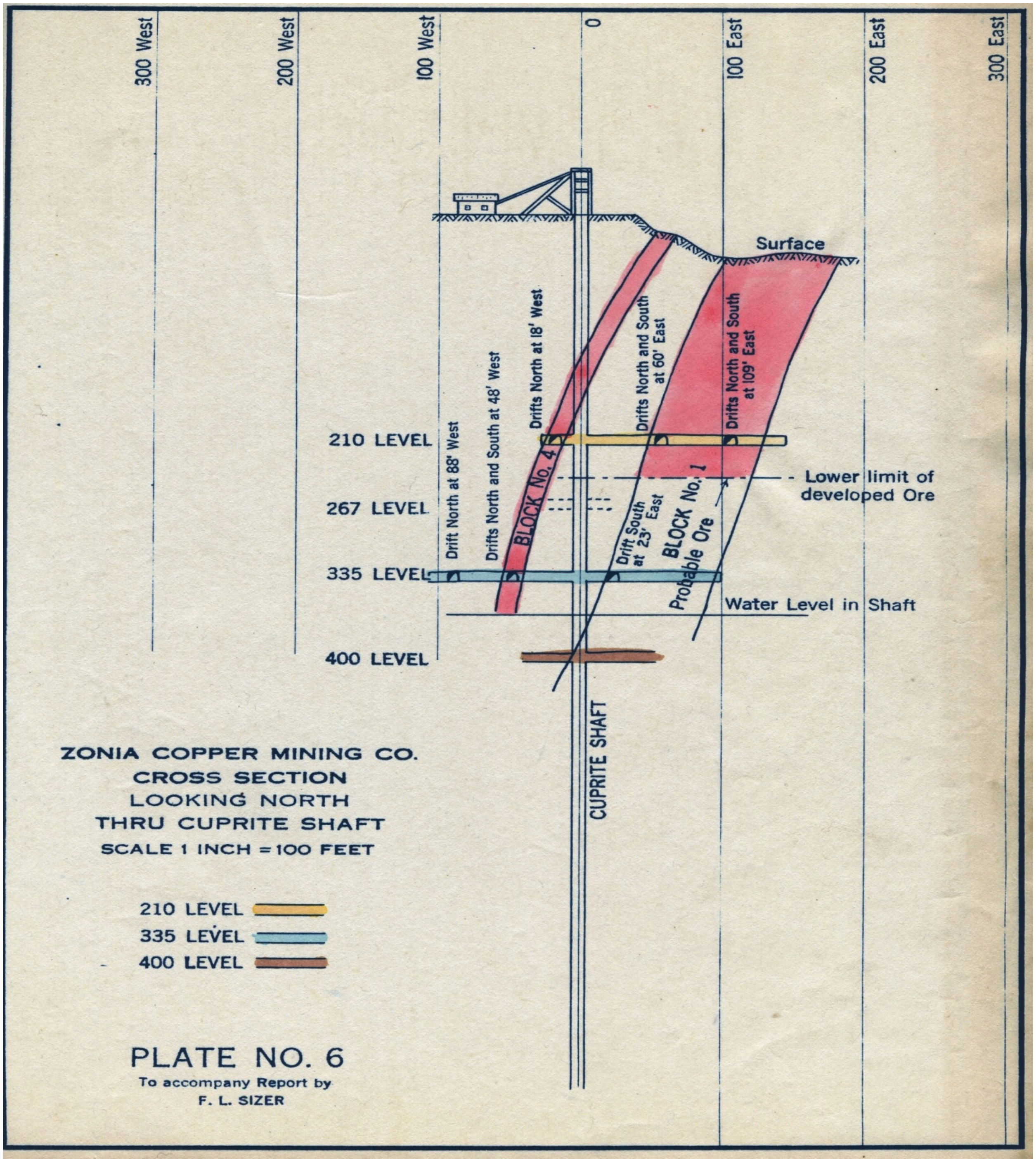
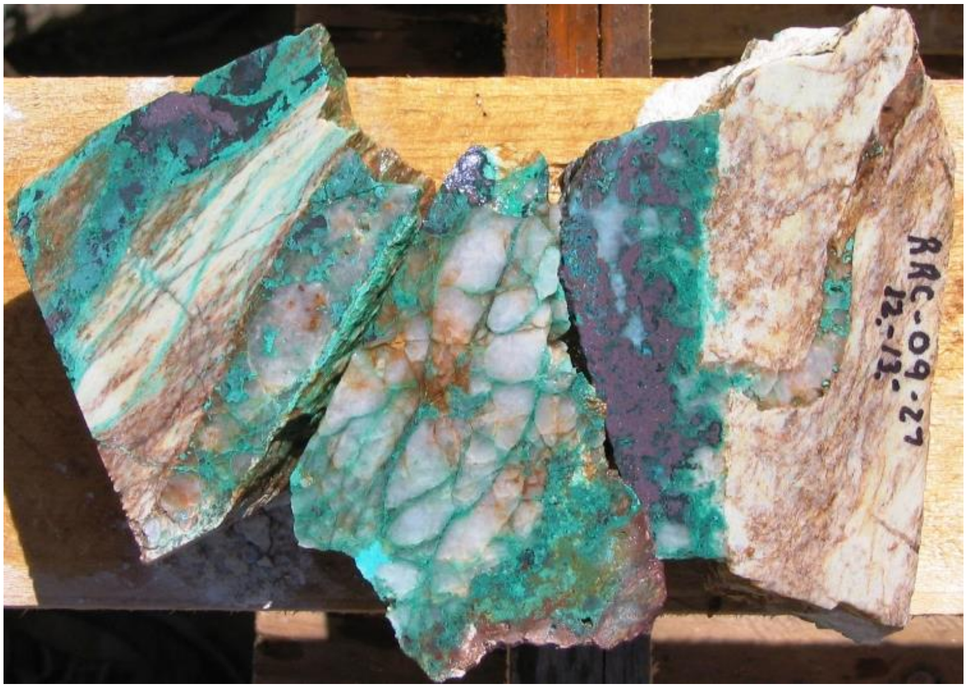


Photo 7-1: Core Mineralization



Fairplay tunnel and shaft, and 800 feet to the southwest (left) originating from the Cuprite shaft. Orientation in the long-section is reversed (view to SE) with the Fairplay to the left side of the drawing (F.L. Sizer, 1930).

Figure 7-12 is a cross section through the Cuprite shaft (looking north). There was a total of 6900 feet of underground workings developed at Zonia on five levels off of the Cuprite shaft, which was developed to 874 feet of depth from the original surface. Current Cuprite shaft is about 90 feet below original topographic surface (source: F.L. Sizer, 1930).

Photo 7-1 shows a high-grade interval from RRC09-27 grading 11.12% Cu over 8.5 feet and 92% sequential copper recovery. Samples show supergene chalcocite (metallic grey) with strata-bound silica, oxidizing to a thin, black copper pitch oxide rim, then maturing to malachite (dark green) and then to chrysocolla (blue-green factures). This is a classic reaction sequence in copper deposits, documented by Schwartz in 1934 (Schwartz, 1934).

The ore body was 1.38 million “proven” tons grading 1.75% Cu, principally as supergene chalcocite strongly oxidized to chrysocolla, malachite, and cuprite, and an additional 0.76 million tons of “probable” ore grading 1.5% Cu.

These historical reserves are not considered current and do not comply with the current standards of NI 43-101 and CIM definitions for reporting mineral resources and reserves, and are included for historical reference and context only. The reported numbers have not been, nor can be, verified and therefore should not be relied on.

Photo 7-2 shows cut core samples from RRC09-27, taken just below the samples pictured in Photo 7-1, showing further oxidation of chalcocite (metallic grey) to cuprite (?), copper "pitch" and malachite. Note the oxidation-leaching of the chalcocite to a distinctive red-earthy hematite.

Photo 7-2: Core Mineralization

Photo 7-3 is a cut core sample from RRC09-X08 from an interval grading 0.33% Cu, from the SW extension on the 4000S section. Siliceous Qss hosts malachite and azurite mineralization and minor sulfides.

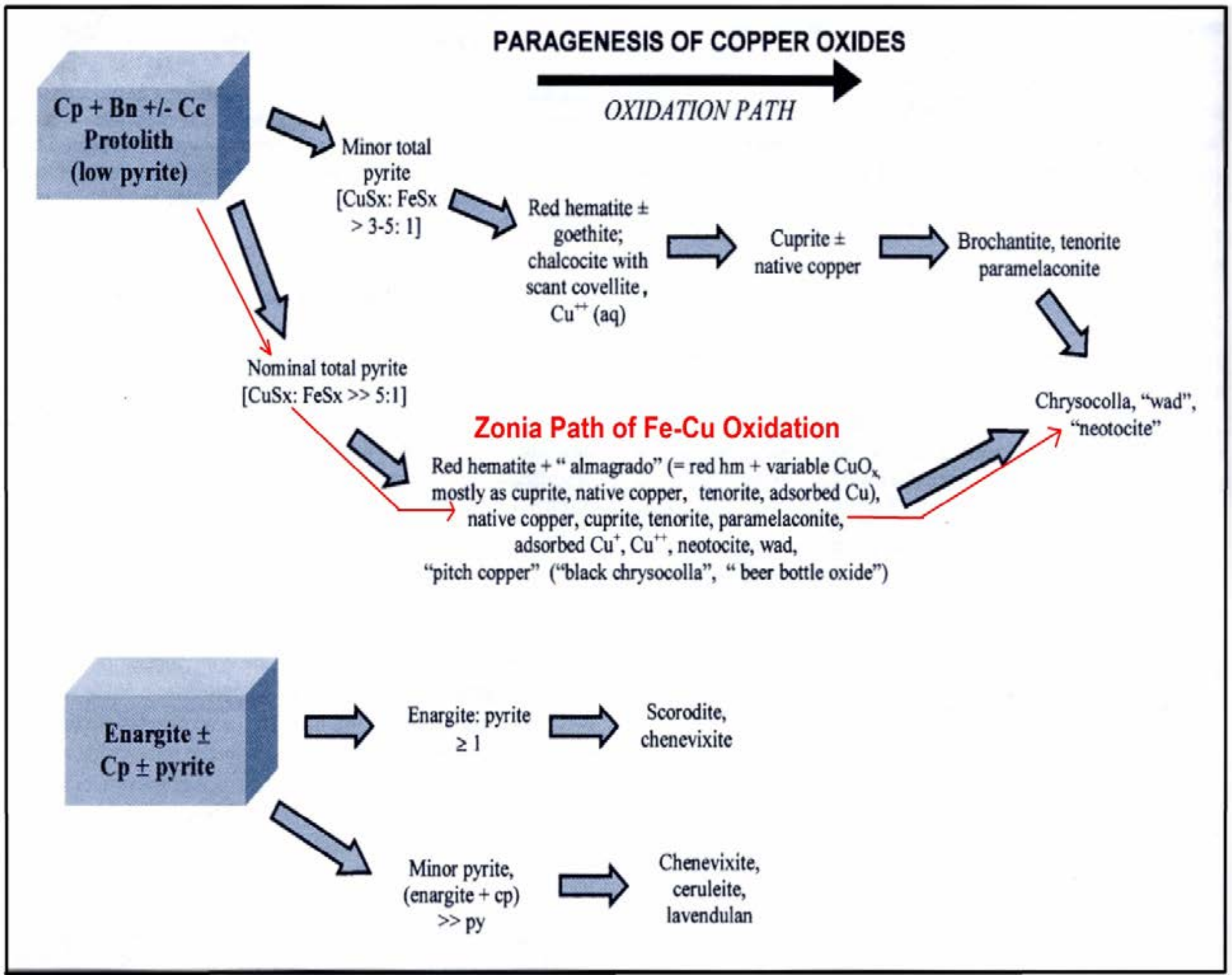
Photo 7-3: Core Mineralization

Photo 7-4 is a bench outcrop of mineralization from the North Pit showing final stages of copper oxide development, with chrysocolla going to "black" (manganese rich) chrysocolla, neotocite, and copper bearing hematite, goethite, and wad.

Photo 7-4: Outcrop Mineralization

Figure 7-13 illustrates the overall development of copper and iron oxides at Zonia. Pyrite content was low and oxidation followed a path that developed distinctive red hematite over zones of leached secondary copper (Figure 7-5) (Chavez, 2000).

Figure 7-13: Paragenesis of Copper Oxides Flow Chart



Source: Chavez, 2000.

8.0 DEPOSIT TYPES

Zonia has seen two long-running conflicting interpretations for the origin of the primary sulfides: porphyry (intrusive) versus volcanogenic exhalative (syngenetic). However, all workers agree there was an early primary sulfide mineralizing event, a long time lapse, and then superimposed oxidation. The current view on the history is:

- Deposition of pyrite-chalcopyrite sulfides in a subvolcanic porphyry environment, and subsequently strongly deformed during the Yavapai orogeny.
- The oxidation, supergene enrichment, and then secondary oxidation of the original sulfide deposit leading to the oxide deposit of current interest and resource.

Zonia was initially interpreted as “mesothermal pyritic replacement” (an early description for porphyry-style mineralization) by Allen and Spencer (1957), then reinterpreted as a VMS deposit based on the high-grade mineralization in the underground mine, which was poorly described and understood, by Chadwick (1964). In 1930, known underground semi-massive mineralization extended 7,000 feet in length, varied from 10 to 40 feet in thickness, and extended as much as 800 feet down-dip. The amount that was mined is not known exactly but was insignificant to the current estimated resources, judging from plans of the historical workings.

The confusion can be at least partly attributed to the fact that deep-seated Proterozoic structures in central Arizona are the crustal controls of the Laramide intrusions (Anderson, (1982). The operating open pit copper mine at Bagdad is in a former massive sulfide district (Baker III, et al., 1968), as is the well-known Copper Basin breccia pipe (Johnson, et al., 1961) 12 miles north of Zonia. Both are Laramide age, porphyry-related deposits.

Homestake Mining Company in 1975 recognized Zonia as an oxidized porphyry copper deposit (Cameron, 1975) and realized that, while the overall mineralogy and zonation was similar to numerous Laramide-age porphyry copper deposits hosted by Proterozoic greenstone rocks in central Arizona, the mineralization was at least in part foliated and the deposit was similar in age to the Precambrian host rocks. Although Homestake attempted Rb-Sr age-dating on the Qmp, initial results were questionable (1100Ma, 840Ma, and Tertiary; Cameron, 1975) and the data for subsequent samples are lost.

Amselco (1980) and Queenstake (1983) returned to a volcanogenic interpretation for features at Zonia. However, succeeding companies (Western States Engineering, 1995) continued to employ the porphyry model in their evaluation of the Zonia mineralization.

Copper Mesa and Redstone Resource personnel, following the examination and recommendation by Davis (2007), followed the volcanogenic exhalative model, interpreting the sulfide portion of the Zonia copper deposit as syngenetic-exhalative in its origin, albeit with only the deeper portions of the system preserved. They correlated Zonia with the nearby Iron King and Bagdad mining districts and the more distant Jerome (United Verde) mining district.

8.1 Oxide Mineralization

The portion of Zonia deposit within the resource estimate is characterized as mostly oxidized, supergene-enriched stringer, vein, fracture, and disseminated oxide mineralization. The original pyrite-chalcopyrite mineralization underwent oxidation, with copper remobilized into fracture conduits and concentrated in more mafic, reactive units (the greenstone and chlorite schist) and at the water table, and resulted in development of chalcocite-rich lenses that are presently up to 800 feet deep along the steep, west-dipping foliation of the host rocks.

This supergene mineralization was subsequently oxidized and partly remobilized due to uplift, erosion, and lowering of the water table, resulting in a large deposit of *in situ* and transported copper oxide mineralization which masks much of the original sulfide depositional environment. The processes of oxidation followed by supergene enrichment and then by second oxidation are well documented by Locke (1926), Blanchard (1968), and Anderson (1982).

9.0 EXPLORATION

The exploration and work included in this section was undertaken since 2008 by the most recent previous operators, Copper Mesa and Redstone, as well as Cardero. Work completed prior to 2008 is included under Item 6: History, because the authors had no involvement with the project at that time.

Cardero, the issuer for whom the report is being completed, acquired the Property in fourth quarter 2015, and has only completed surface exploration consisting of an extensive rock sampling grid and limited mapping.

9.1 2008 - 2009 Exploration

Exploration-related activities on the Zonia project were undertaken by Copper Mesa and Redstone in 2008 and solely Redstone in 2009 and 2010. Redstone focused on compilation of existing data sets and development of interpretive cross sections in GEMS. The sections are based on historical drillhole information, including geologic logs, mineralization, and assay data. Sections included interpretive geology and depiction of oxide (hematite vs. limonite) distribution. The sectional information was subsequently used to develop 3D solids for use in development of a block model for the deposit.

In follow-up to the 2008 compilation work, drilling was completed as strategic twinning of historical drill holes for compliance verification. During the drilling program, issues with collar locations were recognized; these issues precluded full use of the new data for verification purposes.

Follow-up exploration-related work completed in 2009 by Redstone included re-establishment of the grid system for the property by conversion of coordinates from a local grid system to Arizona State Plane. The block model was subsequently rebuilt in the re-established coordinate system.

Additional drilling was completed in 2009 to twin historical drill holes for grade verification to meet compliance standards and for exploration away from previous drilling. Further drill campaigns were undertaken in 2010 using diamond drill and reverse circulation drilling techniques. Additional information regarding drill programs is discussed in Section 11.0.

9.2 2010 Exploration

After 2008, almost all field work at Zonia concentrated on the immediate pit areas, confirming and expanding the existing contiguous resource and placing that resource into a relevant geologic and resource context. Field exploration of the Zonia property outside of the known resource area began in June 2010 and continued through December 2010.

In June 2010, Redstone retained Mr. Gary Bender, R.G., to map the geology, at a scale of 1”=400’, of the 23 Bragg Estate patented claims. These patented claims adjoin the southeast side of the patented Zonia Mine claims. The geological mapping was a requirement of the Arizona Department of Environmental Quality’s Aquifer Protection Permit (APP) process to identify lithologies and geological structures that might potentially be relevant to the understanding of the hydrology of the area. This program included surface lithologic and structural mapping, prospect pit evaluation, and extensive rock sampling of outcrops, pits, trenches, and shafts.

In July and August 2010, Redstone extended the mapping and sampling onto the 10 Newton Claims and on the 78 unpatented Copper Crown Claims adjoining to the north of the Zonia Mine. The mapping and sampling was designed to evaluate the potential for mineralization along and cross-strike to the main Zonia Mine mineralized zones and to assist with drillhole placement in target areas identified during recent reconnaissance by Redstone contract geologists. Work completed by Gary Bender, R.G., is compiled in Figure 9-1, which is annotated to show target zones and 2010 exploration drillholes.

9.2.1 Sampling Methods

A total of 234 grab samples were collected during the reconnaissance mapping and sampling program and submitted to Skyline Laboratories of Tucson, Arizona. Details and values of all of the samples and their locations are available in an internal report entitled "*Exploration Potential of the Zonia Deposit, Yavapai County, Arizona"*, and available at the Zonia mine office.

Reconnaissance sampling was wide-spaced to maximize data coverage. Mineralized zones were easily accessed by roads and trails left over from the historical exploration, and the zones were well exposed in pits, trenches, shafts, and adits. Apparently barren areas were sampled to provide background information and to check for the possibility of disseminated mineralization that might not be visually obvious.

9.2.2 Analytical Methods

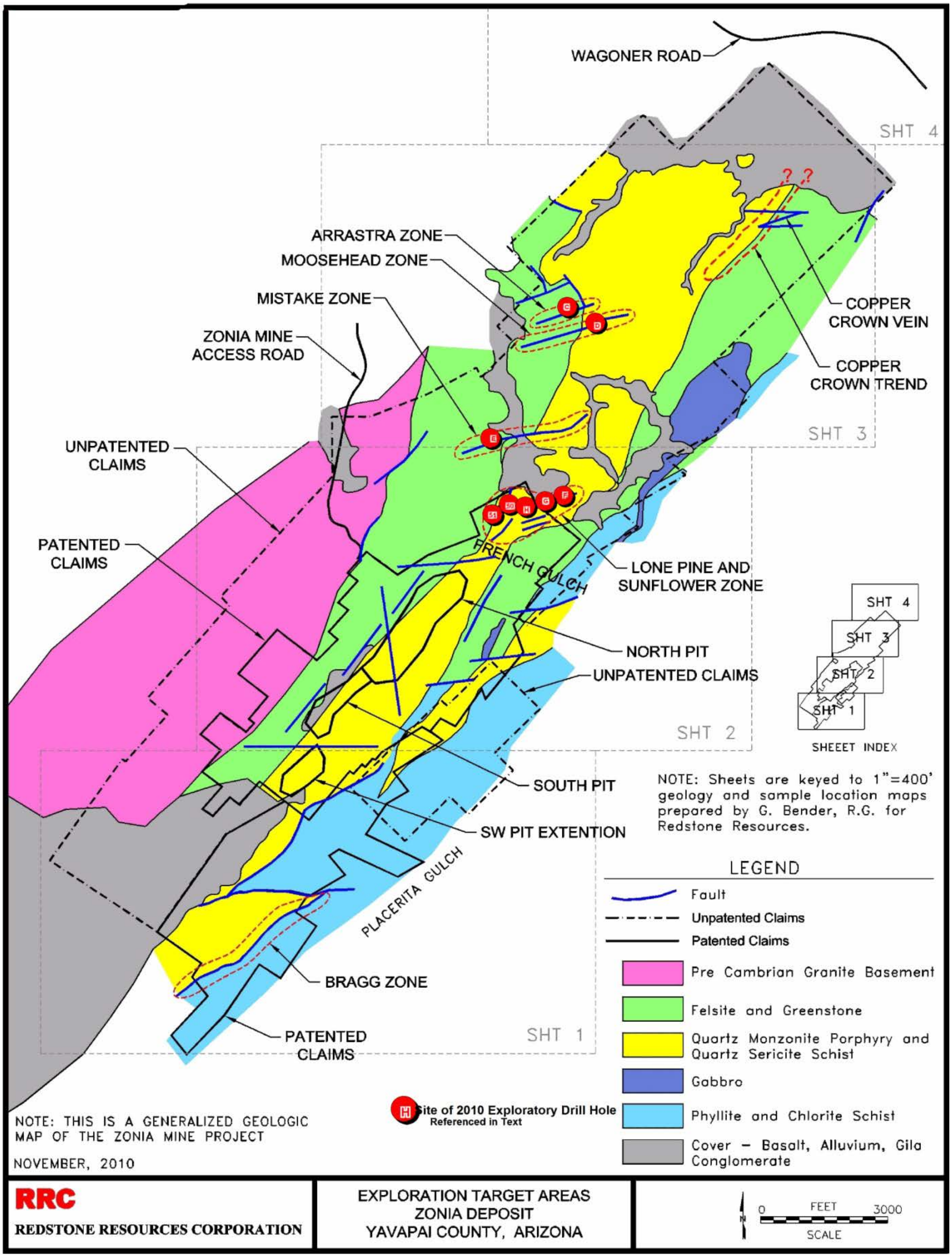
Samples were initially fire assayed for gold with atomic absorption (AA) finish analysis. If results exceeded the method reporting limit (3,000 parts per billion [ppb]), they were rerun with fire assay with gravimetric finish analysis and reported as grams/Metric ton (g/Mt). Each sample was also analyzed for a list of 34 elements (including copper) by Trace Element (TE) ICP-optical emission spectrometry (OES). Silver, if exceeding 200 parts per million (ppm), was re-assayed using fire assay with gravimetric analysis and reported as g/Mt. Copper, zinc, and lead, if exceeding 10,000 ppm, were re-assayed using Skyline's standard TCu-TPb-TZn method and reported as a percentage.

9.2.3 Exploratory Drilling Program

As part of the evaluation of the untested patented and unpatented ground, eight drillholes were planned to test several widespread and easy to access anomalies:, five on the unpatented ground and three on patented ground, all to the northeast of French Gulch. The drilling was poorly scheduled, and the drillholes were completed prior to the completion of the sampling and mapping program. Nonetheless, the grab samples were poor representations of mineral potential and would have been of limited use in discerning other potentially better targets for drilling.mineralization within the east-northeast structures is discontinuous both laterally and at depth, but it did not test the potential for more disseminated oxide mineralization in the area.

Figure 9-1 shows the location of the exploration drillholes. All of the drillholes intersected the targeted zones, but with average to mediocre results. At the least the results demonstrated that copper-gold mineralization within the east-northeast structures is discontinuous both laterally and at depth, but it did not test the potential for more disseminated oxide mineralization in the area.

Figure 9-1: 2010 Exploration Target Areas Map



9.3 2018 Exploration

9.3.1 Surface Sampling

In 2017, Cardero reviewed the results of the Redstone exploration in 2010-2012 and compiled geology and geochemical mapping into a single coherent map. In 2016, Cardero staked the adjoining Silver Queen property, formerly held and explored by Alliance Mining Corp., and incorporated that geology and geochemical, and geophysical data into Zonia. Since the previous exploration sampling was based on unreliable grab samples and focused on structurally controlled mineralization, a more systematic sampling approach was planned.

Property-wide rock geochemical sampling on a 150-meter spaced grid was mostly completed in early 2018; the southwest end has recently been completed but results are pending at time of writing. The grid sampling generated a new porphyry copper target based on coincident anomalous copper, molybdenum and manganese. The 2500- by 1000-meter anomaly, the “Northeast Porphyry Target”, occurs two kilometers northeast of the drill-defined Zonia copper oxide deposit, and shares characteristics of its geochemical footprint.

Coincident areas of elevated molybdenum (Mo) and copper (Cu) values with depressed manganese (Mn) values is a classic geochemical signature of porphyry copper mineralization (Figure 9-2 and Figure 9-3). Copper values are also anomalous (Note the Mn low coincides well with the Mo high. High Mn values typically form haloes around porphyry deposits.

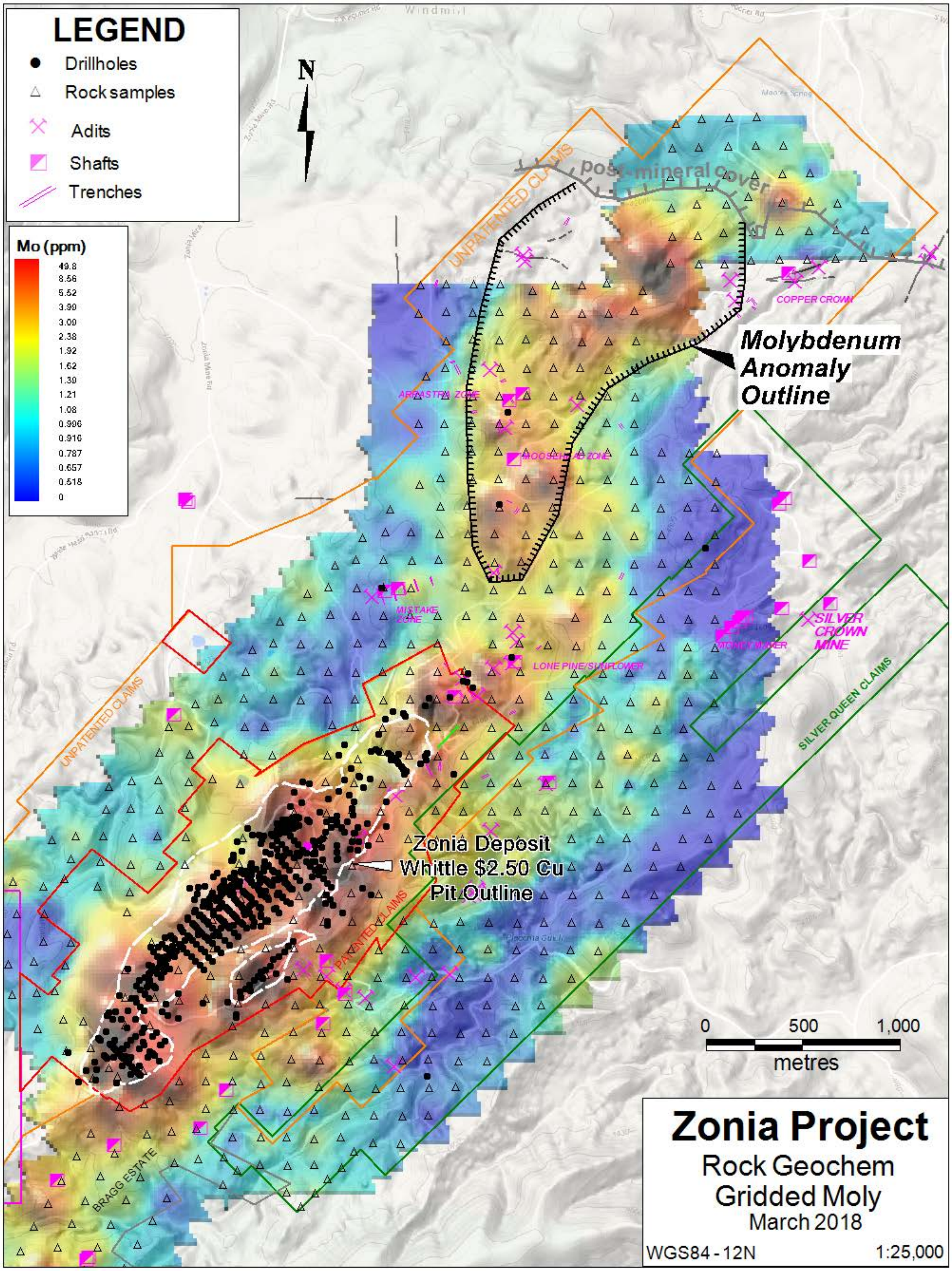
Figure 9-4), but copper is not as reliable as the other metals due to its solubility in the weathering profile. The overlapping anomalies suggest a porphyry copper target size on the order of 2,500 by 1,000 meters. The same quartz-monzonite porphyry that hosts the Zonia copper oxide resource (see NR17-08) underlies the anomaly. The anomaly marks a break in the northeast trend of the mineralization, with a narrow southern “tail” that opens northward to a broader northeast trend. The anomaly is truncated at the north end by younger, post-mineral cover rocks (Gila conglomerate, alluvium, and Tertiary basalt). The east margin of the anomaly contains some narrow high-grade copper bearing structures in the historical Copper Crown mine workings, with associated intense epidote alteration.

9.3.2 Sampling Procedures and Quality Assurance and Quality Control

The work program at Zonia was designed by John Drobe, P.Geo., Cardero’s Chief Geologist, with the field work conducted by Discovery Consultants, of Vernon, B.C. Due to a lack of consistent soil cover over the project, composite rock samples were collected by shovel from 10- to 25-cm depth over a roughly one-meter square area at each station, and the locations marked with flagging and aluminum tags hung from the nearest vegetation. Samples were placed in woven Sentry brand 7- by 12.5-inch Olefin sample bags, which were sealed, transported, and dropped off directly at ALS Minerals laboratories in Tucson, Arizona by Discovery personnel. The samples were dried at high temperature (method DRY-21), crushed, pulverized (methods CRU-31, SPL-21, PUL-31), and then analyzed by ICP-AES for 35 elements (method ME-ICP41) with gold determined by 30 g fire assay and atomic absorption finish (method Au-AA23).

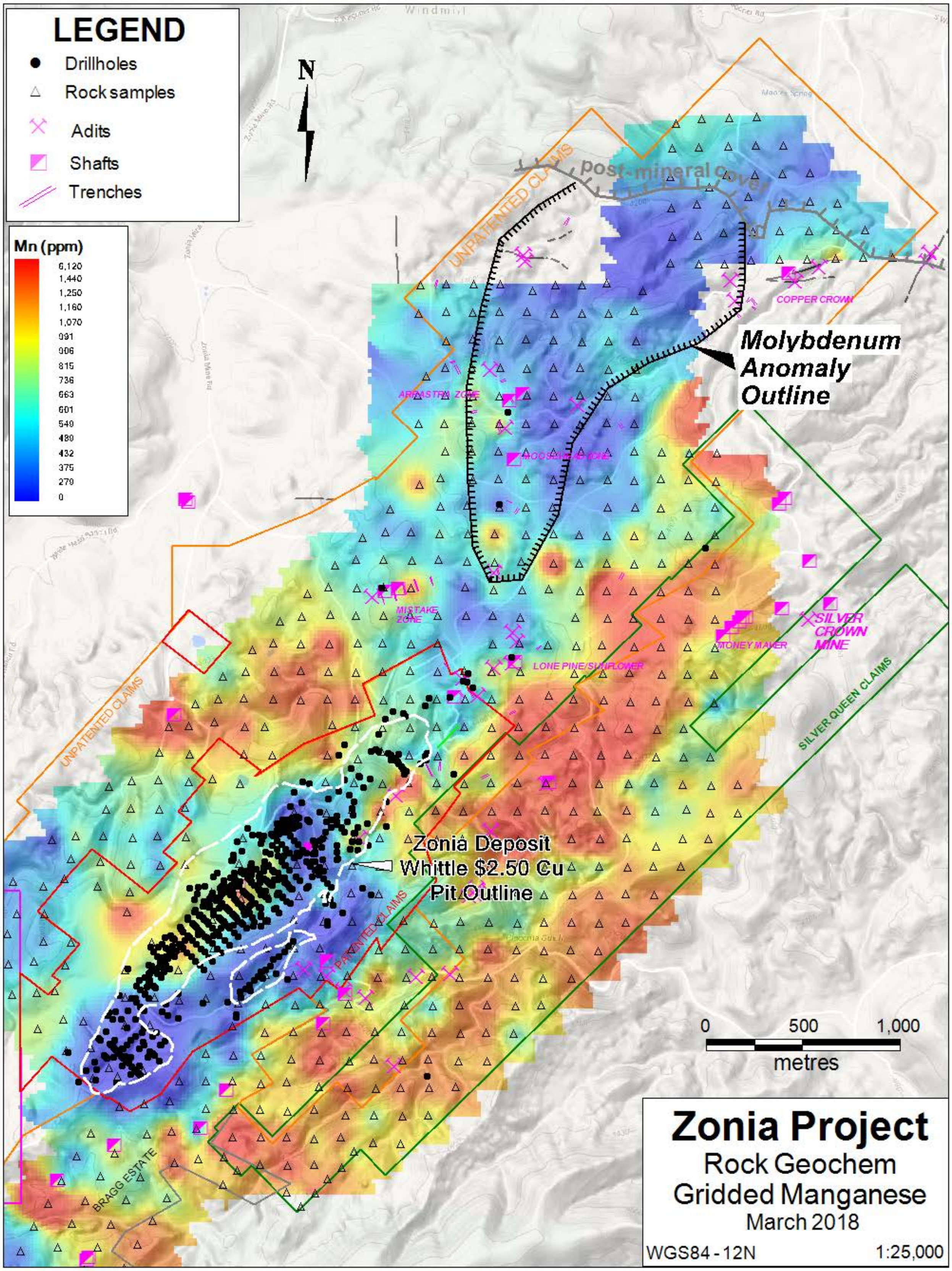
This sampling program did not include a comprehensive QA/QC program; however, ALS Minerals is an ISO 9002 registered laboratory and inserted blanks, standards, and duplicates following their QA/QC protocol. These additional samples returned satisfactory values.

Figure 9-2: Gridded and Thematic Molybdenum (Mo) Rock Values



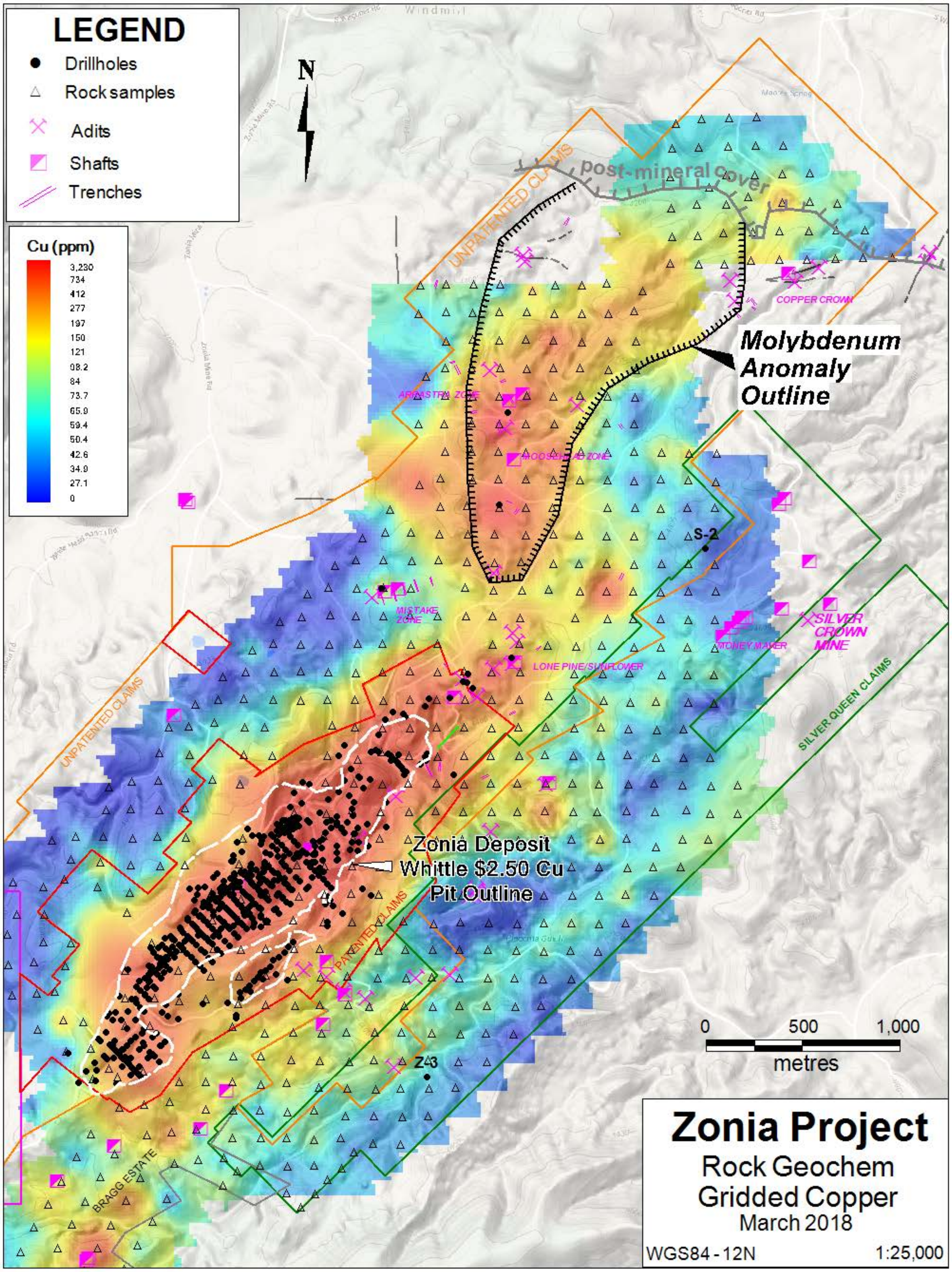
Shown with the Whittle pit outline that defines the current resource estimate at Zonia.

Figure 9-3: Gridded and Thematic Manganese (Mn) Rock Values



Note the Mn low coincides well with the Mo high. High Mn values typically form haloes around porphyry deposits.

Figure 9-4: Gridded and Thematic Copper (Cu) Rock Values



The Cu values are higher over the Zonia deposit within the historical open pit, where the partially leached upper horizon has been removed.

9.3.3 On-going Exploration

The long-term exploration potential of Zonia can be divided into extending the known copper oxide resource at Zonia and brownfield exploration at outlying targets:

9.3.3.1 Resource Extensions

- Down-dip extensions of copper oxide mineralization in the North Pit area, where the boundary between oxide and sulfide mineralization remains open at depth.
- Along-strike extension of copper oxide mineralization to the northeast, where the prospective units continue outside the current planned pit for about 500 meters to near the Sunflower-Lone Pine zone and Z-10 shaft.

9.3.3.2 Outlying Targets

- Drilling of the Northeast Porphyry Target: this will require about 5000 meters of core drilling for the initial program to prove the concept. If successful, the target would need definition drilling at 100-meter spacing.
- Delineation of prospective stratigraphy on the Bragg Estate ground where chloritic, mafic schist could be host to strata-bound copper oxide and sulfide mineralization, and calcareous schist may host skarn mineralization.
- Exploration of favorable units that could host massive copper sulfide mineralization to the north and northeast of the known prospective stratigraphy. The main targets are the contacts of the gabbro and quartz-monzonite porphyry (known to be prospective in the Jerome VMS district) and the Copper Crown area, where there is widespread intense epidote-chlorite-magnetite alteration around the old mine workings.

10.0 DRILLING

The exploration and drilling included in this section was undertaken by the most recent previous operators, Copper Mesa and Redstone. Work completed prior to 2008 is included under Item 6: History because the authors had no involvement with the project at that time. Work completed since that time by Copper Mesa and Redstone is included here under Item 10: Drilling because the authors have knowledge of the work completed. However, it is important to note that Cardero, the issuer for whom the report is being completed, acquired the Property in fourth quarter 2015, and **Cardero has not completed any exploration drilling on the Property. All the results described in this section result from work completed by previous operators.**

Core drilling was contracted to Boart Longyear Diamond Drilling (Boart Longyear) of Peoria, Arizona, USA. The drilling crews ran two 12-hour shifts per day with two drill rigs, including a skid-mounted LF-70 and a truck-mounted LF-90. Boart Longyear has drilled three programs from 2008 to 2010, drilling 77 HQ-sized diamond drill core holes for 25,342.3 feet of drilling. Recovery, measured in 51 core holes, was 96%, which is suitable for reliable results. A possible correlation between grade and recovery has not been investigated, and further study is recommended. It is common for lost fines to be biased with higher grade; however, this has not been studied or determined at Zonia.

Harris Exploration Drilling of Escondido, California and Preston Drilling of Tempe, Arizona were contracted to conduct reverse circulation drilling from July to December 2010; 54 holes were completed for 28,984 feet of drilling.

Exploration by Copper Mesa and Redstone, and solely by Redstone, was conducted from when Copper Mesa acquired the Zonia property up to 2010. The initial campaigns by Copper Mesa in 2008 and 2009 were primarily undertaken to verify historical drilling through twinning of historically drilled holes. In 2008 and 2009, 46 HQ-sized diamond drillholes were drilled toward this end, totaling 13,179.3 feet. The remaining nine holes drilled in 2009 were for exploration and resource expansion purposes, intending to further define mineralization. Copper Mesa's involvement in the project was terminated in August of 2009, and Redstone became sole operator of the property and continued to conduct drilling operations. In 2010, an additional 22 HQ-sized diamond holes were drilled totaling 12,163.0 feet, along with 54 reverse circulation drillholes totaling 28,984 feet. The 2010 campaign was also conducted for exploration and resource expansion purposes, providing data which could be used for the basis of additional resource calculation. A summary of drilling conducted by Copper Mesa and Redstone is included in Table 10-1.

Table 10-1: Summary of Copper Mesa and Redstone Drilling

Year	Company	Type of Drilling	Series	No. of Holes	Amount (feet)
2008	Copper Mesa / Redstone	Diamond HQ	RRC	16	2,971.8
2009	Redstone	Diamond HQ	RRC-09	30	6,524.5
2009	Redstone	Diamond HQ	RRC-09-X	9	3,568.0
2010	Redstone	Diamond HQ	RRC-10	22	12,163.0
2010	Redstone	Reverse Circulation	R-RC10	54	28,984.0

Table 10-2 details the core drilling completed from 2008 to 2010 and identifies the corresponding historical holes that were twinned, the collar positions, orientation, and depth of each drillhole. Table 10-3 details the collar locations, orientation, and depth of the reverse circulation holes drilled in 2010. Drillhole collars were surveyed by Mr. Gary Berg, a licensed Arizona surveyor, in the mine grid. Unfortunately, many of the temporary markers placed after drilling were either difficult to locate or not locatable. Downhole surveys were not performed for holes drilled in the initial 2008 and 2009 campaigns due to the majority of the drillholes being twinned and having vertical orientation and a relatively shallow depth. All core drillholes from the 2010 campaign were surveyed by Boart Longyear. Downhole surveys were not conducted for the reverse circulation drilling in 2010.

Table 10-2: 2008-2010 Diamond Drilling Program

Hole Id	Easting	Northing	Collar Elevation (feet a.m.s.l.)	Azimuth	Dip	Length (feet)	Twin Hole ID
RRC-01	483508.6	1202007.4	4553.2	45.5	-90	200	M - 18
RRC-02	484144.3	1202239.5	4557.3	45.5	-90	150.3	F - 267
RRC-03	483843.8	1202543.9	4607.8	45.5	-90	201	F - 265
RRC-04	483765.2	1201773.1	4523.6	45.5	-90	175	F - 279
RRC-05	484152.7	1201947.9	4520.1	45.5	-90	150.5	F - 272
RRC-06	484182.1	1201819.4	4520.4	45.5	-90	200	F-262/F-189
RRC-07	484540.3	1202122.1	4512.8	45.5	-90	150	F - 301
RRC-08	484467.5	1202196.1	4513.8	45.5	-90	150	F -300
RRC-09	482843.9	1201287.6	4655.4	45.5	-90	200	F - 291
RRC-10	482709.4	1201413.1	4673.0	135	-60	250	E - 527
RRC-11	481690.8	1199618.9	4713.5	45.5	-90	125	F- 171
RRC-12	481906.6	1199400.9	4656.1	45.5	-90	150	F - 176
RRC-13	484807.0	1202858.9	4513.4	135	-60	320	E-529
RRC-14	482494.3	1201074.6	4637.2	45.5	-90	200	F-293/F-085
RRC-15	484577.8	1202367.4	4513.6	45.5	-90	150	F - 336
RRC-16	482314.6	1200688.6	4618.8	45.5	-90	200	F - 097
RRC-09-01	481869.0	1199541.9	4690.1	135	-45	200	RH-103
RRC-09-02	483981.3	1201842.1	4523.0	315	-80	200	F-276
RRC-09-03	484020.4	1202088.9	4554.0	135	-80	200	F-270
RRC-09-04	485457.2	1203397.5	4515.0	0	-90	200	F-353
RRC-09-05	485530.7	1203301.8	4525.0	0	-90	200	F-352
RRC-09-06	485751.5	1203221.5	4583.0	0	-90	300	F-350
RRC-09-07	484339.1	1202338.9	4519.0	0	-90	200	F-299
RRC-09-08	482348.9	1200935.1	4729.0	0	-90	250	F-297
RRC-09-09	482625.9	1201209.6	4657.0	0	-90	200	F-292
RRC-09-10	483806.1	1202305.6	4560.0	0	-90	251	M-009
RRC-09-11	485359.3	1203310.5	4568.0	0	-90	250	F-195
RRC-09-12	485270.9	1203407.0	4534.0	0	-90	250	F-194
RRC-09-13	482366.8	1199468.1	4669.0	0	-90	200	F-165
RRC-09-14	483315.4	1201967.6	4589.0	0	-90	200	F-009
RRC-09-15	482509.9	1199609.6	4626.7	0	-90	200	F-159
RRC-09-16	485802.5	1203317.3	4580.0	0	-90	200	F-197
RRC-09-17	482230.3	1200464.9	4640.0	0	-90	200	F-108
RRC-09-18	482159.6	1200536.5	4640.0	0	-90	200	F-112

Hole Id	Easting	Northing	Collar Elevation (feet a.m.s.l.)	Azimuth	Dip	Length (feet)	Twin Hole ID
RRC-09-19	482231.9	1200749.5	4640.0	0	-90	300	F-098
RRC-09-20	482588.9	1200960.9	4640.0	0	-90	250	F-084
RRC-09-21	482312.0	1200807.9	4641.0	135.5	-80	200	E-525
RRC-09-22	482804.3	1201315.4	4660.0	0	-90	195	F-037
RRC-09-23	483889.1	1202515.0	4599.0	0	-90	350	F-026
RRC-09-24	482954.6	1201455.4	4646.8	0	-90	200	M-002
RRC-09-25	484772.6	1202465.8	4515.0	0	-90	200	F-360
RRC-09-26	483630.7	1201624.4	4549.0	0	-90	200	F-283
RRC-09-27	483449.1	1202094.3	4578.0	0	-90	225	F-005
RRC-09-28	483602.2	1202228.5	4568.0	0	-90	161.5	F-002
RRC-09-29	482076.7	1200396.3	4672.0	0	-90	150	E-524
RRC-09-30	484590.7	1202656.9	4563.0	135	-45	300	RH-123
RRC-09-X01	485822.1	1204169.3	4449.8	135	-60	452	Exploration
RRC-09-X02	485653.3	1203861.0	4458.8	135	-60	458.5	Exploration
RRC-09-X03	485457.2	1203498.1	4468.1	135	-60	415	Exploration
RRC-09-X04	486013.9	1204127.6	4557.6	0	-90	390.5	Exploration
RRC-09-X05	483217.4	1200313.0	4690.6	135	-60	430	Exploration
RRC-09-X06	480939.2	1199468.5	4793.5	135	-60	430	Exploration
RRC-09-X07	481252.5	1199158.5	4749.7	135	-60	300	Exploration
RRC-09-X08	481418.3	1198999.6	4726.5	135	-60	250	Exploration
RRC-09-X09	485212.9	1202765.0	4517.3	135	-60	450	Exploration
RRC-10-01	482262.7	1200589.0	4634.3	135	-70	410	Exploration
RRC-10-02	482312.4	1200822.3	4636.8	135	-60	480	Exploration
RRC-10-03	482482.8	1200935.6	4636.1	135	-70	500	Exploration
RRC-10-04	482459.8	1201257.6	4696.6	135	-55	490	Exploration
RRC-10-05	482561.5	1201422.5	4679.2	135	-60	560	Exploration
RRC-10-06	483119.8	1201473.9	4574.9	135	-70	442	Exploration
RRC-10-07	483268.3	1201424.6	4548.0	135	-60	330	Exploration
RRC-10-08	483375.4	1201453.6	4547.6	135	-60	380	Exploration
RRC-10-09	483463.5	1201505.8	4547.1	135	-60	410	Exploration
RRC-10-10	483049.0	1202071.5	4753.2	135	-80	537	Exploration
RRC-10-11	483517.6	1201595.0	4547.6	135	-60	490	Exploration
RRC-10-12	483099.0	1202157.8	4752.9	135	-60	490	Exploration
RRC-10-13	484827.5	1202860.5	4577.1	135	-70	470	Exploration
RRC-10-14	483543.9	1201708.3	4549.2	135	-70	470	Exploration
RRC-10-15	483223.9	1202292.3	4729.0	135	-60	849	Exploration
RRC-10-16	483367.8	1202457.4	4717.0	135	-70	770	Exploration
RRC-10-17	484175.3	1201925.1	4520.1	135	-70	300	Exploration
RRC-10-18	483666.0	1202440.9	4605.8	135	-70	501	Exploration
RRC-10-19	483732.1	1202515.8	4606.7	135	-80	700	Exploration
RRC-10-20	483671.5	1202722.3	4690.8	135	-60	850	Exploration
RRC-10-21	483702.3	1202830.1	4682.0	135	-80	820	Exploration
RRC-10-22	483767.8	1203049.3	4664.7	135	-60	944	Exploration

Table 10-3: 2010 Reverse Circulation Drilling Program

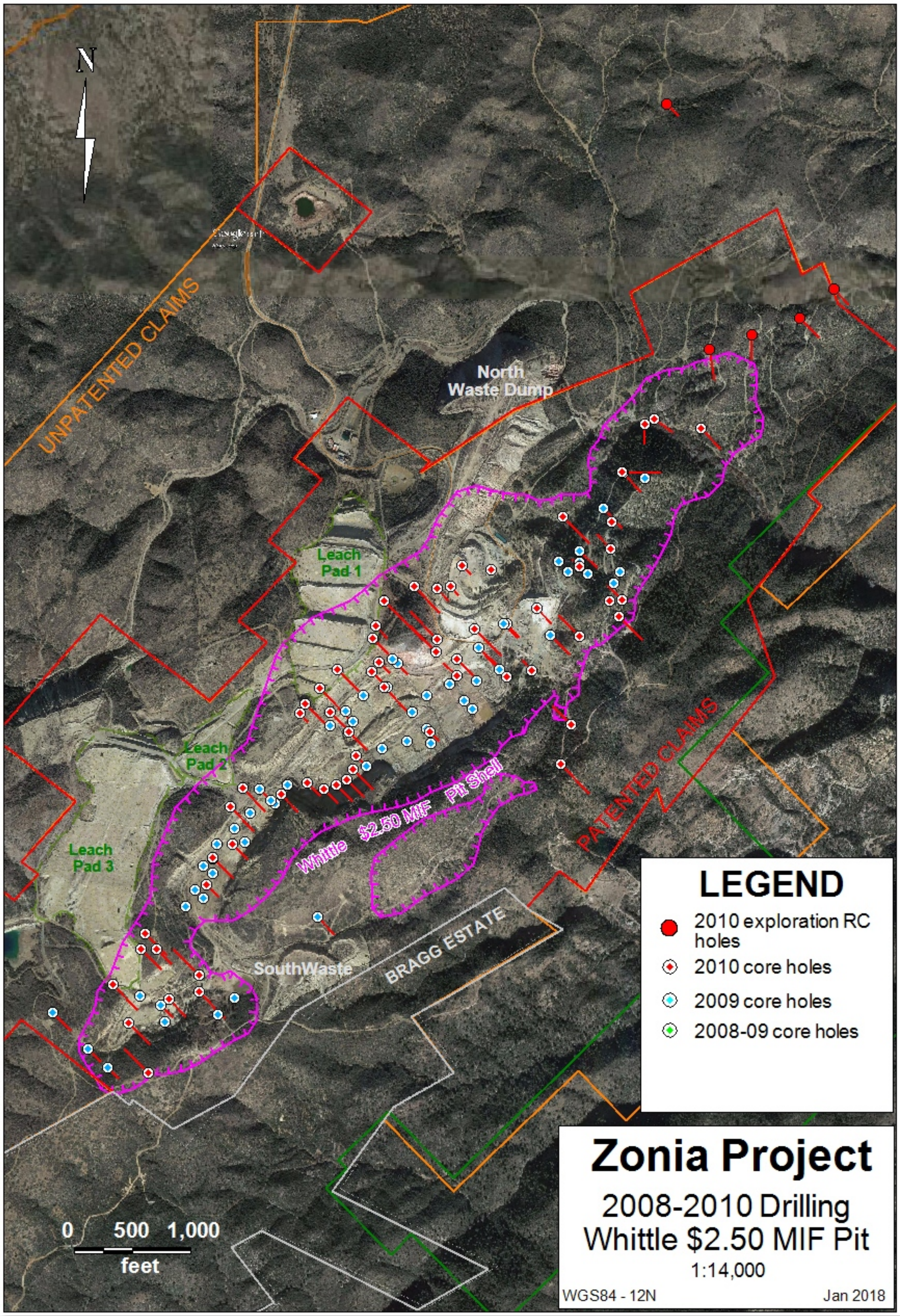
Hole Id	Easting	Northing	Collar Elevation (feet a.m.s.l.)	Azimuth	Dip	Length (feet)
R-RC10-26	484836.6	1202400.1	4498.0	315	-60	550
R-RC10-27	484224.0	1203159.2	4595.5	135	-75	565
R-RC10-28	484556.6	1202816.1	4595.7	135	-60	635
R-RC10-29	484343.6	1203178.8	4554.8	135	-75	369
R-RC10-30	485049.6	1202463.1	4495.0	315	-75	465
R-RC10-31	484442.5	1203362.2	4553.8	135	-75	400
R-RC10-33	484693.0	1203329.0	4490.0	0	-90	485
R-RC10-35	485088.7	1202993.5	4514.1	135	-75	700
R-RC10-39	485457.6	1203357.9	4527.9	315	-75	350
R-RC10-40	485310.4	1203793.2	4393.0	135	-60	635
R-RC10-41	485726.4	1203513.3	4551.3	315	-75	690
R-RC10-42	485730.8	1203747.0	4513.7	0	-90	575
R-RC10-43	485818.0	1204176.2	4449.2	90	-60	655
R-RC10-44	486011.8	1204594.2	4310.8	180	-60	340
R-RC10-46	486490.4	1204559.2	4328.5	135	-60	500
R-RC10-48	485714.0	1203063.8	4591.0	0	-90	265
R-RC10-49	485456.7	1202760.7	4610.6	315	-60	500
R-RC10-50	486907.5	1205352.3	4479.4	180	-60	425
R-RC10-51	486558.4	1205209.1	4447.6	170	-60	540
R-RC10-52	484456.9	1204481.3	4598.2	135	-60	765
R-RC10-53	484893.8	1203982.1	4444.1	135	-60	415
R-RC10-54	484844.2	1203168.0	4549.4	135	-60	585
R-RC10-55	484976.4	1202402.8	4505.0	315	-60	400
R-RC10-56	483883.0	1203074.3	4687.0	135	-60	700
R-RC10-57	481734.3	1200155.6	4763.1	135	-60	435
R-RC10-58	481634.8	1199945.6	4719.7	135	-60	405
R-RC10-AA	482896.1	1201371.0	4654.5	135	-65	500
R-RC10-BB	481695.6	1200024.9	4718.0	135	-65	570
R-RC10-C	488296.2	1210324.5	4524.2	170	-60	535
R-RC10-CC	481609.8	1199402.2	4694.7	135	-65	600
R-RC10-D	488148.4	1208768.3	4510.8	100	-60	455
R-RC10-E	486170.9	1207352.9	4620.2	135	-60	285
R-RC10-EE	481769.4	1198953.6	4768.3	315	-65	700
R-RC10-F	488399.5	1206216.9	4496.4	135	-55	385
R-RC10-FF	482206.2	1199657.8	4628.3	135	-60	615
R-RC10-G	487700.0	1205763.7	4520.2	135	-65	545
R-RC10-GG	481941.9	1199593.6	4682.2	135	-65	500
R-RC10-H	487612.0	1205756.0	4517.8	135	-60	375
R-RC10-HH	481457.1	1199716.4	4726.0	135	-65	700
R-RC10-I	485827.0	1203076.5	4530.4	315	-75	700
R-RC10-II	484218.9	1202616.7	4605.8	315	-60	630
R-RC10-J1	485803.5	1202928.6	4530.4	135	-60	545
R-RC10-L	485392.5	1201996.3	4551.5	315	-60	450
R-RC10-M	485307.9	1201656.8	4578.9	135	-60	700

Hole Id	Easting	Northing	Collar Elevation (feet a.m.s.l.)	Azimuth	Dip	Length (feet)
R-RC10-N	484232.7	1202721.1	4613.7	315	-60	670
R-RC10-Q	484405.2	1202548.8	4563.1	135	-65	525
R-RC10-R	484027.2	1203172.4	4619.2	135	-65	735
R-RC10-S	484400.3	1202409.5	4516.0	315	-75	600
R-RC10-T	483772.9	1202303.6	4558.0	135	-65	765
R-RC10-T3	481832.9	1200028.1	4756.1	135	-60	365
R-RC10-T4	482200.2	1199805.9	4648.2	315	-60	625
R-RC10-U	483312.7	1202086.6	4637.4	135	-65	660
R-RC10-W	483476.7	1201919.8	4551.7	135	-65	520
R-RC10-Y	486092.3	1204635.9	4309.4	125	-60	380

As part of the 2010 drilling and resource expansion effort, all 39 of the Equatorial RC holes and the 7 Homestake NQ core holes were relogged employing the Redstone core logging scheme, and the data incorporated into the Zonia database.

Figure 10-1 shows the locations of the 2008 to 2010 drillholes.

Figure 10-1: 2008-2010 Drillhole Location Map



11.0 SAMPLE PRESERVATION, ANALYSES AND SECURITY

11.1 Sample Method and Details

Redstone explored the Zonia property with diamond core and reverse circulation drilling methods. Boart Longyear drilled three programs from 2008 to 2010, drilling 77 HQ-sized diamond drill core holes for 25,342.3 feet of drilling. Samples were collected and analyzed from all three programs, totaling 3,086 assays.

Harris and Preston were contracted to conduct reverse circulation drilling in 2010; 54 holes were completed for 28,984 feet of drilling. The reverse circulation program provided 5,739 assay samples.

11.2 Sampling Method and Data Collection

11.2.1 Diamond Core Drilling

Core diameter was HQ, which is approximately 2.75-inch in diameter. Following convention, the drill crew at the drill site placed core samples in waxed, ten-foot capacity cardboard boxes (Photo 11-1 provides representative examples of filled core boxes marked according to depth. Sample boxes were delivered to Zonia's secure sample warehouse at the mine site.

Once the sample boxes arrived, they were catalogued, photographed, and readied for logging. Once the logging was complete, the location of individual samples was determined, and the core was split, employing a standard 12-inch core saw.

11.3 Data Collection

Drill core was systematically logged by geologists of Redstone in facilities on the Zonia property. Diamond drill core logging included the following components:

- **Geologic Description,** including estimation of lithology and alteration types along with estimates of quartz veining, and estimates of intensity of mineralization, including sulfides, green copper oxide, dark copper oxide, and iron oxide. In addition, specific observations were recorded.
- **Geotechnical** data, including percent recovery, total length, rock quality designation (RQD) %, number of fractures, and descriptive notes. RQD measurements represent the percentage of the accumulation of unbroken pieces that exceeded 10 centimeters of the total length of core collected in the run. Breaks in the core caused by drilling procedures were not counted. The average recovery of 51 core holes was 96%.
- **Weight of Cores** was collected on a box-by-box basis, including accounting for total weight and that of representative samples.
- **Photographs** of all core were taken sequentially by box.
- **Hand Sample Descriptions** were written to include detailed lithology, alteration, and mineralization of representative rock types as noted by depth in each drillhole.

Photo 11-1: Core Box

- **Density Measurements** were taken by Skyline Labs on a total of 41 samples, 16 of which were inside the mineralized host rock. The average density of the 16 host rock sample determinations is 12.6 ft^3/ton. The existing population is too small to statically support changing the currently accepted value of 12.5 ft^3/ton. The currently accepted density value of 12.5 ft^3/ton was determined by Arimetco Inc. and was used in the 1995 Zonia Feasibility Study (Western States Engineering, 1995). It is recommended that further density data collection procedures include a systematic approach consisting of a much larger data population size equally distributed throughout the mineralized zones and waste rock.

11.4 Reverse Circulation Drilling

Double walled pipe was used for reverse circulation drilling, allowing for the sample to return up the drill string without contacting the annulus of the drillhole. The drilled material passed through a Gilson splitter and was separated, with a ¼ split of the material being bagged and retained for an assay sample. Sample size was determined to most closely match the volume of HQ split core and averaged approximately 15 lbs.

Geologic descriptions were made of chip samples collected from the Gilson splitter, with each sample representing a 5-foot interval length of the drillhole. Geologic description included lithology type, oxidation, mineral observations, and additional notes and description.

11.5 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 85 percent.

11.6 Sample Quality

It is the authors' opinion that Redstone's drill core samples of the Zonia 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.

Further evidence for this belief that both the historical and new data meet NI 43-101 and CIM requirements for the reporting of mineral resources is noted in Section 14.0 of this report. Section 14.0 includes a discussion of how the twin-hole program played a key role in this conclusion.

11.7 Sample Preparation, Analyses and Security

The authors have reviewed Redstone's sample preparation, handling, analyses, and security procedures. It is the authors' opinion that the current practices meet NI 43-101 and CIM defined requirements. During a site visit in May 2008, the authors noted that standard samples were being stored in an unsecured area. It was recommended that the standards be placed in a locked, secure area, and Redstone updated storage procedures and applied these procedures to the remaining 2009 and 2010 drill programs.

11.7.1 Core Sample Preparation and Security

Drill core was transported at the end of each shift by the drill crew to Redstone's secure sample warehouse and logged by a Redstone geologist, who marked approximate five to ten-foot sample intervals with a colored marker. Each group of three core boxes, bearing a label tag showing drillhole number, box number, and box footage interval, was then photographed. RQD and recovery measurements were taken. Core was then split in half using an electric core saw at the warehouse by Zonia personnel.

11.7.2 Sample Analysis

From 2008 to 2010, 3,086 samples from 77 HQ-size core drillholes and 5,739 samples from 54 reverse circulation drillholes were analyzed at Skyline Assayers & Laboratories (Skyline) in Tucson, Arizona. Skyline is a fully accredited independent assay laboratory. Redstone shipped samples to Skyline via lab pick-up service, which assured chain of custody from secured Zonia mine facilities to secured Skyline facilities. A

Quality Assurance and Quality Control (QA/QC) assay protocol was implemented by Redstone, where a standard, blank, and duplicate were inserted in the assay stream along with each set of 20 samples. Control samples account for approximately 13% of the samples submitted. Samples sent to Skyline were separated into batches, with each batch containing three control samples. In the event of a control sample assay failure, all samples from the corresponding failed batch were requested by Redstone to be re-analyzed. Skyline also implemented their own QA/QC protocol, where a control sample assay failure would also initiate a batch re-run. Skyline's test procedures were as follows:

- For %TCu: a 0.2000 to 0.2300-gram 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/atomic absorption spectroscopy (AAS) using standards and blanks for calibration.
- For %AsCu: a 1.00 to 1.05-gram 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 % Quick Leach Time (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: %TCu, %AsCu, Cyanide Soluble Copper (%CNCu), and the difference, or Residual. Following analysis for %TCu and %AsCu, 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 QLT leach, with possible greater leaching of certain sulfides (e.g. chalcocite or bornite) during the cyanide leach step.

11.7.3 Quality Control

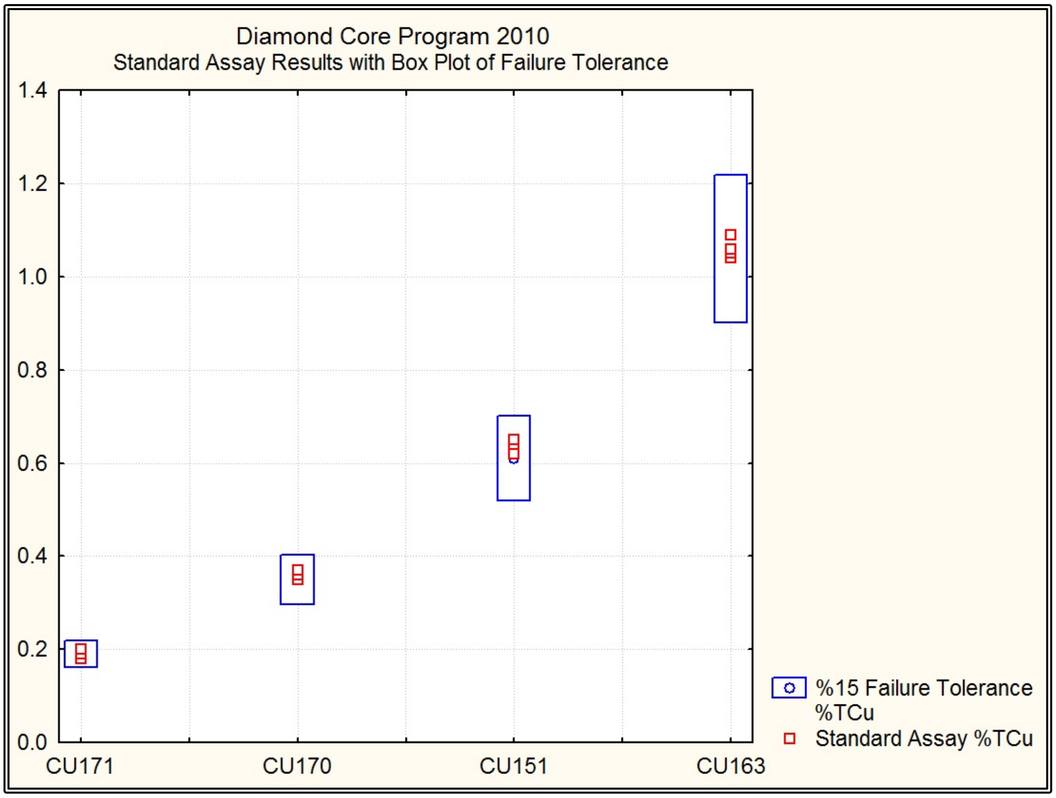
As part of the Redstone's QA/QC program for diamond core drilling from 2008 to 2010, 827 standards and 725 blanks were submitted to Chemex and Skyline Labs. Results from the two laboratories are shown in Table 11-1. Lot failure criteria was established as any standard assay outside of ±15% of the reported value for the standard or any blank assay greater than 0.015 %TCu. Re-analysis was requested for all failed lots, and all re-run samples supersede those from the initial failed lot.

The authors recommend that further drill programs, both diamond and reverse circulation, use the industry standard criteria for lot failure of ±2 standard deviations. When this standard is applied to the 2010 diamond drill program, where no standards are outside ±15%, there are 10 instances of standards being outside the ±2 standard deviation criteria. See Figure 11-1 for comparison of all four standards used for the diamond drill program compared to checked assay samples.

Table 11-1: Blank and Standard Failure Rates by Laboratory for 2008-2010 Core Drilling

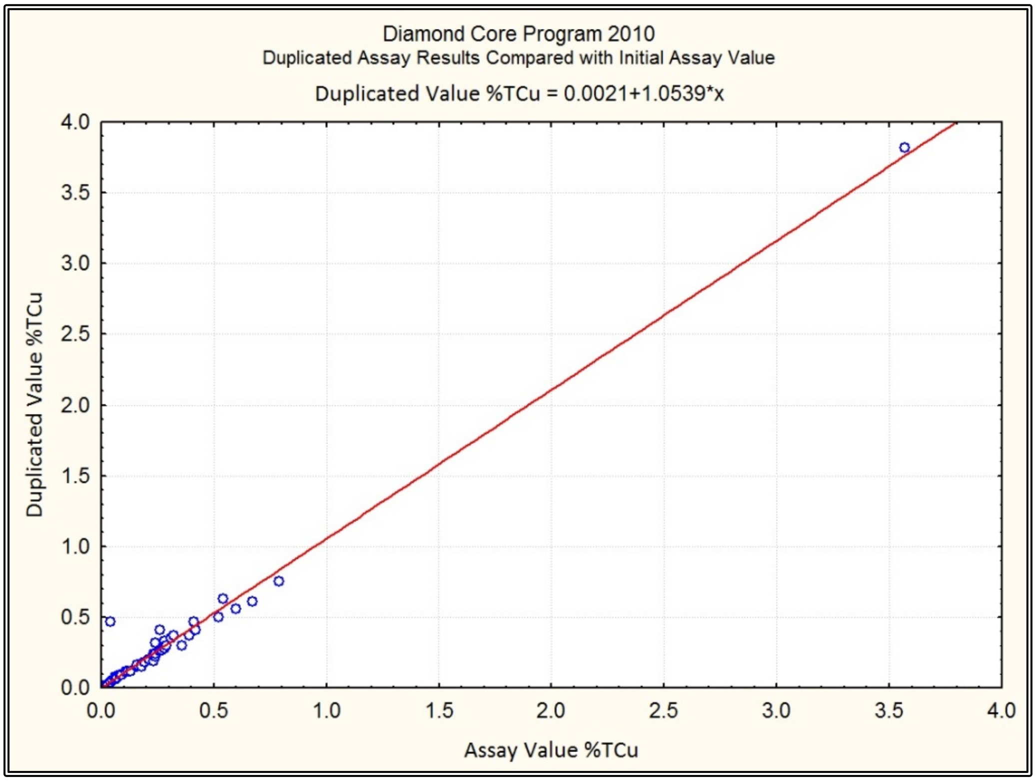
	Chemex Assay Labs	Skyline Laboratories
Submitted Blanks	20	705
Failed Blanks	0	16
% Blank Failure	**0.0**	**2.2**
Submitted Standards	32	795
Failed Standards	7	17
% Standard Failure	**21.9**	**2.1**

Figure 11-1: Diamond Core Program 2010 Standard Results with Accepted Failure Tolerance



The difficulty of comparing duplicated samples in a meaningful way inhibits the use of a simple pass or fail test. The authors reviewed the duplicated values and determined that all but one duplicated sample failed to show significant error that can be directly attributed to anything other than localized variability. Sample #375742 from hole RRC-10-05, where the initial test is 0.04 %TCu and duplicated test is 0.47 %TCu, has significant enough variability that the authors recommended Redstone investigate possible sources of error. When the 60 duplicated assay values are compared with initial assay values, a linear fit with a slope of 1.1 is observed; a perfectly duplicated data set would have a fit line slope of 1. Results are shown in Figure 11-2.

Figure 11-2: Diamond Core Program Duplicate Results Compared to Initial Assay Results



The QA/QC program for the reverse circulation drilling conducted in 2010 included analysis of 264 standard samples, 288 blank samples, and 262 duplicate samples. Standards of eight different %TCu values ranging from 0.19-1.15 %TCu were used. Using criteria for failure as an assay value greater than 15% different than the expected standard value, only one tested standard failed. Standard CU170 has an expected value of 0.34 %TCu and, in one case, returned a value of 0.20 %TCu, an error of 42.9% less than expected; all samples from this batch were re-analyzed. See Figure 11-3 for comparison of all eight standards compared to checked assay samples. Of the 288 blanks that were submitted, only one blank sample failed. Using a failure criterion of any value with a returned value greater than 0.015 %TCu, all samples from this batch were re-analyzed. The difficulty of comparing duplicated samples in a meaningful way inhibits the use of a simple pass or fail test. The authors reviewed the duplicated values and determined that none of the duplicated samples show significant error that can be directly attributed to anything other than localized variability. When the 262 duplicated assay values are compared with initial assay values, a linear fit with a slope of 0.99 is observed; a perfectly duplicated data set would have a fit line slope of 1. Results are shown in Figure 11-4.

In addition to in-stream reverse circulation QA/QC assay procedure, Redstone submitted pulps from 304 samples previously analyzed by Skyline to ALS Minerals of Reno, Nevada, for comparison in 2011. Of the 304 pulps submitted, 17 samples were standards. All standard pulps tested are inside ±8% of the expected value and are not included Figure 11-5. Of the remaining 287 pulps tested, comprising interval samples collected during the 2010 reverse circulation drill program, all but one test showed good replication of the original assay value by Skyline (see Figure 11-5). The authors recommended that sample 810412 be sampled again due to the large discrepancy in values. Skyline's original test indicates a value of 0.06 %TCu,

Figure 11-3: Reverse Circulation Standard Results with Accepted Failure Tolerance

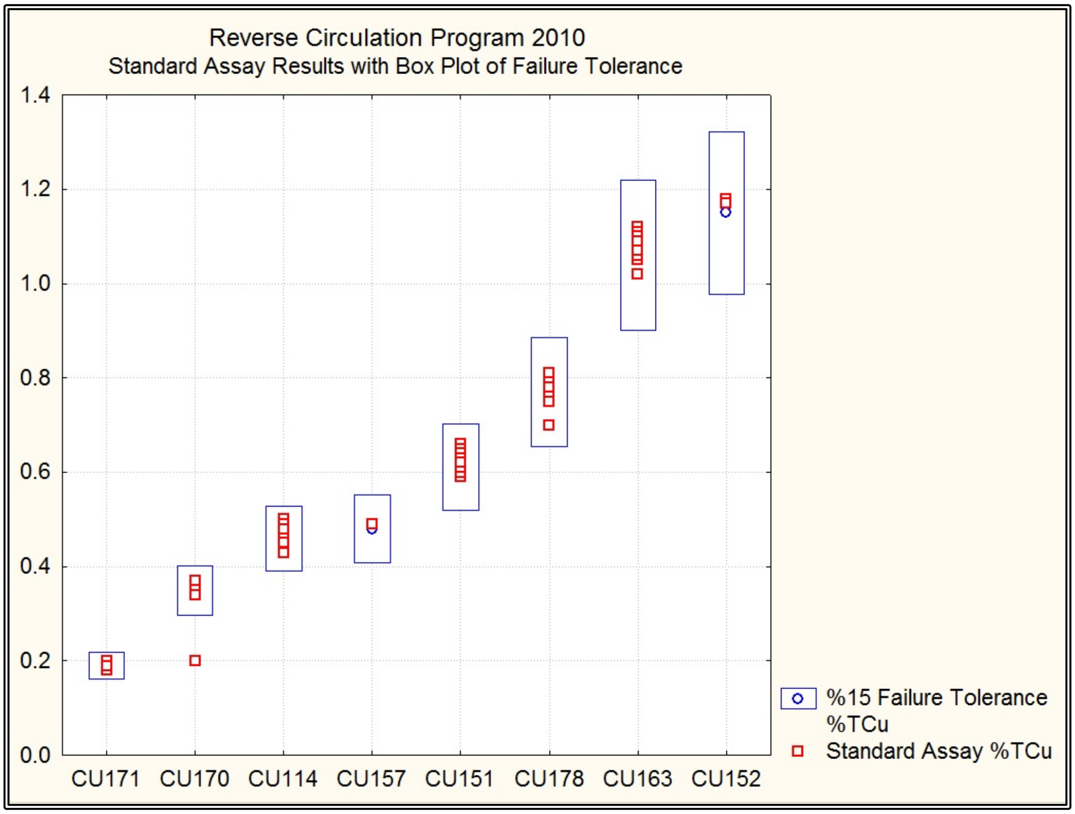


Figure 11-4: Reverse Circulation Duplicate Results Compared to Initial Assay Results

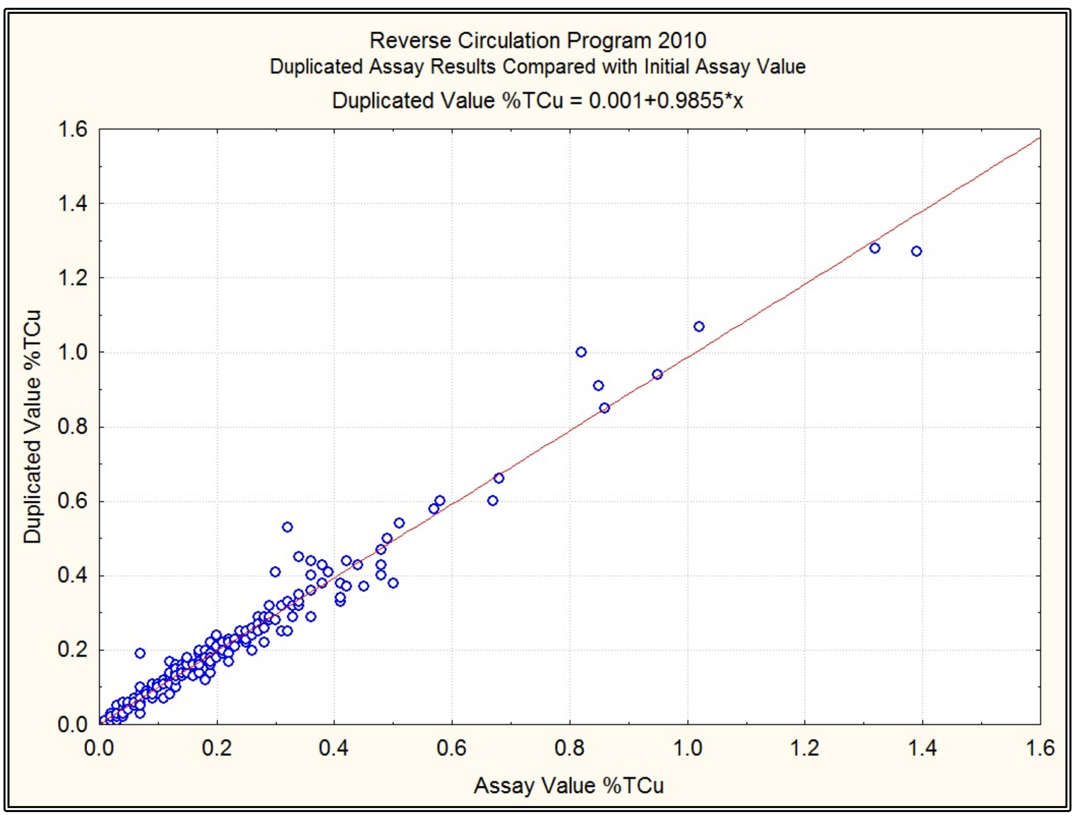
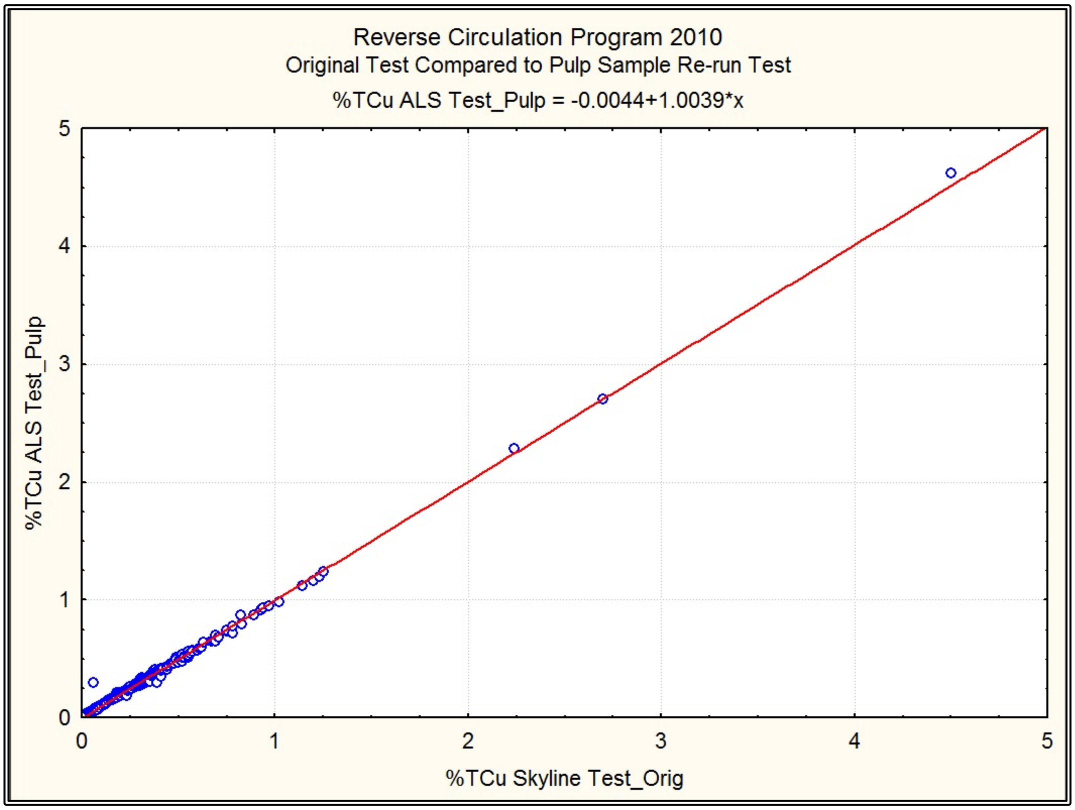


Figure 11-5:Reverse Circulation Original Test Compared to Pulp Re-Run Test



but when re-run by ALS, the value was indicated to be 0.30 %TCu. Other than the isolated incidence mentioned above, the pulp re-run exercise indicates the validity of the initial analyses by Skyline.

12.0 DATA VERIFICATION

This section details the results of the authors' verification of existing data for the Zonia Project.

12.1 Topography

Topographic control on the Zonia project site has developed through several stages. Historical exploration and development work was completed based on an orthogonal mine grid established with a northeast-southwest baseline at a bearing of 055°. Drillholes, pit development, mine facilities, and historical mapping data were recorded in this system.

To facilitate on-going work, the site coordinate system has been upgraded to AZ State Plane (NAD 83, Central Zone) coordinates. This allows translation of all historical coordinates into a modern survey base system amenable to field global positioning system survey control.

The update was completed by comparison of site geographic features located in mine grid with lat.-long. control points identified on a Google-Earth® orthographic image and U.S. Geological Survey (USGS) topographic base maps of the site. Site mine grid coordinates were compared with the coordinates from the maps from which a x-y and rotational shift translation was developed to allow for datum projection. Digital elevation data for the site was obtained from the USGS map sources.

Redstone subsequently contracted an aerial survey of the site, from which Orthoshop Inc. of Tucson, AZ developed a 2-foot contour map of the area encompassing the patented claims and the Bragg Estate. Contours for the patented claim block at the core of the Zonia property were visually and, thereafter, electronically merged with 10-foot contours from USGS sources.

In 2009, Redstone acquired limited ground survey data on the project site. Two control points were established from which a final refinement of the translational adjustment to site maps was developed. Site coordinates are now collected directly in lat.-long. coordinates or in the mine grid system and are electronically converted to AZ State Plane.

12.2 Twinned Hole Correlation Study

Drilling and sampling has occurred at the Zonia site over many decades. To assess the usability of the historical assay data, a twinned drillhole study of the relationship between historical drill assay information and data from associated recently drilled twin holes was conducted. The study incorporates two twin drilling campaigns. The first, completed in 2008 comprised 16 drillholes and are called RRC twins. After a reinterpretation of geology, a second drillhole campaign of 30 drillholes was done in 2009. These drillholes are called RRC09 twins.

12.2.1 Twin Drillhole Locations for the 2008 RCC and 2009 RCC09 Drillholes

Sixteen twin holes were drilled and assayed by Redstone in 2008 and an additional 30 in 2009. These drillholes are tabulated alongside their corresponding historical twin drillhole in Table 12-1. The twinned holes are located on ten sections (3400S through 1200N) as shown in Figure 12-1 for the RRC drillholes and Figure 12-2 for the RRC09 drillholes.

Table 12-1: Twinned Hole Listing

Redstone	Historical	Redstone	Historical
RRC-01	M-018	RRC09-08	F-296
RRC-02	F-267[2]	RRC09-09	F-292
RRC-03	F-265[4]	RRC09-10	M-009
RRC-04	F-279[2]	RRC09-11	F-195
RRC-05	F-272	RRC09-12	F-194
RRC-06	F-189	RRC09-13	F-165
RRC-07	F-301[1]	RRC09-14	F-009
RRC-08	F-300[1]	RRC09-15	F-159
RRC-09	F-291[3]	RRC09-16	F-197
RRC-10	E-527	RRC09-17	F-108
RRC-11	F-171[2]	RRC09-18	F-112
RRC-12	F-176	RRC09-19	F-008
RRC-13	E-529	RRC09-20	F-084
RRC-14	F-085	RRC09-21	E-525
RRC-15	F-336	RRC09-22	F-037
RRC-16	F-097	RRC09-23	F-026
RRC09-01	RH-103	RRC09-24	M-002
RRC09-02	F-276	RRC09-25	F-360
RRC09-03	F-270	RRC09-26	F-283
RRC09-04	F-353	RRC09-27	F-005
RRC09-05	F-352	RRC09-28	F-002
RRC09-06	F-350	RRC09-29	E-524
RRC09-07	F-299	RRC09-30	RH-123

Notes:

1: RCC-07/F-301, RRC-08/F-300 removed for visual non-correlation (Outside Mineralized zone)

2: Twins RRC-02/F-267 , RCC-04/F-279,and RRC-11/F-171 for visual non-correlation (Within Mineralized zone) likely related to analysis of waste rock in twin holes.

3: Twin RRC-09/F-291 kept - possible detection limit difference been old and new data-used lower grade cut.

4: Twin RRC-03/F-265 kept but both collars are below topographic surface and they are also outside of mineralized zone.

Figure 12-1: Location of Twin Drillholes Labelled with Corresponding Historical Twin

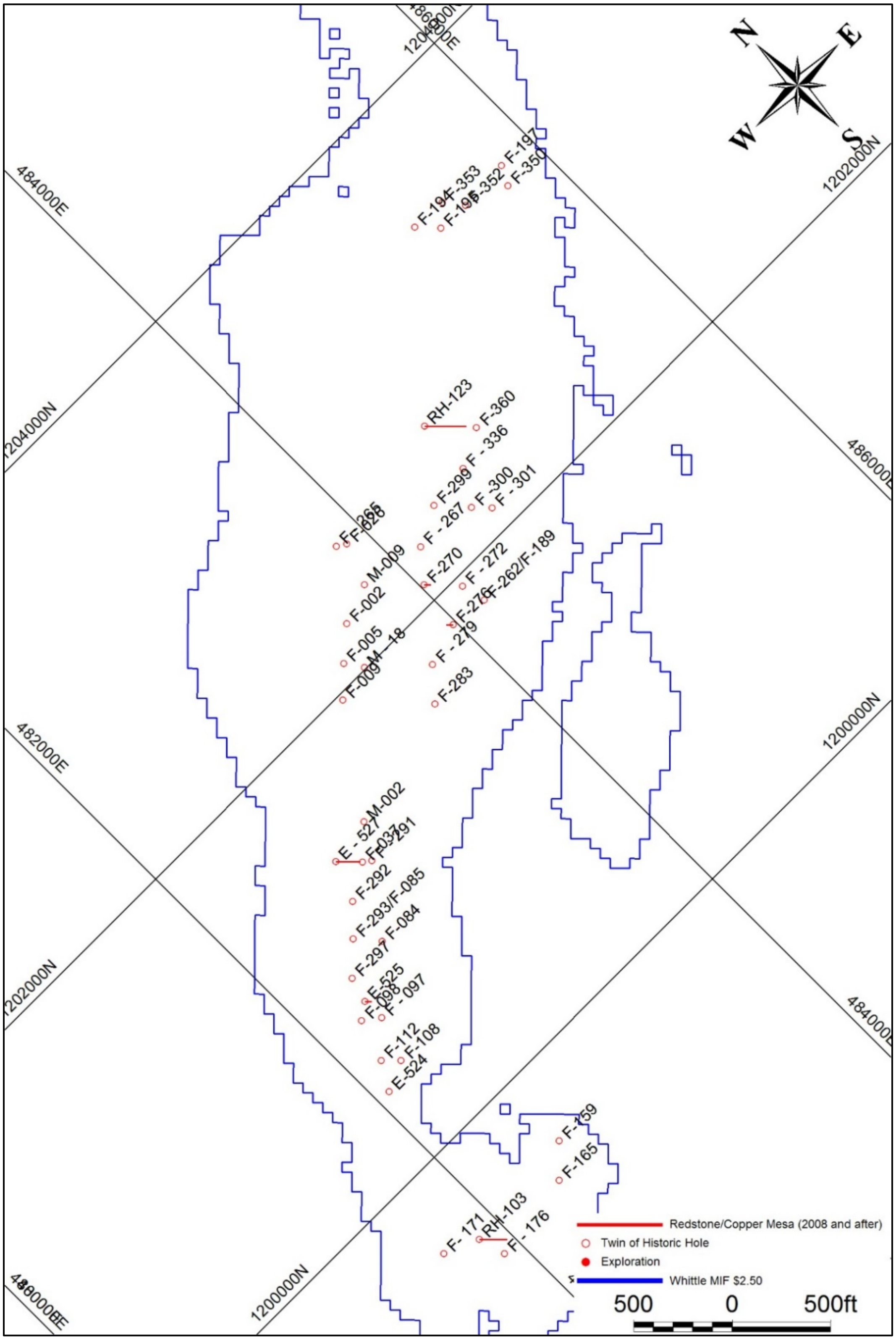
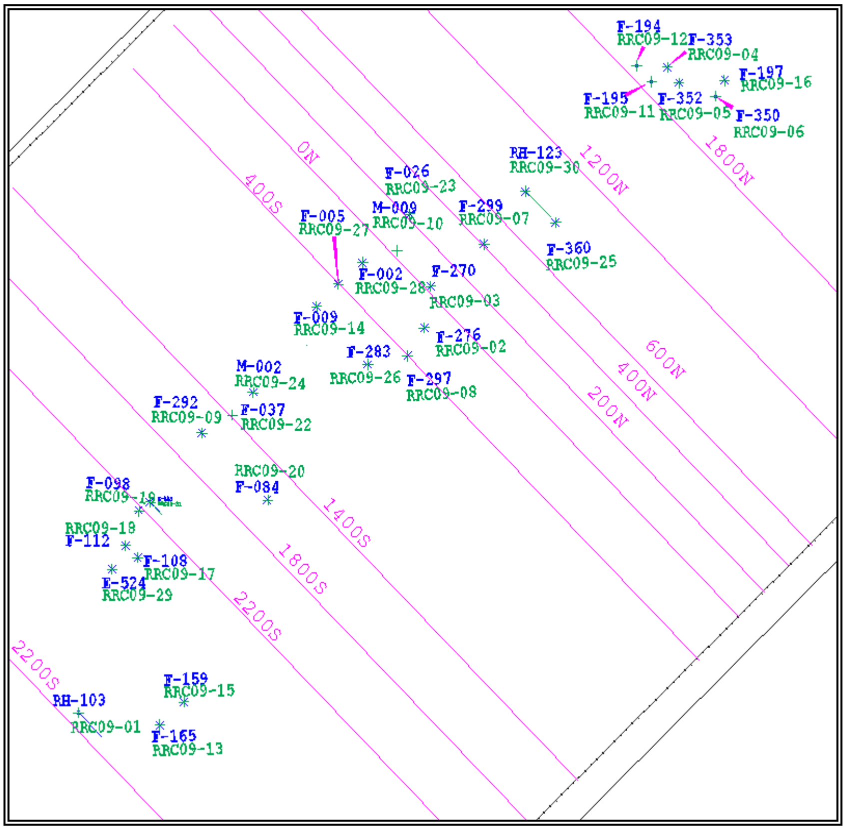


Figure 12-2: Location of Thirty Twin RRC09 Drillholes



12.2.2 Sectional Assay Comparison – RRC Series

New twin drillholes were assayed for %TCu generally over intervals of nine feet as selected by the site geologist. The original twin drillhole assay intervals were generally five feet in length and taken at regular intervals down the hole.

Figure 12-3, Figure 12-4, and Figure 12-5 show drillhole trace histograms and value plots of %TCu on Gemcom® sections viewed towards the north-east. On the sections, original twinned drillhole data (F, E and M series holes) are plotted on the left of the drill trace while the newer twin holes (RCC series) have their values plotted to the right. Figure 12-3, Figure 12-4, and Figure 12-5 show the original assay sample intervals. However; assay sample data were bench-composited to 20-foot intervals for statistical analysis because of the differing sample lengths.

Selected twin holes are shown on Section 0N (Figure 12-3). To clarify visual presentation of the assay data, the newer Redstone twin holes have been displayed alongside historical drill traces. The upper section of historical hole M-108 appears collared above the current topography; previous mining has removed the upper section of the hole. For clarity, assay values for the recent Redstone twin holes are posted to the left, and values for historical holes are provided to the right of drill traces.

Visual inspection of the original RCC data indicates that some may not be representative for such a comparative statistical study. For example, the twinned drillhole pairs RRC-02/M-018 show good grade in the original twinned hole (M-018), but show poor grade with the newer (RCC-02) twin hole. Based on location, it is suspected that the new twin drillhole (RCC-02) was drilled and sampled in rock classified as

Figure 12-3: Example Section (0N) – Excluding RRC-02/F-267 and RRC-04/F-279, as Candidates for Straddling Vertical Mineral Zones

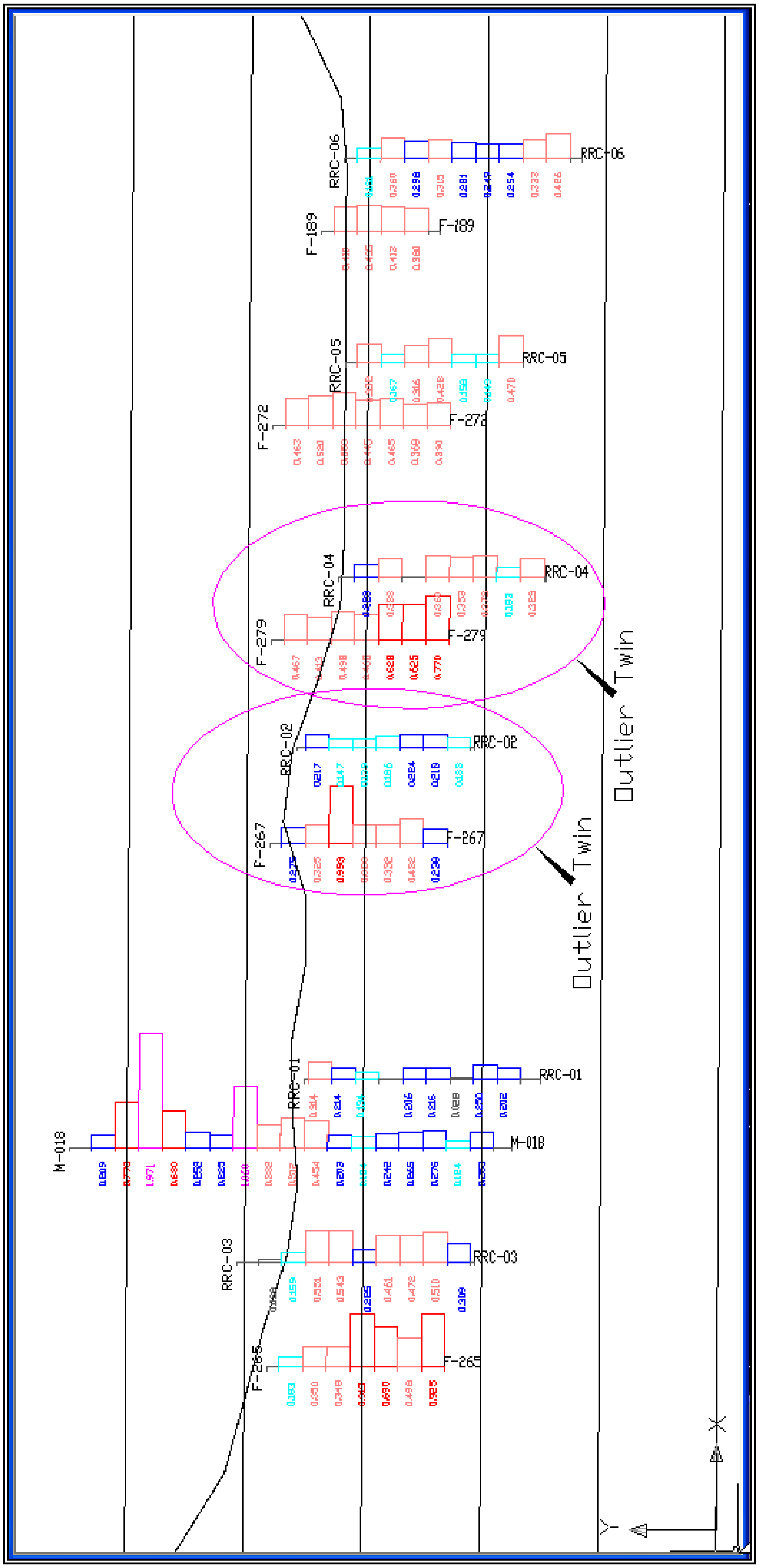


Figure 12-4: Section 1400S (RRC-10/E-527 and RRC-09/E-291)

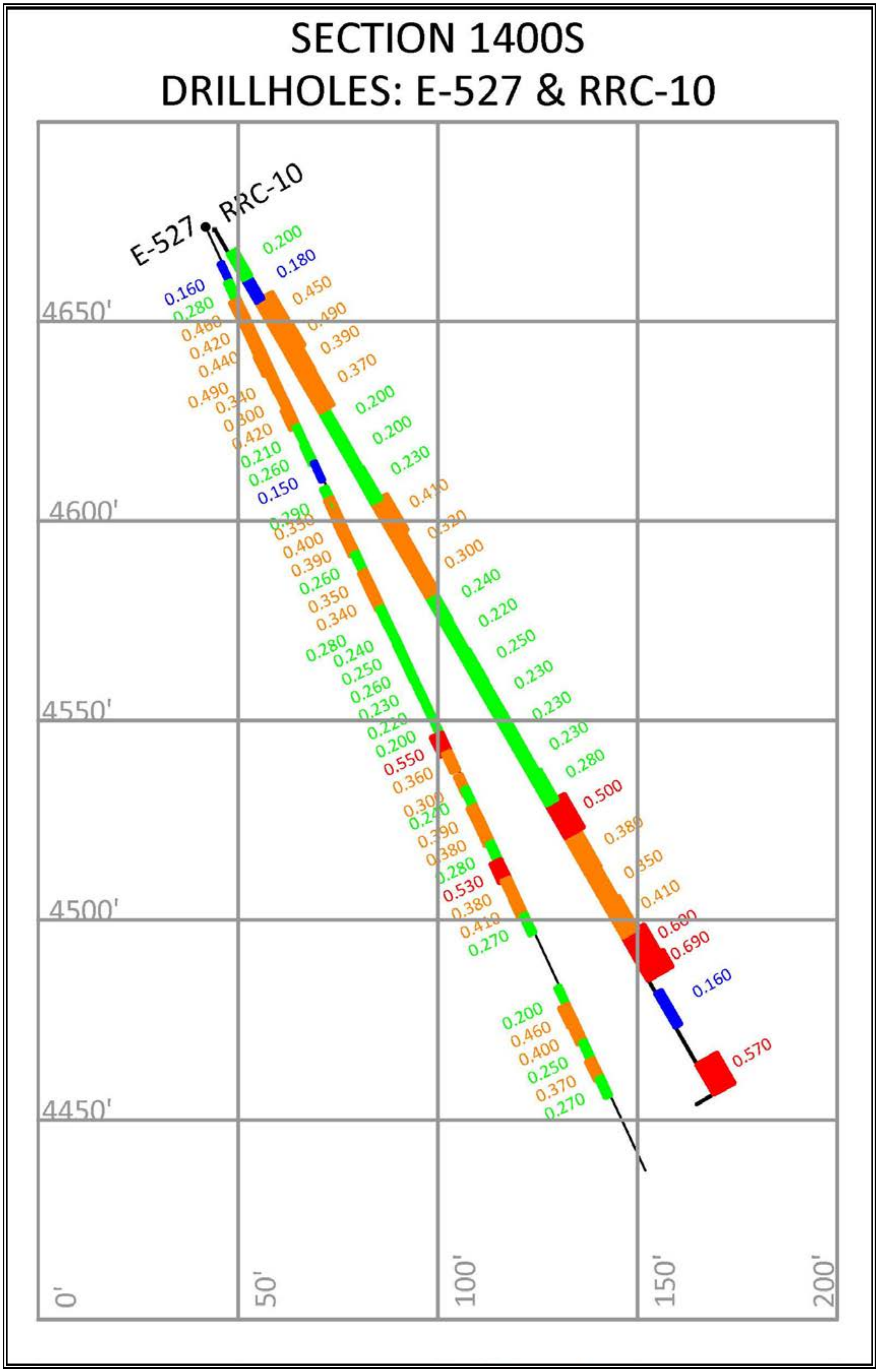
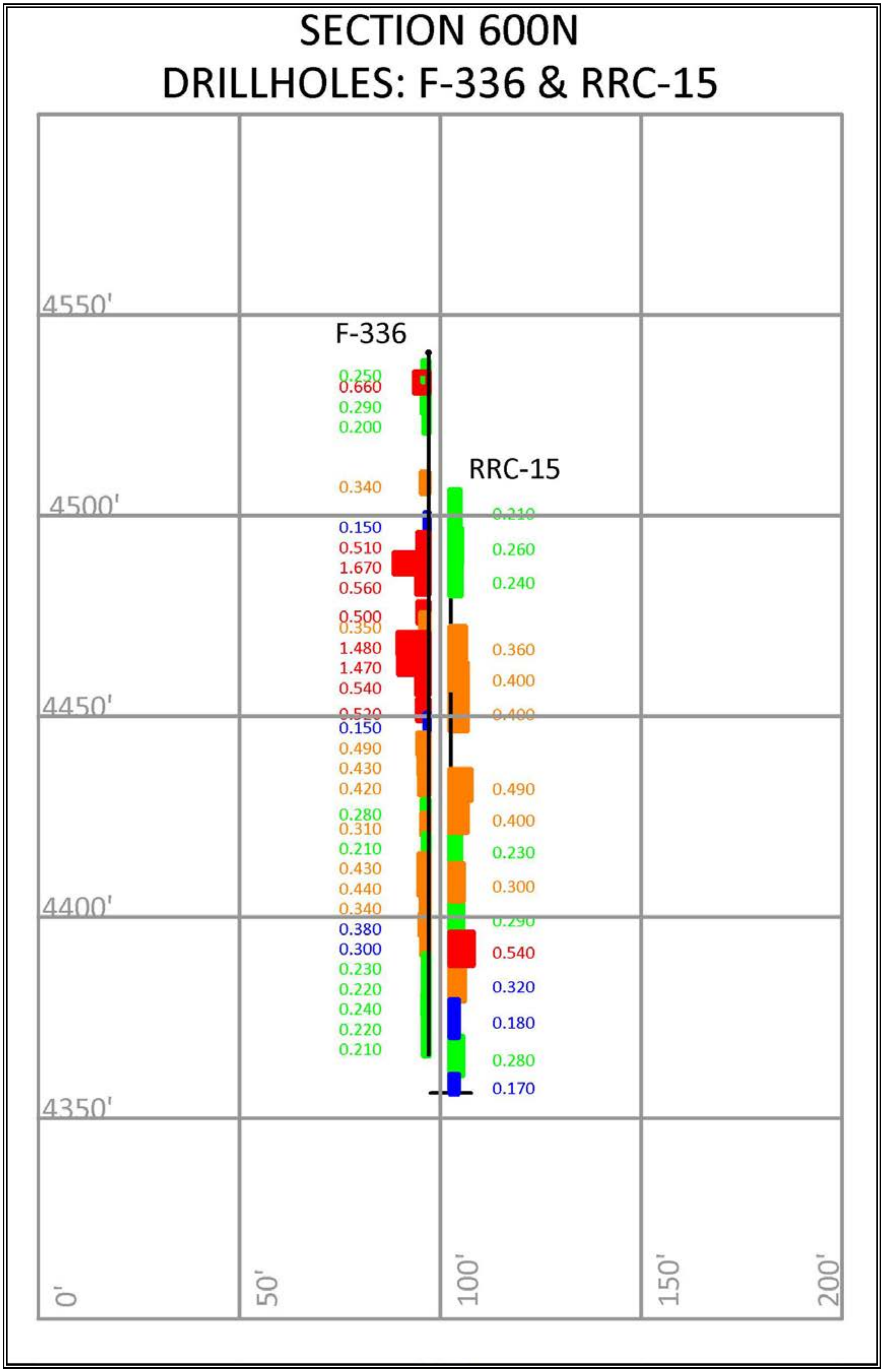


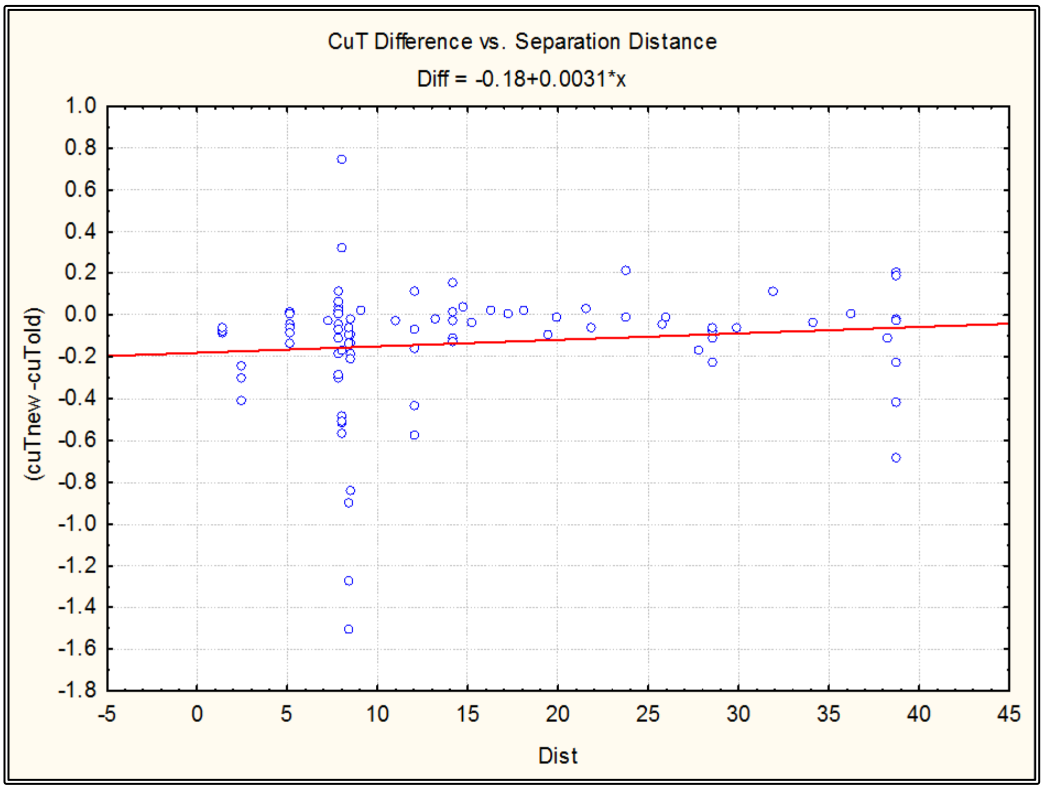
Figure 12-5: Section 600N (RRC-15/F-336)



waste. It is known by pit wall mapping that the mineral zones are sharply defined vertically; RRC-02 appears to have been collared outside of the mineral zone. Likewise, the twins RRC-04/F-297 and RRC-11/F-171 are suspected to have the same issue. To assess the impact of the non-representative RCC data, suspect twins were flagged in the data set and excluded in a second, follow-up statistical analysis. In addition, twins RRC-07/F-300 and RRC-08/F301 were removed based on visual non-correlation as these holes fall outside the interpreted mineral zone wireframe.

Based on visual inspection of the raw data as plotted on the sections, the original, historical twinned holes appear to have higher grades than the newer Redstone twin holes. It is noted that the twin and twinned drillhole pairs did not always have constant separation distance. Figure 12-6 illustrates the difference in copper grade as a function of separation distance for each of the twinned 20-foot composites. Using all of the data, the regression indicates a bias that the older data has higher grade at close distances becoming of similar grade at greater (30-40 feet) separation.

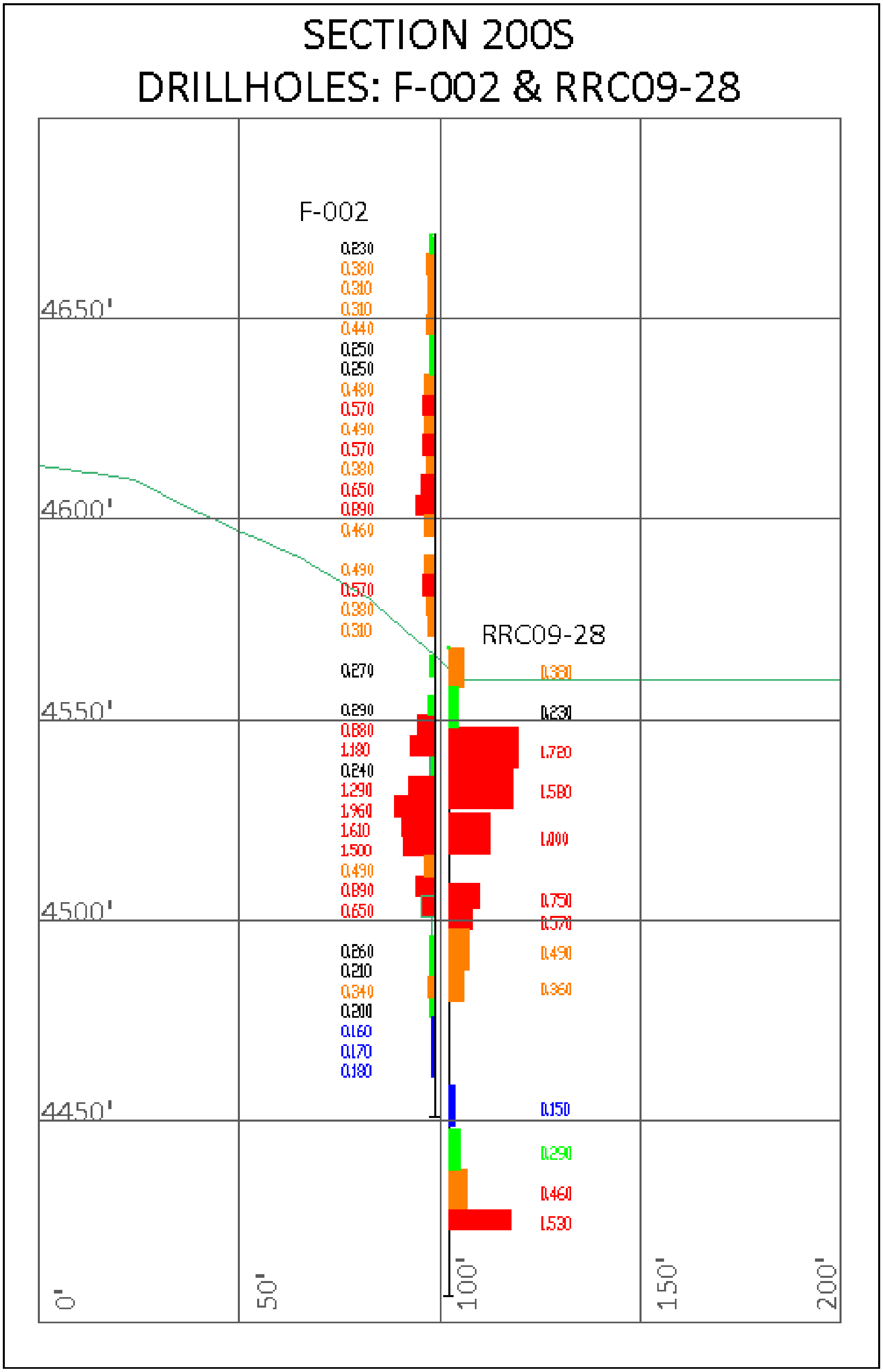
Figure 12-6: Plot of Grade Difference Between Twinned 20-Foot RRC Composites as a Function of Separation Distance



12.2.3 Sectional Assay Comparison – RRC09 Series

Due to the small sample size of only eleven, and the exclusion of suspect twins, the results of the 2008 twinning were inconclusive and subsequently unable to determine whether or not the old data could be used for resource calculations. An additional 30 locations were selected in 2009 to further investigate the validity of the existing assay database. A re-interpretation of geology allowed for a better selection of twin hole locations. Figure 12-7 shows the location of the twin hole pairings drilled in 2009.

Figure 12-7: Section 200S (Drillholes F-002 & RRC09-28)



Unlike the RCC data, all RCC09 twins have been accepted as valid samples. Yet, as in the RCC case, there is an apparent mixture of both good and bad visual correlations. For example, Figure 12-8 and Figure 12-9 show a good visual correlation. Note that the older twins, F-002, M-009 and F-037, have a portion of their assay values above the current topography.

Figure 12-9 displays drillhole F-037 alongside RRC09-22. When compared, the twinned pair does not appear to have as good a visual grade correlation as some of the other drillhole twins. The exact reason in uncertain, but it is suspected that the historical drillhole, an RC hole, may contain some down-the-hole smearing. It should be noted that every second 5-foot interval was not sampled in the zone of poorest correlation in drillhole F-037. This sampling method could account for higher grade mineralization being under represented. It should also be noted that there is a general correlation between zones of higher grade mineralization and areas of moderate mineralization. The poor correlation between twin F-037 and RRC09-22 has been observed as an isolated case.

12.2.4 Statistical Comparison of Assay Data (RRC and RRC09)

Essentially two comparison tests were performed. The first test compared the two populations of old and new twin data. This method develops histograms of the pooled data for both old and new. It then compares the shape of distribution along with the population means. The second method compares the individual composite grades for the new and old twin data in a side-by-side manner. This last method is analogous to the visual correlation previously discussed.

Statistical analysis of the pooled data is shown in Figure 12-10. Note that the new and old data shows a similar log-transformed data distribution. This is shown with both following lognormal distributions that almost overlay each other. However, the distribution of the old composites (LCuTold%) appears to have a slightly higher average grade than that of the new data (LCuTnew%).

A box-and-whisker comparison of the same data (i.e., 20-foot combined RCC and RCC09 composites) also indicates an apparent higher average in the old data compared to the new drillhole data (Figure 12-11). The graph shows the results using the selected data. Yet the boxes containing the mean plus/minus one standard error almost overlaps. To test whether this apparent difference in means is statistically significant, a t-test (Table 12-2) was done. The result indicates that at up to an alpha of 0.15 the means are not significant. The t-test failed to reject the null hypothesis that the new and old twin results are from the same population.

The second test compares the twin's copper data side-by-side as scatter plot of old and new data. This scatter plot is shown in Figure 12-12. A perfect match in grades between the new and old data would show as plotted points along a 45-degree line. The plot shows instead a cloud of points. The correlation of old and new is approximately 0.43. This appears to confirm the more qualitative visual correlation discussed earlier. The plot also has an ellipse plotted to contain 90 percent of the data pairs.

In addition, there may be a difference in detection limits between historical assay data and new assay data. Samples that have assay grades near 0.01 %TCu from new twin drillholes appear to correspond with grades of 0.05 %TCu in assays of original twin drillhole intercepts. In Figure 12-12, points plotted in red have values for %TCu-new that are below 0.05% total copper. The old twin values (%TCu-old) do not

Figure 12-8: Section 0S (Drillholes M-009 & RRC09-10)

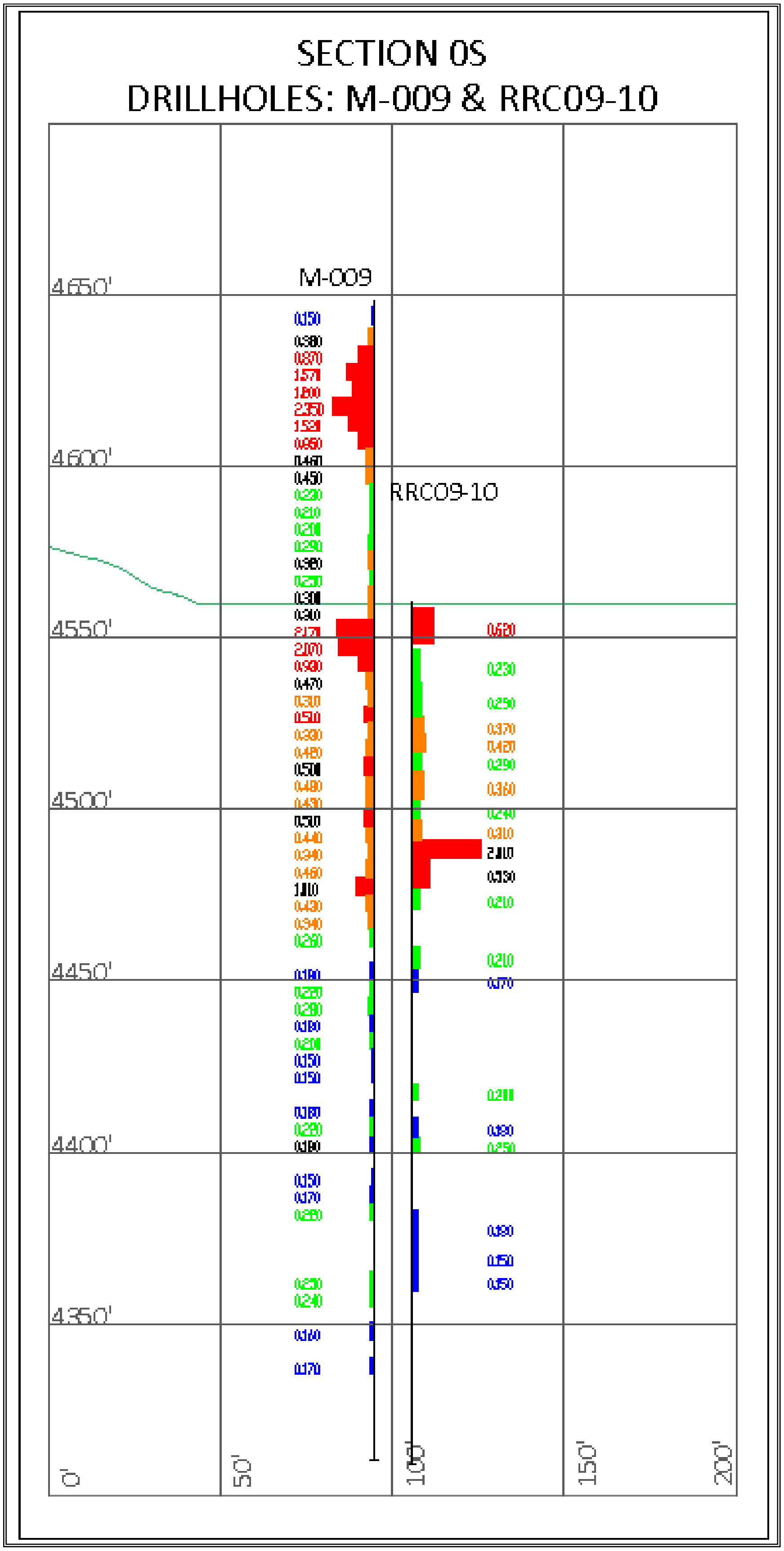


Figure 12-9: Section 1200S (Drillholes M-037 & RRC09-22)

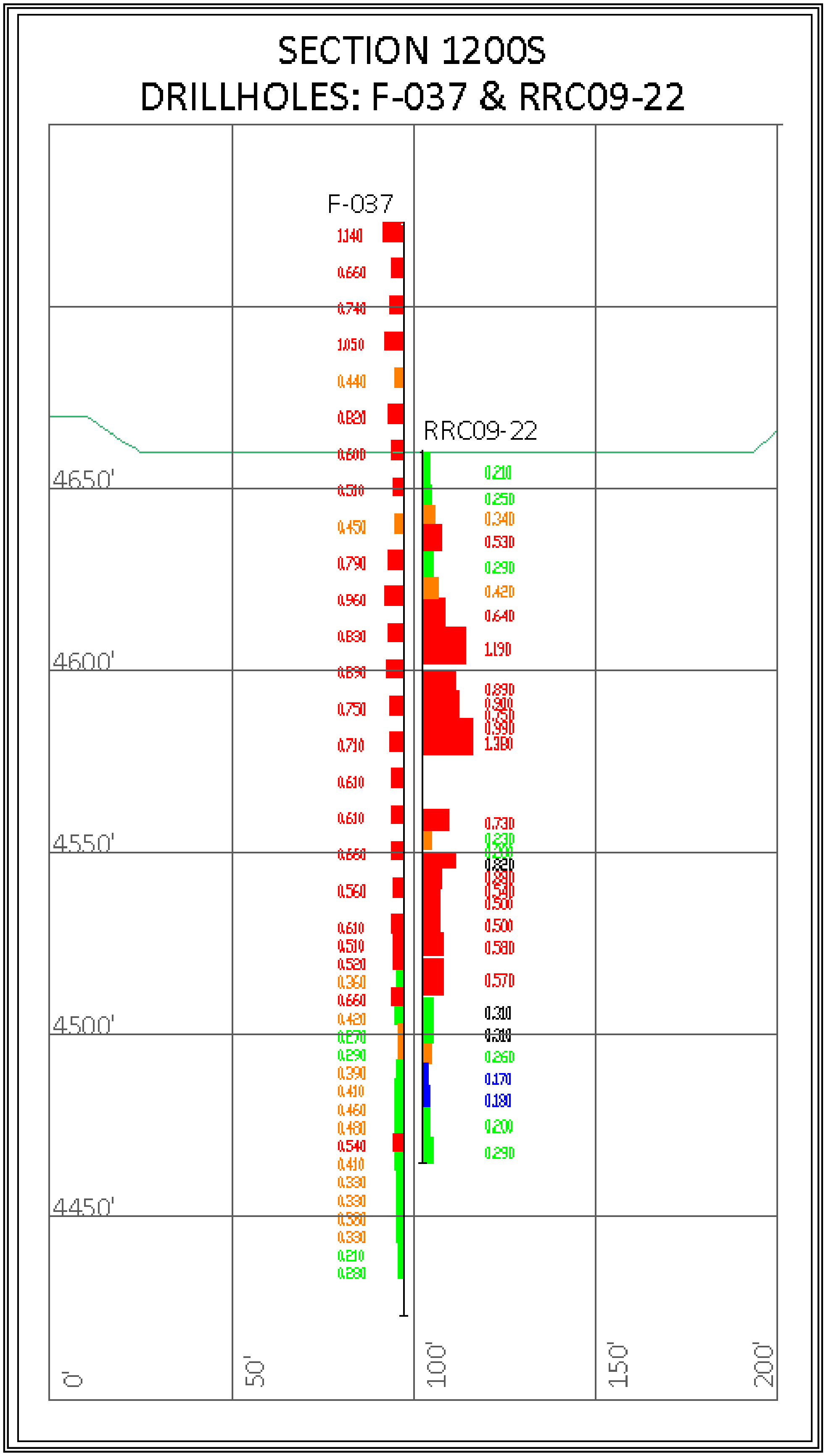


Figure 12-10: Histogram of RRC and RRC09 Twinned Holes (lcut% - 20-foot Composites)

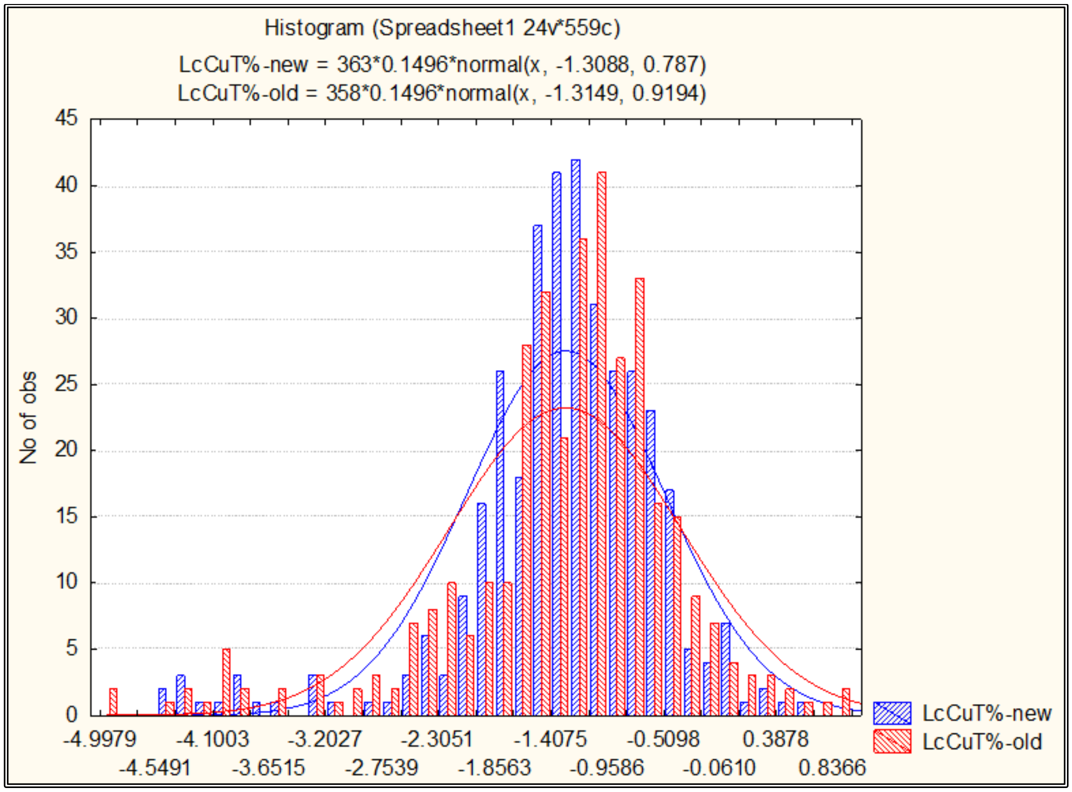


Figure 12-11: Box & Whisker Plots Comparing 20-foot Total Copper Grades

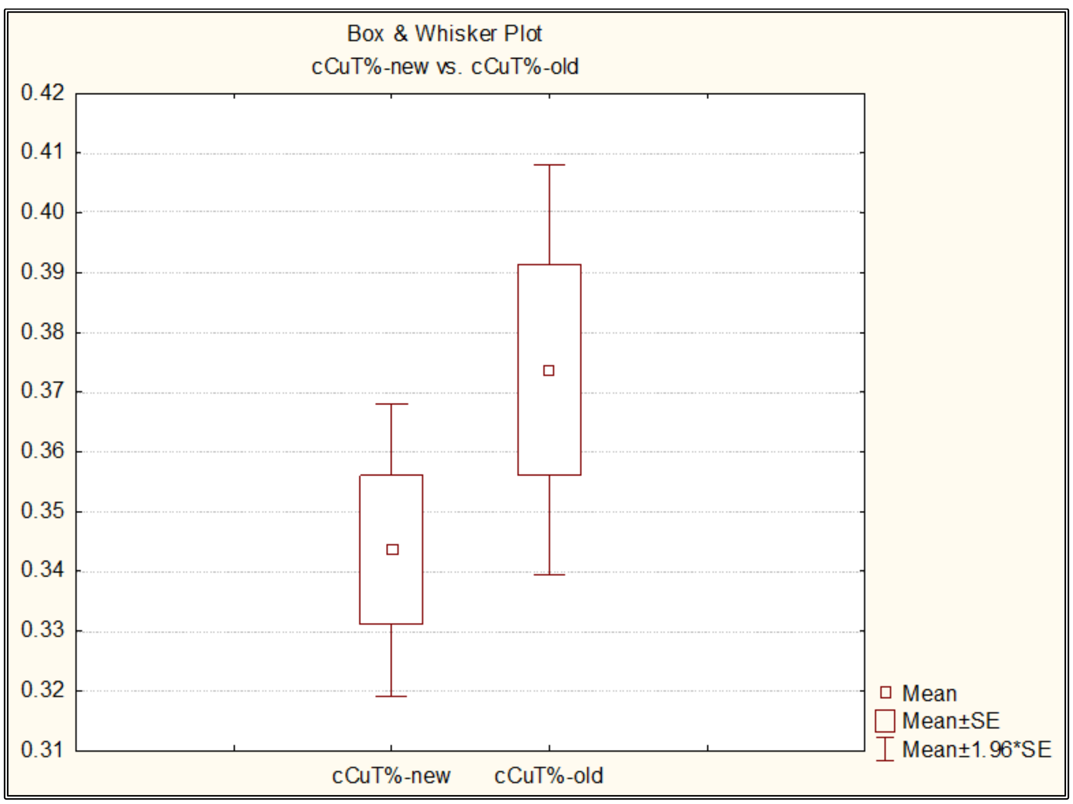
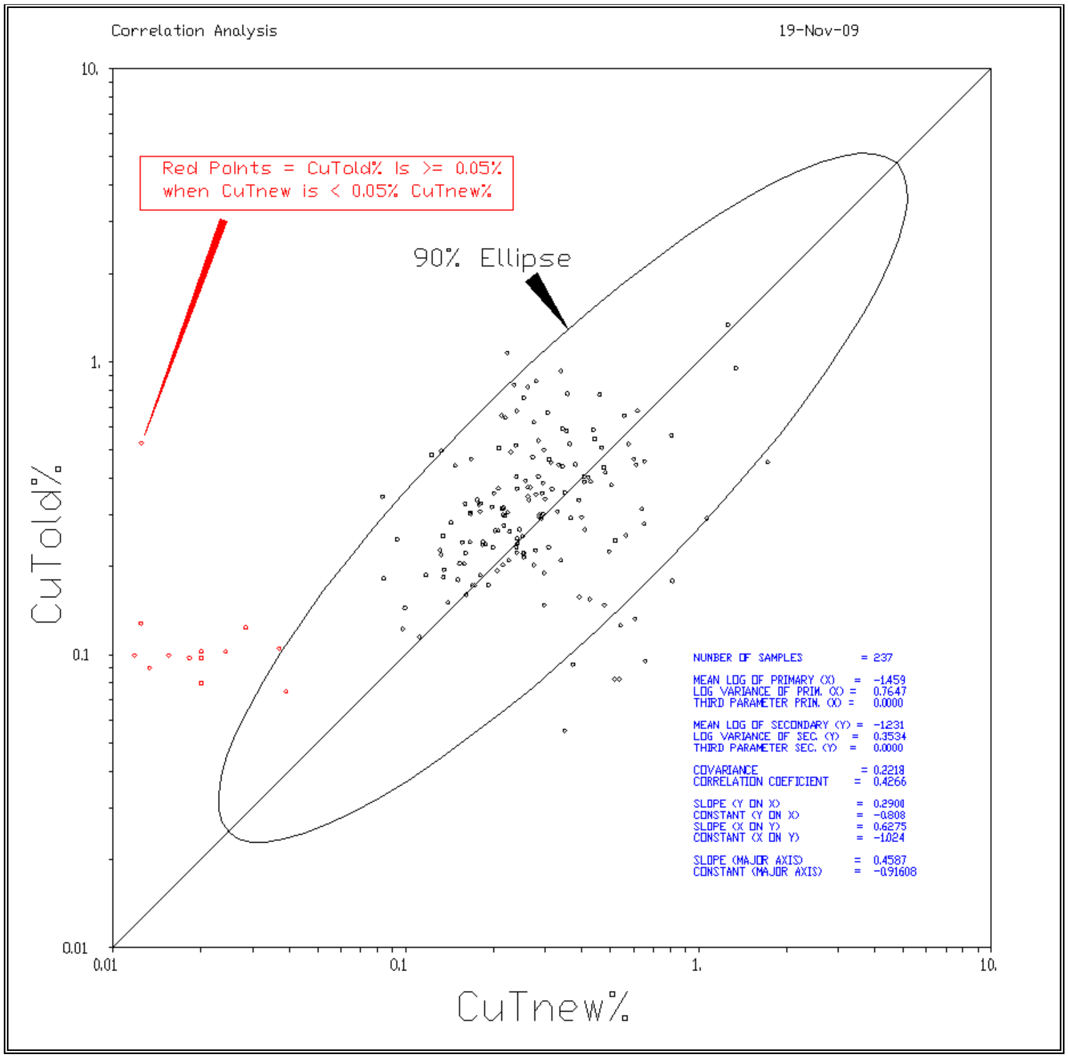


Table 12-2: T-Test of %TCu Old vs. %TCu New

Group 1 vs. Group 2	Mean Group 1	Mean Group 2	t-value	df	p	Valid N Group 1	Valid N Group 2	Std. Dev. Group 1	Std. Dev. Group 2	F ratio Variances
cCuT%-new vs. cCuT%-old	0.343571	0.373729	-1.40540	720	0.160335	363	359	0.236931	0.332265	1.966641

Figure 12-12: Correlation of %TCu-old vs. %TCu-new



go as low, with values around 0.1%. Samples assayed from new twin drillholes may have been analyzed with techniques that report lower copper grade values than the older methods. If this is a laboratory reporting issue, it may partially explain the apparent higher grade of old versus new.

12.2.5 Conclusion of Twinning Study

The weight of evidence supports the conclusion that the new data is sampling the same deposit as the old data. The first two tests contribute the strongest evidence, demonstrating that the old and new assay data populations have the same distributional shape, and their means are statistically the same. This in turn supports using all old and new data without adjustments or other qualifications.

The observation that there is a low correlation between the old and new data when compared side-by-side does suggest that the understanding of local grades is poor. This suggests that further geologic interpretation and higher density of sampling may be required.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Test Work

Several phases of metallurgical testing have been completed on the Zonia deposit. Initial studies were performed by Arimetco Incorporated (Arimetco) in 1995 followed by Constellation Copper Corporation (Constellation) in 2008. Redstone Resources Corporation (Redstone) conducted the most recent investigations on fresh core samples drilled in 2009 and 2010 along with trench samples taken from the deposit.

GRE has reviewed all of the available metallurgical studies related to the Zonia property including those undertaken by Leach Inc. (1995) and Metcon Research (2008) (2011) as documented in the following reports:

- *Zonia Project – Column Leach Tests*, Prepared for Arimetco Inc., Leach Inc., March 1995
- *Zonia Project – Column Leach Study on Surface Bulk Samples*, Prepared for Constellation Copper Corporation, Metcon Research, May 2008
- *Locked Cycle Column Leach Testing on Composite Samples*, Prepared for Redstone Resources Corporation, METCON Research, April 2011

The results of these metallurgical studies are presented in the following sections.

13.1.1 Arimetco Test Work (1995)

A series of five column leach tests were designed to evaluate the impacts of acid curing, particle size, solution acid concentration and lift height on copper extraction from two ore types. The test work was conducted at Mountain States R & D International (MSRDI) under the supervision of Leach, Inc. in 1995.

Two samples were collected by the Arimetco and delivered to MSRDI. Both samples are referred to as ROM but no additional details on their origin are available. The samples were crushed, blended, and split into test charges. The head assays of the two samples are shown in Table 13-1.

Table 13-1 Sequential Copper Assays – (Arimetco 1995)

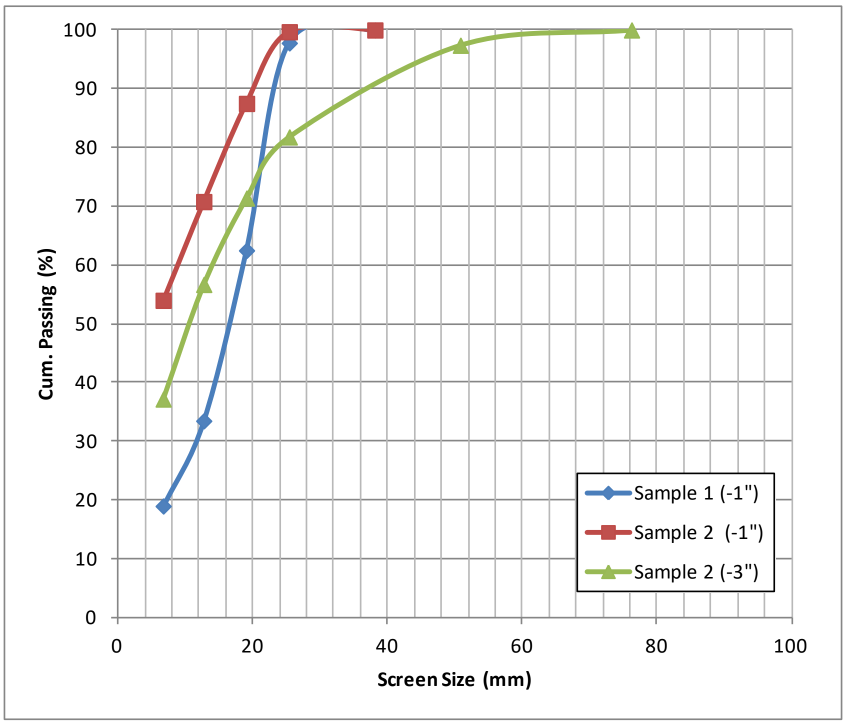
Sample	TCu (%)	AsCu (%)	CNCu (%)	CuRes (%)	Acid Sol (%)
1	0.243	0.186	0.01	0.047	80.7
2	0.330	0.204	0.01	0.116	64.8

These samples exhibited significantly different copper grades and acid soluble content. Total acid soluble content is estimated as the sum of the direct acid soluble copper (ASCu) and the cyanide soluble copper (CNCu). In Sample 1, approximately 80% of the copper is acid soluble, while in Sample 2, only 65% appears acid soluble.

Two tests were conducted on Sample 1 at the minus 25-millimeter (mm) crush to evaluate the effect of leach solution acid concentration. The first test, SK-1, was irrigated with solution maintained at a sulfuric acid concentration of 10 gpl and test SK-2 was irrigated with an acid concentration of 20 gpl. After 60 days of leaching, copper extractions for SK-1 and SK-2 were 62.1% and 64.2%, respectively. The higher acid concentration in the leach solution provided a slight increase in the copper extraction for the 60-day period with a slightly higher acid consumption on a kilogram of acid per kilogram of copper extracted basis.

Three column leach tests were conducted on Sample 2, two at a crush size of minus 25 mm and one test at a minus 76-mm crush. These tests were all operated at an initial acid concentration of 20 gpl. Tests SK-3 and SK-4 were designed to evaluate the impact of the acid cure, and test SK-5 was designed to evaluate copper extraction at a 76-mm crush size in a large column format (500-mm diameter by 5.8 meters [m] high). After 60 days of leaching, test SK-3 (acid cure) resulted in 76.8% copper extraction, and test SK-4 (no acid cure) resulted in 72.4% copper extraction. Test SK-5, at a minus 76-mm crush, resulted in a 58% extraction after 60 days, increasing to 68.1% after 143 days. Figure 13-1 shows the screen analysis of the three crushed samples.

Figure 13-1: Arimetco Sample Screen Analysis (1995)



The operating conditions and results of the Arimetco test work are summarized in Table 13-2.

The Arimetco test work conducted by MSRDI appears to be competently performed and can be summarized as follows:

- The origin of the samples utilized in this test work are not known and, therefore, the results of the work cannot be used for metallurgical evaluation beyond the basic conclusions that the results were generally good and a large format column appears to have been free from percolation issues.
- There is no indication in the report regarding how well these two samples represent the overall mineralization.

Table 13-2: Test Work Results (Arimetco 1995)

			Crush					Acid Cons (60 day)			Cu Extraction			Acid Cons (60 day)		Acid/Cu (kg/kg)	
Test	Column Dia (mm)	Height (m)	Size (mm)	P80	Assay (% Cu)	Acid Cure (kg/t)	Sol Acid (gpl)	Gross kg/t	Net kg/t*	Irrigation (lph/m^2)	60-day (%)	Ultimate (%)	Duration (days)	Gross kg/t	Net kg/t*	60-day	Ultimate
SK-1	203.2	3.4	25	22.2	0.243	No	10	15.1	12.8	9.1	62.1	75.4	143	26.4	23.6	4.1	5.9
SK-2	203.2	3.1	25	22.2	0.243	No	20	18.5	16.1	10.1	64.2	64.4	63	18.6	16.2	4.9	4.9
SK-3	203.2	2.9	25	16.5	0.330	20.8	20	26.0	21.7	8.9	76.8	76.9	64	26.2	22.3	4.2	4.2
SK-4	203.2	3.1	25	16.5	0.330	No	20	13.7	10.0	10.1	72.4	72.5	63	13.8	10.1	2.4	2.4
SK-5	203.2	5.9	76	22.9	0.330	No	20	8.3	5.3	12.0	58	68.1	142	13.6	10.2	1.8	2.5

* - net acid consumption considers acid potentially returned by solvent extraction

- These are preliminary tests that tend to show a benefit of finer crushing and acid curing with little benefit shown with the use of a higher acid concentration in the leach solution.
- The impact of crush size on copper extraction is complicated by the fact that the two samples, although crushed to the same target size, had widely different P_{80} sizes, 22.2 mm vs 16.5 mm, Sample 1 and Sample 2, respectively. Further, the minus 76-mm sample was almost the same P_{80} size as the minus 25-mm sample (22.8 mm). Additionally, when leach times were extended, the impact of crush size difference was reduced.

13.1.2 Constellation Test Work (2008)

Constellation Copper undertook a metallurgical program at METCON in Tucson, Arizona in 2007 to further evaluate the Zonia project. This work included additionalsampling, column testing on surface composites, and bottle roll bulk leach testing on drillhole samples. The primary objective of the program was to evaluate the leaching characteristics of three surface composites (A, B, D). Additionally, eleven reverse circulation (RC) drillhole samples were supplied for bulk leach testing. The test program consisted of agitated leaching, static leaching, and column leach tests.

A bulk surface sampling campaign of Zonia mineralization was conducted in July of 2007 to obtain samples for assay and subsequent metallurgical testing. The samples were taken from four trenches that cut approximately perpendicular to the strike direction of the deposit from within the historical open pit boundaries. Trenching involved a bulldozer to clear alluvium in a 6-meter wide pit of approximately 0.3 meters depth at each of the selected trench locations. Mineralized material removed from each trench was placed in conical piles spaced roughly 8 meters along the length of each trench. Piles that were estimated to contain mineralization greater than 0.1 percent copper were sampled from bottom to top with the backhoe and placed in drums (approximately 1/8 of each selected pile). Metallurgical composites noted as A, B, C, and D were obtained from this sampling campaign. Composite C was not submitted for metallurgical testing presumably due to its anticipated lower grade (roughly 0.15% Cu). The head analyses for the surface test samples are summarized in Table 13-3.

Table 13-3: Trench Sequential Copper Assays (Constellation 2008)

	Total		Copper Sequential Analysis		
Test Composite	Cu (%)	Fe (%)	ASCu (%)	CNCu (%)	CuRes (%)
Trench A	0.37	3.16	0.29	<0.01	0.05
Trench B	0.26	2.37	0.17	<0.01	0.06
Trench D	0.76	2.63	0.61	<0.01	0.08

Preliminary bottle roll bulk leach tests were conducted to define the acid cure dosages to be used in the subsequent column leach tests. METCON also conducted column leach tests on the surface composites. The primary objective of this portion of the test program was to generate copper extraction and acid consumption data at two different crush sizes (80% passing 19 mm and 9.5 mm). The column tests were conducted in nominally 200-mm diameter by 2-meter high columns under the following conditions:

Table 13-4 shows the copper extraction and acid consumption for these initial tests.

METCON also conducted column leach tests on the surface composites. The primary objective of this portion of the test program was to generate copper extraction and acid consumption data at two different crush sizes (80% passing 19 mm and 9.5 mm). The column tests were conducted in nominally 200-mm diameter by 2-meter high columns under the following conditions:

Table 13-4: 48 Hour Static Leach Tests – Pulverized Sample (Constellation 2008)

Sample ID	Test Number	Head Calculated		Extraction		Acid Consumption		
		Cu (%)	Fe (%)	Cu (%)	Fe (%)	Total (kg/t)	Net (kg/t) *	Net (kg/kg Cu)
Trench A	BR-30	0.37	3.20	68.72	5.82	24.78	20.85	8.19
Trench B	BR-31	0.26	2.37	70.83	11.38	25.04	22.22	12.15
Trench C	BR-32	.076	3.55	71.70	6.58	30.20	21.78	3.99

* - net acid consumption considers acid potentially returned by solvent extraction

- Acid cure dosage at 12 to 15 kg per ton (50% of bottle roll acid consumption)
- Leach solution was mature raffinate from Silver Bell mine adjusted to 10 gpl H_2SO_4
- Leach solution application rate: 12 lph/m^2
- Test duration 60 days

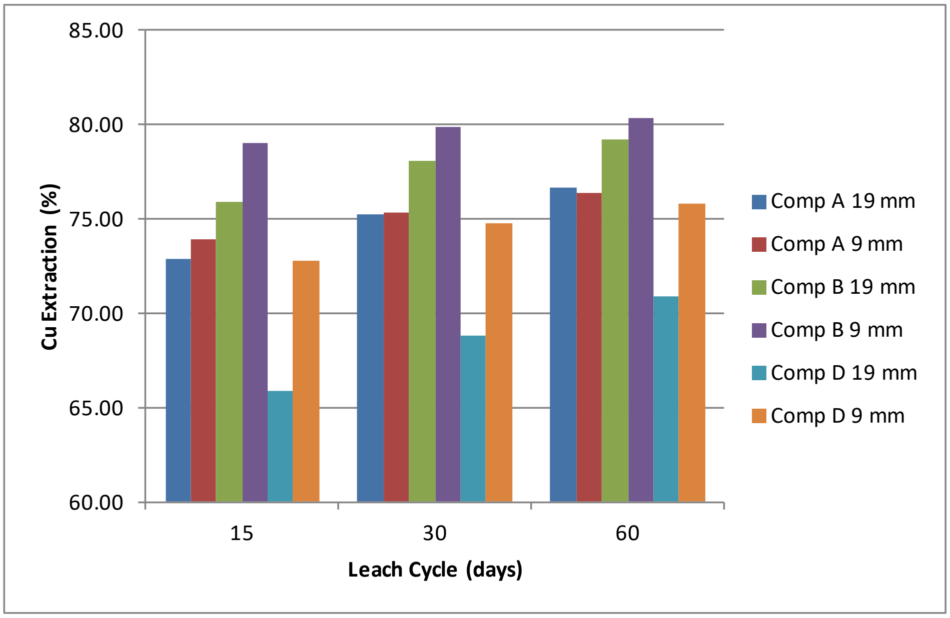
The results of these column tests are summarized in Table 13-5. Copper extractions in the column tests ranged from 71 to 80%, with net acid consumptions ranging from approximately 13 to 16 kg/ton. Copper extractions for Composites A and B were mostly independent of the crush size for a 60-day leach period. Composite D achieved a 7% increase in copper extraction at the finer crush size in the 60-day leach. In most cases, close to ultimate copper extractions were achieved within 30 days. Figure 13-2 shows the copper extractions for each of the composites at three time intervals (15, 30, and 60 days).

Table 13-5: 60 Day Column Leach Tests – Surface Composites (Constellation 2008)

Sample ID	Test Number	Crush Size (P80)	Head				Extraction				Acid Consumption		
			Assays		Calculated								
			Cu (%)	Fe (%)	Cu (%)	Fe (%)	Cu (%)	Fe (%)	Cu (kg/t)	Fe (kg/t)	Gross (kg/t)	Net (kg/t)*	Net (kg/kg Cu)*
Composite A	CL-01	19.1	0.35	3.14	0.40	3.12	76.67	8.59	3.07	2.68	17.55	12.80	4.17
	CL-02	9.5	0.35	3.17	0.39	3.28	76.38	6.00	2.99	1.97	18.05	13.42	4.48
Composite B	CL-03	19.1	0.25	2.25	0.28	2.32	79.21	7.66	2.25	1.78	17.82	14.34	6.38
	CL-04	9.5	0.25	2.22	0.28	2.28	80.34	9.47	2.25	2.16	18.45	14.96	6.65
Composite D	CL-05	19.1	0.71	3.61	0.79	3.81	70.93	7.71	5.60	2.94	24.56	15.91	2.84
	CL-06	9.5	0.71	3.68	0.80	3.66	75.86	6.92	6.05	2.53	25.07	15.71	2.60

* - net acid consumption considers acid potentially returned by solvent extraction

Figure 13-2: Surface Sample Composite Column Tests



Source: Constellation 2008

Bottle roll tests were conducted on drillhole interval assay pulp samples (crushed to 100 micron [µm]) preserved from an earlier drilling program conducted by Equatorial. These tests were designed to evaluate the trend in copper extraction with depth within the Zonia deposit.

Sequential copper analyses for ASCu, CNCu, and residual copper (CuRes) were analyzed for both the head samples and the leach residues for each test. A summary of the bottle roll test results and predicted extractions (based on sequential analysis) are provided in Table 13-6.

There is a poor correlation between the total copper grade and the copper extraction but there is a trend that indicates that a high residual copper grade (primary copper sulfides) results in lower overall copper extraction. Figure 13-3 shows the copper extraction for each of the tests versus the total copper grade (TCu), the acid soluble copper grade (ASCu) and the residual copper grade (CuRes).

The Constellation test work conducted by METCON appears to be competently performed and can be summarized as follows:

- Good copper extractions were achieved on the surface trench samples; ranging from 71% to 80% in a 60-day column leach test.
- Reducing the P_{80} size from 19 mm to 9.5 mm had little impact on the 60-day column leach copper extraction for the surface composites A and B and resulted in a 7% increase for composite D.
- Net acid consumption averaged 14.5 kg/t in the column tests.
- Acid consumptions from bottle roll and static leach tests are generally overstated when compared to column tests and not typically employed directly for acid consumption estimates.
- A high residual copper grade (sulfides) tends to reduce the overall copper extraction.

13.1.3 Redstone Test Work (2011)

The primary objective of the most recent test work conducted by METCON Research for Redstone was to obtain metallurgical data that would more accurately represent extractable copper by mineralization type, depth, and locations within the deposit.

In August 2010, METCON Research received drill core samples and ROM samples from the Zonia project to use for column leach testing. The samples received were identified as:

- Master composite
  - Hole 2009-04 (0-200 ft.)
  - Hole 2009-13 (0-200 ft.)
  - Hole 2009-21 (0-200 ft.)
  - Hole 2010-2 (0-500 ft.)
  - Hole 2010-12 (200-500 ft.)
  - Hole 2010-22 (400-1000 ft.)
- High copper: Hole 2009-30 (0-100 ft.)
- Average copper: Hole 2009-25 (0-200 ft.)
- Low grade copper: Hole 2010-13 (0-100, 200-300 ft.) and Hole 2010-17 (200-300 ft.).
- Intermediate Depth: Hole 2010-05 (300-600 ft.)
- Lower Depth: Hole 2010-15 (600-900 ft.)
- High secondary copper: Hole 2009-01 (100-200 ft.)
- Run of Mine

The head assays including sequential copper analysis are shown in Table 13-7.

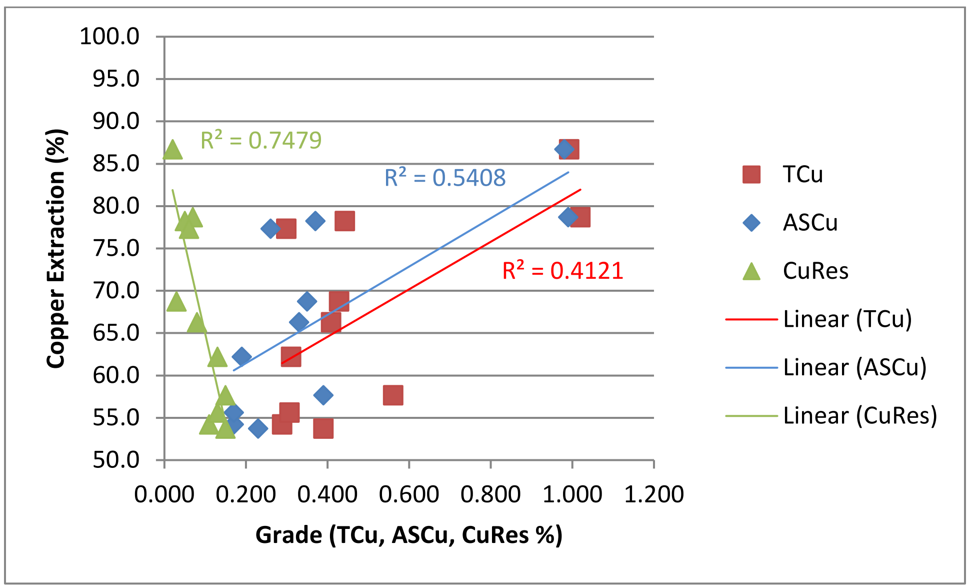
The estimated acid soluble copper is represented by the “Calc CuSOL” column. This column is the sum of the ASCu and CNCu grades divided by the TCu grade. It represents a rough estimate of the maximum extraction of copper achievable from a given sample. As expected, samples with higher proportion of CuRes copper tend to have a lower overall copper extraction potential.

Table 13-6: Drill Sample Bottle Leach Tests (Constellation 2008)

Test Number	Sample ID	Head Assays Cu (%)	Head Assays Fe (%)	Calculated Head Cu (%)	Calculated Head Fe (%)	Copper Sequential Analysis Results Head ASCu (%)	Head CNCu (%)	Head CURes (%)	Leach Residue ASCu (%)	Leach Residue CNCu (%)	Leach Residue CURes (%)	Extraction Cu (%)	Extraction Fe (%)	Acid Consumption Gross (kg/t)	Net (kg/t)*	Net (kg/kg Cu)*
BR-33	E 501 525-555	0.420	3.98	0.408	4.59	0.33	0.01	0.08	0.03	<0.01	0.10	66.3	7.2	68.51	64.33	23.77
BR-34	E-517 50-90	0.584	6.75	0.561	7.03	0.39	0.01	0.15	0.07	<0.01	0.16	57.7	7.4	67.87	62.88	19.45
BR-35	E-525 265-300	0.426	2.87	0.428	3.24	0.35	0.05	0.03	0.02	0.03	0.08	68.8	11.7	38.18	33.63	11.42
BR-36	E-527 20-50	0.396	2.97	0.390	3.39	0.23	<0.01	0.15	0.02	<0.01	0.16	53.7	6.5	37.12	33.89	16.18
BR-37	E-528 55-90	1.068	3.06	1.020	3.43	0.99	<0.01	0.07	0.04	<0.01	0.17	78.7	7.6	36.33	23.94	2.98
BR-43	E-528 125-180	1.043	3.80	0.992	4.32	0.98	0.01	0.02	0.03	<0.01	0.11	86.7	6.4	37.14	23.87	2.77
BR-38	E-529 150-195	0.436	4.20	0.442	4.92	0.37	<0.01	0.05	0.02	<0.01	0.08	78.2	6.1	49.3	43.96	12.72
BR-39	E-529 230-265	0.306	2.74	0.299	3.14	0.26	<0.01	0.06	0.01	<0.01	0.06	77.4	15.3	56.68	53.11	22.96
BR-40	E-529 290-2330	0.283	2.51	0.289	2.94	0.17	0.02	0.11	0.02	0.01	0.11	54.2	16.2	108.09	105.67	67.53
BR-41	E-530 245-275	0.302	3.71	0.310	4.40	0.19	<0.01	0.13	0.01	<0.01	0.10	62.2	15.5	71.57	68.60	35.55
BR-42	E-538 240-275	0.294	2.77	0.306	3.29	0.17	<0.01	0.13	0.02	<0.01	0.12	55.6	12.1	42.17	39.54	23.20

* - net acid consumption considers acid potentially returned by solvent extraction

Figure 13-3: Drillhole Sample Bottle Roll Test Grade Correlations



Source: Constellation 2008

Table 13-7: Sequential Copper Assays (Redstone 2011)

Sample ID	Assays						
	TCu (%)	TFe (%)	ASCu (%)	CNCu (%)	CuRES (%)	Calc TCu (%)	Calc CuSOL (%)
High Secondary Copper	0.380	2.520	0.128	0.164	0.073	0.365	80.0
High Copper	0.499	3.540	0.350	0.010	0.120	0.480	75.0
Average Copper	0.292	2.330	0.199	0.006	0.088	0.293	70.0
Low Grade Copper	0.120	2.260	0.064	0.003	0.056	0.123	54.0
Intermediate Depth	0.349	3.060	0.237	0.013	0.093	0.343	73.0
Lower Depth	0.401	3.040	0.206	0.060	0.074	0.340	78.0
Run of Mine	0.585	3.320	0.466	0.011	0.155	0.592	81.0
Master Compostie	0.483	2.740	0.358	0.018	0.081	0.457	82.0

13.1.3.1 Crusher Work Index

The crusher work index (Wi) was determined for the ROM sample to be 6.97 kilowatt-hours per ton (kw-hr/t). The abrasion indices (Ai) for the ROM material (Ai = 0.0529) and the master composite sample (Ai = 0.1015) indicate that material is moderately abrasive. Table 13-8 shows the crusher work index for the two samples.

Table 13-8: Crusher Indices (Redstone 2011)

Sample ID	Wi (kw-hr/t)	Ai
Run of Mine Composite	7.68	0.0529
Master Composite	NA	0.1015

13.1.3.2 Static Leach Tests

Static leach tests were conducted on the composite samples from the Zonia project. Static leach testing of both 10 days and 20 days were conducted to provide an indication of the acid consumption and copper extraction. These tests were run at nominal 25-mm crush size except for the ROM sample, which was conducted at a coarser size. The data for 20 days extraction is presented in Table 13-9. Lower copper extractions in these tests appear to have resulted from failure to maintain the leach pH. The copper extractions were significantly higher when tests were repeated and pH maintained below pH 1.5, as shown in Table 13-10. The 10-day tests were all conducted at a P80 of 1 inch. These tests provide an indication of the acid required for curing before column testing.

Table 13-9: Static 20-Day Leach Tests (Redstone 2011)

Composite ID	Crush Size (P80 mm)	Extraction		Acid Consumption			
		Cu (%)	Fe (%)	Total (kg/t)	Total (kg/kg Cu)	Net (kg/t)*	Net (kg/kg Cu)*
High Secondary Copper	25	27.29	0.86	3.49	3.58	1.98	2.04
High Copper	25	26.04	0.04	10.13	7.86	8.14	6.32
Average Copper	25	17.50	0.05	14.17	27.77	13.39	26.22
Lower Depth	25	18.75	0.61	13.60	21.04	12.61	19.50
Low Grade Copper	25	18.01	0.12	13.41	59.63	13.07	58.09

Composite ID	Crush Size (P80 mm)	Extraction		Acid Consumption			
		Cu (%)	Fe (%)	Total (kg/t)	Total (kg/kg Cu)	Net (kg/t)*	Net (kg/kg Cu)*
Intermediate Depth	25	19.32	0.08	12.3	17.56	11.22	16.01
Master Composite	50	20.38	0.46	6.52	14.26	5.81	12.72
Master Composite	25	24.16	0.48	9.29	17.20	8.46	15.65
Master Composite	12.5	29.48	0.83	11.32	16.07	10.23	14.52
Run of Mine	As Received	32.55	0.05	2.03	1.64	0.12	0.09
Run of Mine Composite	50	35.11	0.03	5.09	3.74	2.99	2.19
Run of Mine Composite	25	36.46	0.02	5.92	4.32	3.81	2.78
Run of Mine Composite	12.5	45.49	0.03	5.67	2.86	2.61	1.32

* - net acid consumption considers acid potentially returned by solvent extraction

Table 13-10: Redstone Static 10 Day Leach Tests (2011)

Composite ID	Extraction		Acid Consumption			
	Cu (%)	Fe (%)	Total (kg/t)	Total (kg/kg Cu)	Net (kg/t)	Net (kg/kg Cu)
High Copper	56.4	0.5	12.5	5.4	8.9	3.9
Average Copper	42.6	0.4	19.6	13.5	17.3	12.0
Lower Depth	36.7	1.9	17.9	12.5	15.7	11.0

Net acid consumption ranged from approximately 2 kg/t to 17 kg/t, with consumption increasing with leach time and finer crush sizing.

13.1.3.3 Column Leach Tests

The main objective of the column leach tests was to determine the impact of crush sizes on copper extraction and acid consumption. Three crush sizes of P_{80} passing 50 mm, 25 mm and 12.5 mm were examined. Ten locked-circuit column leach tests were conducted on the various samples. The samples were cured with approximately 60% of the static leach test acid consumption for a period of 5 days prior to application of the leach solution (raffinate from an existing operation) containing 5 g/L H_2SO_4 and 5 g/L Fe^{3+}. Table 13-11 shows the results of the locked cycle column tests.

Table 13-11: 90-Day Locked Cycle Column Tests (Redstone 2011)

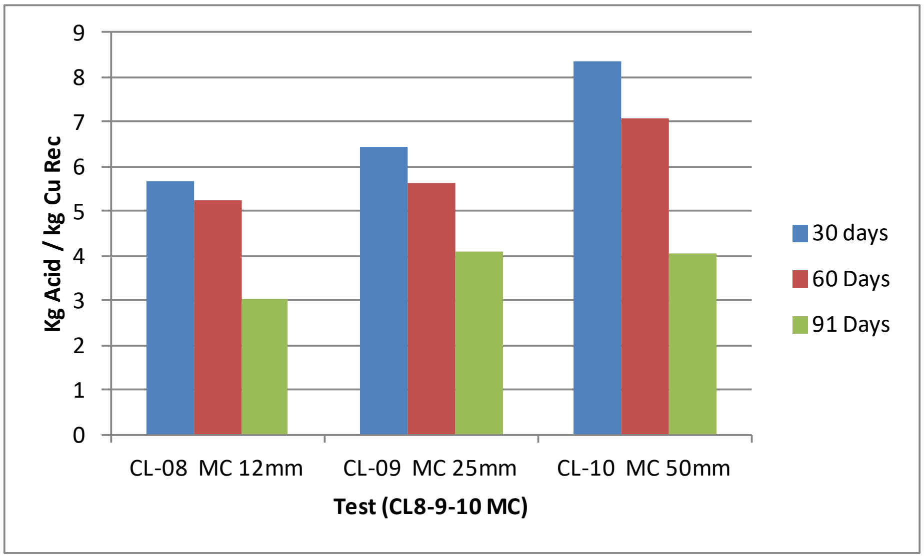
Test No.	Sample	Cure		Crush Size (P80 mm)	Irrigation Flow (l/hr/m²)	Leach Cycle (days)	Cu Extraction (%)	Acid Cons	
		Dosage (kg/t)	Time (days)					Net (kg/t)*	Net (kg/kg Cu)*
CL-01	High Secondary Copper	2.25	5	25	9.78	107	69.5	7.7	2.7
CL-02	High Copper	8.06	5	25	9.78	107	69.6	9.1	3.0
CL-03	Average Copper	12.64	5	25	9.78	107	63.5	16.6	7.9
CL-04	Lower Depth	11.6	5	25	9.78	107	54.0	17.9	9.8
CL-05	Low Grade Copper	8.68	5	25	9.78	107	47.6	14.2	23.1
CL-06	Intermediate Copper	7.94	5	25	9.78	107	58.8	14.5	7.1
CL-07	Run of Mine	3.29	5	50	9.78	105	67.2	7.6	1.9
CL-08	Master Composite	5.41	5	12	9.78	91	81.3	11.3	3.0
CL-09	Master Composite	5.41	5	25	9.78	91	77.8	14.7	4.1
CL-10	Master Composite	5.42	5	50	9.78	91	72.6	11.7	4.1

* - net acid consumption considers acid potentially returned by solvent extraction

The copper extractions ranged from 47.6% to 81.3%. The master composite sample, which was constructed to represent the majority of the deposit, achieved a copper extraction of 77.8% at a P_{80} of 25 mm. Percolation problems were not observed on any of the cycle column leach tests.

Figure 13-4 shows the acid consumption for the master composite sample in kilograms acid per kilogram copper extracted for the three crush sizes and three time periods.

Figure 13-4: Column Tests – Master Composite Net Acid Consumption



Source: Redstone 2011

In general, reducing the crush size or increasing the leach time results in a higher gross acid consumption. However, when the results are normalized to account for the copper extracted, the results are reversed. Finer crushing sizes tended to produce more copper while not increasing the acid consumption proportionally; similarly, longer leach times resulted in a reduction in the normalized acid consumption.

The Redstone test work conducted by METCON appears to be competently performed and can be summarized as follows:

- Good copper extractions were achieved from the majority of the samples, ranging from 59% to 81% in a 91-day column leach test (excluding high sulfide and low-grade samples).
- Reducing the P_{80} size from 50 mm to 12 mm improved the copper extraction in the master composite from 72.6% to 81.3%.
- Net acid consumption (kg acid/ kg Cu) averaged 7.6 to 17.9 kg/t in the column tests. With the master composite tests averaging 12.6 kg/t.
- The average extractable copper content in the composites is approximately 74% (CuSOL) and the master composite average was 80.3%. The column leach tests indicate that 60% to 95% of the leachable copper can be extracted at a nominal crush size of 25 mm.
- An overall copper extraction of 73% has been employed for the oxide materials and 70% for the transitional materials (ASCu and CNCu) and no credit has been given for copper sulfide in the process design.

13.1.4 Recommendations

A significant amount of test work has been conducted on the Zonia deposit. The results of the work are generally good, exhibiting relatively high copper extractions with moderate acid consumptions. The scope of the work was preliminary in nature and further work should be conducted in the following areas:

- Additional drillholes may be required to allow a better sample representation of the deposit to be developed. These samples would provide a higher degree of confidence for copper extraction across the entire deposit.
- The impact of utilizing a larger crush size should be evaluated. The original test work shows a trend of increased copper extraction with reduced crush size but that benefit is reduced if leach times are extended. The cost benefit analysis of coarser crush sizes should be investigated. Larger diameter drill core or surface trench sampling would need to be utilized to provide nominal 150-mm material.
- Large format column testing to ensure that permeability at the design lift height is maintained.
- Lock-cycle testing with SX to determine acid balance and SX parameters.
- SX/EW evaluation using PLS developed from test work.
- Additional crusher work index and abrasion index analysis across a larger proportion of samples.
- Confirmation of various mineralization type densities should be completed.
- Analysis of bioheap options for the lower elevations of the deposit where secondary copper sulfides predominate.

14.0 MINERAL RESOURCE ESTIMATE

An independent mineral resource estimate of the contained copper in the Zonia deposit was completed in 2017 (Tetra Tech, 2017). **The block model developed for the 2017 Resource Estimate was used to re-optimize pits for this 2018 Technical Report; the resource estimate has not otherwise been updated for this Technical Report. The following Section is taken from the 2017 Resource Estimate Technical Report** (Tetra Tech, 2017).

For the 2017 Resource Estimate, three-dimensional wireframes and model visualization was done with GemCom® software; geostatistics and resource estimation was done with MicroModel®; additional statistical analysis was done with Statistica® and Excel®. For this 2018 Technical Report, pit optimization was done with Techbase® software; model visualization and mine design were done with GEMS® software; additional analysis was done with Excel®.

14.1 Model Parameters

The block model is based on blocks 50 x 50 x 20 feet in dimension. The complete block model comprises 210 rows, 100 columns and 80 levels, with a total of 1,386,000 possible blocks. The block model is rotated 045.35° azimuth from true north. The block model parameters used in the Tetra Tech estimate are compiled in Table 14-1. Block model and project coordinates are in Arizona State Plane North American Datum 1983 feet (USSP NAD83). The block model has been rotated to align with a local grid established at the mine.

Two topographic surfaces are considered in this model: pre- and post-historical open pit. Potential mineralization has been estimated in 444,834 blocks below the present topographic surface.

Table 14-1: Zonia Model Parameters

```
CURRENT TIME  :  14-Mar-11 02:02 PM
PROJECT TITLE : Zonia 50x50x20 (2011 _ new UG data & GEMS origin)

SAMPLE    LABELS       COMPOSITE LABELS       GRADE     LABELS

 1         CuT%          1         cCuT%        1        kCuT%  [Cu Total 50x50x20          ]
 2         CuAS%         2         cCuAS%       2        kCuAS% [Cu Acid Soluble 50x50x20    ]
 3         CuCN%         3         cCuCN%       3        kCuCN% [cyanide soluble             ]
 4         CuAS/T        4         cAS/T        4        kAs/T  [proportion of AS to Total Cu]

ORIGIN IS LOCATED AT    478860.16  EAST    1200226.14  NORTH       3680.00  ELEVATION
ROTATION ANGLE FROM NORTH CLOCKWISE TO THE LEFT BOUNDARY IS :   45.38

NUMBER OF COLUMNS  :  100                                COLUMN DIMENSION   :    50.00  FEET
NUMBER OF ROWS     :  210                                ROW DIMENSION      :    50.00  FEET
NUMBER OF LEVELS   :   66                                LEVEL DIMENSION    :    20.00  FEET


NUMBER OF SAMPLE DRILLHOLES CURRENTLY ENTERED    :        603
NUMBER OF COMPOSITE DRILLHOLES CURRENTLY ENTERED :        603
NUMBER OF SAMPLE ASSAY VALUES                    :      28813
NUMBER OF COMPOSITE ASSAY VALUES                 :       8265
```

14.2 Database

The Zonia block model incorporates geologic and assay results from a long history of drilling on the Zonia property and the most recent drilling completed by Redstone in 2010. This study uses 603 drillholes, totaling 163,566.4 feet, with an average depth of approximately 271 feet per hole (Table 14-2). A large percentage of the holes were drilled vertical. The majority of 28,813 assays for %TCu analysis were done on five-foot assay intervals. Of the 28,813 samples assayed for %TCu, 26,085 had values greater than 0.0 %TCu, and, of these, nearly 20% were assayed for %AsCu and 13% for cyanide leach copper (%CNCu).

Table 14-2: Zonia Drillhole and Assay Statistics

	EASTING	NORTHING	ELEVATION	AZIMUTH	DIP	DEPTH
MINIMUM	480939.2	1198953.6	4283.0	0.0	-45.0	0.0
MAXIMUM	491632.9	1208073.6	4794.0	325.2	90.0	1528.0
AVERAGE	483875.9	1201973.3	4593.7	77.0	75.2	271.3
RANGE	10693.7	9120.0	511.0	325.2	135.0	1528.0
TOTAL COUNT	603					
TOTAL LENGTH	163566.4					

DH CLASS LIMITED BY
1 = Old dhf 2 = Old RCC 3 = New RCC 4 = Reject RCC 5 = Old RCC09 6 = New RCC09
7 = 2010 Holes 8 = 2010Oct18 9 = RRC-10 10 = RRC-X 11 = underground 12 = 2011
13 = channel_UG

TOTAL DRILLHOLES = 603
AVERAGE VALUES OF SELECTED DATA

LABEL	NUMBER	AVERAGE	STD DEVIATION	MIN. VALUE	MAX. VALUE	# MISS.
CuT%	26085	0.26980	0.38632	0.00000	11.12000	2728
CuAS%	5163	0.21189	0.32664	0.00000	10.23000	23650
CuCN%	3389	0.01807	0.04318	0.00000	0.86000	25424
CuAS/T	3902	0.59335	0.34222	0.00000	8.00000	24911

The average grades of %TCu, %AsCu, and %CNCu are 0.27%, 0.21%, and 0.018%, respectively. The average ratio of %AsCu to %TCu is 0.59. The %AsCu to %TCu ratio shown in the table above applies to the entire assay database and is not subject to a grade cutoff or constrained by the block model and does not differentiate mineralization type or location above or below oxide sulfide boundary. In addition, %AsCu to %TCu was calculated only in instances where both %TCu and %AsCu were assayed from the same sample interval. Detailed metallurgical analysis of %AsCu to %TCu is available in Section 13.0. Due to the relatively low number of acid soluble assays, only total copper was analyzed spatially in this estimate.

14.3 Wireframe Solids and Drillhole Coding

Geologic interpretation of lithology and mineralized zones at various copper percent cutoffs was completed by Redstone geologic staff and checked by Tetra Tech geological staff. Interpretation was done in vertical section as the basis for three-dimensional wire-framing using GemCom®.

A numerical code system was established based on lithology and grade shell (Table 14-3). For a particular location, a total-value was established based on summation of the block codes. For instance, if a particular block was within a grade shell of 0 to 0.15 %TCu (code 100) and within a quartz sericite schist lithology (code 42), the codes were concatenated to produce a value of 142.

Table 14-3: Zonia Drillhole and Block Codes

Lithology Codes:	
20	Basalt
22	Chlorite Schist
24	Dacite Porphyry
26	Felsite
28	Gossan
30	Greenstone
32	Metasediment
34	Overburden
36	Precambrian Granite
38	Quartz Porphyry (not used)
40	Quartz Monzonite Porphyry
42	Quartz Sericite Schist
44	Quartz Sericite Talc
Grade Shell Zone Codes:	
100	TCu% 0.00 – 0.15
200	TCu% 0.15 – 0.30
300	TCu% 0.30 – 0.45
400	TCu% 0.45+
0	Previously Mined (above current topo)
9999	Outside of Wireframes
Mineral Type Codes:	
1000	Oxidized: 73% recovery for Lerchs-Grossman Optimization
2000	Mixed (Transition): 70% recovery for Lerchs-Grossman Optimization
3000	Sulfide (Primary): 0% recovery for Lerchs-Grossman optimization

14.4 Assay Statistics

Log transformed statistics and a histogram (Figure 14-1) analysis was performed for %TCu on 25,690 original five-foot assays with values noted to be above zero in the assay database. The data includes assays from inside the modeled zone, including rock mined from above the current topography. The histogram follows a lognormal distribution with an average of 0.274 %TCu. The coefficient of variability (CV) for this distribution is 1.42.

Log transformed statistics and histogram analysis (Figure 14-2) for %AsCu were completed on 4,879 assays representing approximately 20% of the total copper data. The statistical distribution is more complex than for %TCu, with a mean equal 0.224 and a CV of 1.48.

Figure 14-1: Total Copper Percent Histogram for Drillhole Assays

```
PROJECT TITLE : Zonia 50x50x20 (2011 _ new UG data & GEMS origin)
    DATA TYPE IS SAMPLE
    STATISTICS FOR LABEL : CuT%
```

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.
ALL	2728	395	0	25690	0.00500	11.120	0.27396	0.15039	0.38781	1.4156	-1.8246	1.1627	1.0783	0.2884	1.4827

ROCK TYPE	SAMPLE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	MEDIAN STATISTICS MINIMUM	5TH PERCENTILE	25TH PERCENTILE	50TH PERCENTILE	75TH PERCENTILE	95TH PERCENTILE	99TH PERCENTILE	MAXIMUM
ALL	2728	395	0	25690	0.00500	0.02000	0.09000	0.18000	0.32000	0.77000	2.0000	11.120

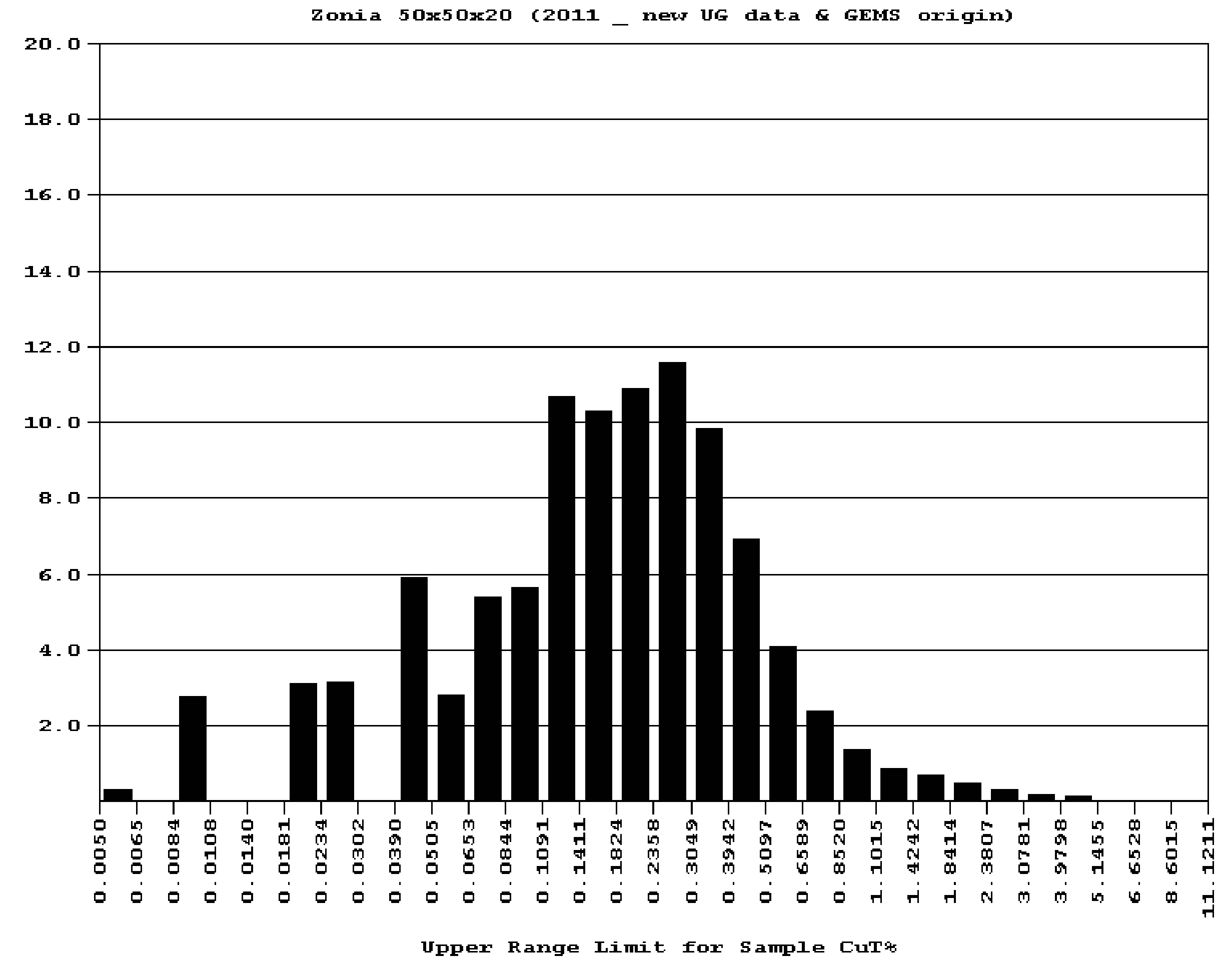
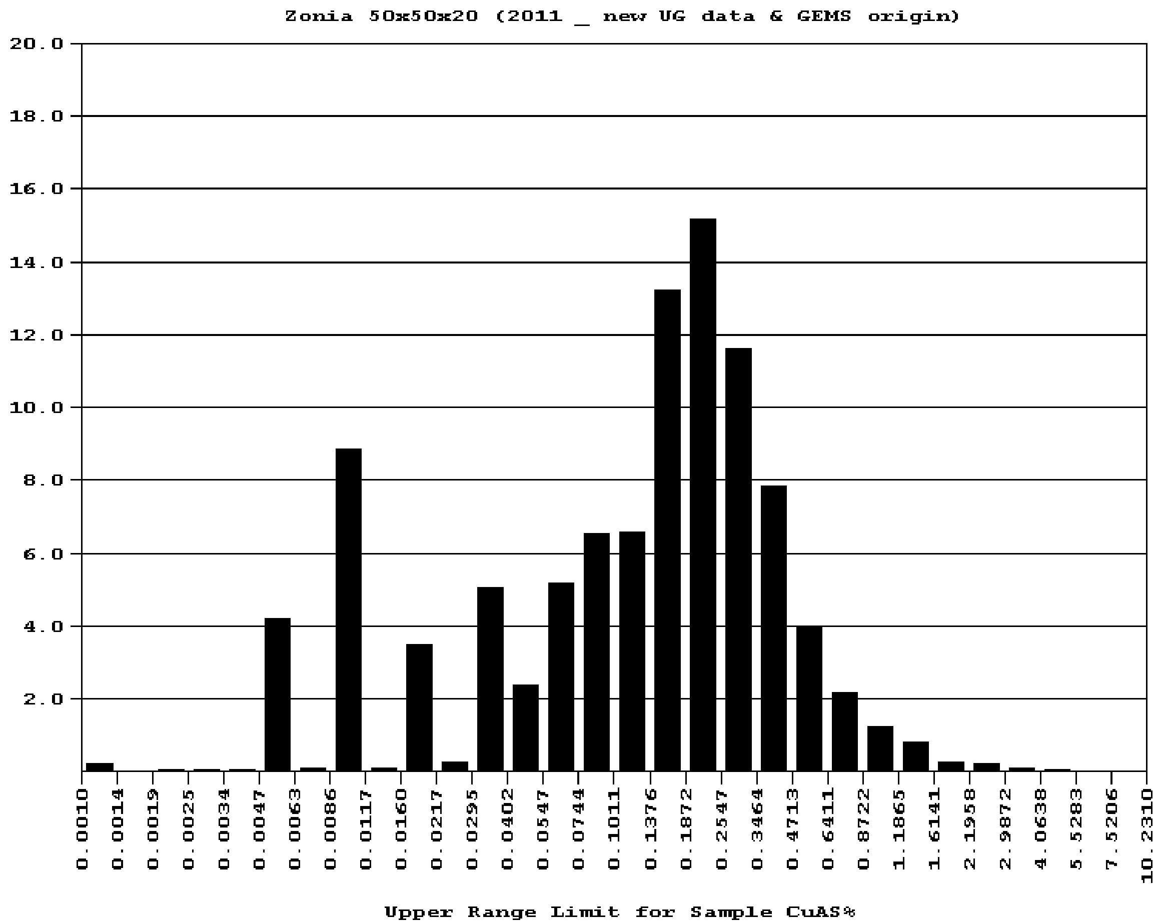


Figure 14-2: Acid Soluble Copper Percent Histogram for Drillhole Assays

PROJECT TITLE : Zonia 50x50x20 (2011 _ new UG data & GEMS origin)
DATA TYPE IS SAMPLE
STATISTICS FOR LABEL : CuAS%

	SAMPLE 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.
ALL	23650	284	0	4879	0.00100	10.230	0.22422	0.11014	0.33188	1.4801	-2.2035	1.9123	1.3828	0.2872	2.4017

	SAMPLE COUNT				MEDIAN STATISTICS							
ROCK TYPE	MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	MINIMUM	5TH PERCENTILE	25TH PERCENTILE	50TH PERCENTILE	75TH PERCENTILE	95TH PERCENTILE	99TH PERCENTILE	MAXIMUM
ALL	23650	284	0	4879	0.00100	0.01000	0.06000	0.16000	0.28000	0.64000	1.4222	10.230



14.5 Composite Statistics

Bench composites of 20 feet were calculated. The statistics (Table 14-4) for the 7,215 composites of %TCu indicate that an individual assay maximum of 11.12% is reduced by averaging over a 20-foot composite to a value of 9.51%. The mean of the bench composites is slightly reduced to 0.269 %TCu, and the CV is also reduced to 1.24. The mean copper grades contained for composites within all rock codes are listed in Table 14-4. The code 9999 is used to designate composite data that is under the current topography but outside of geologic modeling. Code 0 is composites above the current topography and below the historical pre-mining topography. These historical code 0 composites were used to estimate present day block values.

Table 14-4: Composited Total Cu for 20-Foot Composites Broken Out by Rock Code

PROJECT TITLE : Zonia 50x50x20 (2011 _ new UG data & GEMS origin)
DATA TYPE IS COMPOSITE
STATISTICS FOR LABEL : cCuT%

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	141	1	0	643	0.00270	2.6385	0.33772	0.10030	0.31670	0.9378	-1.4890	0.9672	0.9835	0.3659	1.2769
122	0	0	0	16	0.01000	0.07038	0.04414	0.000252	0.01589	0.3600	-3.2057	0.2138	0.4624	0.0451	0.4882
124	1	6	0	17	0.01641	0.41060	0.11499	0.01363	0.11674	1.0152	-2.5062	0.6158	0.7847	0.1110	0.9225
126	0	1	0	46	0.001000	0.29700	0.04514	0.00277	0.05260	1.1652	-3.5812	1.0585	1.0288	0.0473	1.3719
130	157	4	0	244	0.00196	0.35000	0.05997	0.00249	0.04985	0.8312	-3.1598	0.7904	0.8891	0.0630	1.0974
134	23	0	0	4	0.00500	0.02500	0.01272	0.000085	0.00921	0.7236	-4.5692	0.4154	0.6445	0.0128	0.7177
136	2	0	0	43	0.00150	0.14078	0.04637	0.00125	0.03536	0.7626	-3.4903	1.1150	1.0559	0.0532	1.4316
140	55	13	0	449	0.000500	1.2125	0.13503	0.02144	0.14644	1.0845	-2.4634	1.0947	1.0463	0.1472	1.4101
142	49	24	0	286	0.00250	0.63250	0.09901	0.01168	0.10806	1.0914	-2.8103	1.0706	1.0347	0.1028	1.3846
144	0	0	0	4	0.01770	0.10566	0.04925	0.00161	0.04015	0.8153	-3.2559	0.4830	0.6950	0.0491	0.7880
222	0	0	0	5	0.01000	0.06745	0.04341	0.000570	0.02387	0.5498	-3.3304	0.4960	0.7043	0.0458	0.8013
224	1	0	0	17	0.03718	0.37171	0.12177	0.00708	0.08416	0.6911	-2.2986	0.3768	0.6138	0.1212	0.6765
226	0	0	0	5	0.06099	0.16000	0.10186	0.00152	0.03904	0.3833	-2.3426	0.1172	0.3424	0.1019	0.3526
230	23	0	0	328	0.00500	1.0597	0.18065	0.01718	0.13107	0.7256	-2.0214	0.8049	0.8972	0.1981	1.1120
234	5	1	0	4	0.01000	0.13500	0.05891	0.00286	0.05351	0.9083	-3.2012	0.8618	0.9284	0.0626	1.1694
236	0	0	0	4	0.02900	0.04550	0.03962	0.000053	0.00728	0.1838	-3.2427	0.0307	0.1752	0.0397	0.1765
240	52	11	0	533	0.00597	0.80500	0.17231	0.01040	0.10197	0.5918	-1.9606	0.5081	0.7128	0.1815	0.8137
242	46	7	0	470	0.01000	1.1800	0.17513	0.01692	0.13008	0.7427	-2.0241	0.6645	0.8152	0.1842	0.9714
322	0	0	0	12	0.05500	0.56750	0.26611	0.02687	0.16392	0.6160	-1.5487	0.5281	0.7267	0.2768	0.8341
324	2	0	0	58	0.01527	1.6350	0.29692	0.07307	0.27031	0.9104	-1.5801	0.8943	0.9457	0.3221	1.2024
326	4	0	0	26	0.01422	0.53998	0.20275	0.01655	0.12865	0.6345	-1.8326	0.5925	0.7697	0.2152	0.8992
330	13	0	0	181	0.01459	1.0025	0.24153	0.02061	0.14355	0.5944	-1.6211	0.4900	0.7000	0.2526	0.7952
340	60	1	0	981	0.00500	1.5250	0.25972	0.03437	0.18538	0.7138	-1.5890	0.5697	0.7548	0.2714	0.8762
342	52	13	0	663	0.00400	1.7100	0.25620	0.03782	0.19448	0.7591	-1.6881	0.8785	0.9373	0.2868	1.1863
420	1	0	0	3	0.02075	0.24449	0.10300	0.01515	0.12308	1.1950	-2.8043	1.0668	1.0329	0.1032	1.3807
422	2	0	0	10	0.20200	1.5362	0.65762	0.21623	0.46501	0.7071	-0.6463	0.4599	0.6781	0.65943	0.7641
424	2	0	0	50	0.01308	0.94100	0.30652	0.07271	0.26964	0.8797	-1.7753	1.6219	1.2735	0.3812	2.0157
426	1	0	0	19	0.09912	2.1400	0.63440	0.23618	0.48599	0.7661	-0.7051	0.5317	0.7292	0.64452	0.8378
430	26	1	0	480	0.00700	9.5060	0.45764	0.40901	0.63954	1.3975	-1.2263	0.8782	0.9371	0.4551	1.1860
432	1	0	0	23	0.05760	2.2640	0.58269	0.27686	0.52618	0.9030	-0.9055	0.8126	0.9015	0.60705	1.1197
434	0	0	0	2	0.03439	0.13500	0.08469	0.00506	0.07114	0.8400	-2.6862	0.4675	0.6838	0.0861	0.7720
440	14	0	0	290	0.02073	3.4096	0.37072	0.10614	0.32578	0.8788	-1.2869	0.6629	0.8142	0.3846	0.9697
442	49	0	0	774	0.00500	5.0624	0.50809	0.33688	0.58041	1.1423	-1.0680	0.7636	0.8738	0.5035	1.0705
9999	174	11	0	525	0.00500	1.2681	0.21156	0.02731	0.16527	0.7812	-1.8522	0.7090	0.8420	0.2237	1.0159
ALL	956	94	0	7215	0.000500	9.5060	0.26857	0.11146	0.33386	1.2431	-1.7830	1.0801	1.0393	0.2885	1.3946

The distribution of the composited data for TCu Figure 14-3) is clearly more log-normal than the assay data. The composited acid soluble data (Figure 14-4) has a distribution that appears weakly bimodal.

Figure 14-3: Histogram for 20-Foot Composites for Total Cu

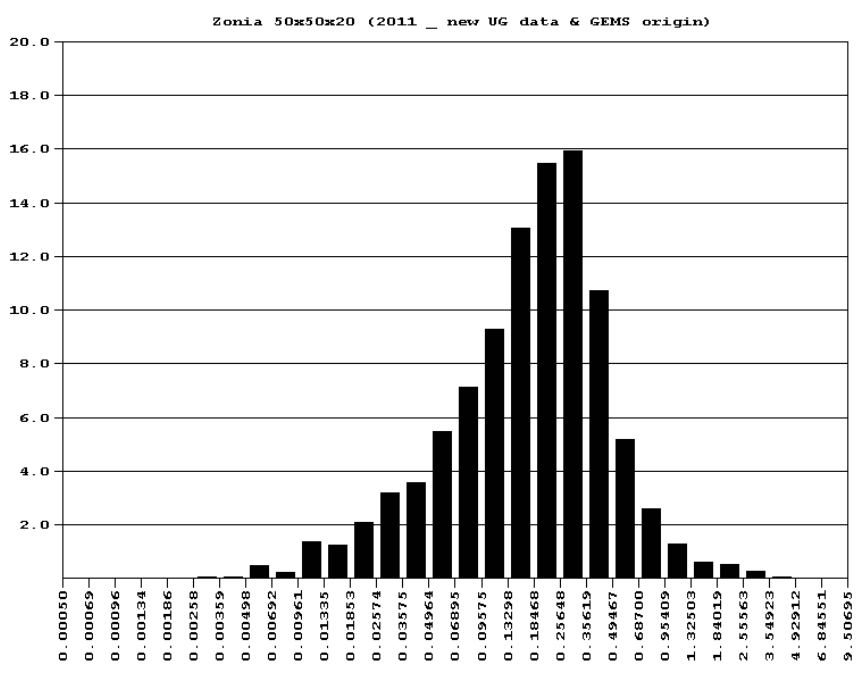
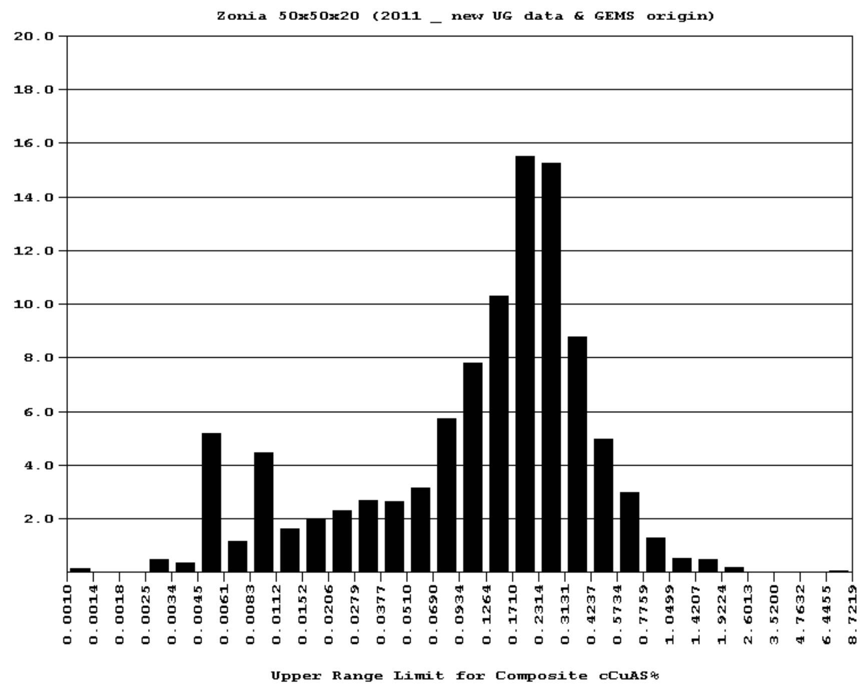


Figure 14-4: Hisogram for 20-Foot Composites for Acid Soluble Cu



The histogram for the ratio between TCu% and AsCu% is shown in Figure 14-5. The histogram is complex in shape can cannot be considered as normal or lognormal as it fails a Shapiro-Wilks normality test for both distributions.

A Box and Whiskers analysis (Figure 14-6) has been completed for log10 transformed %TCu within each of the detailed codes. On the graph, the x-axis indicates the code for the composite, and the y-axis indicates the mean and range of the log10 (L) copper grade.

Except for the 9999 and 0 codes, there is a general increase in the average concentration of copper as rock code increases. This increase in copper concentration is easier to see in Figure 14-7, where detailed rock codes are grouped by general rock type and consolidated within the individual grade envelopes.

14.6 Geostatistics

Tetra Tech completed geostatistical analysis on TCu using the simplified 0 and 100-400 coded data. The 9999 code data was excluded. The geostatistics of AsCu and the ratio of acid soluble to total copper was explored and it was determined that there is insufficient data to warrant further analysis.

A general relative variogram for the mineral zone 400 (+0.45% copper envelope) looking to the NE-SW is shown in Figure 14-8 and Figure 14-9. The variogram is modeled with a nugget and three nested spherical models. The model parameters are shown in Table 14-5 for the variography for each mineral zone. This variogram has its maximum range of the three spherical models of 600, 300, and 150 feet. Each of these models has an anisotropy ratio of 1:0.5:0.25 for the primary, secondary, and tertiary axes. The values for the nugget and sills change for each of the grade envelope zones. The ranges, however, appear fairly constant across each of the grade envelopes. Supporting this observation, Figure 14-10 shows the NE-SW directional variogram for the consolidated mineral zones with similar ranges for mineral zone 4.

Figure 14-5: Histogram of the Ratio of Acid Soluble Cu to Total Cu for 20-Foot Composites

PROJECT TITLE : Zonia 50x50x20 (2011 _ new UG data & GEMS origin)
DATA TYPE IS COMPOSITE
STATISTICS FOR LABEL : cAS/T

ROCK TYPE	COMPOSITE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR
ALL	7084	0	19	1162	0.	1.0000	0.56996	0.07149	0.26737	0.4691

ROCK TYPE	COMPOSITE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	MEDIAN STATISTICS MINIMUM	5TH PERCENTILE	25TH PERCENTILE	50TH PERCENTILE	75TH PERCENTILE	95TH PERCENTILE	99TH PERCENTILE	MAXIMUM
ALL	7084	0	19	1162	0.	0.13184	0.34790	0.60431	0.79520	0.94456	0.99438	1.0000

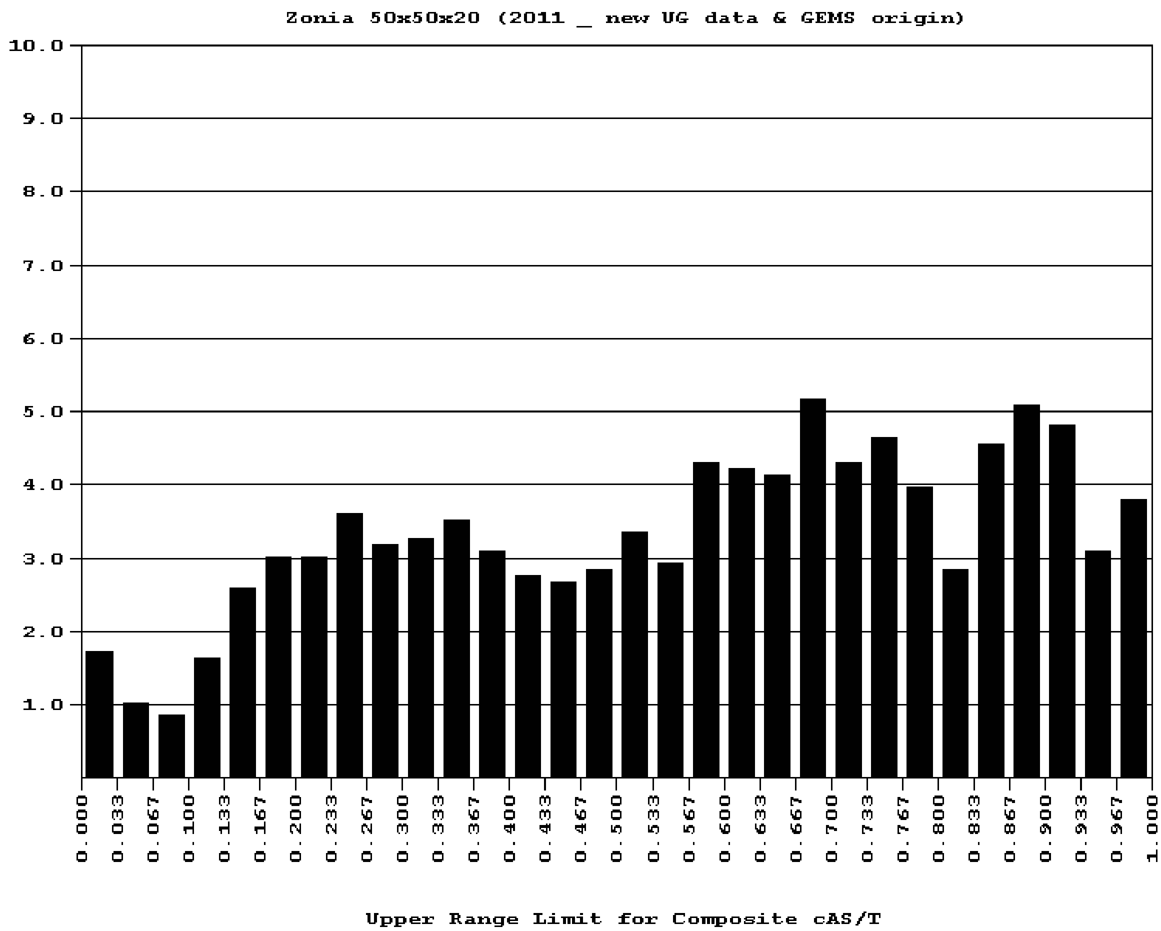


Figure 14-6: Box & Whisker Plot of Total Copper (LCu T%) Broken Out by Detailed Rock Code

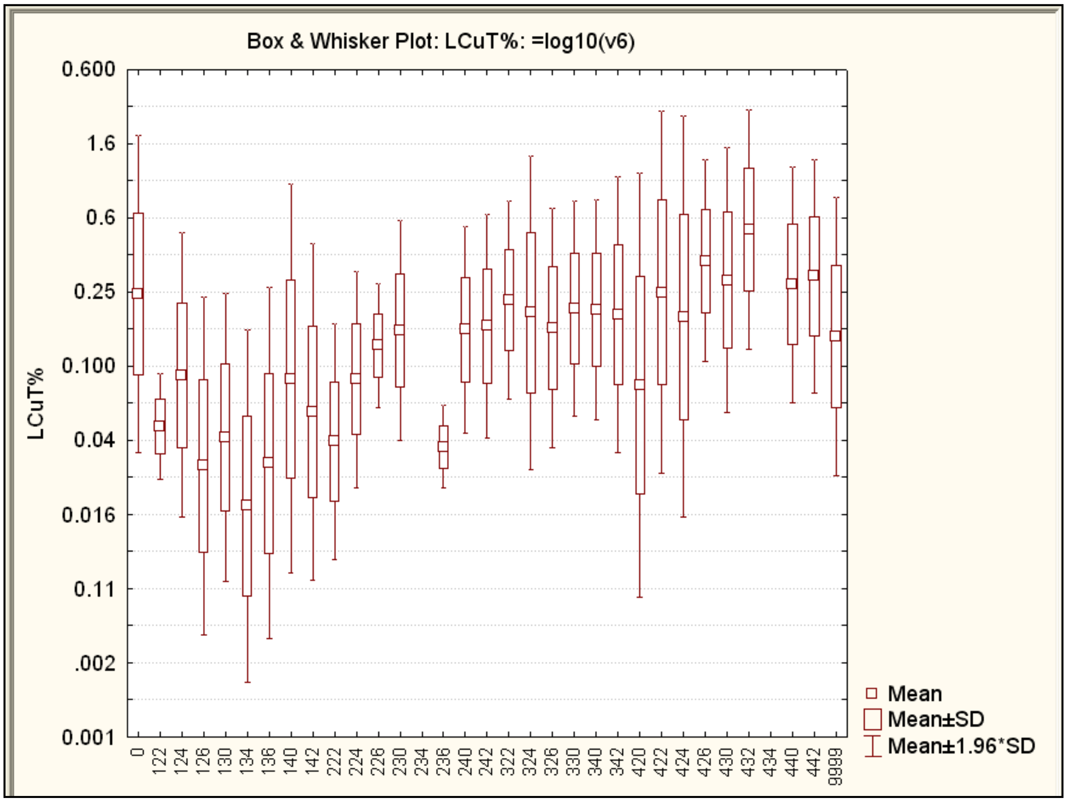


Figure 14-7: Box & Whisker Plot of LCu T% by Mineralized Grade Envelopes (MINZ)

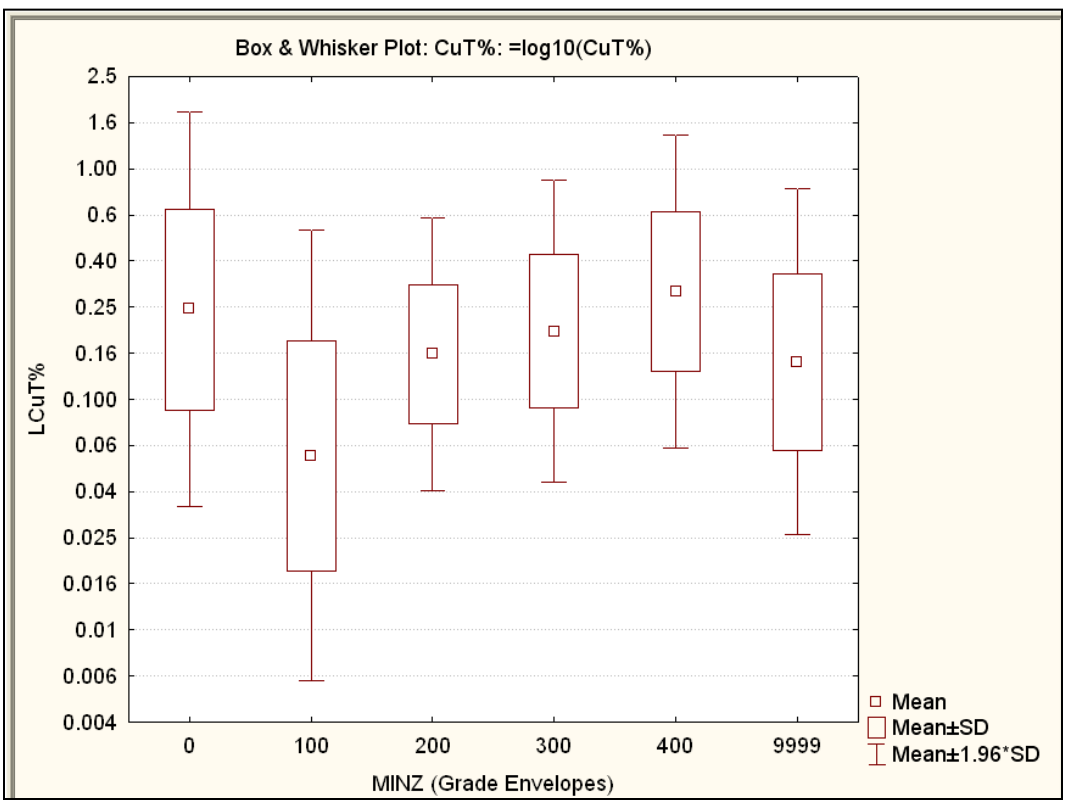


Figure 14-8: General Relative Variogram SE (Dip 45°) – (Cut% > 0.45)

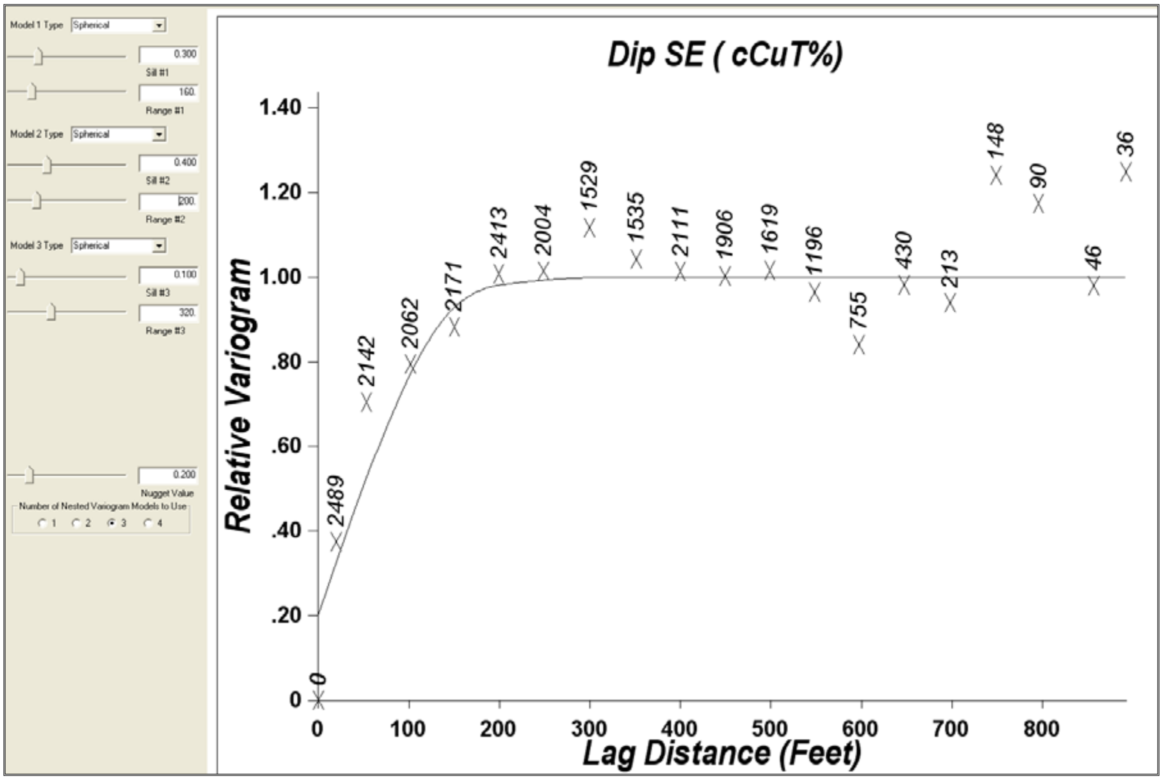


Figure 14-9: General Relative Variogram NE (Dip 45°) – (Cut% > 0.45)

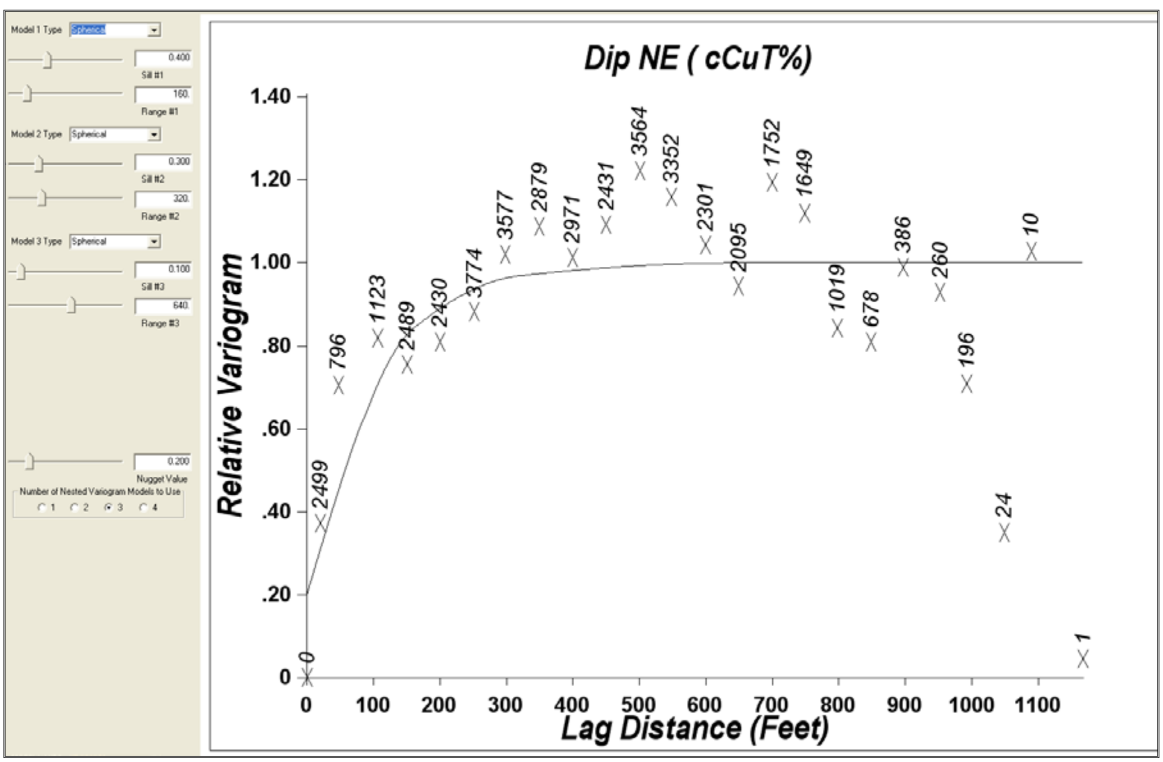


Table 14-5: Kriging Search and Variogram Parameters by Consolidated Zone

			Search Parameters (CuT%)						Variogram Parameters (CuT%)							
Composite Codes	Block Codes	Zone Name	Direction	Search Range	Direction	Preliminary Resource Code **	Times Indicated Range	#Points NearestSearch:Max/DH:Min Points	Nugget	Rotation	Range (1)	Sill (1)	Range (2)	Sill (2)	Range (3)	Sill (3)
A	A'	Within Mineralized Zone	Along Strike	320	Measured	1	0.50	16:99:8	0.20	45	60	0.30	350	0.14	600	0.07
			Along Dip	160	Indicated	3	1.00	8:99:8		0	30		175		300	
			Across Dip	80	Inferred	5	2.00	8:99:4		0	15		87.5		150	
B	B'	Within Mineralized Zone	Along Strike	320	Measured	1	0.50	16:99:8	0.20	45	60	0.30	350	0.14	600	0.07
			Along Dip	160	Indicated	3	1.00	8:99:8		0	30		175		300	
			Across Dip	80	Inferred	5	2.00	8:99:4		0	15		87.5		150	
C	C'	Within Mineralized Zone	Along Strike	320	Measured	1	0.50	16:99:8	0.20	45	70	0.10	175	0.14	600	0.10
			Along Dip	160	Indicated	3	1.00	8:99:8		0	35		87.5		300	
			Across Dip	80	Inferred	5	2.00	8:99:4		0	17.5		43.8		150	
D	D'	Within Mineralized Zone	Along Strike	320	Measured	1	0.50	16:99:8	0.20	45	140	0.30	300	0.40	600	0.30
			Along Dip	160	Indicated	3	1.00	8:99:8		0	70		150		300	
			Across Dip	80	Inferred	5	2.00	8:99:4		0	35		75		150	
Cmp	A	0,420,422,424,426,430,432,434,440,442				Blk	A'	420,422,424,426,430,432,434,440,442								
Cmp	B	0,320,322,324,326,330,432,334,340,342				Blk	B'	320,322,324,326,330,332,334,340,342								
Cmp	C	0,220,222,224,226,230,232,234,240,242				Blk	C'	220,222,224,226,230,232,234,240,242								
Cmp	D	0,120,122,124,426,130,132,134,140,142				Blk	D'	120,122,124,126,130,132,134,140,142								
	*	Unitize General Relative (All variogram structures are transformed to relative variograms from log variograms)														
	**	Kriging Error is used to adjust final resource class														

Figure 14-10: General Relative Variogram of Total Copper for NE (Dip 45°) All MINZ Combined

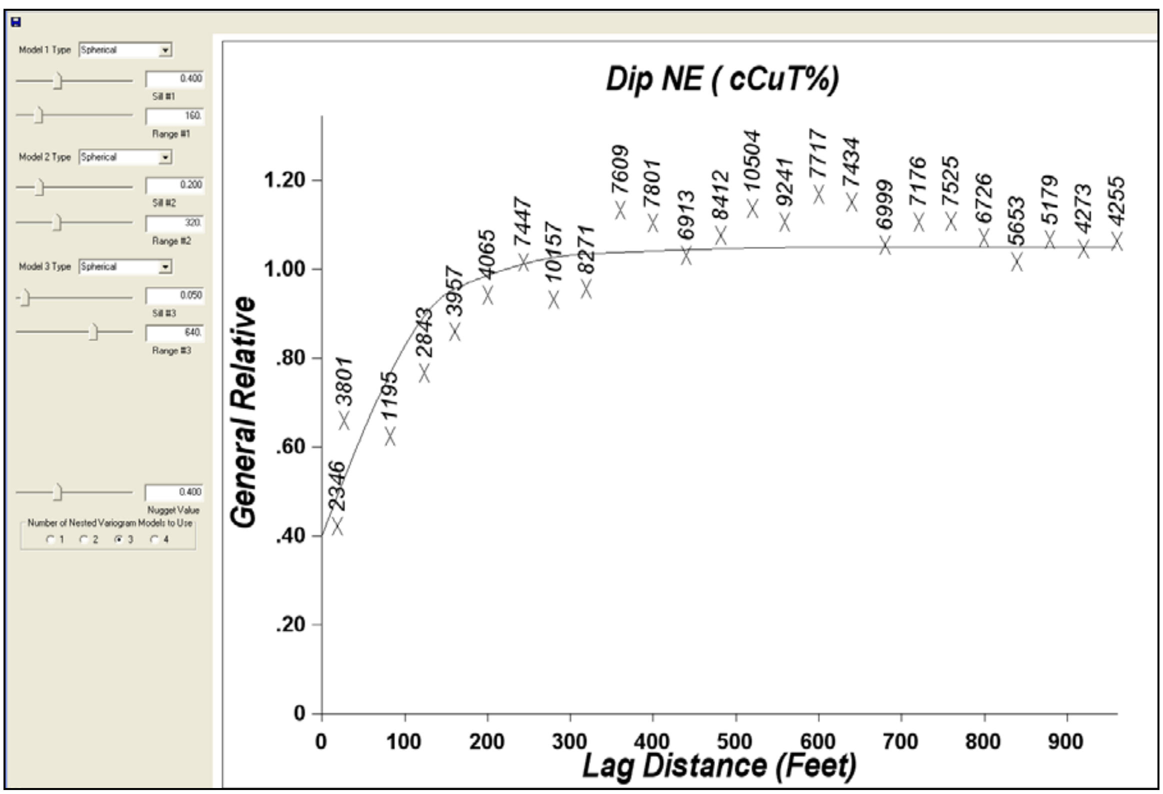


Figure 14-11 shows the results of using the mineral zone variogram model to estimate known composite values. This model validation technique is also called the jackknife method. In jackknifing, known values of %TCu are removed and then estimated using the variogram mode with kriging and the remaining data. The result is a table that lists estimates and original values where the estimate versus the original data can be shown as a difference. This difference ideally centers on zero; the data center is at -0.006 in the figure, is symmetric in distribution, and has a small variance. Figure 14-12 shows the original and estimated distributions plotted side-by-side. The conclusion is that the jackknife method confirms the selected variogram model parameters.

Figure 14-11: Histogram of Jackknifing (Model Validation) Error, Total Copper

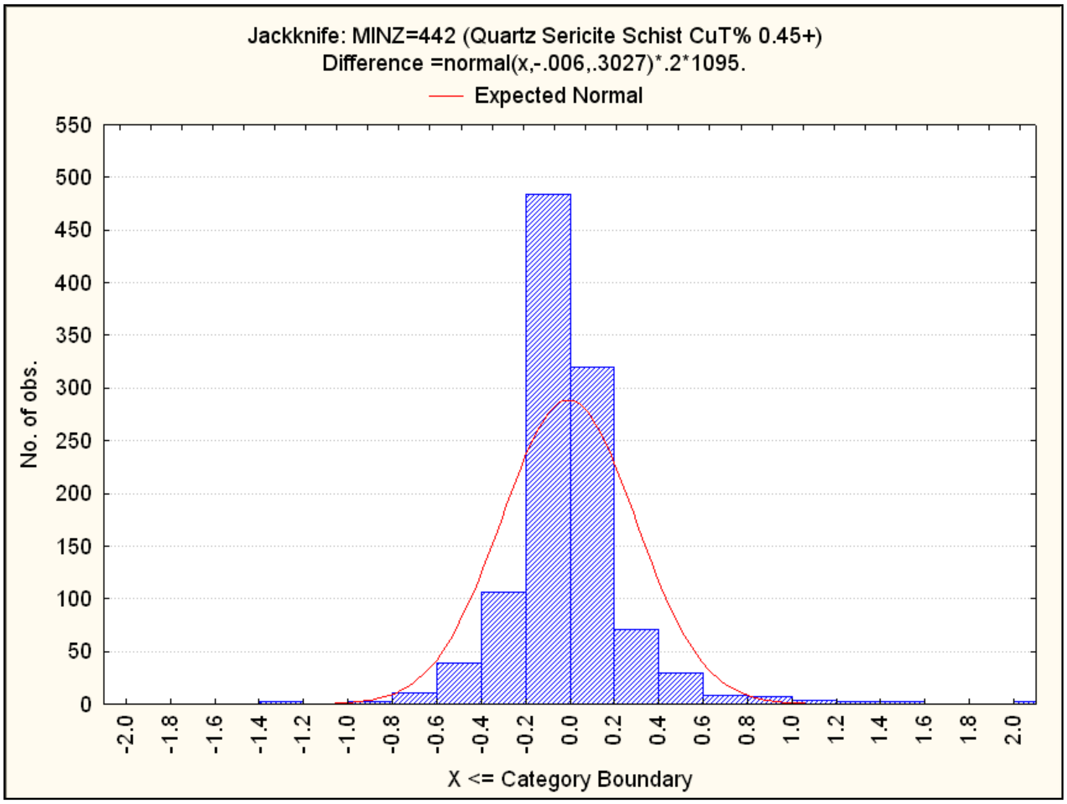
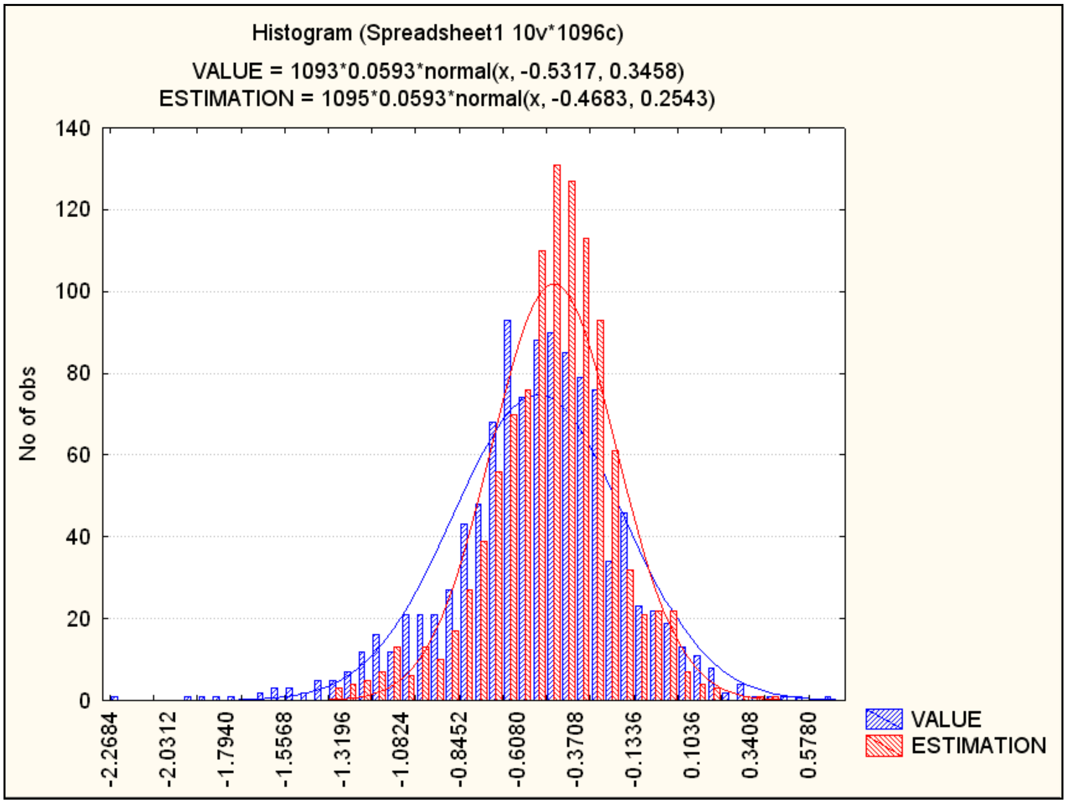


Figure 14-12: Side-by-Side Histograms of Jackknifed Total Copper (Estimation) and Original Composite Data (Value) – MINZ 3 and 4



Using kriging parameters shown in Table 14-5, a total copper block model was estimated. The statistics of this model were broken down using the detailed rock codes as noted in Table 14-6. Figure 14-13 shows the quantity of blocks and their corresponding grade intervals. Figure 14-14 through Figure 14-16 show the total copper block estimates at selected cross-sections.

The estimation was done in three passes using increasing search windows for blocks estimated as measured, indicated, and inferred. Each pass is given a preliminary resource code of 1, 3, and 5, respectively (Table 14-7). The selection of the various search ranges was done by using the Jackknife method. Figure 14-17 shows scatter plot of the original value plotted against the kriged value for the three passes. An estimate is better when the plotted points fall nearer the plotted 45-degree line, which also is reflected in a higher correlation. For this study, the measured search parameters (shown in Figure 14-17 as red dots) have a correlation coefficient of 0.71. The correlation for indicated classification is 0.6 and is 0.2 for inferred classification.

In addition to the three passes, each block's estimate also had a kriging error value. This error value represents a measure of the quality of each block's estimate. Review of the cumulative frequency plot of the kriging errors (Figure 14-18) indicates the estimates are less than acceptable when kriging errors are above 1.07. Any block that exceeded a kriging error of 1.07 was reclassified to a lower resource classification code. For example, consider a kriged block that has a preliminary resource classification of measured, i.e., a code of 1. If the kriging error is above 1.07, then the resource code is incremented to a

Table 14-6: Statistics for Kriged Total Copper Percent Block Values Broken Out by Detailed Code

PROJECT TITLE : Zonia 50x50x20 (2011 _ new UG data & GEMS origin)
CURRENT LABEL : (G101) Kriged Grade kCuT%

	BLOCK 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	444834	0	0	0	0.	0.	0.	0.	0.	0.0000	0.0000	0.0000	0.0000	0.	0.0000
120	9	0	0	14	0.07903	0.10663	0.09331	0.000070	0.00838	0.0898	-2.3756	0.0074	0.0859	0.0933	0.0861
122	30	0	0	320	0.01000	0.13564	0.05987	0.000832	0.02884	0.4817	-2.9775	0.4103	0.6406	0.0625	0.7123
124	1028	5	0	841	0.00149	0.46454	0.09165	0.00416	0.06447	0.7034	-2.7385	0.9669	0.9833	0.1049	1.2766
126	584	0	0	2400	0.00461	0.20020	0.04948	0.000946	0.03076	0.6218	-3.1943	0.4001	0.6326	0.0501	0.7014
130	51035	216	0	31488	0.00158	0.45040	0.06362	0.00271	0.05201	0.8175	-3.0762	0.7211	0.8492	0.0662	1.0280
132	0	0	0	5	0.03674	0.08361	0.06481	0.000359	0.01896	0.2926	-2.7772	0.0896	0.2993	0.0651	0.3061
134	217	9	0	592	0.01194	0.43222	0.11859	0.00791	0.08896	0.7501	-2.5083	0.9069	0.9523	0.1281	1.2152
140	25359	203	0	31590	0.00164	0.95486	0.08968	0.00670	0.08188	0.9130	-2.8024	0.9064	0.9521	0.0954	1.2147
142	17876	345	0	20731	0.00102	0.73750	0.07593	0.00431	0.06564	0.8645	-2.8864	0.6548	0.8092	0.0774	0.9617
222	48	0	0	46	0.01332	0.23870	0.09689	0.00426	0.06529	0.6739	-2.5437	0.4280	0.6542	0.0973	0.7309
224	39	0	0	326	0.01543	0.33244	0.13908	0.00365	0.06040	0.4343	-2.0927	0.2860	0.5348	0.1423	0.5754
226	77	0	0	340	0.02950	0.36868	0.16211	0.00664	0.08148	0.5026	-1.9368	0.2371	0.4869	0.1623	0.5172
230	1092	0	0	5803	0.01213	0.71096	0.18461	0.00607	0.07793	0.4221	-1.7952	0.2541	0.5040	0.1886	0.5378
232	3	0	0	32	0.09382	0.26780	0.18089	0.00196	0.04426	0.2447	-1.7411	0.0655	0.2560	0.1812	0.2602
234	0	0	0	136	0.03186	0.25925	0.11791	0.00395	0.06281	0.5327	-2.2977	0.3461	0.5883	0.1195	0.6431
240	806	0	0	14932	0.00846	0.55203	0.18133	0.00562	0.07498	0.4135	-1.8161	0.2689	0.5186	0.1861	0.5554
242	1569	0	0	9008	0.01133	0.57808	0.15517	0.00606	0.07781	0.5015	-2.0152	0.3657	0.6048	0.1600	0.6645
320	3	0	0	22	0.12309	0.18030	0.14734	0.000218	0.01475	0.1001	-1.9197	0.0094	0.0967	0.1473	0.0969
322	0	0	0	114	0.10729	0.37754	0.18081	0.00590	0.07683	0.4249	-1.7846	0.1350	0.3675	0.1796	0.3802
324	36	0	0	764	0.01530	0.94789	0.26585	0.02402	0.15499	0.5830	-1.4991	0.3995	0.6320	0.2727	0.7007
326	23	0	0	258	0.07624	0.85963	0.25465	0.02296	0.15154	0.5951	-1.5169	0.2890	0.5375	0.2535	0.5788
330	185	0	0	2524	0.03209	0.61000	0.22099	0.00715	0.08453	0.3825	-1.5885	0.1754	0.4188	0.2230	0.4379
332	0	0	0	2	0.21640	0.25038	0.23339	0.000577	0.02402	0.1029	-1.4577	0.0053	0.0729	0.2334	0.0730
334	0	0	0	3	0.10890	0.15791	0.12891	0.000661	0.02571	0.1994	-2.0613	0.0248	0.1574	0.1289	0.1584
340	552	0	0	11453	0.01339	1.0989	0.22847	0.01277	0.11301	0.4946	-1.6004	0.2735	0.5230	0.2314	0.5609
342	983	0	0	8438	0.01140	0.70974	0.19945	0.01098	0.10479	0.5254	-1.8055	0.5127	0.7160	0.2124	0.8184
420	0	0	0	13	0.14790	0.31316	0.19414	0.00184	0.04287	0.2208	-1.6584	0.0359	0.1895	0.1939	0.1912
422	0	0	0	127	0.06808	1.6856	0.31177	0.08023	0.28325	0.9085	-1.4503	0.5289	0.7272	0.3055	0.8349
424	16	0	0	196	0.01507	0.64051	0.28574	0.02229	0.14930	0.5225	-1.4646	0.5874	0.7664	0.3101	0.8940
426	0	0	0	182	0.01505	1.3573	0.36113	0.05029	0.22425	0.6210	-1.2662	0.7315	0.8553	0.4064	1.0383
430	112	0	0	2975	0.01507	3.1362	0.35183	0.06290	0.25080	0.7128	-1.2404	0.4126	0.6424	0.3555	0.7147
432	17	0	0	69	0.03040	0.87351	0.25880	0.03788	0.19462	0.7520	-1.6237	0.5667	0.7528	0.2618	0.8732
434	0	0	0	5	0.18361	0.35436	0.28226	0.00401	0.06335	0.2245	-1.2879	0.0493	0.2221	0.2827	0.2249
440	247	0	0	2062	0.01504	1.6868	0.31572	0.02360	0.15362	0.4866	-1.2788	0.2827	0.5317	0.3206	0.5716
442	543	0	0	4908	0.01505	2.7205	0.34164	0.04058	0.20144	0.5896	-1.2093	0.2925	0.5408	0.3454	0.5829
ALL	1232503	778	0	152719	0.00102	3.1362	0.13560	0.01502	0.12255	0.9037	-2.4074	0.9827	0.9913	0.1472	1.2929

Figure 14-13: Kriged Total Copper Percent Block Values Histogram

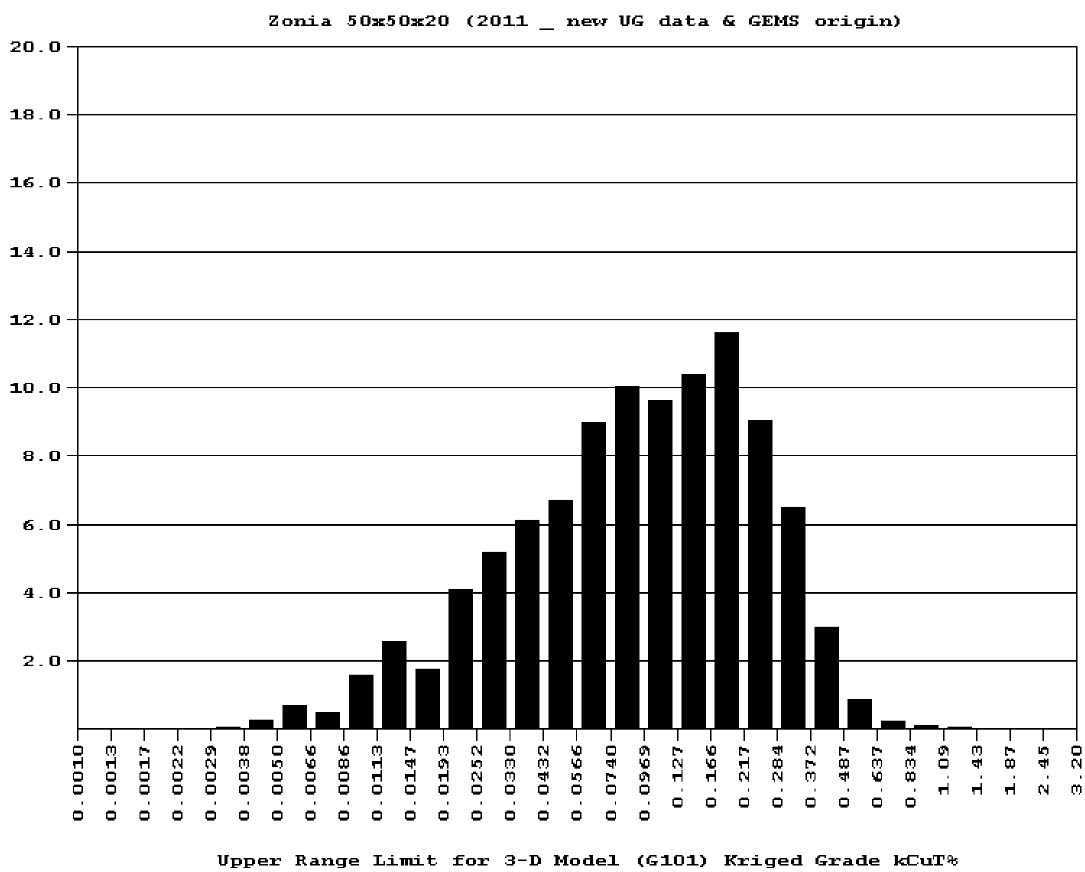


Figure 14-14: Model Section _0000 (Looking Model Northeast) Total Copper Grade Blocks

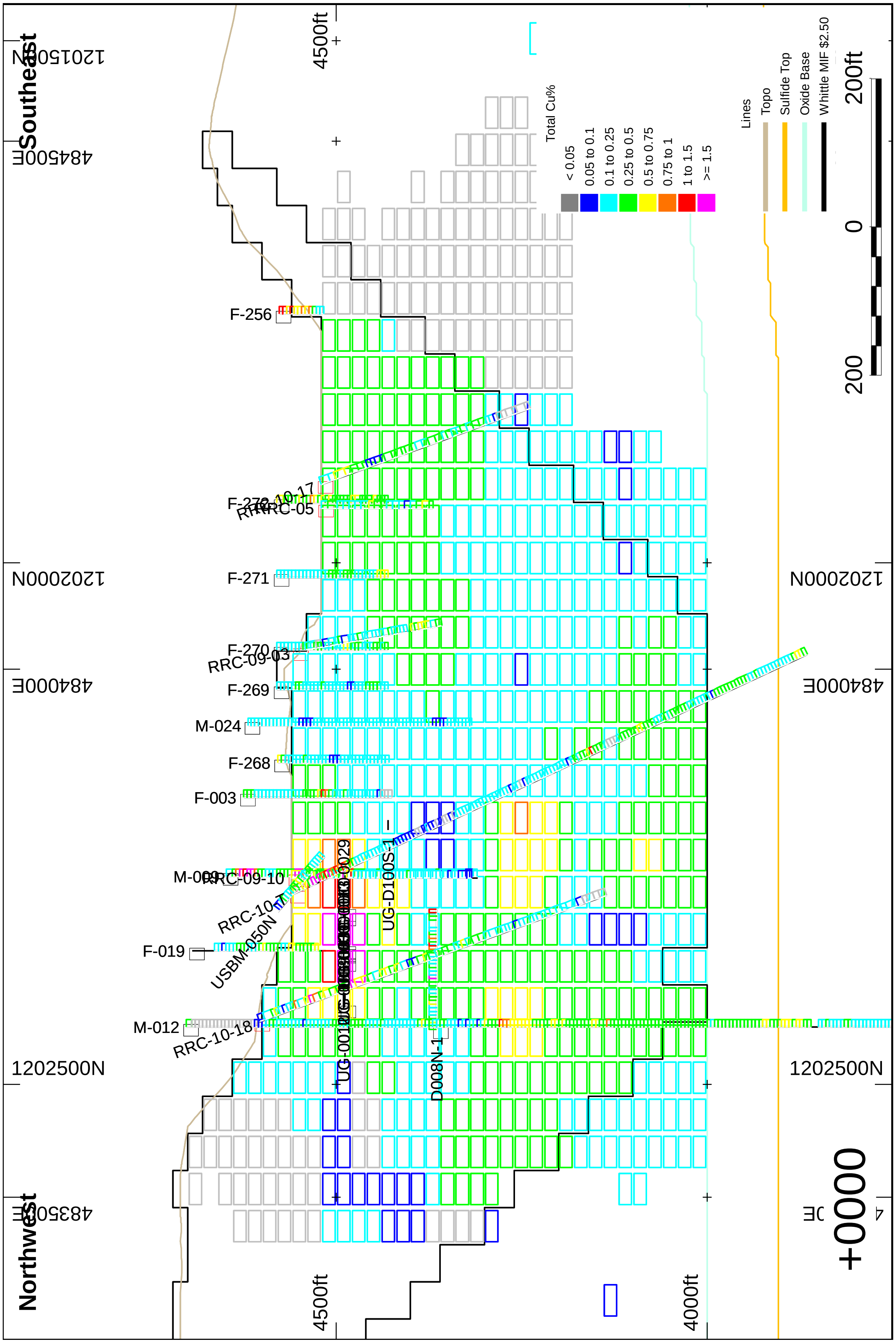
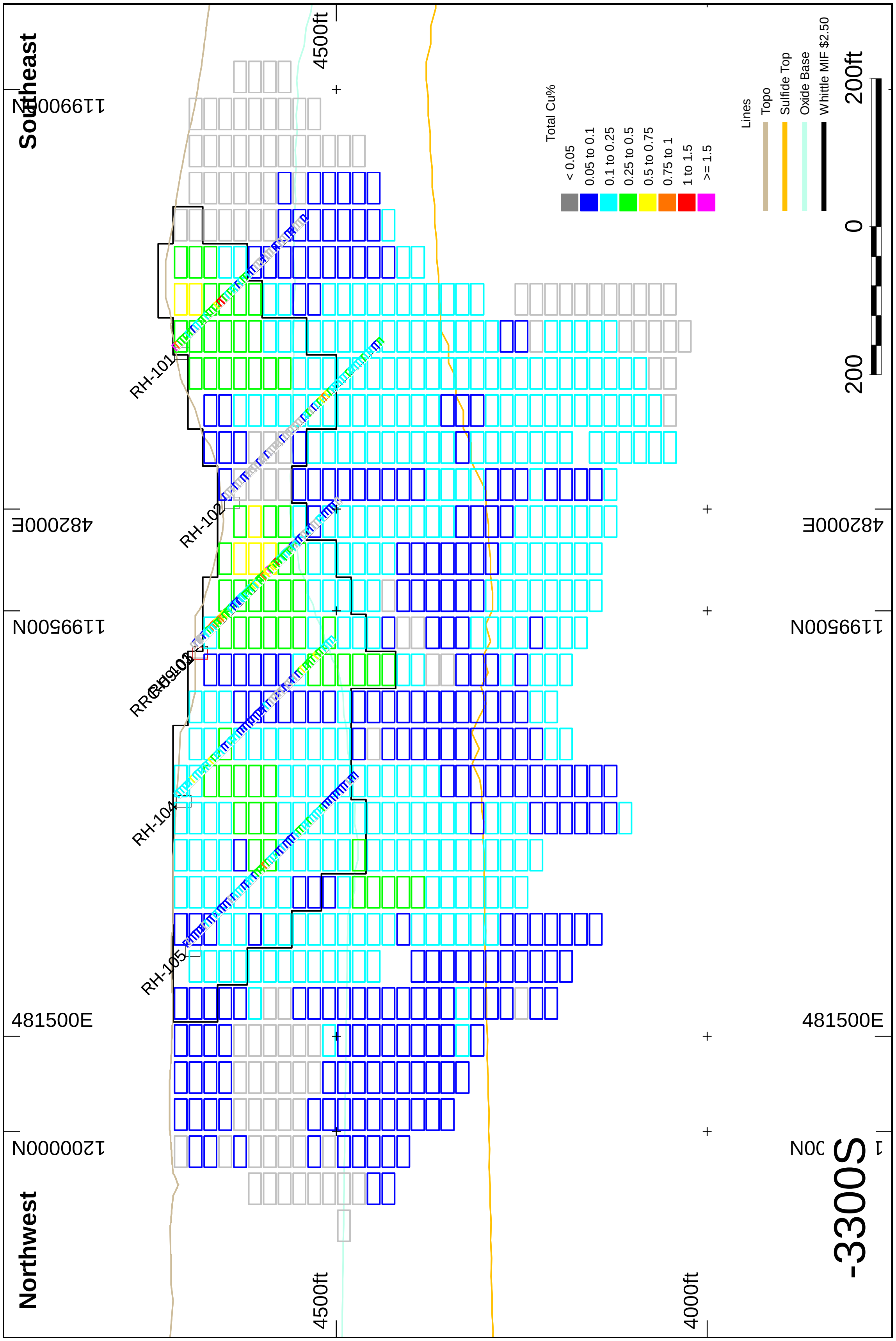


Figure 14-15: Model Section -3300S (Looking Model Northeast) Total Copper Grade Blocks



GLOBAL RESOURCE ENGINEERING

Figure 14-16: Model Section +0800N (Looking Model Northeast) Total Copper Grade Blocks

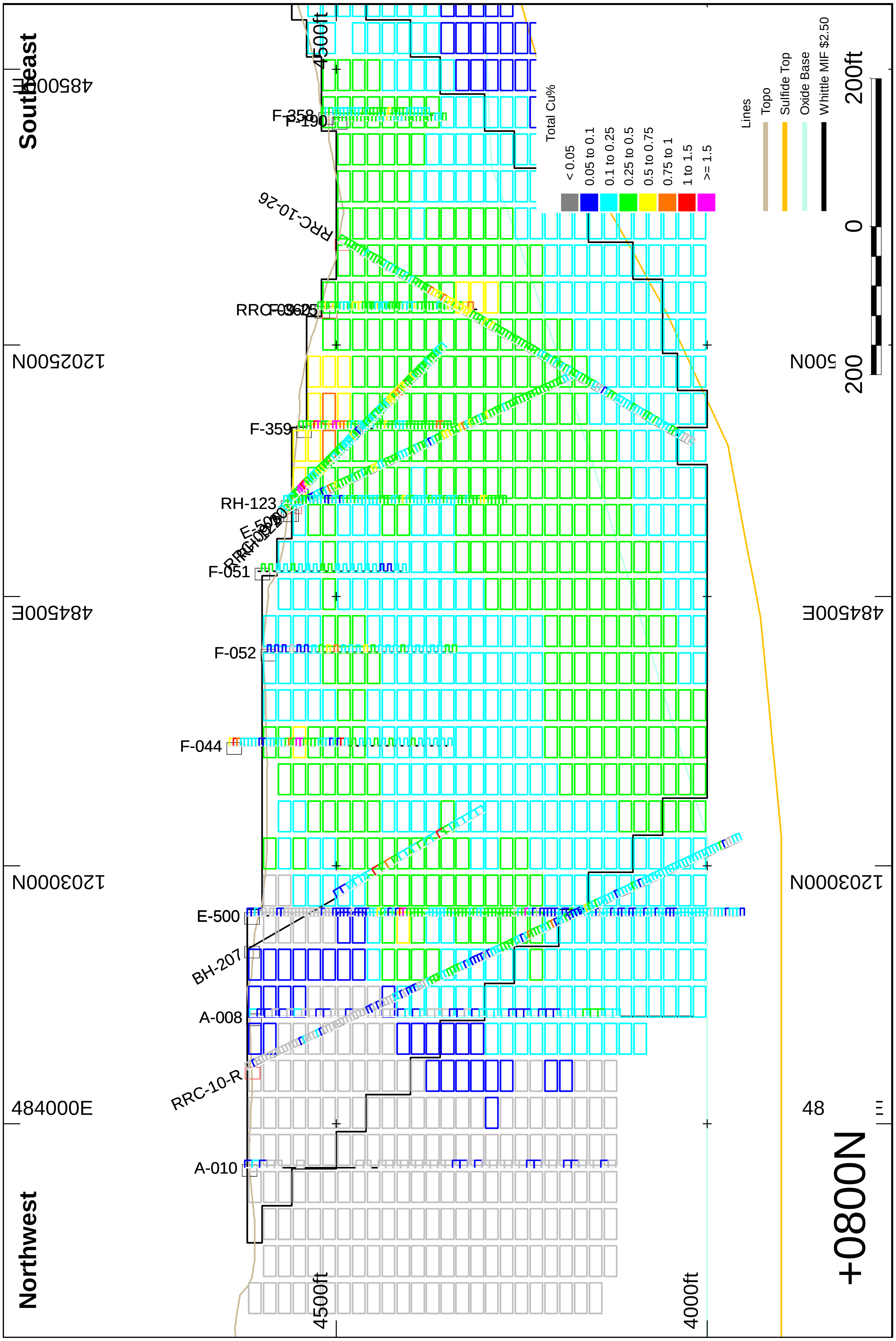


Table 14-7: Final Resource Classification Codes

Code	Category	Comment
1	Measured	original designation
2	Indicated	shifted from measured by kriging error
3	Indicated	original designation
4	Inferred	shifted from indicated by kriging error
5	Inferred	original designation
6	No Class	Shifted from inferred by kriging error

Figure 14-17: Nested Jackknife Error Ellipses Containing ~80% of Trials for Measured, Indicated, and Inferred

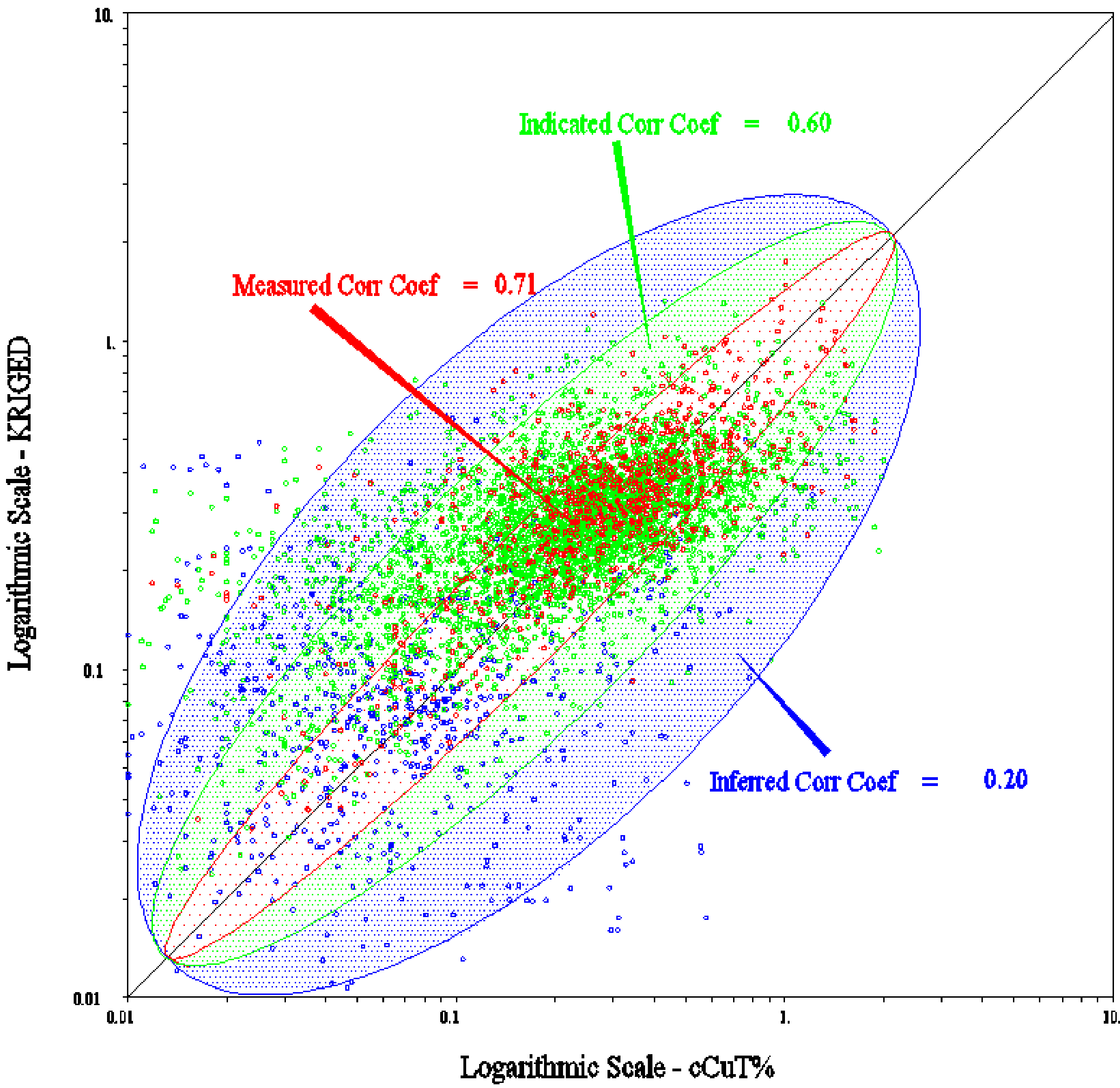
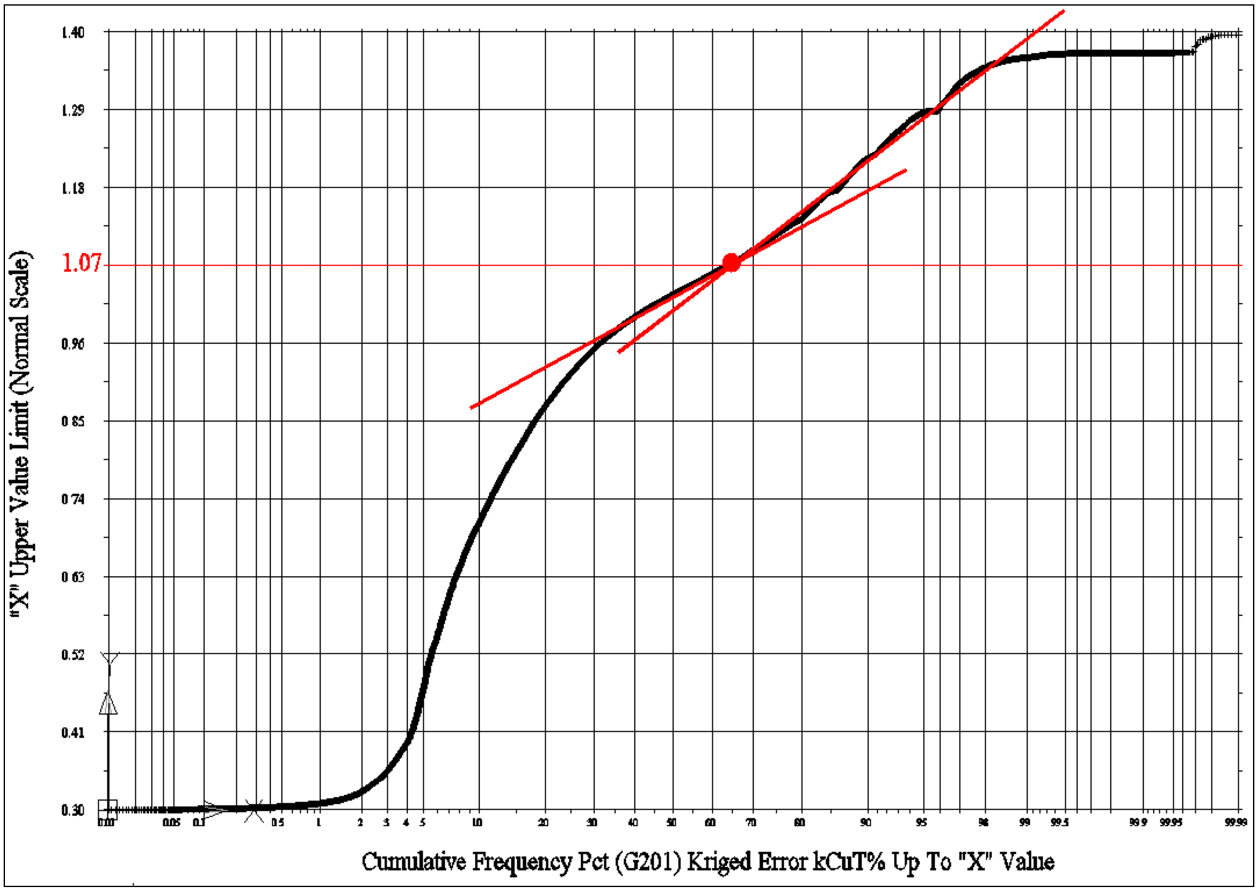


Figure 14-18: Cumulative Frequency Plot of Total Copper Kriging Error



code of 2, which is considered to be an indicated class block. Likewise, an initial inferred class with a kriging error above 1.07 is not considered as a resource block.

14.7 Resource Classification

Each estimated block has been assigned measured, indicated or inferred classification for its contained mineral resources. The Canadian Institute of Mining, Metallurgy and Petroleum (CIM May 10, 2014) defines mineral resources as:

- Mineral Resource: Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource. Please note the Cautionary statements regarding inferred mineral resource estimates.
- Inferred Mineral Resource: An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

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

- Indicated Mineral Resource: An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

  Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

- Measured Mineral Resource: A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

  Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

- Modifying Factors: Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

This classification method has produced minor amounts of isolated blocks of each category that future estimates will need to address. Block model cross-sections of resulting classification are shown in Figure 14-19 to Figure 14-21.

Figure 14-19: Model Section +0000 (Looking Model Northeast) Block Classification

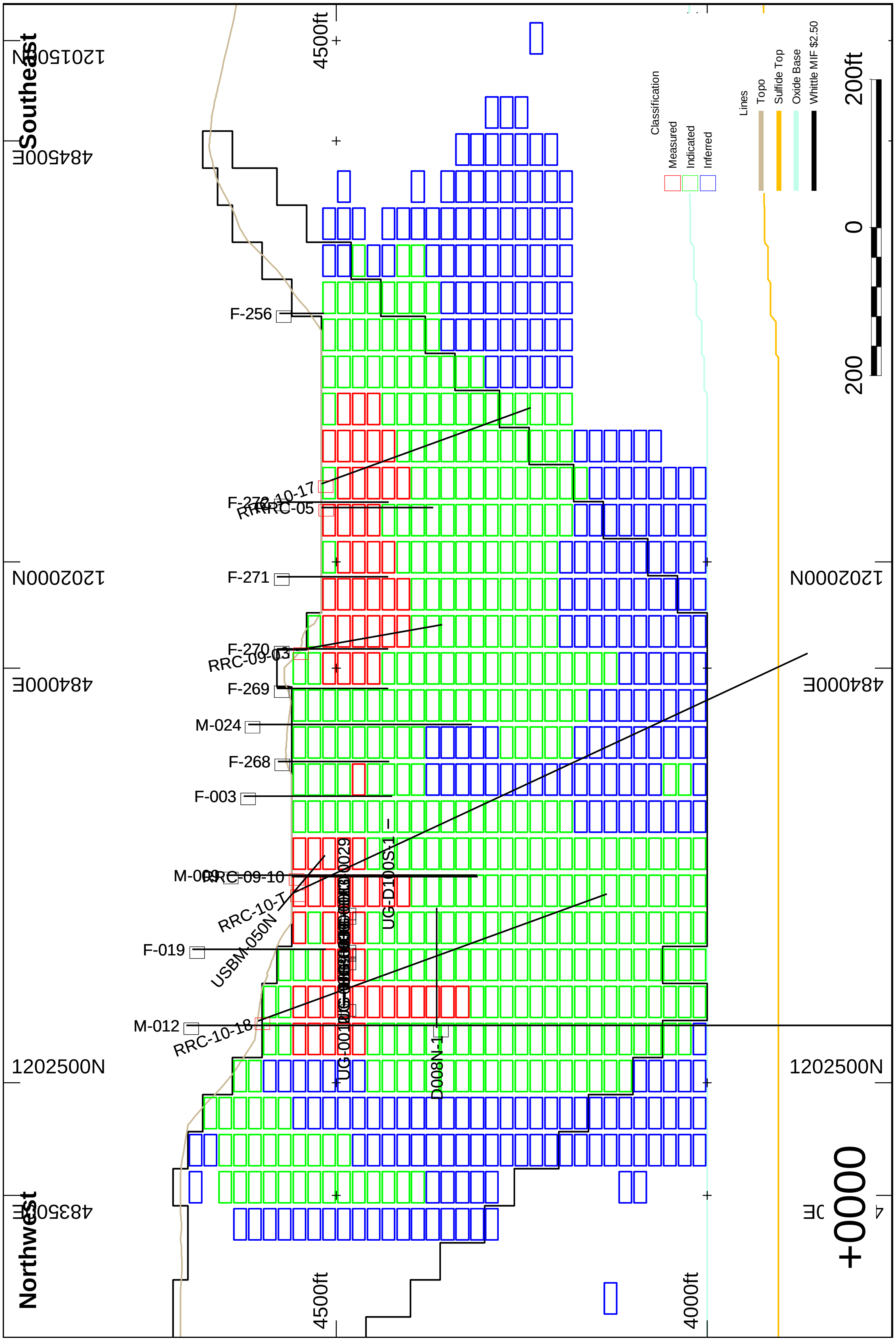


Figure 14-20: Model Section -330S (Looking Model Northeast) Block Classification

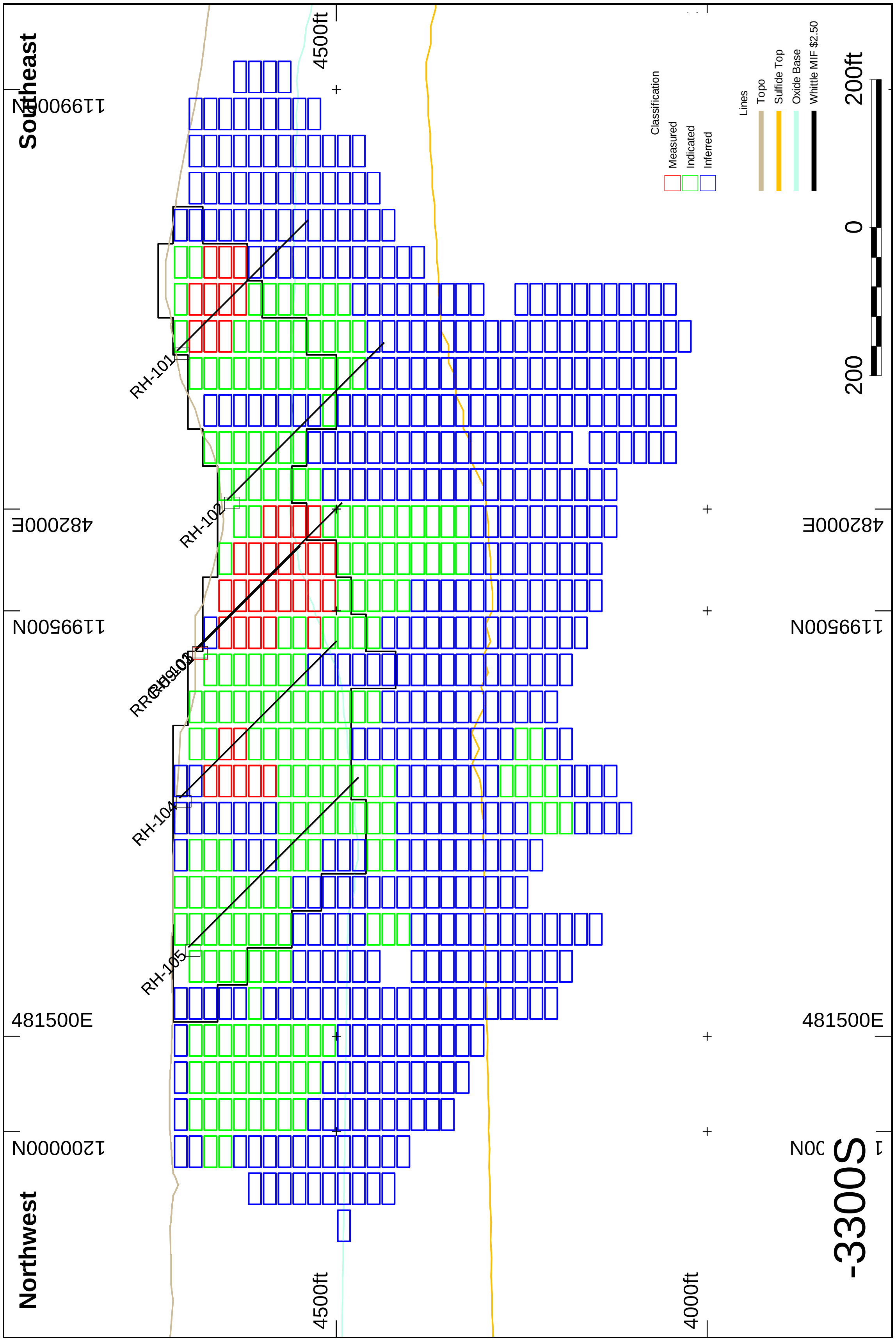
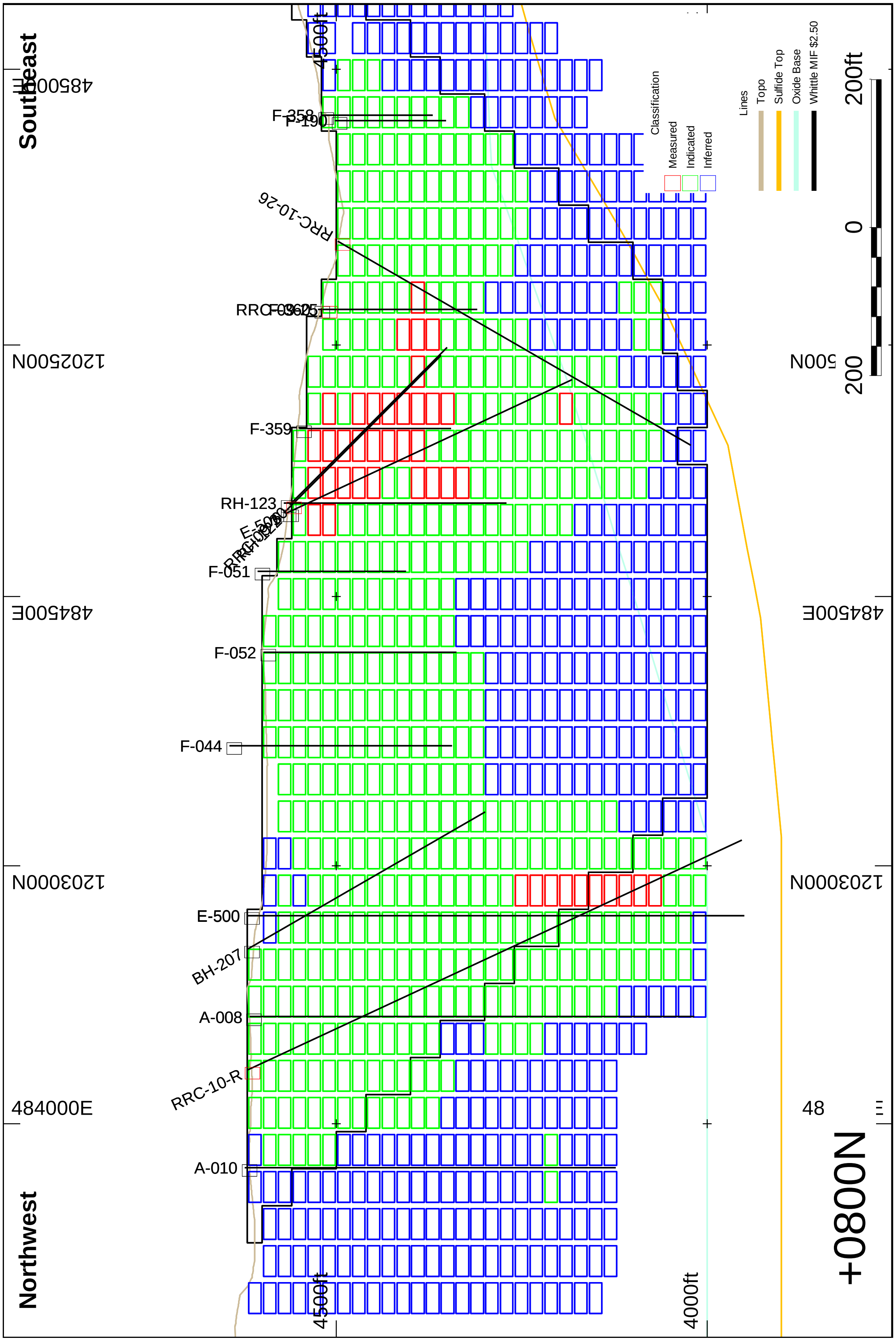


Figure 14-21: Model Section +0800N (Looking Model Northeast) Block Classification



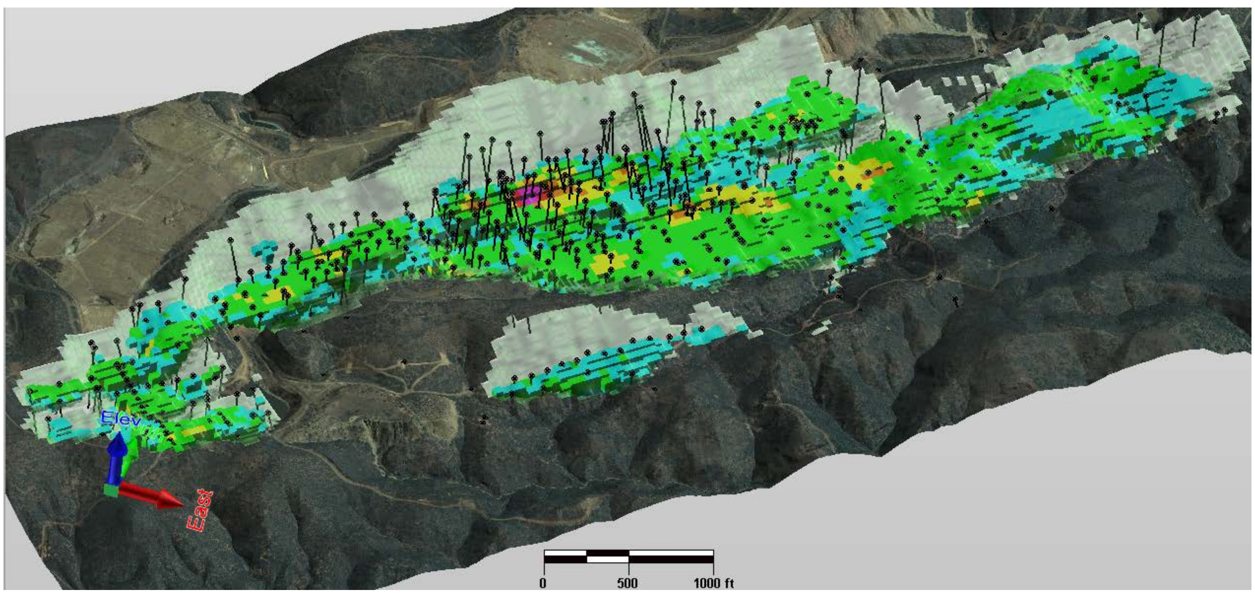
14.8 Cutoff Grade and Reasonable Prospects for Economic Extraction

Mineral resources have been constrained to a Lerchs-Grossman pit optimization run on measured, indicated, and inferred blocks using the parameters shown in Table 14-8. Blocks that fall within the optimized shell have been reported using a base case block cutoff of 0.2% TCu (see Figure 14-22).

Table 14-8: Lerchs-Grossman Optimization Parameters

Input	Value	Unit
Mining Cost	1.5	$/ton
Process Cost	3.4	$/ton
G&A	0.45	$/ton
Recovery Oxide	73	%
Recovery Transition	70	%
Recovery Primary Sulfide	0	%
Pit Slope	45	Degrees
Cu Price	2.5	$/lbs

Figure 14-22: 3D View of Block Model within Optimized Shell



14.9 Mineral Resource Statement

Table 14-9 shows the 2017 Tetra Tech estimated Zonia classified mineral resources at a base case cutoff of 0.2 % TCu. Mineral resources have been reported within an open-pit shell generated using the Lerchs-Grossman algorithm. *Mineral resources within an optimized shell are not mineral reserves and do not have demonstrated economic viability.* It is the authors' opinion that the reported mineral resource classifications comply with current CIM definitions for each mineral class. The authors have evaluated the sensitivity of the cases below the 0.20% Cu cutoff and determined these scenarios meet the threshold of reasonable prospects for eventual economic extraction.

Table 14-9: Zonia Classified Mineral Resource Base Case

Classification	Cutoff Grade Cu%	Tons M	Grade Cu%	Cu lbs M
Measured	0.2	15.4	0.42	129.3
Indicated	0.2	61.4	0.31	380.6
Measured + Indicated	**0.2**	**76.8**	**0.33**	**510.0**
Inferred	0.2	27.2	0.28	154.6

Notes:
Resources are stated within a Lerchs-Grossman optimized shell using the following parameters: Mining (ore and waste) $1.5/ton, processing $3.4/ton, General and Administrative $0.45/ton, oxide recovery 73%, transition recovery 70%, and Cu price $2.50/lbs
Columns may not total due to rounding.
One Ton is equal to 2,000 lbs or 0.9071847 Tonnes.
Inferred Mineral Resources: It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

14.10 Relevant Factors Affecting Resource Estimates

There are currently no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors which could affect the mineral resource estimate.

15.0 MINERAL RESERVE ESTIMATES

There are no Mineral Reserves for the Zonia project at this time. The Zonia project is at a Preliminary Economic Assessment phase of project development. As defined by NI 43-101, a Prefeasibility Study or Feasibility Study is required to state Mineral Reserves.

16.0 MINING METHODS

16.1 Pit Shell Generation

As part of this NI 43-101 Technical Report PEA, GRE used the 2017 Tetra Tech block model to generate new pit shells at metal prices from $0.50/lb to $5.00/lb Cu, in $0.25/lb increments. A description of the process is provided below in Section 17. Table 16-1 shows the estimated classified mineral resources within the $2.50/lb pit at various cutoffs. This pit was chosen to compare to the Tetra Tech pit shell at the 0.20 Cutoff. GRE's pit has slightly fewer tons and pounds of copper.

Table 16-1: GRE $2.50 Pit Shell Mineral Resource at Various Cutoffs

Classification	Tons (millions)	Cu Grade (%)	Cu Pounds (millions)
0.15 Cutoff			
Measured	15.1	0.415	125.4
Indicated	63.7	0.297	378.3
Measured+Indicated	78.8	0.319	503.7
Inferred	28.9	0.265	153.3
0.175 Cutoff			
Measured	14.9	0.418	124.7
Indicated	58.5	0.309	362.0
Measured+Indicated	73.5	0.331	486.7
Inferred	25.6	0.279	143.1
0.20 Cutoff			
Measured	14.8	0.419	124.5
Indicated	58.3	0.310	361.1
Measured+Indicated	73.2	0.332	485.6
Inferred	24.4	0.284	138.6
0.225 Cutoff			
Measured	14.1	0.430	121.3
Indicated	48.9	0.329	321.4
Measured+Indicated	63.0	0.351	442.6
Inferred	19.2	0.304	117.0
0.25 Cutoff			
Measured	12.9	0.449	115.5
Indicated	38.9	0.353	274.3
Measured+Indicated	51.8	0.377	389.8
Inferred	14.6	0.326	95.4

Notes:

(4) Resources are stated within a floating cone optimized shell using the following parameters: Mining (ore and waste) $1.8/ton, processing $2.89/ton plus $0.12/lb copper SX/EW, General and Administrative $0.80/ton, oxide recovery 73%, transition recovery 70%, and Cu price $2.50/lbs

(5) Columns may not total due to rounding

(6) Inferred Mineral Resources: It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Pit shells were generated in Techbase® using the parameters shown in Table 16-2.

Table 16-2: Pit Generation Parameters

Input	Value	Unit
Mining Cost	1.80	$/ton
Process Cost	2.89 Plus 0.12	$/ton $/lb Cu SX/EW
G&A	0.80	$/ton
Recovery Oxide	73	%
Recovery Transition	70	%
Recovery Primary Sulfide	0	%
Pit Slope	45	Degrees
Cu Price	2.5	$/lb

16.2 Pit Shell Selection for Design

Each pit was evaluated at cutoff grades varying in 0.01% increments from 0.12% to 0.22% and using preliminary surface mining costs, processing costs, and G&A costs to calculate preliminary Net Present Value (NPV) at varying discount rates and Internal Rate of Return (IRR). This preliminary analysis was conducted to identify the pits with the greatest potential for economic success. The preliminary analysis indicated that the $2.00/lb pit had the greatest potential for economic success.

The pit shell for the $2.00/lb pit was imported into Geovia GEMS™ to design the ultimate pit layout using 45 degrees batter angle, 20-foot bench height, 12.7-foot bench width, 10% ramp grade, and ramp width of 100 feet for all but the lowest four benches, which were given a single-wide 50-foot ramp width.

16.3 The designed $2.00/lb pit is shown in Economic Cutoff Grade

GRE calculated the economic cutoff grade as follows:

Typical open pit heap leach operating costs:

Mining	$1.80/ton
Process	$3.66/ton
G&A	$0.80/ton
Total	$6.26/ton

With a weighted average 72.9% recovery, the cost is $8.59/ton contained, and with a metal price of $3.00/lb, the calculated cutoff grade is: $\frac{\$8.59/\text{ton}}{\$3.00/\text{lb}} = 2.86 \frac{\text{lb}}{\text{ton}} / 20 = 0.14\%$.

Figure 16-1.

A starter pit using the $1.00/lb pit shell was designed to provide high-grade material for the project startup. The designed starter pit is shown in Figure 16-2.

Table 16-3 summarizes the resources in the GEMS designed pit at various cutoff grades. The design pit is used to plan pit infrastructures, but does not affect the reported /tables resources.

16.4 Economic Cutoff Grade

GRE calculated the economic cutoff grade as follows:

Typical open pit heap leach operating costs:

Mining	\$1.80/ton
Process	\$3.66/ton
G&A	\$0.80/ton
Total	\$6.26/ton

With a weighted average 72.9% recovery, the cost is \$8.59/ton contained, and with a metal price of \$3.00/lb, the calculated cutoff grade is: $\frac{\$8.59/\text{ton}}{\$3.00/\text{lb}} = 2.86\frac{\text{lb}}{\text{ton}}/20 = 0.14\%$.

Figure 16-1: Designed $2.00/lb Zonia Pit

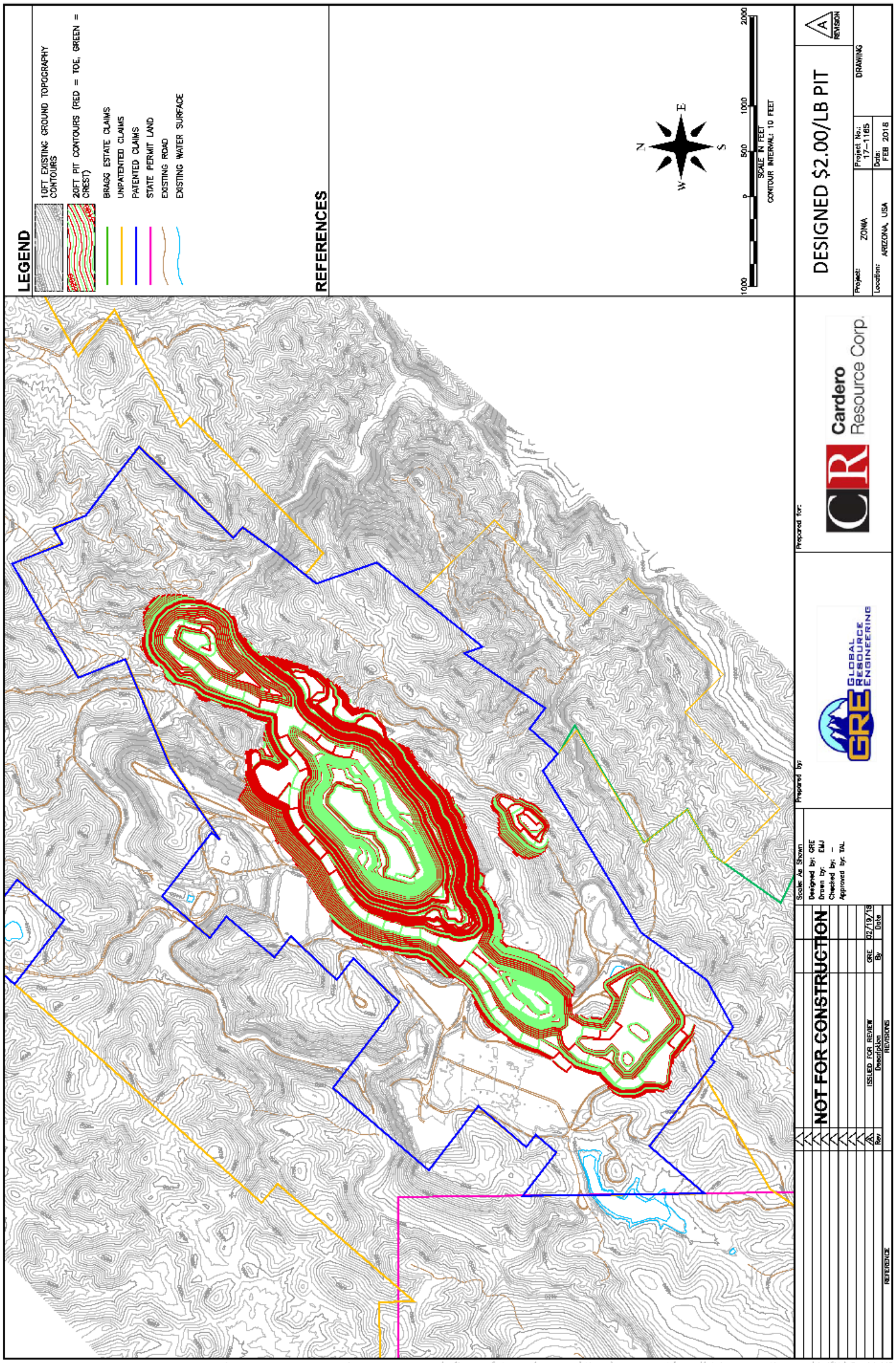


Figure 16-2: Designed Zonia Starter Pit

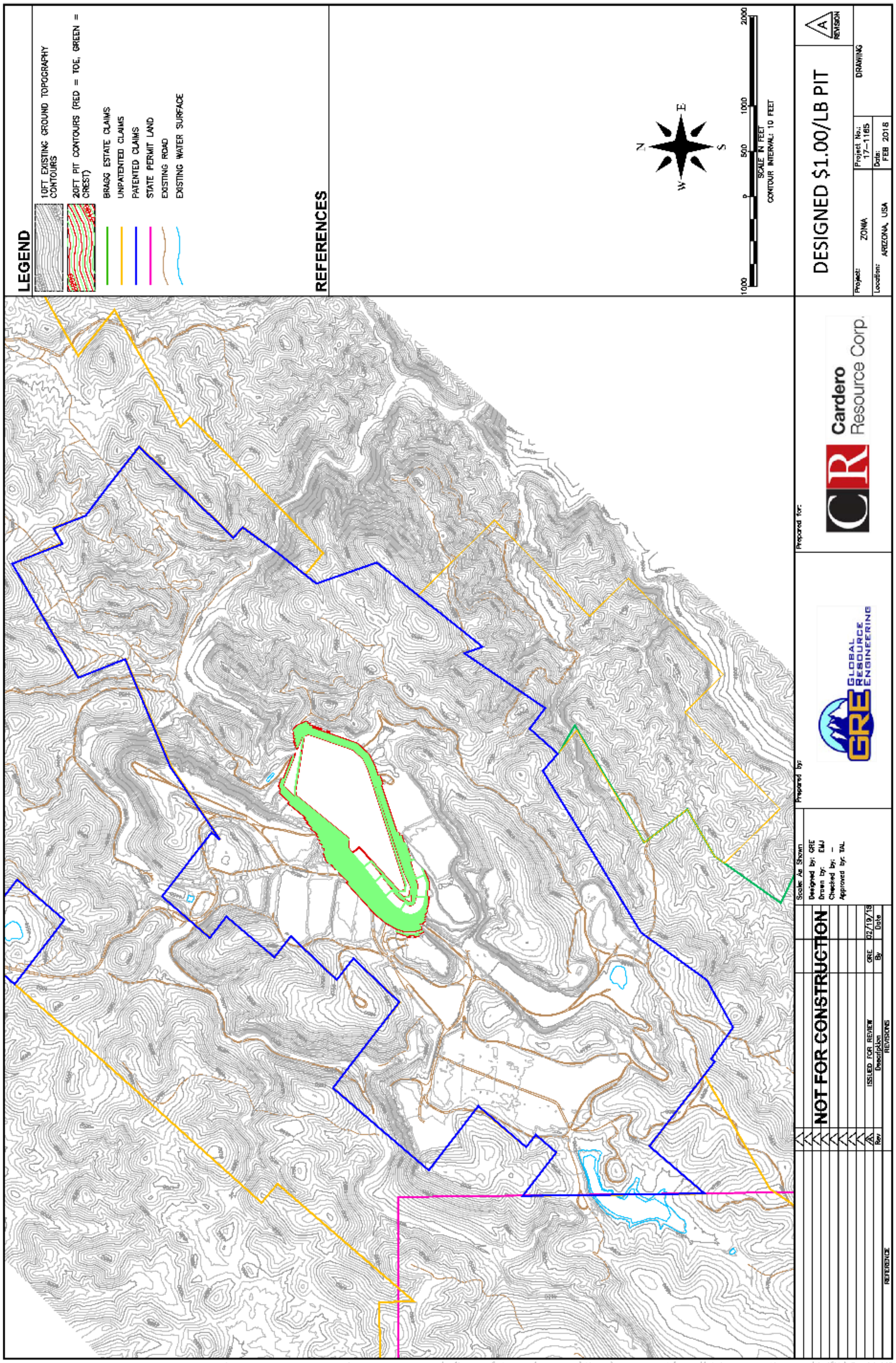


Table 16-3: Mineral Resources within the $2.00/lb Designed Pit

Category	Leachable Tons	Contained Copper lbs	Grade
0.12 Cutoff			
Measured	15.5	126.2	0.408
Indicated	65.1	362.3	0.278
Measured + Indicated	80.5	488.5	0.303
Inferred	26.4	131.1	0.248
0.13 Cutoff			
Measured	15.3	125.9	0.410
Indicated	63.5	358.3	0.282
Measured + Indicated	78.8	484.2	0.307
Inferred	25.6	129.0	0.252
0.14 Cutoff			
Measured	15.2	125.6	0.412
Indicated	61.9	354.1	0.286
Measured + Indicated	77.2	479.8	0.311
Inferred	24.6	126.3	0.257
0.15 Cutoff			
Measured	15.1	125.3	0.414
Indicated	60.1	348.7	0.290
Measured + Indicated	75.2	474.0	0.315
Inferred	23.4	122.9	0.262
0.16 Cutoff			
Measured	15.0	124.9	0.416
Indicated	58.2	342.9	0.295
Measured + Indicated	73.2	467.8	0.320
Inferred	22.2	119.1	0.268
0.17 Cutoff			
Measured	**14.9**	**124.6**	**0.418**
Indicated	**56.4**	**337.1**	**0.299**
Measured + Indicated	**71.3**	**461.7**	**0.324**
Inferred	**21.3**	**116.2**	**0.273**
0.18 Cutoff			
Measured	14.8	124.2	0.420
Indicated	54.0	328.7	0.304
Measured + Indicated	68.8	452.9	0.329
Inferred	20.0	111.5	0.279
0.19 Cutoff			
Measured	14.6	123.3	0.423
Indicated	50.9	317.0	0.312
Measured + Indicated	65.4	440.4	0.337
Inferred	18.2	105.0	0.288
0.20 Cutoff			
Measured	14.3	122.2	0.428
Indicated	48.3	306.9	0.318
Measured + Indicated	62.5	429.1	0.343

Category	Leachable Tons	Contained Copper lbs	Grade
Inferred	17.0	100.1	0.295
0.21 Cutoff			
Measured	14.0	121.2	0.432
Indicated	45.0	293.3	0.326
Measured + Indicated	59.0	414.5	0.351
Inferred	15.6	94.5	0.303
0.22 Cutoff			
Measured	13.7	120.0	0.436
Indicated	42.0	280.4	0.334
Measured + Indicated	55.7	400.4	0.359
Inferred	14.4	89.6	0.310

GRE rounded the calculated cutoff grade to the nearest 0.5%, resulting in a minimum economic cutoff grade of 0.15%. The base case uses a cutoff grade of 0.17% because that cutoff resulted in the best overall project economics. GRE considers this cutoff grade to be reasonable based on our experience with other mines of similar scale and type.

16.5 Block Size and Dilution

The Tetra Tech block model uses 20-foot downhole composites and 40 x 40 x 20-foot blocks, which are slightly larger than a selective mining unit for this deposit and mining fleet. The estimation method combined with block size and compositing has resulted in a fully diluted resource estimate. GRE has not added any other dilution. GRE believes because of the methods used, the resource is recoverable without ore loss.

16.6 Base Case Mine Operation and Layout

For the base case, the $2.00/lb pit with a grade cutoff of 0.17% was selected. Estimates of the resources for the designed pit were exported from GEMS™ by bench to an excel file. A production schedule was generated using the following assumptions:

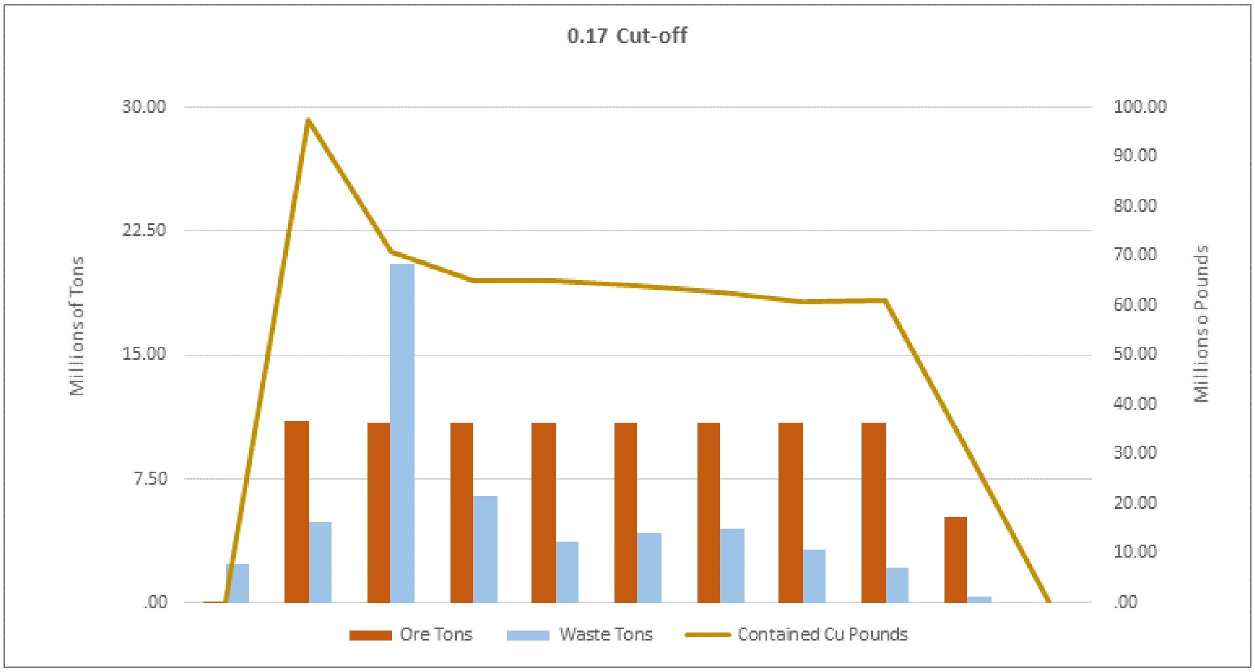
- Leachable material production rate (the “production rate”): 30,000 tpd
- Mine operating days per week: 7
- Mine operating weeks per year: 52
- Mine operating shifts per day: 2
- Mine operating hours per shift: 10

The schedule was broken into two phases, with the starter pit as Phase 1 and the remaining material in the $2.00/lb pit as Phase 2. Additional design work will likely better balance the annual leach and waste stripping and metal production.

Pre-stripping of waste was included if either of the following criteria were met: 1) waste occurred on a bench that had no corresponding leachable material or 2) the tonnage of waste on a bench exceeded 10 times the tonnage of leachable material on that bench. The mining rate for pre-strip benches (the “pre-strip rate”) was set to two times the production rate, or 60,000 tpd.

Schedules were produced for 0.1% incremental cutoffs from 0.12% to 0.22%. The production schedule for the base case at a cutoff of 0.17% is shown in Figure 16-3.

Figure 16-3: Mining Schedule



The mine life for this case is 8.6 years, as shown in Figure 16-3. Total material quantities for the base case are estimated to be 145 million total tons, 92.6 million leachable tons, 52.4 million waste tons, and 577.9 million pounds of contained copper. Of the contained copper pounds, 567.4 million tons are oxide material and 10.5 million tons are mixed material. The recovery rate for oxide material is projected to be 73% and for mixed material is projected to be 70%, resulting in 421.5 million pounds of recovered copper.

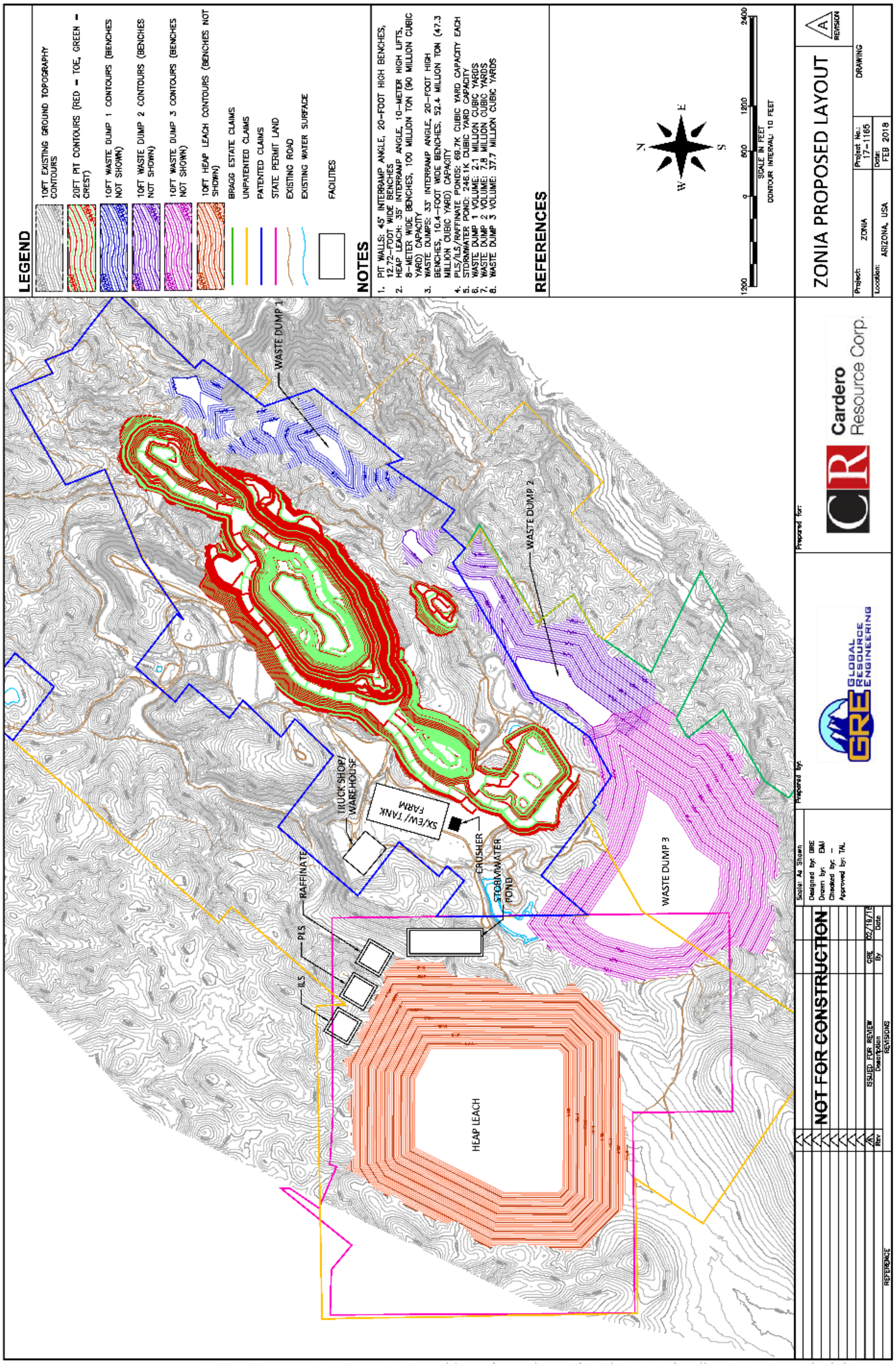
The Zonia project would be mined using conventional open pit methods using off-highway trucks and loaders or shovels. Drilling, blasting, load, and haul would be used to remove overburden waste and leachable material. Waste would be hauled to disposal sites located as near as possible outside of the largest hypothetical pit rim. Ground pressure from stacking the waste is not expected to impact pit wall stability.

Leachable material would be hauled from the pit to the crusher. Crushed material would be transported via conveyors to the leach pad.

For the base case, contractor mining operations were selected.

A proposed layout of the facility is shown in Figure 16-4.

Figure 16-4: Preliminary Site Plan



16.7 Mine Equipment Productivity

A cycle time formulation was generated to determine the sizes and numbers of trucks and loaders required to meet the productivity schedule. A simplified approach to cycle calculations was used. It considered productivity variables such as average daily production of leachable material and waste, average truck haul distance and travel speed, hours per shift and shifts per day, availability variables such as breaks during the day, and truck and loader/shovel capacities. Hourly production rates and truck and loader wait times were calculated to optimize the design. The haul truck and loader productivity calculations are summarized in Table 16-4.

Table 16-4: Summary of Zonia Haul Truck and Loader Productivity Calculations

		Results
Truck Haul Distance	Feet	5,000
Truck Haul Time Loaded	Minutes	6.3
Average Speed	mph	9.0
Truck Haul Time Empty	Minutes	3.0
Average Speed	mph	19.0
Ton	Ton per Day	45,119
No. of Shifts		2
Equipment Availability		95.00%
Effective work minutes / hour		57
Basic Shift Time	Hours	10
Effective Shift Time	Hours	9.50
Total Time Available	Hours	8.00
Cat Loader Model		993K
Loader Bucket Capacity	Cubic Yards	23.5
Loader Bucket Fill Factor		0.95
Net Loader Bucket Capacity	Cubic Meters	22.37
Cat Truck Model		789D
Truck Capacity - Heaped	LCY	170.0
Truck Maximum Load	Tons	209
Truck Maximum Capacity	Cubic Yards	135.7
Buckets/ Truck	Buckets	6.00
Fill Amount	Cubic Yards	134.2
Underfill	Cubic Yards	1.6
No. of Trucks	Trucks	4
Truck Load	Tons	207.04
Load Time/ Truck	Minutes	4.00
Loader Wait for Spot	Minutes	0.25
Truck Cycle/ Load Cycle		3.89
Truck Wait/ Truck Cycle	Minutes/Truck Cycle	0.00
Loader Wait Time	Minutes/Loaded Truck	0.00
Average Truck Cycle with Wait	Minutes	16.55
Effective Production	Tons/hour	2,922.98
Effective Production	Tons/day	46,767.75
Required hours	Hours/day	15.89
Available hours	Hours/day	16.00
Required Fleet		1

17.0 RECOVERY METHODS

17.1 Process Description

17.1.1 The Zonia project would employ open pit mining with a conventional copper acid heap leach system on a 365 day per year 24 hour per day basis. The run of mine material would be crushed in a three-stage crushing circuit to a nominal P_{80} size of 25 mm (1 inch) and then be agglomerated with acid containing solutions as either raffinate or fresh sulfuric acid. The crushing circuit would operate five days per week, 16 hours per day. This operating design would allow sufficient time for maintenance and production catch-up should it be necessary. The crushed rock would be delivered to the heap via overland conveyor and grasshopper conveyors and stacked in 10-m lifts with a radial stacker operating in retreat mode. The heap is designed to contain up to 10 lifts for an ultimate height of 100 m. Provision has been made to use interlift liners between lifts. Crushing and Agglomeration Circuits

The gyratory crusher would crush to a nominal 150 millimeter (mm) (6 inch), with the crushed product reporting by conveyor to a double-deck vibrating screen. Screen oversize, +50 mm (2 inch), would be fed to the secondary standard-head cone crusher (CSS 50 mm (2 inch)), and the screen middlings (-50mm +25 mm) (-2 inch + 1 inch) would report to tertiary crushing. Screen undersize (-25 mm, -1 inch) would report to the fine ore stockpile. The tertiary crushing circuit would be operated open-circuit with a single-deck screen. Oversize material (+25 mm, +1 inch) from secondary crushing reports to the short-head cone crusher (CSS 25mm). Undersize material (-25mm, -1 inch) reports to the fine ore stockpile. Ore would be reclaimed from the fine ore stockpile by a series of two vibrating feeders. Figure 17-2 shows the crushing flowsheet.

Figure 17-1 shows the complete conceptual flowsheet.

The heap leach pad and ponds are designed with a dual layer polyethylene liner system (LDPE and HDPE) with leak detection. Leach solution is transferred by gravity to either the pregnant solution (PLS) pond or the intermediate solution (ILS) pond. PLS is transferred to a conventional solvent extraction (SX) circuit for copper recovery from the solution. The depleted copper solution (raffinate) is transferred to the raffinate pond for reuse on the heap as the primary lixiviant. Solution is recycled to the heap via drip irrigation at a nominal rate of 12 liters per square meter per hour (12 lph/m^2). A storm water pond is provided, designed to handle a 24-hour – 1 in a 100-year precipitation event.

The SX circuit consists of two extraction stages and one stripping stage using a conventional mixer/settler arrangement. The loaded organic from the extraction stage is transferred to the stripper vessel, producing a rich electrolyte solution for subsequent electrowinning. The copper-depleted raffinate from the extraction circuit is recycled to the raffinate pond. Prior to electrowinning, the rich electrolyte is purified to remove entrained organic through column flotation and filtration.

The electrowinning (EW) circuit consists of two parallel banks of 50 polycement cells with 1 square meter cathodes. The plated copper cathodes are stripped using a mechanized stripping system after being washed. Copper cathodes are then sampled and bundled for shipment.

17.1.2 Crushing and Agglomeration Circuits

The gyratory crusher would crush to a nominal 150 millimeter (mm) (6 inch), with the crushed product reporting by conveyor to a double-deck vibrating screen. Screen oversize, +50 mm (2 inch), would be fed to the secondary standard-head cone crusher (CSS 50 mm (2 inch)), and the screen middlings (-50mm +25 mm) (-2 inch + 1 inch) would report to tertiary crushing. Screen undersize (-25 mm, -1 inch) would report to the fine ore stockpile. The tertiary crushing circuit would be operated open-circuit with a single-deck screen. Oversize material (+25 mm, +1 inch) from secondary crushing reports to the short-head cone crusher (CSS 25mm). Undersize material (-25mm, -1 inch) reports to the fine ore stockpile. Ore would be reclaimed from the fine ore stockpile by a series of two vibrating feeders. Figure 17-2 shows the crushing flowsheet.

Figure 17-1: Conceptual Flowsheet for the Zonia Copper Oxide Project

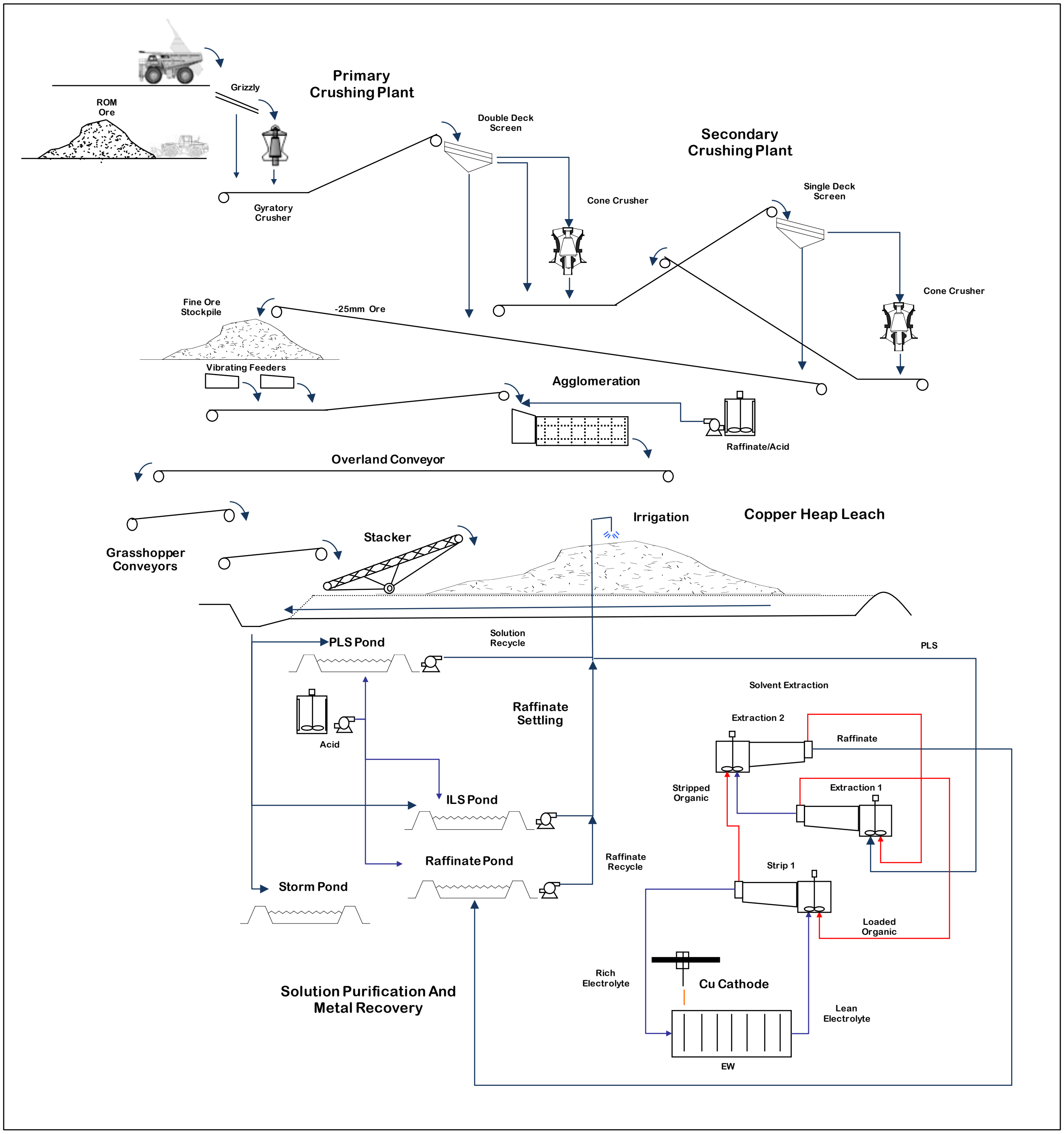
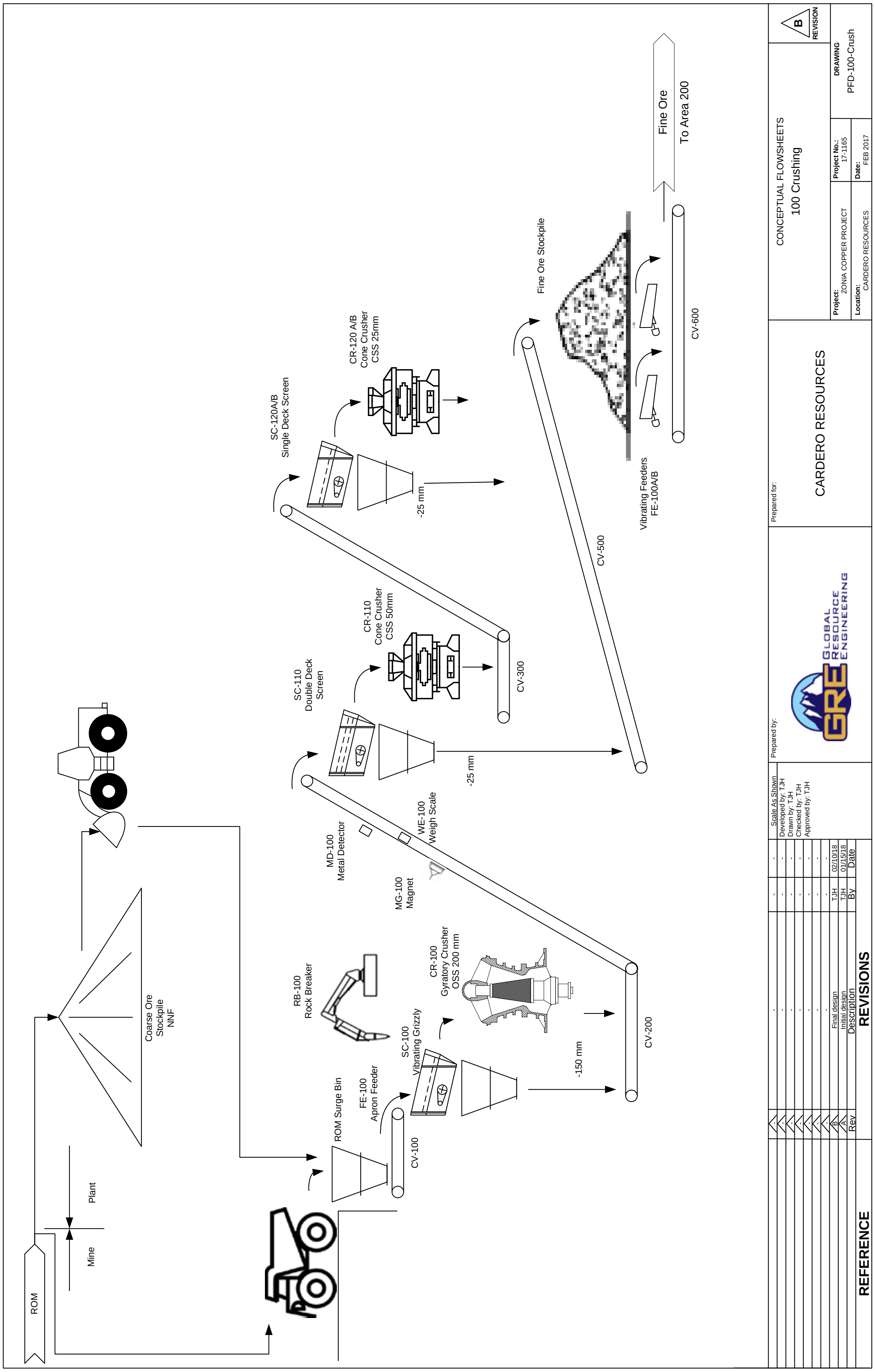


Figure 17-2: Crusher Flowsheet



Crusher material would be reclaimed from the stockpile and fed to the agglomeration drum along with raffinate and/or fresh sulfuric acid. The target is to deliver approximately 60% of the total acid demand to the agglomerated ore while not exceeding 8-10% moisture by weight. The ore would be conveyed via overland and grasshopper conveyors to a prepared permanent leach pad. The ore is stacked using a slewing radial stacker to lift heights of 10 meters (m). Stacking would be conducted in retreat mode during the creation of each leach cell. The conceptual agglomeration circuit is shown in Figure 17-3.

17.1.3 Heap Leach Circuit

Agglomerated ore would be allowed to cure for five days prior to irrigation being introduced. Irrigation is provided by an emitter-type irrigation system designed to deliver 12 liters per square meter per hour (lph/m^2) (0.005 gallons per square foot per hour [gph/ft^2]). Emitter layout is designed to provide suitable ore wetting. The heap would be placed under irrigation for a period of approximately 80 days. At the termination of the leach, irrigation would be discontinued, the heap cell allowed to drain, and the cell may also be rinsed with raffinate or process water to remove residual copper and acid. Figure 17-4 shows the heap leach flowsheet.

High concentration copper leach solutions or PLS flow from the pad to the PLS pond by gravity. As the heap leaching cycle progresses, lower grade solution may be diverted to an ILS pond. ILS is used to maintain copper solution grade targets by allowing the solution to be recycled to the leach pad to increase solution copper concentrations. Solution is collected from each heap cell by a series of drain pipes under the heap that transport the solution to perimeter ditches. The solution can be placed in either the PLS, ILS, or storm water ditch. PLS and ILS solutions flow in the ditches by gravity to the respective ponds (PLS and ILS Ponds). Storm water collected from the pad during heavy precipitation events can be diverted to a storm water pond. The storm water can be utilized as fresh make up water to the circuit or treated and discharged.

17.1.4 Solvent Extraction Circuit

During normal operations, PLS solution is pumped to the solvent extraction (SX) circuit, and the ILS is recirculated to the heap. The SX circuit consists of two extraction stages to recover the copper from the leach solution and a single strip stage to produce electrowinning (EW) feed. Copper-rich leach solutions are contacted with an organic oxime mixed with a kerosene carrier in a conventional mixer/settler tank. The copper is extracted from the aqueous phase of the PLS and carried forward in the organic phase to the stripper. The striping circuit mixes the loaded organic with a high concentration acid solution returning from the EW circuit (lean electrolyte) to cause the copper to transfer from the organic phase to the aqueous phase. The copper-rich aqueous solution is delivered to the electrowinning circuit for copper recovery (rich electrolyte). The conceptual SX circuit is show in Figure 17-5.

Figure 17-3: Agglomeration Flowsheet

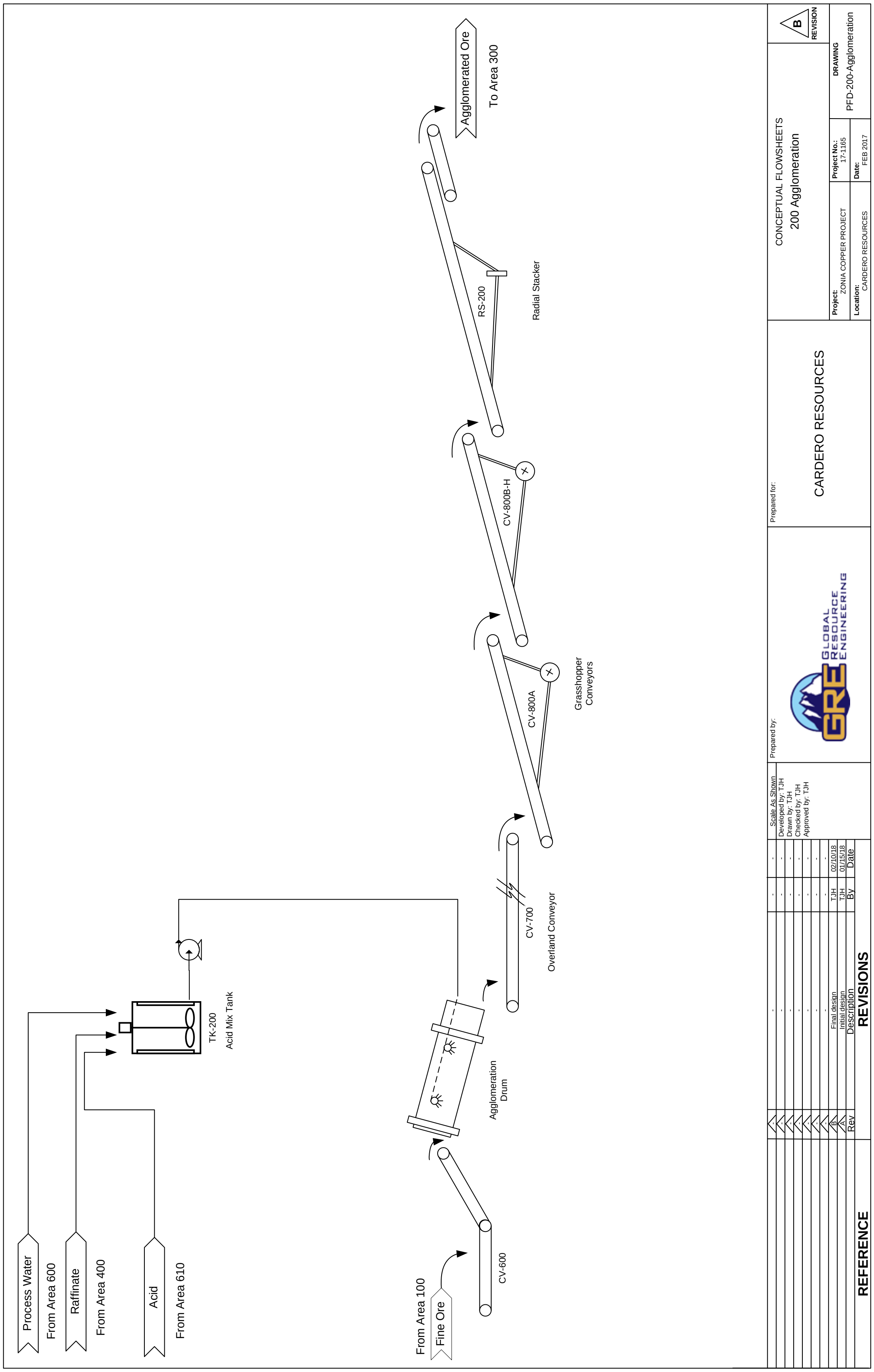


Figure 17-4: Leach Flowsheet

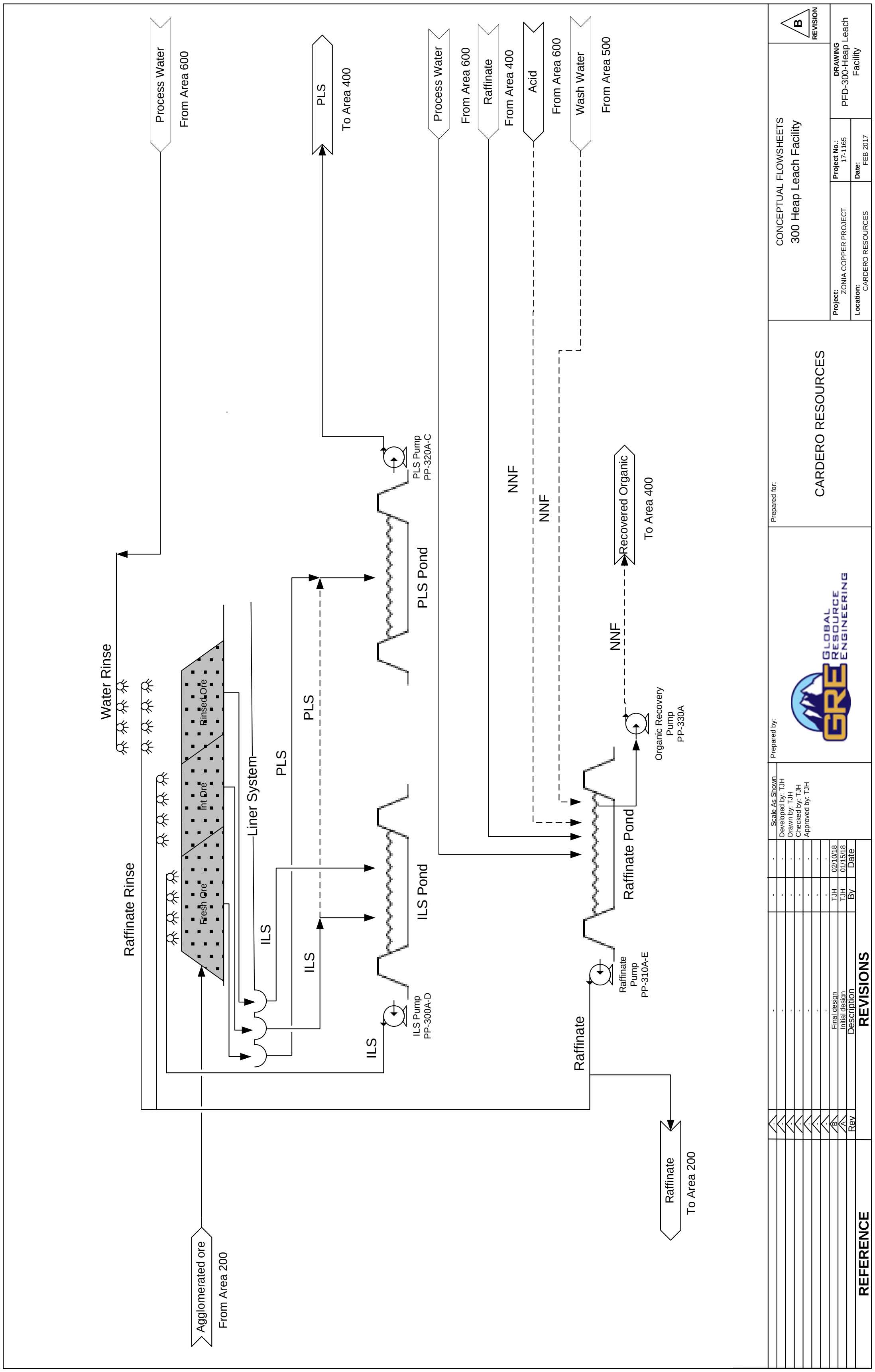
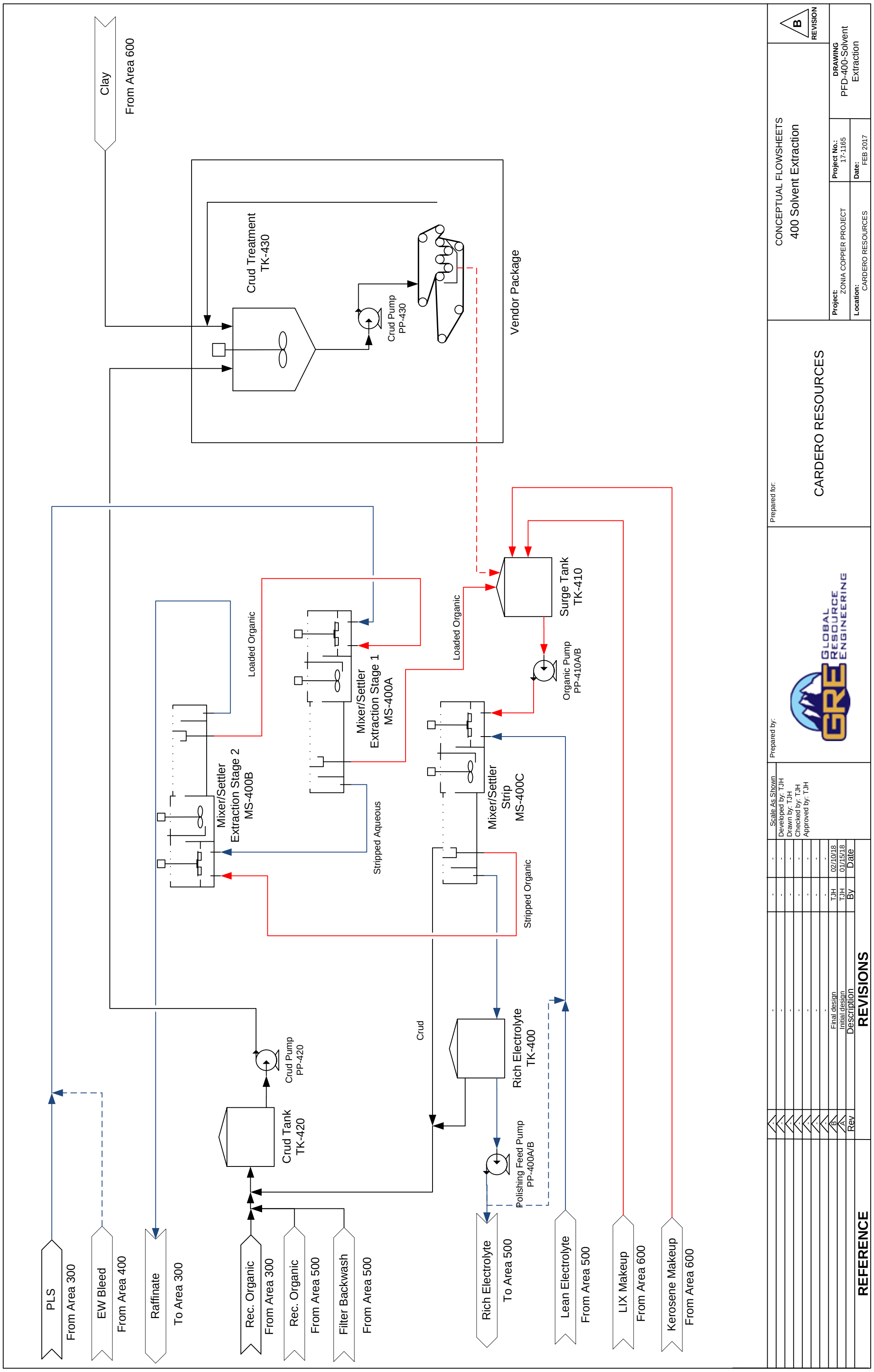


Figure 17-5: Solvent Extraction Flowsheet



17.1.5 Electrowinning Circuit

The rich electrolyte is treated by column flotation and filters to remove any entrained organic and solids. The polished rich electrolyte is pumped to the electrolyte surge tank and combined with a portion of lean electrolyte as required. The copper-rich solution is then plated onto cathodes in the EW tank house. Copper plating continues for a given cycle (typically 7 days), after which the cathode is removed from the circuit and washed and the cathode copper stripped off the stainless steel starter sheet. Cobalt sulphate solution would be added to the electrolyte to maintain a concentration of up to 200 ppm cobalt. This helps to protect the anodes from corrosion. Polyacrylamide solution would also be added to the electrolyte as a growth modifier helping to create dense flat deposits of copper. The EW circuit is designed to produced London Metal Exchange (LME) grade A cathodes that are weighed, sampled, and prepared for shipment. The EW flowsheet is shown in Figure 17-6.

The leach-SX/EW circuit is design as a closed circuit zero-discharge facility with all solution being recirculated within the process. Make up water would be added to compensate for evaporative losses and field moisture losses within the heap.

17.2 Conceptual Heap Leach Pad and Pond Design

The Heap Leach Facility (HLF) consists of the following system components:

- Heap leach pad
- Liner system
- Leachate (solution) collection system
- Storm pond
- Stormwater management system
- Freshwater supply

To minimize capital expenditure, the heap leach pad has been designed in phases, with each phase requiring advanced expansion of the engineered pad. The HLF would be constructed in three phases, with the pad foundation preparation, liner installation, and collection piping advanced as the leach pad expands. The capacity of each stacking stage includes an initial four-year period with two additional three year periods.

The initial HLF development (Phase 1) would also include the full development of the solution handling system, storm pond, and perimeter diversion ditches prior to commencing ore stacking and leaching. Table 17-1 presents the stacking schedule, the respective development footprints, and ore volume capacities. Table 17-2 shows the development phases and the lift capacity in ore volume and duration.

Design details for each of the HLF components are discussed further in the following sections.

17.2.1 Heap Leach Pad

The heap leach pad consists of a perimeter berm, pad liner system, and leachate collection system to collect and convey the leachate solution to the copper SX/EW plant, which should be located adjacent to the heap leach facility. The preliminary location for the heap leach pad would be on the western side of the property, due west of the southern end of the proposed open pit. The pad would be located on

Figure 17-6: Electrowinning Flowsheet

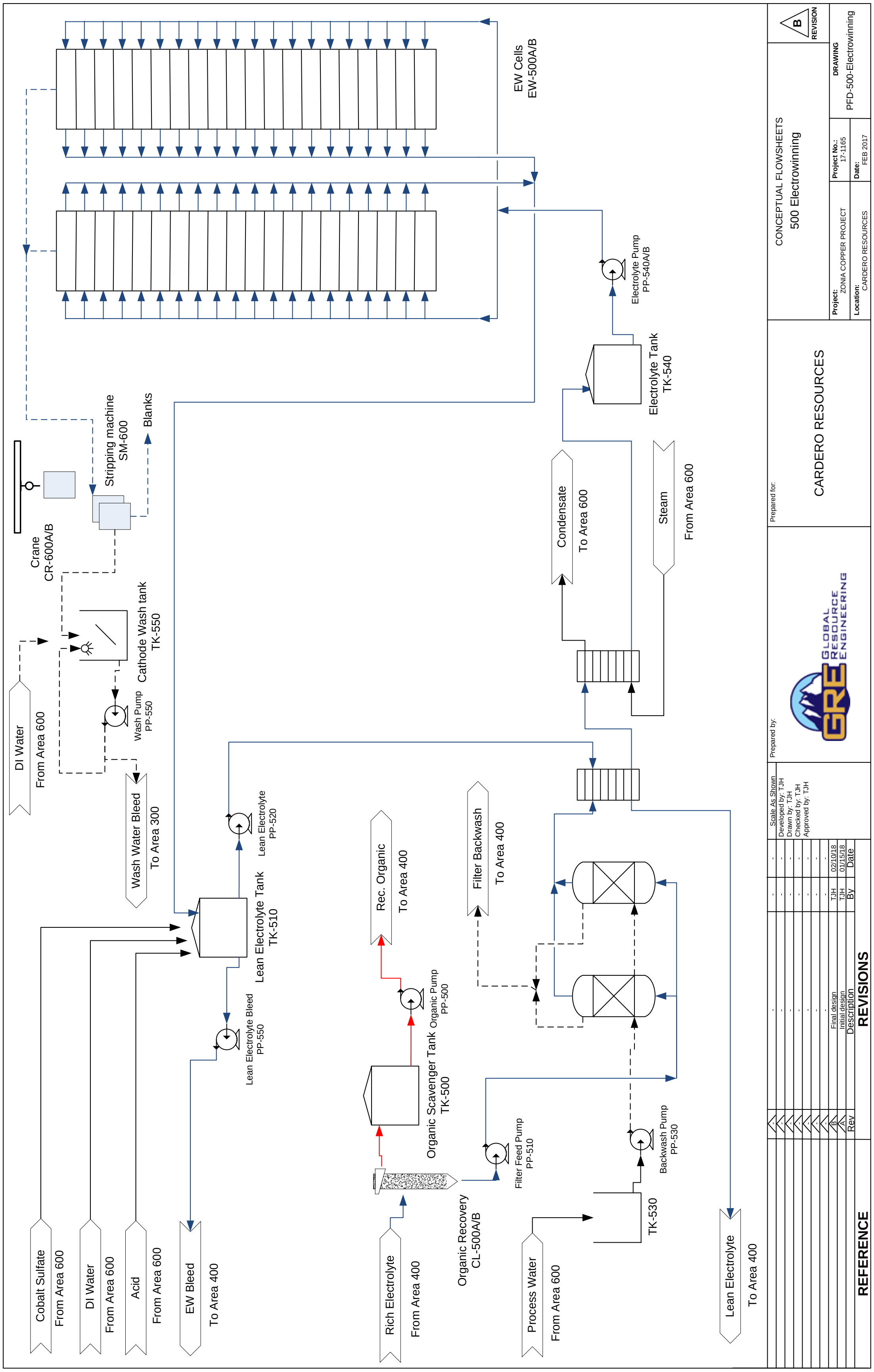


Table 17-1: Heap Stacking Schedule

Year	Development Phase	Liner Footprint (m^2)	Stacked Ore (t)	Stacked Ore (m^3)	Stacked Ore (Cumulative m^3)
1	1	450,000	10,163,050	7,009,000	7,009,000
2			10,163,050	7,009,000	14,018,000
3			10,163,050	7,009,000	21,027,000
4	2	315,000	10,163,050	7,009,000	28,036,000
5			10,163,050	7,009,000	35,045,000
6			10,163,050	7,009,000	42,054,000
7	3	315,000	10,163,050	7,009,000	49,063,000
8			10,163,050	7,009,000	56,072,000
9			10,163,050	7,009,000	63,081,000
10			8,532,550	5,884,517	68,965,517
Total		1,080,000	100,000,000	68,965,517	

*Note year phase 2 is 3.5 years long.

Table 17-2: Heap Capacity

Development Phase	Elevation (absm)	Lift Capacity (days)	Mine Life (years)	Ore Volume (m^3)
1	10	228	0.6	4,302,652
	20	425	1.2	3,711,338
	30	593	1.6	3,159,735
	40	733	2.0	2,647,841
	50	849	2.3	2,175,654
	60	941	2.6	1,743,166
	70	1013	2.8	1,350,366
	80	1066	2.9	997,231
	90	1102	3.0	683,705
	100	1124	3.1	409,608
2	10	1288	3.5	3,100,140
	20	1444	4.0	2,944,190
	30	1592	4.4	2,788,245
	40	1732	4.7	2,632,308
	50	1863	5.1	2,476,382
	60	1986	5.4	2,320,473
	70	2101	5.8	2,164,594
	80	2208	6.0	2,008,767
	90	2306	6.3	1,853,048
	100	2396	6.6	1,697,616
3	10	2561	7.0	3,099,987
	20	2717	7.4	2,944,014
	30	2865	7.8	2,788,040
	40	3004	8.2	2,632,067
	50	3136	8.6	2,476,092
	60	3259	8.9	2,320,117
	70	3373	9.2	2,164,141
	80	3480	9.5	2,008,164

Development Phase	Elevation (absm)	Lift Capacity (days)	Mine Life (years)	Ore Volume (m³)
	90	3578	9.8	1,852,186
	100	3668	10.0	1,696,205

undulating land that generally slopes from the southwest to the northeast, with an overall grade change of 28H:1V (3.5%), and has an approximate final footprint area of 1,150,000 m^2. The heap leach pad is designed to be operated as a fully drained system with no leachate storage within the HLF.

Prior to the start of each of the development stages, the pad foundation must be prepared. Foundation preparation involves stripping the topsoil and vegetation and the removal of any rocks. The topsoil would be stockpiled at a convenient location and used for reclamation of the HLF at closure. The underlying soils would be excavated down to a competent, stable bedrock foundation to provide a uniform and graded surface for the pad liner. Grading and backfill would be used to level the bedrock surface and to ensure that the pad grading will promote leachate flow towards the collection piping system and sump located near the western edge of the Phase 1 heap. A minimum pad grade of 2% is required.

17.2.2 Liner System

A liner system is planned to maximize solution recovery and minimize environmental impacts by minimizing leachate losses through the bottom of the leach heap pad. The liner system consists of both barrier and drainage layers using a combination of synthetic and natural materials to provide leachate solution containment that meets the accepted standards for leach pad design. The pad is designed to operate with minimal solution storage within the pad structure during normal operating conditions. The liner system is designed to meet the required performance standards assuming fully saturated solution storage conditions.

17.2.2.1 Liner Design

A liner system has been developed for the pad using an engineered composite double liner design. The double liner system is designed to be installed as the primary liner system under the entirety of the HLF. The double liner system consists of the following components:

- 1-meter thick overliner (38 mm minus with less than 10% fines content)
- 80-mil (2-mm) linear low-density polyethylene (LLDPE) geomembrane
- 0.3-meter thick compacted low permeability soil liner
- Non-woven, needle punched geotextile layer
- Leak Detection and Recovery System (LDRS)
- 60-mil (1.5-mm) LLDPE geomembrane.

LLDPE was proposed for the geomembrane liner systems for the heap leach pad because it has the following benefits (Lupo, et al., 2005):

- Generally higher interface friction values, compared to other geomembrane materials
- Ease of installation in cold climates due to added flexibility,
- Good performance under high confining stresses (large heap height), and
- Higher allowable strain for projects where moderate settlement may become an issue.

17.2.2.2 Construction

Development of the heap leach liner would be constructed in three phases, with pad expansions proposed every three to four years to meet ore stacking requirements.

The liner system would be constructed with both the synthetic and natural layers extending to the top of the perimeter berms to provide full containment. The synthetic liners and geotextiles would be anchored and backfilled in a trench along the heap leach pad perimeter and perimeter berms to ensure that ore loading does not compromise the liner coverage of the heap leach pad footprint by pulling the liner into the pad. Along the pad toe, all liners would be tied into their corresponding liner layer along the foundation of the pad to provide a continuous seal and drainage connection.

The perimeter berm would be constructed as part of the liner tie-in around the perimeter of the pad footprint to ensure that heap solution is contained within the pad and to prevent surface runoff entering the pad collection system. A 0.3-meter thick bedding sand layer would be placed on the face of the confining embankment directly underneath the second (bottom) geomembrane liner to provide additional integrity protection to the liner.

17.2.2.3 Overliner

A protective layer of approximately one-half meter of coarse crushed waste would be placed over the entire liner system footprint to protect the liner's integrity from damage during ore placement. The overliner acts as the drainage layer, allowing solution drainage into the pipe collection system. The overliner material must be competent, have low acid consumption, and be free from fines.

17.2.3 Solution Collection System

Collection and recovery of the leach solution is facilitated by the solution collection system in conjunction with the heap leach liner, overliner, and LDRS. The collection system consists of the following pipe and sump components:

- Lateral collection pipes
- Collection header pipes
- Main header collection pipes
- Leachate collection sumps

The solution collection system is designed to facilitate quick and efficient solution conveyance off the pad to reduce the potential risk of solution losses through liner system. The entire piping system is constructed from perforated corrugated plastic tubing (CPT), which is embedded within the overliner layer.

The lateral collection pipes, which are spaced approximately five meters apart under the entire pad footprint, feed directly into the collection header pipes which then flow into the main header. The main header pipes are positioned along the centerline of each heap leach pad cell and terminate at the upstream toe of the perimeter berm at the leachate collection ditch.

Three leachate collection ditches allow solution to flow by gravity to the required storage pond. The collection pipes are fitted with gate valves to allow solution to be directed to one of the three perimeter collection ditches – PLS, ILS, or Storm.

17.2.3.1 Leak Detection and Recovery System

The LDRS is designed to capture and convey any solution that may leak through the overlying geomembrane and low permeability soil layers.

The LDRS consists of a 0.3-meter thick sand layer embedded with 100-mm diameter perforated CPT collection pipes. A non-woven needle punched geotextile overlies the LDRS sand layer to prevent particles from the above soil layer from entering the LDRS.

Any leakage recovered by the LDRS would be conveyed into the LDRS sump at the downstream toe of the HLF. A level-switch controlled submersible sump pump would transfer the recovered solution via a pipe installed within the LDRS sand layer and connect into the main solution recovery line for processing. Monitoring of the leakage recovery would be undertaken by recording pump operating hours.

17.2.3.2 Leakage Detection Cells

To facilitate more accurate leak identification, the entire pad solution collection system is typically sub-divided into multiple independently monitored areas (cells) separated by small berms. Each of these cells has a dedicated leakage detection collection system comprising a drain gravel layer beneath the inner composite liner system which conveys the leakage to a 100-mm diameter perforated collection pipe within the LDRS collection trench. The LDRS ditches flow by gravity at a minimum 0.5 % slope towards the LDRS collection sump, located along the sides of the leach pad. The flow rates from the dedicated collection pipes are continuously monitored and measured prior to discharging into a sump.

17.2.4 Ponds

17.2.4.1 Storm Pond

The Storm Pond is designed to provide storage for excess leachate and runoff generated as a result of rainfall events. The pond is situated immediately down gradient of the HLF, and pond flows are conveyed via solution collection piping and ditches. The Storm Pond is designed to meet the following design criteria:

- Storage capacity to contain the excess HLF leachate and surface runoff from the 1 in 100-year 24-hour storm event without discharge
- Overflow designed to discharge the 1 in 200-year 24-hour storm event

The storage requirements for the Storm Pond were established based on containment of the entire estimated surface runoff generated from the HLF (at the Phase 3 footprint) during the 1 in 100-year 24-hour storm event. Based on the surface runoff estimates, the following storage requirements for the events pond were identified:

- Total runoff estimate for 1 in 100-year 24- hour storm event 172,000 m^3
- 10% additional factor of safety 17,200 m^3
- Total pond storage capacity 189,200 m^3

Solution stored in the Storm Pond would be pumped back to the heap leach pad using the Storm Pond pump station. The pump station is designed to be able to drain the storm volume over a period of approximately ten days.

17.2.4.2 PLS, ILS and Raffinate Ponds

The PLS, ILS, and Raffinate ponds are designed to provide storage for leachate and EW return solutions. The ponds are situated immediately down gradient of the HLF, and pond flows are conveyed via solution collection piping and ditches. The PLS and ILS ponds are designed to meet the following design criteria:

- Storage capacity to contain sufficient solution volumes to maintain irrigation and feed to the SX circuits
- The PLS and ILS Ponds are designed to contain up to 24 hours of solution assuming a maximum irrigation rate of 15 lph/m^2
- PLS, ILS, and Raffinate Ponds are designed with a capacity of approximately 53,000 m^3

Excess solution flows to any of these ponds would be diverted to the Storm Pond for recycle back to the heap.

17.2.4.3 Pond Liner System

The engineered double liner system designed for the ponds uses the same design principles as the HLF pad liner system. The liner design consists of the following layer configuration:

- 60-mil (1.5 mm) high-density polyethylene (HDPE) geomembrane
- 0.3-meter thick low permeability soil liner
- Geosynthetic 'geonet' drainage layer
- 60-mil HDPE geomembrane.

The liner system installed on the upslope of the pond embankment will have an additional 0.3-meter thick bedding sand layer which will interface with the lower geomembrane layer to protect the integrity of the liner.

Installation of a LDRS is not required for the Storm Pond as the pond is operated as a dry-facility and would only receive and store runoff water during significant storm events. In the event that leakage does occur through the double liner system, this water would be conveyed via the geonet layer to a 1-meter thick drainage blanket which underlies the Storm Pond embankment. This drainage blanket discharges to a sump for solution return to the pond.

It is recommended that HDPE geomembrane be used for the pond liner system rather than LLDPE. Unlike the heap leach pad, the pond liner system would not be subjected to high confining stresses from ore stacking, and HDPE has a higher ultraviolet resistance, which is critical for exposed surfaces like that of the ponds.

17.2.5 Runoff Collection and Diversion

The surface water management system proposed for the site consists of a series of ditches constructed around the perimeter of the HLF to intercept overland surface runoff around the HLF pad and to convey surface water away from the active site. The ditches are designed to meet the following design criteria:

- Conveys the 1 in 100 24-hour duration storm event
- Minimum freeboard = 0.3 m
- Minimum ditch grade = 0.01 m/m
- Side slopes = 2H:1V
- Channel shape = trapezoidal.

Lining and protection of the ditch channels from erosion and scouring may be required for all permanent ditches. Temporary ditches would be constructed between heap phases.

18.0 PROJECT INFRASTRUCTURE

Sufficient water appears to be available on the Zonia property. Groundwater wells would be developed to meet the project water requirements. Water flow has been noted in some of the Equatorial drillholes and have potential for water production. Investigations cited in supporting documentation by Eric Swanson of AquaLithos Consulting in Appendix D of the Arizona Aquifer Protection Permit Application indicate the potential for an on-site water source, initially taking advantage of existing wells and supplementing by two or three new modern wells for continuing operations.

Power is available at the mine site from the Arizona Public Service grid through a 33kV power line. There are electrical substations at the mine. Local labor for mining is available and some people who live in the area have previously worked at the mine.

18.1 Water Supply

Modeling of the heap operation on a monthly basis over the projected mine life indicates that operation of the HLF requires a water supply with an approximate average flowrate of 123 m^3/h (580 gpm). An additional 24 m^3/h (115 gpm) is required for mine, shop, and office water consumption.

18.2 Water Balance

An operational average monthly water balance model was developed for the HLF. The intent of the modeling was to estimate the magnitude and extent of any water surplus or deficit conditions in the HLF based on annual average climatic conditions. The modeling timeline was for 10 years of HLF operations. The model incorporates the following major project components:

- Heap Leach Pad
- Mine Usage
- Shop Usage
- General Usage
- Fresh Water Supply
- Pond Storage – PLS, ILS, Raffinate, and Storm

The findings of the water balance were that the HLF would operate in a water deficit. The deficit is most pronounced in the early years and is reduced as water stored within the ore is released from the earlier leaching stages. The total make-up required by the HLF is estimated at 10.8 million m^3 over the life of the facility. The HLF water requirement ranges from 940,000 m^3 to 1,200,000 m^3 annually. The project requires a significant amount of water at start up due to the initial ore wetting requirements and the solution retention in the heap. GRE estimates that approximately 150,000 m^3 of fresh water would be necessary at the start of heap operations.

The water balance was based on assumed moisture content values for the stacked ore and climatic conditions for the site. The model is sensitive to these values and they should be reviewed and confirmed for future design studies. The following criteria were employed in the water balance:

- Natural Moisture Content – Ore 5%
- Field Moisture Content – Ore 15%

- Drain-Down Final Moisture Content 10%
- Evaporation Losses – Irrigation 5%
- Average Irrigation Rate 12 lph/ m^2
- Pad Area – Phase 1,2,3 450,000, 765,000, 1,080,000 m^2
- Climate Conditions monthly temperature, precipitation and evaporation

18.3 Mine Facilities

GRE has provided conceptual design of facilities required for mine operations. These include access roads, offices, warehouses, shops, leach pad, and waste dumps.

19.0 MARKET STUDIES AND CONTRACTS

The Zonia project would produce copper cathode. A long-established, active, worldwide market exists for the buying and selling of copper. Cardero expects this to continue throughout the life of the Zonia project. Further market studies are not deemed necessary to establish the existence of a market for the product.

The base case copper price used was $3.00/lb, which is slightly less than the market price at the time of this study. The 3-year trailing average is $2.51/lb. GRE has provided sensitivity analysis from -40% ($1.85/lb) to + 50% ($4.63/lb). The price of copper has been rising, and GRE believes the $3.00/lb base case price reflects the consensus market forecast for copper.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Several documents were reviewed to provide an indication of the existing environmental conditions at the Zonia property near Prescott, Arizona. These reports included:

- Zonia Mine, Copper Oxide Deposit, Property Resource Summary, Steven R. Davis RPG. (2007)
- Technical Report on the Zonia Copper Deposit, Arizona, USA, prepared for Ste-Genevieve Resources Ltd., by Scott Wilson, Roscoe Postle Associates Inc. (RPA) (2006)
- Feasibility Study for the Zonia Property prepared by Western States Engineering (1995)
- APP Application prepared by Arimetco, Inc. (1995)
- APP Application prepared by Mining & Environmental Consultants, Inc. (2010)
- ADEQ Water Quality Division: Monitoring & Assessment (http://www.azdeq.gov/environ/water/assessment/index.html)

In October 1988, the Zonia Company acquired the property title to the Zonia Mine. In 1992, a lease agreement between the Zonia Company and Arimetco was in the process of being entered when the U.S. Environmental Protection Agency (EPA) cited the Zonia Company for violation of the Clean Water Act and ordered the company to perform certain measures to contain discharges. Arimetco agreed to conduct remediation activities under a Consent Order from the EPA which required them to do certain solution containment works, including a hydrological study, construction of pump-back wells and piping, installation of monitoring points, and on-going monitoring. Arimetco completed this work in January 1993.

Review of historical water quality data collected from 1993 through 2006 identified copper, manganese, zinc, and cadmium exceedances over Arizona drinking water standards at various locations throughout the site. Arizona's Integrated 305(b) Assessment and 303(d) Listing Report (reference Arizona Department of Environmental Quality [ADEQ] website) describes the status of surface water in relation to state water quality standards. According to Appendix B of the "2012/14 Status of Water Quality in Arizona 305(b) Assessment Report," French Gulch is located within the Middle Gila Watershed Hassayampa River Drainage Area (HUC 15070103-239). French Gulch (from its headwaters to the Hassayampa River) is listed as "Not Attaining" water quality standards. The causes of impairment are listed as copper, zinc, and cadmium, which were first listed in the water body in 1994. The Total Maximum Daily Load (TMDL) for this reach was completed in 2005. Earlier ADEQ nonpoint source annual reports reference the need to remediate mining issues at the Zonia Mine.

The Zonia property consists of private land (patented mineral claims and other lands purchased by Redstone) and unpatented mineral claims. The unpatented mineral claims are located on BLM-administered public lands. Under current mine planning, the initial years of mine operations could occur entirely on private land. With respect to maintaining mine activities on private land, this provides the advantage of avoiding a federal nexus that would require environmental analysis under the National Environmental Policy Act (NEPA). Permitting for an operation on private land will require the following major permits and certifications:

- ADEQ Aquifer Protection Permit (APP) (application submitted by Mining & Environmental Consultants, Inc. in November, 2010)
- ADEQ Air Quality Control Permit
- Arizona Pollutant Discharge Elimination System (AZPDES) permits (construction and Multi-Sector General Permit)
- ADEQ State of Arizona Clean Water Act Section 401 Water Quality Certification
- U.S. Army Corps of Engineers Section 404 Permit
- Landfill (Solid Waste Disposal)
- Arizona Department of Water Resources (ADWR) Dam Safety Permit to operate water containment structures over 25 feet high (the PLS ponds)
- Possible ADWR Surface Water Appropriation Permit
- Arizona State Mine Inspector Reclamation Plan.

The critical path for the first phase of permitting would be the APP. Cardero would also need to establish environmental baseline conditions to support design, permitting, operation, and closure of the Zonia Project.

No environmental fatal flaws that would materially impede the advancement of the project have been identified.

If future operations expand onto public land, a second phase of permitting would be required. In addition to modification of existing permits to account for the expanded scale of the operation, a Plan of Operations would be required by the BLM for operations on BLM land. NEPA analysis, either an Environmental Assessment (EA) or Environmental Impact Statement (EIS), would be required to consider impacts of the entire project, both on public and private lands, on the environment. Preparation and approval of the Plan of Operations and NEPA document with attendant environmental studies would control the critical path.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

The project would be developed over a period of 18 to 24 months following Feasibility Study and permitting. The initial capital costs shown in the economic model are all incurred in year -1. GRE expect s the additional time for exploration, engineering, permitting, and construction to be 3 – 5 years, resulting in start-up not before 2021.

21.1.1 Mining

Mining capital costs for the base case, which uses contractor operations, include development costs and owner support equipment, such as surveying equipment, computers, software, dispatch system, and radios. These costs were developed based on in-house expertise and are summarized in Table 21-1. Mining equipment would be provided by the contractor, resulting in no capital costs for the major equipment items. Costs for major equipment items are, however, included as operating costs distributed over the life of the project (see Section 21.2.1).

Table 21-1: Summary of Mining Capital Costs (1000s)

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Total Development	$1,150	$0	$0	$0	$0	$0	$0	$0	$0	$0	$1,150
Total Other Mining Equipment	$470	$76	$76	$76	$116	$126	$76	$76	$116	$76	$1,284
Total Mining Equipment Capital Costs	**$1,620**	**$76**	**$76**	**$76**	**$116**	**$126**	**$76**	**$76**	**$116**	**$76**	**$2,434**

21.1.2 Heap Leach and Copper Recovery

The preliminary design presented in this document is limited to the facilities and equipment required to establish a conventional acid copper heap leach facility at the Zonia Project. In general, these include the following:

- Site Improvements
- ROM Handling
- Tertiary Crushing Plant and Ancillaries
- Fine Ore Stockpile with Reclaim
- Agglomeration System
- Overland Conveyors
- Grasshopper Conveyors
- Radial Stacker
- Heap Leach Pad
- Solution Collection and Storage
- Irrigation System

- Solvent Extraction System
- Electrowinning Tank House
- Tank Farm
- Solution Pumps and Piping
- Reagents Storage, Preparation, and Distribution Systems

The following items are included in the scope of facilities:

- Land acquisition
- Water supply
- Power supply
- Ancillary facilities such as workshops, offices, laboratory, warehouse
- Owner's costs including project finance costs

For the purposes of this study, the heap leach feed rate is 10,000,000 tons per year (dry basis) operating 365 operating days per year, producing approximately 27,500 tons per annum of copper cathode (60.5 million pounds). The copper extraction and acid consumption is based on current laboratory results. In the absence of a more detailed engineering design and quotes for equipment, materials, and construction labor, a factored estimating technique has been employed.

Table 21-2: Summary of Heap Leach and Copper Recovery Capital Costs (1000s)

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Crushing	$8,827	$0	$0	$0	$0	$0	$0	$0	$0	$0	$8,827
Discharge Conveyor & Magnet	$6,309	$0	$0	$0	$0	$0	$0	$0	$0	$0	$6,309
Heap	$13,296	$0	$9,451	$0	$0	$17,128	$0	$0	$0	$0	$39,875
SX/EW	$23,459	$0	$0	$0	$0	$0	$0	$0	$0	$0	$23,459
Total Process Equipment Capital	**$51,892**	**$0**	**$9,451**	**$0**	**$0**	**$17,128**	**$0**	**$0**	**$0**	**$0**	**$78,470**

Indirect costs have been estimated using information provided by Infomine's Mining Cost Service Development Series for Leach Pad Construction Estimating and in-house expertise.

21.1.3 G&A

G&A capital costs include office facilities and owner costs, such as permitting, bonding, feasibility study, exploration and met testing, closure, warehouse, training, emergency vehicle and supplies, security, etc. the estimated G&A capital costs are summarized in Table 21-3. These costs were estimated using in-house expertise and prior experience.

Table 21-3: Summary of G&A Capital Costs (1000s)

	Year -1	Total
Diff. GPS – Survey	$55	**$55**
Guard House / Security	$110	**$110**
Startup Training	$1,278	**$1,278**
Emergency Vehicle/Supplies	$110	**$110**

	Year -1	Total
Office	$350	**$350**
Warehouse	$520	**$520**
Fire Protection	$500	**$500**
Water Supply	$2,000	**$2,000**
Power line to site	$2,500	**$2,500**
Substation (15 MW)	$2,000	**$2,000**
Electrical Switch Gear	$300	**$300**
Reclamation Bond	$3,750	**$6,450**
Permitting	$4,000	**$4,000**
Exploration and Met Testing	$2,000	**$2,000**
Feasibility Study	$2,000	**$2,000**
Closure	$0	**$10,000**
Total G&A Capital	**$21,473**	**$34,173**

21.1.4 Facilities

The facilities required include mining facilities (heavy equipment shop, dry, cap magazine and ANFO storage, and fuel station), plant facilities (SX building, EW building, and heap earthworks and infrastructure), and laboratory. Also included in the facilities costs are site-wide infrastructure (power to buildings, water to buildings, earthworks for buildings, security system), and engineering/management. These costs were estimated using InfoMine's Mining Cost Service and are summarized in Table 21-4.

Table 21-4: Summary of Facilities Capital Costs (1000s)

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Heap Earthwork and Infrastructure	$6,897	$0	$0	$3,448	$0	$0	$3,448	$0	$0	$0	**$13,793**
SX Building	$14,223	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$14,223**
EW Building	$17,067	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$17,067**
Mine Facilities	$2,115	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$2,115**
Site-Wide Infrastructure	$950	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$950**
Laboratory	$48	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$48**
Engineering/ Management	$17,278	$0	$0	$2,280	$0	$0	$2,280	$0	$0	$0	**$21,838**
Total Facilities	**$58,578**	**$0**	**$0**	**$5,728**	**$0**	**$0**	**$5,728**	**$0**	**$0**	**$0**	**$70,035**

21.1.5 Laboratory Equipment

Laboratory equipment capital costs include all equipment needed to operate the laboratory for the project. These costs were estimated using InfoMine's Mining Cost Service and in-house expertise and total approximately $516 thousand.

21.1.6 First Fills

Materials and reagents needed for initiating mining and production were estimated based on the quantity needed during the first month of operation for mining and first quarter of operation for processing. The estimated first-fill items and quantities total $9.8 million.

21.1.7 Loss Carry Forward

Cardero's auditors have identified $27.6 million of sunk costs that can be recovered early in the project life cash flow. These loss carry forwards have expiration dates beginning in 2026, well beyond the currently modeled time period the project generates cash flow. GRE has included credit for these loss carry forwards in year 1 and year 2.

21.1.8 Total Capital Costs

The total capital costs include all items identified above and include working capital, estimated as 3 months of operating costs, sustaining capital (estimated as 10% of mobile equipment costs per year, which for the base case is $0), and contingency, which was set to 20%. The total capital costs are summarized in Table 21-5.

Table 21-5: Summary of Total Capital Costs (1000s)

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Mine Capital	$1,620	$76	$76	$76	$116	$126	$76	$76	$116	$76	$2,434
Plant Capital	$51,892	$	$9,451	$	$	$17,128	$	$	$	$	$78,470
Facilities Capital	$58,578	$	$	$5,728	$	$	$5,728	$	$	$	$70,035
G&A Capital	$21,473	$300	$300	$300	$300	$300	$300	$300	$300	$10,300	$34,173
First Fills	$9,787	$	$	$	$	$	$	$	$	$	$9,787
Working Capital	$21,008	$	$	$	$	$	$	$	$	$	$21,008
Capital Contingency	$46,885	$685	$685	$685	$690	$694	$685	$685	$690	$147	$52,531
Total Capital Costs	**$211,243**	**$1,061**	**$10,512**	**$6,789**	**$1,106**	**$18,248**	**$6,789**	**$1,061**	**$1,106**	**$10,523**	**$268,438**

21.2 Operating Costs

21.2.1 Mining

Mining production equipment hours were estimated from the equipment productivity estimates (see Table 16-4), the scheduled leach and waste tonnage, and the number of pieces of equipment required. Table 21-6 summarizes the production equipment operating hour requirements.

Table 21-6: Summary of Mine Production Equipment Operating Hours

Item	Maximum Hours per Year for Fleet	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Haul Truck 789D	24,898	3,211	21,759	43,076	23,812	20,008	20,656	21,171	19,320	17,857	198,508
Loader 993K	6,224	803	5,440	10,769	5,953	5,002	5,164	5,293	4,830	4,464	49,627
Blast Hole Drill	6,224	64	4,068	6,224	4,418	3,727	3,862	3,958	3,608	3,264	33,727

Mining support equipment hours were calculated from the number of pieces of equipment times the operating hours per day, assuming utilization of 90% and availability of 95%, times the operating days per year. Table 21-7 summarizes the support equipment operating hour requirements.

Table 21-7: Summary of Mine Support Equipment Operating Hours

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Dozer w/Ripper D10T (pit/dump)	1,319	12,449	12,449	12,449	12,449	12,449	12,449	12,449	12,449	106,835
Dozer D8T (leach)	660	6,224	6,224	6,224	6,224	6,224	6,224	6,224	6,224	53,418
Dozer (rubber tired) 844k	660	6,224	6,224	6,224	6,224	6,224	6,224	6,224	6,224	53,418
Loader (crush) 992K	660	6,224	6,224	6,224	6,224	6,224	6,224	6,224	6,224	53,418
Grader 16ft	660	6,224	6,224	6,224	6,224	6,224	6,224	6,224	6,224	53,418
Water Truck 10,000 gal	660	6,224	6,224	6,224	6,224	6,224	6,224	6,224	6,224	53,418
Service/Tire Truck	660	6,224	6,224	6,224	6,224	6,224	6,224	6,224	6,224	53,418
ANFO Truck	330	3,112	3,112	3,112	3,112	3,112	3,112	3,112	3,112	26,709
Light Plant 10 kw	1,319	12,449	12,449	12,449	12,449	12,449	12,449	12,449	12,449	106,835
Pump (submersible)	165	1,556	1,556	1,556	1,556	1,556	1,556	1,556	1,556	13,354
Pickup Truck	1,649	15,561	15,561	15,561	15,561	15,561	15,561	15,561	15,561	133,544

Blasting materials requirements were determined in a drilling and blasting schedule that used the parameters and assumptions detailed in Table 21-8. The drilling and blasting requirements are summarized in Table 21-9.

Table 21-8: Parameters for Drilling and Blasting Calculations

Constants		
ANFO density	0.76	ton/yd^3
ANFO powder factor - leach	0.45	lb/ton rock
ANFO powder factor - waste	0.36	lb/ton rock
Bench height	15	ft
Drilling rate	1.97	ft/minute
Drill available %	0.9	
Minutes used/hour	50	minutes/hour
Available labor hours per shift	9	hour
Blasthole depth	19.69	feet
Blasthole diameter	10.66	inch
Rod length	45	feet
ANFO thickness	16.40	feet
ANFO setup min/hole	15	minutes
Calculated		
Blasthole diameter	0.89	feet
Blasthole volume	12.20	ft^3
ANFO volume	10.17	ft^3
ANFO weight tons	0.286	tons
Tons of rock blasted/hole	1,259.4	tons/hole
Volume of rock blasted/hole	15,742.1	ft^3/hole
Drillhole grid spacing	32.4	feet

Table 21-9: Summary of Drilling and Blasting Requirements

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Number drillholes per year -- leach	25	8,708	8,519	8,566	8,629	8,671	8,671	8,659	8,438	3,168
Number drillholes per year -- waste	1,838	3,918	16,324	5,146	2,938	3,314	3,613	2,540	1,691	304
Number of Drill Rigs Required	1	1	2	1	1	1	1	1	1	1
Number of Drilling Crews Required	1	2	3	2	2	2	2	2	2	1
Number of ANFO Trucks	1	1	1	1	1	1	1	1	1	1
ANFO Truck workers	1	1	1	1	1	1	1	1	1	1
ANFO consumption - leach (1000 lb/yr)	14.4	4,974	4,866	4,893	4,929	4,953	4,953	4,946	4,820	1,810
ANFO consumption - waste (1000 lb/yr)	840	1,791	7,460	2,352	1,343	1,515	1,651	1,161	773	139
Caps (1000s)	1.9	13	25	14	12	12	12	11	10	3
Primers (1000s)	2.2	13	25	14	12	12	12	11	10	4

Mine operating costs include production and support equipment. Costs were based on InfoMine's Mining Cost Service hourly operating cost data and include overhaul parts, maintenance parts, diesel, electric power, lube, tires, and wear parts, where applicable. In addition, a 20% markup for contractor overhead and profit was applied. The estimated mine operating costs are summarized in Table 21-10.

Table 21-10: Summary of Mine Operating Costs (1000s)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Haul Truck 789D	$856	$6,164	$10,932	$6,623	$5,772	$5,917	$6,033	$5,619	$5,292	$2,326	$55,534
Loader 993K	$276	$2,025	$3,476	$2,165	$1,906	$1,950	$1,985	$1,859	$1,760	$779	$18,183
Blast Hole Drill	$26	$732	$1,035	$782	$684	$703	$717	$668	$619	$151	$6,118
Dozer w/Ripper (pit/dump) D10T	$69	$647	$647	$647	$647	$647	$647	$647	$647	$308	$5,552
Dozer (leach) D8T	$58	$552	$552	$552	$552	$552	$552	$552	$552	$263	$4,734
Dozer (rubber tired) 844K	$126	$1,185	$1,185	$1,185	$1,185	$1,185	$1,185	$1,185	$1,185	$564	$10,172
Loader (crush) 992K	$139	$1,315	$1,315	$1,315	$1,315	$1,315	$1,315	$1,315	$1,315	$626	$11,284
Grader	$101	$958	$958	$958	$958	$958	$958	$958	$958	$456	$8,218
Water Truck	$57	$540	$540	$540	$540	$540	$540	$540	$540	$257	$4,634
Service/Tire Truck	$14	$130	$130	$130	$130	$130	$130	$130	$130	$62	$1,115
ANFO Truck	$17	$163	$163	$163	$163	$163	$163	$163	$163	$78	$1,401
Light Plants	$3	$31	$31	$31	$31	$31	$31	$31	$31	$15	$269
Pumps (submersible)	$1	$10	$10	$10	$10	$10	$10	$10	$10	$5	$90
Pickup Truck	$37	$348	$348	$348	$348	$348	$348	$348	$348	$166	$2,988
Total Mine Operating Costs	**$1,781**	**$14,801**	**$21,323**	**$15,449**	**$14,242**	**$14,450**	**$14,614**	**$14,025**	**$13,550**	**$6,054**	**$130,291**

Mine manpower costs include hourly and salaried labor and a payroll burden of 35%. Labor rates were estimated based on in-house knowledge and data provided by third-party sources and are typical of the mining industry. Mine labor and supervision costs are summarized in Table 21-11.

Table 21-11: Summary of Mine Labor and Supervision Costs (1000s)

Item	Quantity at Steady State	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Hourly												
Driller	8	$41	$780	$780	$780	$780	$780	$780	$780	$780	$186	$6,566
Blaster	2	$21	$195	$195	$195	$195	$195	$195	$195	$195	$93	$6,468
Excavator/Loader Operator	8	$41	$390	$780	$390	$390	$390	$390	$390	$390	$186	$1,674
Truck Driver	16	$41	$1,560	$2,730	$1,560	$1,560	$1,560	$1,560	$1,560	$1,170	$371	$3,738
Grader/Dozer Operator	6	$62	$585	$585	$585	$585	$585	$585	$585	$585	$278	$13,675
Water Truck Operator	2	$21	$195	$195	$195	$195	$195	$195	$195	$195	$93	$5,021
Mechanic	23	$343	$3,235	$3,235	$3,235	$3,235	$3,235	$3,235	$3,235	$3,235	$1,540	$1,674
Laborer/ Maintenance	2	$14	$135	$135	$135	$135	$135	$135	$135	$135	$64	$27,760
Salaried												
Mine Superintendent	1	$19	$176	$176	$176	$176	$176	$176	$176	$176	$84	$1,506
Foreman	2	$31	$297	$297	$297	$297	$297	$297	$297	$297	$141	$2,549
Maintenance Foreman	2	$31	$297	$297	$297	$297	$297	$297	$297	$297	$141	$2,549
Engineer	2	$24	$230	$230	$230	$230	$230	$230	$230	$230	$109	$1,970
Geologist	2	$23	$216	$216	$216	$216	$216	$216	$216	$216	$103	$1,854
Surveyor/ Technician	4	$37	$351	$351	$351	$351	$351	$351	$351	$351	$167	$3,012
Total Mine Labor Costs		**$751**	**$8,641**	**$10,202**	**$8,641**	**$8,641**	**$8,641**	**$8,641**	**$8,641**	**$8,251**	**$3,556**	**$74,607**

Mine consumables include ANFO, caps, primers, and snap line. Estimated costs for these items were estimated based on InfoMine's Mining Cost Service data and in-house knowledge and are summarized in Table 21-12.

Table 21-12: Summary of Mine Consumables Costs (1000s)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
ANFO	$543	$4,302	$7,839	$4,608	$3,989	$4,114	$4,200	$3,884	$3,557	$1,239	$38,277
Caps	$5	$35	$69	$38	$32	$33	$34	$31	$28	$10	$317
Primers	$13	$76	$151	$83	$70	$72	$74	$68	$61	$22	$690
Snap Lines	$18	$121	$238	$131	$111	$115	$118	$107	$97	$33	$1,090
Total Mine Consumables Costs	**$580**	**$4,535**	**$8,297**	**$4,861**	**$4,202**	**$4,334**	**$4,427**	**$4,090**	**$3,744**	**$1,304**	**$40,373**

21.2.2 Heap Leach and Copper Recovery

The crushing circuit is designed to operate on a 5 day per week basis with two 8-hour shifts per day. The benefit of this work schedule is that it allows sufficient time for maintenance and provides additional capacity for production shortfall catch up. The agglomeration and stacking schedule is designed to match the crushing circuit, and the use of a fine ore stockpile decouples the two circuits to provide additional flexibility. Table 21-13 shows the operating cost for the Zonia heap leach.

Table 21-13: Summary of Heap Leach and Copper Recovery Operating Costs (1000s)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Crushing	$279	$2,630	$2,630	$2,630	$2,630	$2,630	$2,630	$2,630	$2,630	$1,252	$22,571
Discharge Conveyor & Magnet	$132	$1,248	$1,248	$1,248	$1,248	$1,248	$1,248	$1,248	$1,248	$594	$10,711
Heap	$51	$480	$480	$480	$480	$480	$480	$480	$480	$229	$4,121
SX/EW	$52	$486	$486	$486	$486	$486	$486	$486	$486	$232	$4,175
Total Process Operating Costs	**$514**	**$4,845**	**$4,845**	**$4,845**	**$4,845**	**$4,845**	**$4,845**	**$4,845**	**$4,845**	**$2,306**	**$41,578**

Labor and Supervision: A payroll burden of 35% of basic salary has been assumed as per typical practice. Labor rates were estimated based on in-house knowledge and data provided by third-party sources and are typical of the mining industry. Labor and supervision costs are summarized in Table 21-14.

Table 21-14: Summary of Heap Leach and Copper Recovery Labor Costs (1000s)

Item	Qty	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Metallurgical Staff												
Superintendent	1	$19	$176	$176	$176	$176	$176	$176	$176	$176	$84	$1,506
General Foreman	1	$16	$149	$149	$149	$149	$149	$149	$149	$149	$71	$1,274
Maintenance Foreman	1	$16	$149	$149	$149	$149	$149	$149	$149	$149	$71	$1,274
Shift Foreman	4	$46	$432	$432	$432	$432	$432	$432	$432	$432	$206	$3,707
Chief Assay Chemist	1	$11	$101	$101	$101	$101	$101	$101	$101	$101	$48	$869
Senior Metallurgist	2	$26	$243	$243	$243	$243	$243	$243	$243	$243	$116	$2,085
Metallurgist	2	$23	$216	$216	$216	$216	$216	$216	$216	$216	$103	$1,854
Process Technician	3	$21	$203	$203	$203	$203	$203	$203	$203	$203	$96	$1,738
Instrument Technician	3	$21	$203	$203	$203	$203	$203	$203	$203	$203	$96	$1,738
Crusher												
Operator	6	$52	$486	$486	$486	$486	$486	$486	$486	$486	$231	$4,171
FEL Operator	2	$17	$162	$162	$162	$162	$162	$162	$162	$162	$77	$1,390
Maintenance	2	$20	$189	$189	$189	$189	$189	$189	$189	$189	$90	$1,622
Electrician	1	$10	$95	$95	$95	$95	$95	$95	$95	$95	$45	$811
Heap												
Stacking	3	$26	$243	$243	$243	$243	$243	$243	$243	$243	$116	$2,085
Agglomeration	3	$26	$243	$243	$243	$243	$243	$243	$243	$243	$116	$2,085
Irrigation Operator	3	$26	$243	$243	$243	$243	$243	$243	$243	$243	$116	$2,085
Reagent Operator	3	$18	$170	$170	$170	$170	$170	$170	$170	$170	$81	$1,460
FEL Operator	2	$17	$162	$162	$162	$162	$162	$162	$162	$162	$77	$1,390
Assayers	6	$52	$486	$486	$486	$486	$486	$486	$486	$486	$231	$4,171
Samplers	3	$14	$132	$132	$132	$132	$132	$132	$132	$132	$63	$1,130
Reagent Operator	3	$14	$132	$132	$132	$132	$132	$132	$132	$132	$63	$1,130
Mechanic	3	$30	$284	$284	$284	$284	$284	$284	$284	$284	$135	$2,433
Electrician	2	$20	$189	$189	$189	$189	$189	$189	$189	$189	$90	$1,622
SX/EW												
SX Operator	3	$26	$243	$243	$243	$243	$243	$243	$243	$243	$116	$2,085
EW Operator	3	$26	$243	$243	$243	$243	$243	$243	$243	$243	$116	$2,085
Cathode Stripping	6	$52	$486	$486	$486	$486	$486	$486	$486	$486	$231	$4,171
Samplers	3	$14	$132	$132	$132	$132	$132	$132	$132	$132	$63	$1,130
Reagent Operator	3	$14	$132	$132	$132	$132	$132	$132	$132	$132	$63	$1,130

Item	Qty	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Mechanic	2	$20	$189	$189	$189	$189	$189	$189	$189	$189	$90	$1,622
Electrician	2	$20	$189	$189	$189	$189	$189	$189	$189	$189	$90	$1,622
Total Plant Labor	**82**	**$710**	**$6,697**	**$6,697**	**$6,697**	**$6,697**	**$6,697**	**$6,697**	**$6,697**	**$6,697**	**$3,188**	**$57,477**

Electrical Energy: The total connected load of is 17,850 kilowatts (kW), and consumes 121,200 megawatt-hours (MWh) per year (the majority of which is employed for cathode production). The assumed electrical energy unit cost is US$ 55/MWh. The power costs are summarized in Table 21-15.

Table 21-15: Summary of Heap Leach and Copper Recovery Power Costs (1000s)

Item	Installed kW	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Crushing	1,818	$39	$369	$369	$369	$369	$369	$369	$369	$369	$3,164	$3,164
Heap	3,361	$130	$1,231	$1,231	$1,231	$1,231	$1,231	$1,231	$1,231	$1,231	$10,563	$10,563
SX/EW	12,250	$524	$4,941	$4,941	$4,941	$4,941	$4,941	$4,941	$4,941	$4,941	$42,402	$42,402
Total Power Costs		**$693**	**$6,540**	**$6,540**	**$6,540**	**$6,540**	**$6,540**	**$6,540**	**$6,540**	**$6,540**	**$3,113**	**$56,130**

Reagents and Consumables: These include consumables and reagents utilized in crushing, heap leaching, solution purification, solvent extraction, and electrowinning. Reagent costs are summarized in Table 21-16. Water has been assumed to be available. No costs have been included for water rights. Annual costs for maintenance parts and materials were estimated at an assumed amount equal to 1.5% of the capital cost of the mechanical equipment. This is for supplies beyond those included in the operating cost.

Table 21-16: Summary of Heap Leach and Copper Recovery Reagent and Consumable Costs (1000s)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Sec Crusher	$40	$380	$380	$380	$380	$380	$380	$380	$380	$181	$3,261
Heap	$984	$9,282	$9,282	$9,282	$9,282	$9,282	$9,282	$9,282	$9,282	$4,418	$79,658
SX/EQ	$360	$3,397	$3,397	$3,397	$3,397	$3,397	$3,397	$3,397	$3,397	$1,617	$29,151
Plant Consumables	$215	$2,028	$2,028	$2,028	$2,028	$2,028	$2,028	$2,028	$2,028	$965	$17,407
Total Reagent and Consumables Costs	**$1,599**	**$15,087**	**$15,087**	**$15,087**	**$15,087**	**$15,087**	**$15,087**	**$15,087**	**$15,087**	**$7,181**	**$129,478**

Qualifications and Exclusions: Laboratory test work has provided the basis for a preliminary process and plant design. Continued work is necessary for the scale-up and verification of these numbers. The results of such work may necessitate changes to the assumptions on which the present operating requirement estimates are based.

21.2.3 G&A

Administrative operating costs include administrative labor, such as general manager, personnel manager, and environmental, purchasing, and accounting personnel, and services and supplies, such as office supplies, insurances, training, legal costs, etc. These costs were estimated based on in-house knowledge and are summarized in Table 21-17.

Table 21-17: Summary of General and Administrative Operating Costs (1000s)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
G&A Labor	$180	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$809	$14,586
Services and Supplies	$180	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$809	$14,593
Total G&A Operating Costs	**$360**	**$3,400**	**$3,400**	**$3,400**	**$3,400**	**$3,400**	**$3,400**	**$3,400**	**$3,400**	**$1,618**	**$29,179**

21.2.4 Transportation

Transportation costs of the copper cathode were estimated based on in-house knowledge and were applied to the economic model. Further discussion of these costs can be found in Section 22.0.

21.2.5 Total Operating Costs

The total operating costs include all items identified above and include contingency, which was set to 10%. The total operating costs are summarized in Table 21-18.

Table 21-18: Summary of Total Operating Costs (1000s)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Mine Op	$3,111	$27,977	$39,822	$28,951	$27,086	$27,425	$27,682	$26,756	$25,545	$10,915	$245,271
Process Op	$3,516	$33,170	$33,170	$33,170	$33,170	$33,170	$33,170	$33,170	$33,170	$15,788	$284,662
Admin	$360	$3,400	$3,400	$3,400	$3,400	$3,400	$3,400	$3,400	$3,400	$1,618	$29,179
Operating Contingency	$699	$6,455	$7,639	$6,552	$6,366	$6,400	$6,425	$6,333	$6,211	$2,832	$55,911
Total Operating Costs	**$7,686**	**$71,002**	**$84,031**	**$72,073**	**$70,021**	**$70,395**	**$70,677**	**$69,659**	**$68,326**	**$31,154**	**$615,024**

22.0 ECONOMIC ANALYSIS

Readers are advised that mineral resources that are not mineral reserves do not have demonstrated economic viability under National Instrument 43-101. This PEA is preliminary in nature and 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 under National Instrument 43-101. Readers are advised that there is no certainty that the results projected in this preliminary economic assessment will be realized.

22.1 Economic Model

GRE has constructed a discounted cash flow economic model for the Zonia project. The model evaluates the project on a standalone basis, as if a decision to proceed were obtained following completion of a Feasibility Study. The model assumes capital costs are incurred in year -1 and includes loss carry forward of amounts Cardero auditors provided to GRE. The project will require three to five years of exploration, metallurgical testing, permitting, feasibility study, and development to achieve production. Costs for these activities have been included as an aggregate cost in year -1.

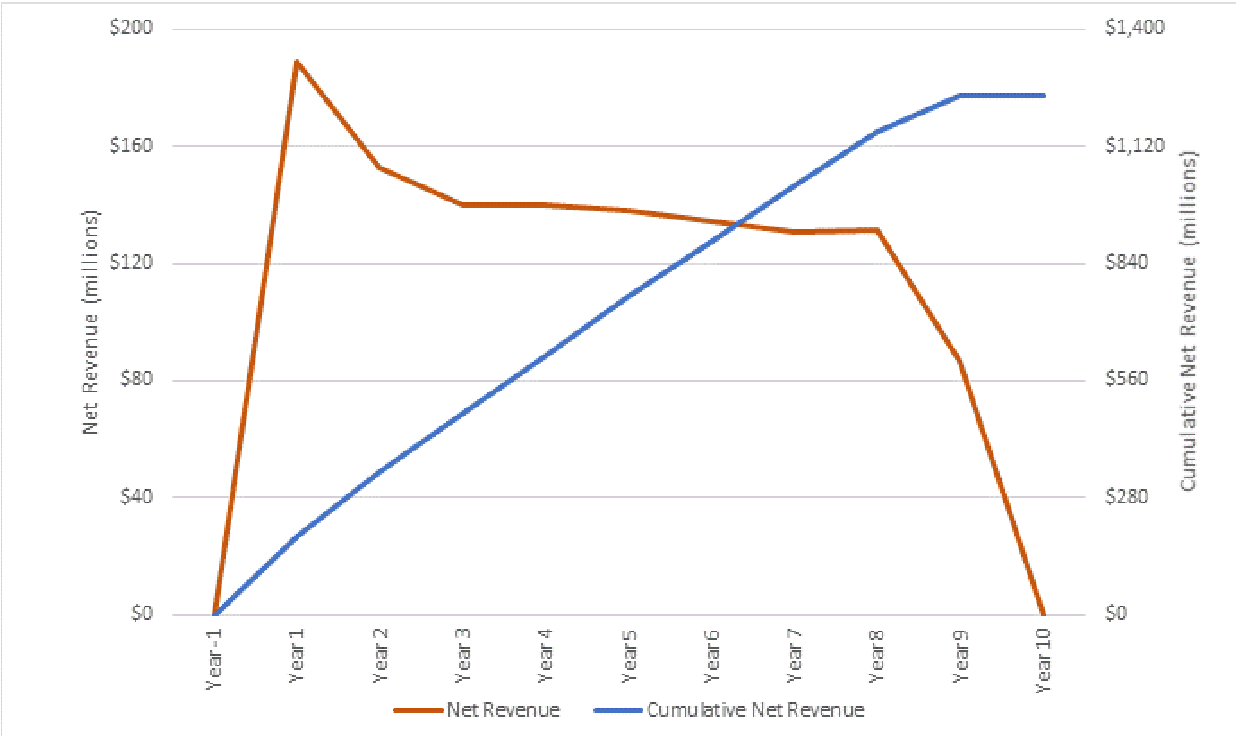
The economic model uses recoveries of 73% for oxide material and 70% for mixed (or transition) material. Sulfide materials were given no recovery. All material categorized within the block model as mineralized, i.e., all measured, indicated, and inferred categorized material, was used in developing the economic model.

To generate the economic model, GRE calculated revenues for the recovered copper using a copper price of $3.00/lb, refining charges of $0.032/lb, and transportation charges of $0.10/ton-mile and assuming an 80-mile transport to Phoenix, AZ. A ramp up was applied to year one, delaying 10% of the revenue that year and recovering it in year 9. The base case recoveries and revenues are summarized in Table 22-1, and the revenues are illustrated in Figure 22-1.

Table 22-1: Base Case Copper Recovery and Revenues (millions)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Recovered Oxide Copper	$71.3	$51.0	$46.9	$47.1	$46.8	$45.7	$44.2	$43.6	$17.6	$71.3	$414.2
Recovered Mixed Copper	$0.0	$0.8	$0.6	$0.3	$0.0	$0.0	$0.0	$0.9	$4.7	$0.0	$7.3
Payable Copper	$71.0	$51.6	$47.3	$47.2	$46.6	$45.5	$44.0	$44.3	$22.1	$71.0	$419.4
Gross Revenue	$212.9	$154.7	$141.8	$141.5	$139.8	$136.4	$132.1	$132.8	$66.3	$212.9	$1,258.2
Refining	-$2.3	-$1.7	-$1.5	-$1.5	-$1.5	-$1.5	-$1.4	-$1.4	-$0.7	-$2.3	-$13.4
Transport	-$0.3	-$0.2	-$0.2	-$0.2	-$0.2	-$0.2	-$0.2	-$0.2	-$0.1	-$0.3	-$1.7
Revenue Delay	-$21.3	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$21.3	$0.0
Net Revenue	$189.0	$152.8	$140.1	$139.8	$138.1	$134.8	$130.5	$131.2	$86.8	$189.0	$1,243.1

Figure 22-1: Base Case Copper Revenues



Operating costs were deducted from Net Revenue, yielding before-tax cash flow. Taxes were applied as follows:

- Depreciation of facilities capital costs was calculated on a straight-line, 10-year basis. Depletion allowance was calculated as 15% of revenues up to a maximum of 50% of before-tax income minus depreciation. Depreciation and depletion were deducted from before-tax cash flow to obtain taxable income.
- Federal tax at 21% was applied to the taxable income and Arizona severance tax at 7% was applied to the taxable income. The taxes were deducted from the taxable income, then the depreciation and depletion allowance were added back from taxable income to obtain after-tax cash flow.
- There are no royalties associated with this project.

Capital costs were deducted from the after-tax cash flow to obtain net cash flow after taxes. NPV at discount rates of 6%, 8%, and 10% and IRR were calculated from the net cash flow after taxes. Table 22-2 summarizes the tax calculations and after-tax cash flow, and Table 22-3 shows the economic model results; the entire economic model is provided in Appendix C.

Table 22-2: Summary of Tax Calculations (millions)

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Before Tax Cash Flow	-$7.7	$118.0	$68.8	$68.0	$69.7	$67.7	$64.1	$60.9	$62.9	$55.7	$628.1
Depreciation	$0.0	$19.8	$21.0	$21.7	$21.8	$23.9	$24.6	$24.7	$24.7	$26.0	$0.0
Loss Carry Forward	$0.0	-$7.7	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Depletion Allowance	$0.0	$28.4	$22.9	$21.0	$21.0	$20.7	$19.7	$18.1	$19.1	$13.0	$183.9
Taxable Income	$0.0	$62.2	$24.9	$25.2	$27.0	$23.1	$19.7	$18.1	$19.1	$16.7	$235.9

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total
Federal Tax	$0.0	$13.1	$5.2	$5.3	$5.7	$4.8	$4.1	$3.8	$4.0	$3.5	$49.5
State Tax	$0.0	$4.4	$1.7	$1.8	$1.9	$1.6	$1.4	$1.3	$1.3	$1.2	$16.5
Add Back Depreciation	$0.0	$19.8	$21.0	$21.7	$21.8	$23.9	$24.6	$24.7	$24.7	$26.0	$208.3
Add Back Depletion	$0.0	$28.4	$22.9	$21.0	$21.0	$20.7	$19.7	$18.1	$19.1	$13.0	$183.9
After Tax Cash Flow	-$7.7	$100.6	$61.8	$60.9	$62.2	$61.2	$58.6	$55.8	$57.6	$51.0	$562.1

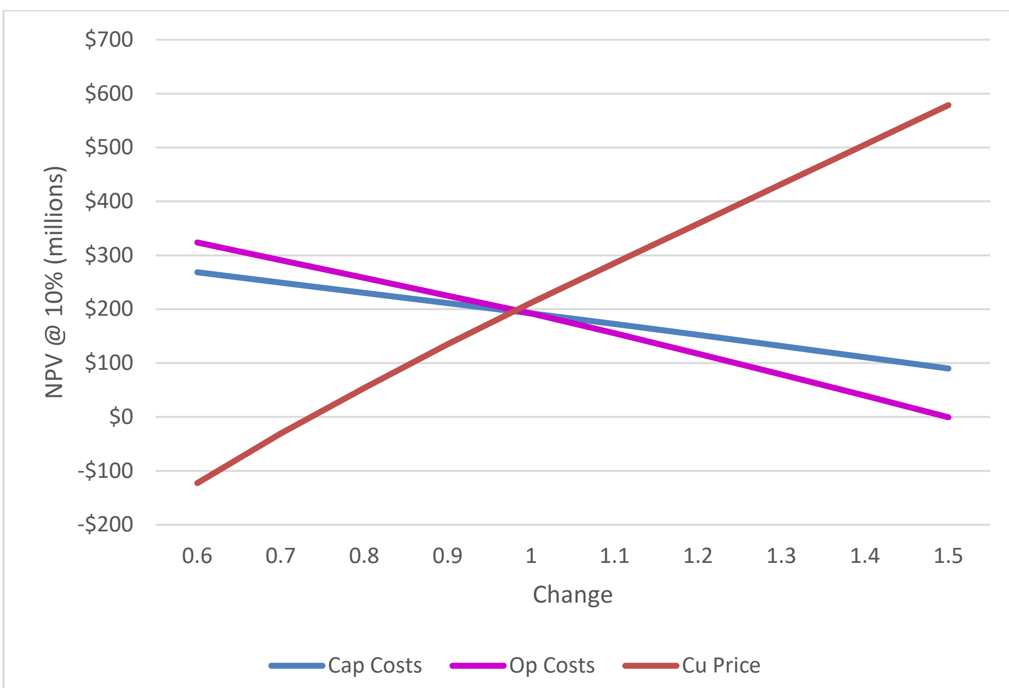
Table 22-3: Economic Model Results

Item	Result
NPV@6%	$225 million
NPV@8%	$192 million
NPV@10%	$163 million
IRR	29.0%
Initial Capital	$198 million
Cumulative Net Cash Flow After Taxes	$331 million
Payback Period	2.89 years
Op Cost/Lb	$1.46
All in Cost/Lb	$2.06

22.2 Sensitivity Analysis

GRE evaluated the after-tax NPV@8% sensitivity to changes in copper price, capital costs, and operating costs. The results are shown in Figure 22-2.

Figure 22-2: NPV@8% Sensitivity to Changes in Copper Price, Capital Costs, and Operating Costs



The results in Figure 22-2 indicate that after-tax NPV@8% is least sensitive to changes in capital costs, ranging from $268.4 million at 60% of the base case capital costs to $89.8 million at 150% of the base case capital costs, or approximately $20 million for every 10% change in capital costs. The after-tax NPV@8% stays positive for the full range of capital costs examined.

The after-tax NPV@8% is moderately sensitive to changes in operating costs, ranging from $323.7 million at 60% of the base case operating costs to -$0.6 million at 150% of the base case operating costs, or approximately $36 million for every 10% change in operating costs. The after-tax NPV@8% goes negative when the operating costs increase to approximately 150% of the base case costs.

The after-tax NPV@8% is most sensitive to changes in copper price, ranging from -$122.7 million at 60% of the base case copper price to $578.6 million at 150% of the base case copper price, or approximately $78 million for every 10% change in copper price. The after-tax NPV@8% goes negative when the copper price dips to approximately 51% of the base case copper price ($1.52/lb).

22.3 Conclusions of Economic Model

The base case project scenario produces 92.6 million tons of leachable material over an 8.6-year mine life.

At a copper price of $3.00/lb, the project shows an after-tax NPV@6% of $225 million, an NPV@8% of $192 million, an NPV@10% of $163 million, and an IRR of 29.0%. The project payback period is 2.89 years. The project is most sensitive to copper price, then operating costs, then capital costs.

The project appears to be economically viable using a combination of open pit mining and sulfuric acid heap leach methods. The economic indicators suggest that further development of the project is warranted.

23.0 ADJACENT PROPERTIES

Scott Wilson RPA (2006) reported that there are a number of patented claims which adjoin the Zonia property to the southwest. The ownership of these claims was not able to be determined. GRE is unaware of any other claims near the Zonia Copper Project site.

24.0 OTHER RELEVANT DATA AND INFORMATION

Section 27, References, provides a list of documents that were consulted in support of the PEA. No further data or information is necessary, in the opinion of the Authors, to make the PEA understandable and not misleading.

25.0 INTERPRETATION AND CONCLUSIONS

Economic analysis of the Zonia project indicate a positive NPV@8% of $185 million with an initial capital cost of $190 million. The project generates 289,000 tons of copper and has an IRR of 29%.

Current test work shows the deposit is amenable to heap leaching with sulfuric acid, producing good copper recoveries with moderate acid consumption.

GRE has determined that conventional open pit mining is the most suitable method for this deposit. Contract mining has higher operating costs, but reduces upfront capital.

There appears to be the potential to expand the current resource into a reserve in areas adjacent to and below the current resource area.

GRE recommends the following budget to advance the project to a PFS.

Exploration/Infill/Below 4000 – 50 drill holes average 500 feet @ $100/ft =	$2,500,000
Metallurgical Test Work/Large Column	$500,000
Permitting	$500,000
PFS	$500,000
Owners Cost	$1,000,000
Total	$5,000,000

26.0 RECOMMENDATIONS

The Zonia project needs additional information related to the following:

- *In situ* density determination/validation for each mineral type
- Expanded test work on optimization of particle size on heap leach recovery
- Confirm heap scale up parameters using bulk sampling and large particle size column testing
- The resource model currently limits mineralization to elevations above 4000 feet while geologic models and drillhole data show oxide and transition mineralization extending below that elevation. Deeper infill drilling and resource modeling below the 4000-foot elevation is recommended.
- More detailed mine designs are required to optimize the production schedule. Further, alternative annual production schedules should be investigated.
- Additional geotechnical information is needed for confirmation of pit slopes, heap leach set backs, and plant site placement and design.
- Investigate alternatives for acid supply and on-site acid production.

27.0 REFERENCES

Anderson, C. A. 1968. Arizona and Adjacent New Mexico. [ed.] J. D. Ridge. *Ore Deposits of the United States, 1933 - 1967 (Graton-Sales Volume).* 1968, Vol. 2, pp. 1163-1190.

Anderson, J. A. 1982. Characteristics of Leached Capping and Techniques of Appraisal. [ed.] S. R. Titley. *Advances in Geology of the Porphyry Copper Deposits, Southwestern North America.* s.l. : University of ARizona Press, 1982, pp. 275-295.

Anderson, P. 1989. Stratigraphic Framework, Volcanic-Plutonic Evolution, and Vertical Deformation of the Proterozoic Volcanic Belts of Central Arizona. [ed.] J. P. Jenny and S. J. Reynolds. *Geologic Evolution of Arizona, Arizona Geological Society Digest.* 1989, Vol. 17, pp. 57-147.

Arimetco, Inc. 1995. *Aquifer Protection Permit Application, Zonia Mine, Yavapai county, Arizona.* 1995.

Baker III, A. and Clayton, R. L. 1968. Massive Sulfide Deposits of the Bagdad District, Yavapai County, Arizona. [ed.] J. D. Ridge. *Ore Deposits of the United States, 1933 - 1967 (Graton-Sales Volume).* 1968, Vol. 2.

Blanchard, R. 1968. Interpretation of Leached Outcrops. *Nevada Bureau of Mines Bulletin.* 1968, 66.

Cameron, Jonathan. 1975. *Homestake Mining Company, Final Report on the Zonia Copper Deposit, Yavapai County, Arizona, April 22.* 1975.

Chavez, William X. 2000. Supergene Oxidation of Copper Deposits: Zoning and Distribution of Copper Oxide Minerals. *Society of Economic Geologists Newsletter.* 2000, 41.

CIM. 2010. *Definition Standards for Mineral Resources and Mineral Reserves.* s.l. : CIM Standing Committe on Reserve Definitions, 2010.

Donnelly, M. E. and Hahn, G. A. 1991. A Review of the Precambian Volcanogenic Massive Sulfide Deposits in Central Arizona and the Relationship to their Depositional Environment. [ed.] W. R. Dickinson and W. D. Payne. *Relations of Tectonics to Ore Deposits in the Southern Cordillera.* Arizona Geological Society Digest, 1991, Vol. 14, pp. 11-21.

Franklin, J. M., et al. 2005. Volcanogenic Massive Sulfide Deposits. [ed.] J.W. Hedenquist, et al. *Economic Geology 100th Anniversay Volume 1905-2005.* 2005, pp. 523-560.

Gifkins, C. C., Herrmann, W. and Large, R. R. 2005. *Altered Volcanic Rocks: A Guide to Description and Interpretation.* s.l. : Centre for Ore Deposit Research, Univeristy of Tasmania, 2005.

Homestake Mining Company. 1975. *Final Report on the Zonia Copper Deposit, Yavapai County, Arizona.* 1975.

Johnson, W. P. and Lowell, J. D. 1961. Geology and Origin of Mineralized Breccia Pipes in Copper Basin, Arizona. *Economic Geology.* 1961, 56, pp. 916-940.

Karlstrom, K. E. and Bowring, S. A. 1991. Styles and Timing of Early Proterozoic Deformation in Arizona: Constraints on Tectonic Models. *Arizona Geological Digest.* 1991, Vol. 19.

Leach, Inc. 1995. *Zonia Project Column Leack Tests, Zonia Project Feasibility Study, Arimetco Inc.* 1995.

Locke, A. 1926. *Leached Outcrops as Guides to Copper Ore.* Baltimore : Williams and Wilkins, 1926.

Lupo, J. F. and Morrison, K. F. 2005. Innovative Geosynthetic Liner Design Approaches and Construction in the Mining Industry. *Proceedings of the ASCE Geo-Frontiers, Austin, TX, 24-26 January.* 2005, p. 16.

McAlester. 1979. *Zonia Copper Mine, Yavapai County, Arizona.* 1979.

METCON Research. 2008. *Zonia Project, Column Leach Study on Surface Bulk Samples.* 2008.

—. 2011. *Zonia Project, Locked Cycle Column Leach Testing on Composite Samples.* 2011.

Mining & Environmental Consultants, Inc. 2010. *Aquifer Protection Permit Application, Zonia Mine, Arizona.* 2010.

Mintec. 2001. *Preliminary Resource and Reserve Estimate - Zonia Project.* 2001.

MRA. 1994. *Ore Reserve Estimate and Mine Plan for the Zonia Project, Yavapai County: Report prepared for Arimetco International, inc.* 1994.

Queenstake Resources. 1983. *Preliminary REport on the Zonia Mine Yavapai County, Arizona.* 1983.

Schwartz, G. M. 1934. Paragenesis of the Oxidized Ores of Copper. *Economic Geology.* 1934, Vol. 29, pp. 55-75.

Scott Wilson RPA. 2006. *Technical Report on the Zonia Copper Deposit, Arizona, U.S.A., NI 43-101 Report Prepared for Ste-Genevieve Resources Ltd.* Roscoe Postle Associates Inc. 2006.

Tetra Tech. 2017. *Zonia Copper Project NI 43-101 Technical Report, Yavapai County, Arizona USA, Draft Amended.* 2017. October.

Western States Engineering. 1995. *Zonia Project Feasibility Study Summary: Report for Arimetco, Inc.* 1995.

APPENDIX A
LIST OF PATENTED AND UNPATENTED MINING CLAIMS

ZONIA MINE - CLAIMS

Name of Claim	Number of Acres	BLM Mineral Survey	Book of Deeds	Page No.	Patent No.	Book of Official Record	Page	AMC Number
Patented Claims								
Georgia		3866	134	557-566	954817	134	557-566	
Georgia No.2		3866	134	557-566	954817	134	557-566	
Georgia No.3		3866	134	557-566	954817	134	557-566	
Yankee Girl		3866	134	557-566	954817	134	557-566	
Sunrise		3866	134	557-566	954817	134	557-566	
Sunrise No.2		3866	134	557-566	954817	134	557-566	
Sunrise No.3		3866	134	557-566	954817	134	557-566	
Sunrise No.4		3866	134	557-566	954817	134	557-566	
Richmond	98.1	3867	134	369-372	951190	134	369-372	
Virginia	10.4	3867	134	369-372	951190	134	369-372	
Polar Star	13.5	1342	49	485	31584			
Toumaline	17.7	1342	49	485	31584			
Copper Glance	17.5	1342	49	485	31584			
Sunset	18.5	1342	49	485	31584			
Manilla	16.5	1342	49	485	31584			
Copperopolis	20.2	1342	49	485	31584			
Defiance	18.6	1342	49	485	31584			
Fairplay	20.5	1342	49	485	31584			
Quartette	20.2	1321	77	114-117	31479			
Sunflower	20.4	1323A	49	478	31583			
Lone Pine	20.4	1323A	49	478	31583			
Fraction	13.5	1323A	49	478	31583			
Iron Hat	20.1	1323A	49	478	31583			
Fountain	20.7	762	27	633	15269			
Arrastra	17.5	767	27	636	15270			
Cuprite	19.9	4659A	1294	739	02-80-0005	1294	686	
Black Prince	20.6	4659A	1294	744	02-80-0005	1294	686	
Shamrock	20.1	4659A	1294	745	02-80-0005	1294	686	
Zonia No. 26	20.6	4681B	1294	693	02-80-0005	1294	686	
Zonia	2.3	4659A	1294	743	02-80-0005	1294	686	
Fraction	2.3	4659A	1294	741	02-80-0005	1294	686	
Victor Copper	20.6	4659A	1294	746	02-80-0005	1294	686	
Victory Copper No.1 One	20.2	4659A	129A	747	02-80-0005	1294	686	
Zonia MS No.2	4.8	4659B	1294	750	02-80-0005	1294	686	
Zonia MS No.3	4.8	4659B	1294	751	02-80-0005	1294	686	
Zonia MS No.4	4.8	4659B	1294	753	02-80-0005	1294	686	
Zonia MS No.5	4.8	4659B	1294	754	02-80-0005	1294	686	
Zonia MS No.6	4.8	4659B	1294	755	02-80-0005	1294	686	
Zonia MS No. 12	4.8	4659B	1294	760	02-80-0005	1294	686	
Zonia MS No. 13	4.8	4659B	1294	762	02-80-0005	1294	686	
Zonia MS No. 14	4.9	4659B	1294	763	02-80-0005	1294	686	

Name of Claim	Number of Acres	BLM Mineral Survey	Book of Deeds	Page No.	Patent No.	Book of Official Record	Page	AMC Number
Zonia MS No. 15	4.9	4659B	1294	764	02-80-0005	1294	686	
Zonia MS No. 16	4.8	4659B	1294	765	02-80-0005	1294	686	
Zonia MS No. 21	4.8	4659B	1294	770	02-80-0005	1294	686	
Zonia MS No. 22	4.8	4659B	1294	771	02-80-0005	1294	686	
Zonia MS No. 23	4.8	4659B	1294	772	02-80-0005	1294	686	
Zonia MS No. 24	4.8	4659B	1294	773	02-80-0005	1294	686	
Zonia MS No. 25	4.8	4659B	1294	774	02-80-0005	1294	686	
Zonia MS No. 26	4.8	4659B	1294	775	02-80-0005	1294	686	
Zonia MS No. 27	4.8	4659B	1294	776	02-80-0005	1294	686	
Zonia MS No. 28	4.8	4659B	1294	777	02-80-0005	1294	686	
Zonia MS No. 29	4.8	4659B	1294	778	02-80-0005	1294	686	
Zonia MS No. 30	5.0	4659B	1294	779	02-80-0005	1294	686	
Zonia MS No. 31	5.0	4659B	1294	780	02-80-0005	1294	686	
Zonia MS No. 32	1.5	4659B	1294	782	02-80-0005	1294	686	
Zonia MS No. 37	4.8	4659B	1294	787	02-80-0005	1294	686	
Zonia MS No. 38	4.8	4659B	1294	788	02-80-0005	1294	686	
Zonia MS No. 39	4.8	4659B	1294	789	02-80-0005	1294	686	
Zonia MS No. 43	4.8	4659B	1294	793	02-80-0005	1294	686	
Zonia MS No. 46	4.8	4659B	1294	796	02-80-0005	1294	686	
Zonia MS No. 47	4.8	4659B	1294	796	02-80-0005	1294	686	
Zonia MS No. 48	4.8	4659B	1294	797	02-80-0005	1294	686	
Zonia MS No. 49	4.8	4659B	1294	798	02-80-0005	1294	686	
Zonia MS No. 50	4.8	4659B	1294	799	02-80-0005	1294	686	
Zonia MS No. 51	5.0	4659B	1294	800	02-80-0005	1294	686	
Zonia MS No. 52	4.8	4659B	1294	802	02-80-0005	1294	686	
Zonia MS No. 53	4.8	4659B	1294	803	02-80-0005	1294	686	
Zonia MS No. 54	4.8	4659A&B	AMD1294	836	02-80-0005	1294	686	
Zonia MS No. 55	4.8	4659A&B	AMD1294	837	02-80-0005	1294	686	
Zonia MS No. 56	4.8	4659A&B	AMD1294	838	02-80-0005	1294	686	
Zonia MS No. 59	4.8	4659A&B	AMD1294	841	02-80-0005	1294	686	
Zonia MS No. 60	4.8	4659A&B	AMD1294	842	02-80-0005	1294	686	
Zonia MS No. 61	4.8	4659A&B	AMD1294	843	02-80-0005	1294	686	
Zonia MS No. 63	4.8	4659A&B	AMD1294	844	02-80-0005	1294	686	
Zonia MS No. 70	4.8	4681B	1294	695	02-80-0005	1294	686	
Zonia MS No. 71	4.9	4681B	1294	697	02-80-0005	1294	686	
Zonia MS No. 72	4.9	4681B	1294	699	02-80-0005	1294	686	
Zonia MS No. 73	1.7	4681B	1294	700	02-80-0005	1294	686	
Unpatented Lode Mining Claims								
Mistake Fraction No.1	3.5					761	114	75989
Mistake Fraction No.2	10.0					761	115	75990
Mistake No.1	20.7					761	116	75991
Mistake No.2	20.7					761	117	75992
Mistake No.3	20.7					761	118	75993
Mistake No.4	20.7					761	119	75994

Name of Claim	Number of Acres	BLM Mineral Survey	Book of Deeds	Page No.	Patent No.	Book of Official Record	Page	AMC Number
Mistake No.5	20.7					761	120	75995
Mistake No.6	19.7					761	121	75996
Mistake No.7	20.7					761	122	75997
Mistake No.8	20.7					761	123	75998
Mistake No.9	15.8					761	124	75999
Mistake No. 10	16.6					761	125	76000
Mistake No. 11	14.9					761	126	76001
Mistake No. 12	18.2					761	127	76002
Mistake No. 13	18.2					761	128	76003
Mistake No. 14	18.2					761	129	76004
Mistake No. 15	20.4					761	130	76005
Mistake No. 16	20.4					761	131	76006
Mistake No. 17	20.4					761	132	76007
Mistake No. 18	20.4					761	133	76008
Last Mistake	20.4					761	134	76009
Lois No.1	19.8					464	551	75979
Lois No.2	10.9					464	552	75980
Lois No.3	15.7					464	553	75981
Lois No.4	20.7					464	554	75982
Lois No.5	20.7					464	555	75983
Lois No.6	16.0					464	556	75984
Lois No. 17	20.7					464	557	75985
Lois No. 18	20.7					464	558	75986
Lois No. 19	17.9					464	559	75987
Lois No. 20	20.7					464	560	75988
Zonia No.2	1.8					1358	591-592	124258
Zonia No.6	18.2					1358	595-596	124260
Zonia No.7	20.4					1358	597-598	124261
Zonia No.8	20.7					1358	599-600	124262
Zonia No.9	20.7					1358	601-602	124263
Zonia No. 10	20.7					1358	603-604	124264
Zonia No. 11	20.7					1358	605-606	124265
Zonia No. 14	17.5					1358	607-608	124266
Zonia No. 15	18.2					1358	609-610	124267
Zonia No. 16	19.7					1358	611-612	124268
Zonia No. 17	19.3					1358	613-614	124269
Zonia No. 18	0.5					1358	615-616	124270
Zonia No. 19	0.8					1358	617-618	124271
Zonia No. 20	3.7					1358	619-620	124272
Zonia No. 21	20.7					1358	621-622	124273
Zonia No. 22	20.7					1358	623-624	124274
Zonia No. 23	20.7					1358	625-626	124275
Zonia No. 24	20.7					1358	627-628	124276
Copper Bar No.2	5.5					1358	645-646	124285

Name of Claim	Number of Acres	BLM Mineral Survey	Book of Deeds	Page No.	Patent No.	Book of Official Record	Page	AMC Number
Copper King No.1	9.5					1358	635-636	124280
Copper King No.3	5.8					1358	637-638	124281
Copper King No.4	20.7					1358	639-640	124282
Scott No.1	3.0					1358	641-642	124283
Scott No.2	13.5					1358	643-644	124284
Copper Crown Group Unpatented Lode Mining Claims								
Copper Crown No.1	20.7					147	155	76047
Copper Crown No.2	20.7					147	156	76048
Copper Crown No.3	20.7					147	157	76049
Copper Crown No.4	20.7					151	331	76050
Copper Crown No.5	20.7					151	332	76051
Copper Crown No.6	20.7					151	333	76052
Copper Crown No.7	20.7					151	334	76053
Copper Crown No.8	20.7					151	335	76054
Copper Crown No.9	20.7					55	111	76055
Copper Crown No. 10	20.7					7	186	76056
Copper Crown No. 12	20.7					55	112	76057
Copper Crown No. 13	20.7					560	929	76058
Copper Crown No. 14	20.7					63	204	76059
Copper Crown No. 15	20.7					64	179	76060
Copper Crown No. 16	20.7					64	180	76061
Copper Crown No. 17	20.7					64	181	76062
Copper Crown No. 18	20.7					68	385	76063
Copper Crown No. 19	20.7					68	386	76064
Copper Crown No. 20	20.7					68	387	76065
Copper Crown No. 21	20.7					68	388	76066
Copper Crown No. 22	20.7					68	389	76067
Copper Crown No. 23	20.6					68	390	76068
Copper Crown No. 24	20.7					68	391	76069
Copper Crown No. 25	20.7					68	392	76070
Copper Crown No. 26	20.7					68	393	76071
Copper Crown No. 27	20.7					83	74	76072
Copper Crown No. 28	20.7					73	402	76073
Copper Crown No. 29	20.7					73	403	76074
Copper Crown No. 30	20.7					73	404	76075
Copper Crown No. 31	20.7					73	405	76076
Copper Crown No. 32	20.7					83	75	76077
Copper Crown No. 33	20.7					112	374	76078
Copper Crown No. 34	20.7					112	375	76079
Copper Crown No. 35	20.7					112	376	76080
Copper Crown No. 36	20.7					560	930	76081
Copper Crown No. 37	20.7					560	931	76082
Copper Crown No. 38	20.7					560	932	76083
Copper Crown No. 39	20.7					560	933	76084

Name of Claim	Number of Acres	BLM Mineral Survey	Book of Deeds	Page No.	Patent No.	Book of Official Record	Page	AMC Number
Copper Crown No. 40	20.7					560	934	76085
Copper Crown No. 41	20.7					560	935	76086
Copper Crown No. 42	20.7					560	935	76087
Copper Crown No. 43	20.7					560	937	76088
Copper Crown No. 44	20.7					560	938	76089
Copper Crown No. 45	20.7					560	939	76090
Copper Crown No. 46	20.7					560	940	76091
Copper Crown No. 47	20.7					560	941	76092
Copper Crown No. 48	20.7					560	942	76093
Copper Crown No. 49	20.7					560	943	76094
Copper Crown No. 50	20.7					560	944	76095
Copper Crown No. 51	20.7					560	945	76096
Copper Crown No.51 Amend.	20.7					560	945	76096
Copper Crown No. 53	20.7					1484	185	188442
Gold Crown						159	400	76046
Unpatented Lode Mining and Millsite Claims								
N-30	3.1		3798	672				354858
N-31	20.7		3798	671				354859
N-32	20.7		3798	670				354860
N-34	20.7		3798	669				354861
N-35	20.7		3798	668				354862
N-36	20.7		3798	667				354863
N-37	20.7		3798	666				354884
N-38	20.7		3798	665				354885
N-39	20.7		3798	664				354886
N-40	20.7		3798	663				354887
Triad No. 1	15.6		3799	235				353382
Triad No. 2	20.7		3799	234				353383
Triad No. 3	20.7		3799	233				353384
Receiving Shop	5.0		3799	236				353385
Pump Station	5.0		3799	237				353388
Zonia MS No.1 Amended	4.8	4659B	1294	748				76098
Zonia MS No.7 Amended	4.8	4659B	1294	756				76104
Zonia MS No.8 Amended	4.8	4659B	1294	757				76105
Zonia MS No.9 Amended	4.8	4659B	1294	758				76106
Zonia MS No. 10 Amended	4.8	4659B	1294	759				76107
Zonia MS No. 11 Amended	4.8	4659B	1294	759				76108
Zonia MS No. 17 Amended	4.8	4659B	1294	767				76114
Zonia MS No. 18 Amended	4.8	4659B	1294	767				76115
Zonia MS No. 19 Amended	4.8	4659B	1294	768				76116
Zonia MS No. 20 Amended	4.8	4659B	1294	769				76117
Zonia MS No. 33 Amended	4.8	4659B	1294	783				76130
Zonia MS No. 34 Amended	4.8	4659B	1294	784				76131
Zonia MS No. 35 Amended	4.8	4659B	1294	785				76132

Name of Claim	Number of Acres	BLM Mineral Survey	Book of Deeds	Page No.	Patent No.	Book of Official Record	Page	AMC Number
Zonia MS No. 36 Amended	4.8	4659B	1294	786				76133
Zonia MS No. 40	4.8	4659B	1294	790				76137
Zonia MS No. 41	4.8	4659B	1294	791				76138
Zonia Ms No. 42	4.8	4659B	1294	792				76139
Zonia MS No. 44	4.8	4659B	1294	794				76141
Zonia MS No. 45	4.1	4659B	1294	795				76142
Zonia MS No. 57 Amended	5.0	4659A&B AMD	1294	840				76154
Zonia MS No. 58	4.8	5659B	1294	808				76155

Note: MS stands for Mill Site

APPENDIX B
ECONOMIC MODEL

Zonia Project

Case 50, $200 Pit, 0.17 Cutoff, Contractor Operated, $3 Copper Price

			Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Production														
Ore Tons Oxidized			31,803	10,966,055	10,728,203	10,788,187	10,867,613	10,920,000	10,920,000	10,904,288	10,626,977	3,989,781	-	90,742,906
Ore Tons Oxidized per Day			824	37,035	30,000	30,000	30,000	30,000	30,000	30,000	30,000	30,000	-	
Copper lbs Oxidized			144,501	97,549,687	69,861,082	64,249,754	64,529,719	64,143,220	62,592,707	60,563,184	59,679,144	24,055,082	-	567,368,080
Copper CU_grade (%)			0	0.445	0.326	0.298	0.297	0.294	0.287	0.278	0.281	0.301	-	0.313
Ore Tons Mixed			-	-	191,797	131,813	52,387	-	-	15,712	293,023	1,208,011	-	1,892,743
Ore Tons Mixed per Day			-	-	-	-	-	-	-	-	-	-	-	
Copper lbs Mixed			-	-	1,182,192	837,148	405,046	-	-	66,853	1,337,053	6,660,933	-	10,489,226
Copper CU_grade (%)			-	-	0.308	0.318	0.387	-	-	0.213	0.228	0.276	-	0.277
Waste Tons			2,314,866	4,934,304	20,557,469	6,480,552	3,700,498	4,173,952	4,550,528	3,198,235	2,129,234	383,397	-	52,423,036
Total Tons			2,346,669	15,900,359	31,477,469	17,400,552	14,620,498	15,093,952	15,470,528	14,118,235	13,049,234	5,581,189	-	145,058,685
Stripping Ratio			72.8	0.4	1.9	0.6	0.3	0.4	0.4	0.3	0.2	0.1	-	0.6
Process Copper Recovery														
Recovered Copper Cathode Produced - Ox	73.0%		-	71,316,758	50,998,590	46,902,320	47,106,695	46,824,551	45,692,676	44,211,124	43,565,775	17,560,210	-	414,178,699
Recovered Copper Cathode Produced - Mix	70.0%		-	-	827,535	586,004	283,532	-	-	46,797	935,937	4,662,653	-	7,342,458
Total Recovered Copper Cathode Produced			-	71,316,758	51,826,125	47,488,324	47,390,227	46,824,551	45,692,676	44,257,922	44,501,712	22,222,863	-	421,521,157
Payable Copper Cathode	99.5%		-	70,960,174	51,566,994	47,250,883	47,153,276	46,590,428	45,464,213	44,036,632	44,279,204	22,111,749	-	419,413,551
Revenue														
Gross Revenue	$3.000 /lb		$0	$212,880,522	$154,700,982	$141,752,648	$141,459,827	$139,771,283	$136,392,638	$132,109,896	$132,837,611	$66,335,246	$0	$1,258,240,653
Refining Charges	$0.032 /lb		$0	($2,270,726)	($1,650,144)	($1,512,028)	($1,508,905)	($1,490,894)	($1,454,855)	($1,409,172)	($1,416,935)	($707,576)	$0	($13,421,234)
Transport	$0.100 /ton-mile		$0	($283,841)	($206,268)	($189,004)	($188,613)	($186,362)	($181,857)	($176,147)	($177,117)	($88,447)	$0	($1,677,654)
Royalty	0.0%		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Revenue Delay				($21,288,052)	$0	$0	$0	$0	$0	$0	$0	$21,288,052	$0	$0
Net Revenue			**$0**	**$189,037,903**	**$152,844,570**	**$140,051,616**	**$139,762,309**	**$138,094,028**	**$134,755,926**	**$130,524,578**	**$131,243,560**	**$86,827,275**	**$0**	**$1,243,141,765**
Operating Costs														
Mine			$3,111,262	$27,977,488	$39,821,999	$28,951,161	$27,085,827	$27,425,488	$27,682,165	$26,756,472	$25,544,716	$10,914,918	$0	$245,271,495
Process			$3,515,729	$33,169,742	$33,169,742	$33,169,742	$33,169,742	$33,169,742	$33,169,742	$33,169,742	$33,169,742	$15,788,407	$0	$284,662,068
Admin			$360,373	$3,400,000	$3,400,000	$3,400,000	$3,400,000	$3,400,000	$3,400,000	$3,400,000	$3,400,000	$1,618,360	$0	$29,178,733
Op Cost Contingency			$698,736	$6,454,723	$7,639,174	$6,552,090	$6,365,557	$6,399,523	$6,425,191	$6,332,621	$6,211,446	$2,832,168	$0	$55,911,230
Total Operating Costs			**$7,686,100**	**$71,001,952**	**$84,030,915**	**$72,072,993**	**$70,021,125**	**$70,394,753**	**$70,677,097**	**$69,658,835**	**$68,325,903**	**$31,153,853**	**$0**	**$615,023,525**
Prior Year Non-Capital Costs		$27,966,655												
Before Tax Cash Flow		($27,966,655)	($7,686,100)	$118,035,952	$68,813,655	$67,978,624	$69,741,184	$67,699,276	$64,078,830	$60,865,743	$62,917,657	$55,673,422	$0	$628,118,240
Depreciation			$0	$19,830,001	$21,009,220	$21,741,709	$21,791,629	$23,898,079	$24,630,569	$24,675,689	$24,725,609	$25,970,729	$0	
Loss Carry Forward - max 30% of BTCF			$0	($35,410,785)	($241,970)	$0	$0	$0	$0	$0	$0	$0	$0	
Depletion Allowance (Percentage)	15%		$0	$28,355,686	$22,926,685	$21,007,742	$20,964,346	$20,714,104	$19,724,130	$18,095,027	$19,096,024	$13,024,091	$0	$183,907,836
Taxable Income			$0	$34,439,480	$24,635,780	$25,229,172	$26,985,208	$23,087,092	$19,724,130	$18,095,027	$19,096,024	$16,678,602	$0	$207,970,514
Federal Tax	21%		$0	$7,232,291	$5,173,514	$5,298,126	$5,666,894	$4,848,289	$4,142,067	$3,799,956	$4,010,165	$3,502,506	$0	$43,673,808
Allowance for Property and State Tax	7%		$0	$2,410,764	$1,724,505	$1,766,042	$1,888,965	$1,616,096	$1,380,689	$1,266,652	$1,336,722	$1,167,502	$0	$14,557,936
Income after Tax			$0	$24,796,425	$17,737,762	$18,165,004	$19,429,350	$16,622,706	$14,201,374	$13,028,419	$13,749,137	$12,008,593	$0	$149,738,770
Add Back Depreciation			$0	$19,830,001	$21,009,220	$21,741,709	$21,791,629	$23,898,079	$24,630,569	$24,675,689	$24,725,609	$25,970,729	$0	$208,273,234
Add Back Depletion			$0	$28,355,686	$22,926,685	$21,007,742	$20,964,346	$20,714,104	$19,724,130	$18,095,027	$19,096,024	$13,024,091	$0	$183,907,836
After Tax Cash Flow			($7,686,100)	$108,392,897	$61,915,637	$60,914,456	$62,185,325	$61,234,890	$58,556,073	$55,799,135	$57,570,770	$51,003,413	$0	$569,886,496

Zonia Project

Case 50, $200 Pit, 0.17 Cutoff, Contractor Operated, $3 Copper Price

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Capital Costs												
Mine	$1,620,000	$76,000	$76,000	$76,000	$116,000	$126,000	$76,000	$76,000	$116,000	$76,000	$0	$2,434,000
Plant	$51,891,691	$0	$9,450,825	$0	$0	$17,127,750	$0	$0	$0	$0	$0	$78,470,266
Laboratory	$516,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$516,000
Facilities	$58,578,467	$0	$0	$5,728,078	$0	$0	$5,728,078	$0	$0	$0	$0	$70,034,624
G&A	$21,473,220	$300,000	$300,000	$300,000	$300,000	$300,000	$300,000	$300,000	$300,000	$10,300,000	$0	$34,173,220
First Fills	$9,786,901	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$9,786,901
Working Capital	$21,007,729	$0	$0	$0	$0	$0	$0	$0	$0	$0	($21,007,729)	$0
Mobile Eq Sustaining Capital	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Cap Cost Contingency	$32,974,802	$75,200	$1,965,365	$1,220,816	$83,200	$3,510,750	$1,220,816	$75,200	$83,200	$2,075,200	$0	$43,284,548
Total Capital Cost	**$197,848,810**	**$451,200**	**$11,792,190**	**$7,324,894**	**$499,200**	**$21,064,500**	**$7,324,894**	**$451,200**	**$499,200**	**$12,451,200**	**($21,007,729)**	**$238,699,559**
Net Cash Flow	($205,534,910)	$107,941,697	$50,123,447	$53,589,561	$61,686,125	$40,170,390	$51,231,179	$55,347,935	$57,071,570	$38,552,213	$21,007,729	$331,186,937
Cumulative Cash Flow	($205,534,910)	($97,593,213)	($47,469,766)	$6,119,795	$67,805,921	$107,976,311	$159,207,490	$214,555,425	$271,626,995	$310,179,208	$331,186,937	

NPV@6%	$225,187,255
NPV@8%	$192,041,665
NPV@10%	$162,962,442
IRR	29.0%

CERTIFICATE OF QUALIFIED PERSON

I, Terre A Lane, of 600 Grant St., Suite 975, Denver, Colorado, 80203, the co-author of the report entitled "Preliminary Economic Assessment, NI43-101 Technical Report, Zonia Copper Project, Yavapai County, Arizona, USA" with an effective date of March 6, 2018 (the "PEA"), DO HEREBY CERTIFY THAT:

1. I am a MMSA Qualified Professional in Ore Reserves and Mining, #01407QP and a Registered member of SME - 4053005.
2. I hold a degree of Bachelor of Science (1982) in Mining Engineering from Michigan Technological University.
3. I have practiced my profession since 1982 in capacities from mining engineer to senior management positions for engineering, mine development, exploration, and mining companies. My relevant experience for the purpose of this PEA is as the mine planner and economic modeler with 25 or more years of experience in each area.
4. I have created or overseen the development of mine plans for several hundred open pit and underground projects and operating mines.
5. I have been involved in or managed several hundred studies including scoping studies, prefeasibility studies, and feasibility studies.
6. I have been involved with the mine development, construction, startup, and operation of several mines.
7. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
8. I visited the property on March 21 and 22, 2018.
9. I am responsible for Sections 4, 5, 6, 15, 16 and 20 of the PEA and have contributed to Sections 1, 2, 3, 14, 18, 19, 21, 22, 24, 25, 26, and 27.
10. I am independent of Cardero Resources Corp. as described in section 1.5 by National Instrument 43-101.
11. I have no prior experience with the Zonia Copper Project.
12. I have read National Instrument 43-101 and Form 43-101F1. The PEA has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
13. As of the effective date of the PEA, to the best of my knowledge, information and belief, the PEA contains all scientific and technical information that is required to be disclosed to make the PEA not misleading.

Terre A. Lane

"Terre A. Lane"

Mining Engineer

Global Resource Engineering, Ltd.

Denver, Colorado

Date of Signing: March 22, 2018

CERTIFICATE OF QUALIFIED PERSON

I, Jeffrey Todd Harvey, PhD, of 600 Grant St., Suite 975, Denver, Colorado, 80203, the co-author of the report entitled "Preliminary Economic Assessment, NI43-101 Technical Report, Zonia Copper Project, Yavapai County, Arizona, USA" with an effective date of March 6, 2018 (the "PEA"), DO HEREBY CERTIFY THAT:

1. I am a Society of Mining Engineers (SME) Registered Member Qualified Professional in Mining/Metallurgy/Mineral Processing, #04144120.
2. I hold a degree of Doctor of Philosophy (PhD) (1994) in Mining and Mineral Process Engineering from Queen's University at Kingston. As well as an MSc (1990) and BSc (1988) in Mining and Mineral Process Engineering from Queen's University at Kingston.
3. I have practiced my profession since 1988 in capacities from metallurgical engineer to senior management positions for production, engineering, mill design and construction, research and development, and mining companies. My relevant experience for the purpose of this PEA is as the test work reviewer, process designer, process cost estimator, and economic modeler with 25 or more years of experience in each area.
4. I have taken classes in mineral processing, mill design, cost estimation and mineral economics in university, and have taken several short courses in process development subsequently.
5. I have worked in mineral processing, managed production and worked in process optimization, and I have been involved in or conducted the test work analysis and flowsheet design for many projects at locations in North America, South America, Africa, Australia, India, Russia and Europe for a wide variety of minerals and processes.
6. I have supervised and analyzed test work, developed flowsheets and estimated costs for many projects including International Gold Resources Bibiani Mine, Aur Resources Quebrada Blanca Mine, Mineracao Caraiba S/A, Avocet Mining Taror Mine, Mina Punta del Cobre Pucobre Mine, and others, and have overseen the design and cost estimation of many other similar projects.
7. I have worked or overseen the development or optimization of mineral processing flowsheets for close to one hundred projects and operating mines, including copper flotation and acid heap leach SX/EW processes.
8. I have been involved in or managed many studies including scoping studies, prefeasibility studies, and feasibility studies.
9. I have been involved with the mine development, construction, startup, and operation of several mines.
10. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
11. I visited the property on March 21 and 22, 2018.
12. I am responsible for Sections 13 and 17 of the PEA and have contributed to Sections 1, 2, 3, 18, 19, 21, 22, 24, 25, 26, and 27.
13. I am independent of Cardero Resources Corp. as described in section 1.5 by National Instrument 43-101.
14. I have no prior experience with the Zonia Copper Project.
15. I have read National Instrument 43-101 and Form 43-101F1. The PEA has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.

16. As of the effective date of the PEA, to the best of my knowledge, information and belief, the PEA contains all scientific and technical information that is required to be disclosed to make the PEA not misleading.

Jeffrey Todd Harvey, PhD

"Todd Harvey"

Director of Process Engineering

Global Resource Engineering, Ltd.

Denver, Colorado

Date of Signing: March 22, 2018

CERTIFICATE OF QUALIFIED PERSON

I, Rex C. Bryan, PhD, of Golden, Colorado, the co-author of the report entitled "Preliminary Economic Assessment, NI43-101 Technical Report, Zonia Copper Project, Yavapai County, Arizona, USA" with an effective date of March 6, 2018 (the "PEA"), DO HEREBY CERTIFY THAT:

1. I am a Senior Geostatistician with Tetra Tech, Inc. with a business address at 350 Indiana Street, Suite 500, Golden, Colorado 80401, USA.
2. This certificate applies to the technical report entitled "Zonia Copper Project, NI 43-101 Technical Report, Yavapai County, Arizona, USA" dated January 13, 2016 (the "Technical Report").
3. 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. 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. I am a Registered Member (#411340) of the Society for Mining, Metallurgy, and Exploration, Inc. (SME). I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
4. My most recent personal inspection of the Property was on November 5, 2015 for one day.
5. I am responsible for all Sections 7 through 12 and 23 of the Technical Report and have contributed to Sections 1, 2, 3, 14, 24, 25, 26, and 27.
6. I am independent of Cardero Resource Corp. as defined by Section 1.5 of the Instrument.
7. I have had prior involvement with the Property that is the subject of the Technical Report acting as a subject matter expert contributing to the estimation of resources.
8. I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Rex C. Bryan, PhD

"Rex C. Bryan"

Senior Geostatistician

Tetra Tech, Inc..

Date of Signing: March 22, 2018