M3-PN210313
Effective Date:
February 1, 2023
Issue Date:
February 22, 2023
Revision 0

Gunnison Copper Project

NI 43-101 Technical Report

Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment

Cochise County, Arizona, USA

Qualified Persons:

Richard Zimmerman, SME-RM
Jeffrey Bickel, CPG
Thomas L Dyer, PE, SME-RM
Neil Prenn, MMSA-QPM
Robert J. Bowell, PhD, C.Chem., C.Geol
Dr. Terence P. McNulty, PE, DSc
R. Douglas Bartlett, CPG

Prepared For:

Excelsior
MINING CORP

DATE AND SIGNATURES PAGE

The effective date of this report is February 1, 2023. The issue date of this report is February 22, 2023. See Appendix A, Prefeasibility Study and Preliminary Economic Assessment Contributors and Professional Qualifications, for certificates of qualified persons. These certificates are considered the date and signature of this report in accordance with Form 43-101F1.

GUNNISON COPPER PROJECT
FORM 43-101F1 TECHNICAL REPORT

TABLE OF CONTENTS

SECTION PAGE

LIST OF FIGURES AND ILLUSTRATIONS

LIST OF TABLES

TABLE DESCRIPTION PAGE

LIST OF APPENDICES

1 SUMMARY

M3 Engineering & Technology Corporation (M3) was commissioned by Excelsior Mining Corp. ("Excelsior") to prepare an updated Prefeasibility Study (PFS) in accordance with the Canadian National Instrument 43-101 ("NI 43-101") standards for reporting mineral properties, for the Gunnison Copper Project (the "Gunnison Project" or the "Project") in Cochise County, Arizona, USA. The Project utilizes in-situ recovery (ISR) methods to leach copper from a buried copper oxide deposit and extract the copper by conventional solvent extraction-electrowinning (SX-EW) technology. The ISR process involves injecting leach solutions acidified with sulfuric acid into the oxidized mineralization to get soluble copper into solution. Recovery wells pump the copper-bearing pregnant leach solution (PLS) to the surface for copper recovery by SX-EW into salable copper cathodes.

The Gunnison Project is located about 65 miles east of Tucson, Arizona on the southeastern flank of the Little Dragoon Mountains in the Cochise Mining District. The property is within the copper porphyry belt of Arizona. The Gunnison Project hosts the North Star (formerly known as the I-10) deposit and contains copper oxide and sulfide mineralization with associated molybdenum in potentially economic concentrations.

Oxidized, mineralized bedrock that lies 300 feet to 800 feet beneath alluvial basin fill with ISR using a staggered series of injection and recovery wells to circulated acidified leach solution that dissolves the copper. The basin fill is typically above the water table and most of the oxidized mineralization is below the water table. The North Star copper deposit host rocks show significant fracturing and jointing resulting in broken ground that is below the water table (saturated zone) and permeable. The copper silicates and oxides occur preferentially as coatings on the fracture planes and as veinlets or matrix fill to the broken fragments. This should result in preferential exposure of the copper minerals to the leaching solution (lixiviant), thus reducing the amount of acid consumed by the un-exposed gangue rocks. The above features, combined with the large size of the deposit, suggest ISR is a viable approach to mining this deposit.

ISR is a closed-loop mining system, where metal-bearing minerals are dissolved within the host formation using an appropriate leach solution (lixiviant). Production wells constructed in an alternating array are used to deliver (inject) the lixiviant to the ore zone to be drawn toward the recovery wells in the array. Leached metals in the pregnant leach solution (PLS) are recovered to the surface for processing by wells that are equipped with submersible electric pumps. After processing, the solution is recycled to the wellfield to continue the leaching cycle, making ISR a continuous mining operation.

Several ISR operations for copper have operated or been permitted in Arizona including Miami (BHP-Billiton), San Manuel (BHP-Billiton), Silver Bell (ASARCO), Old Reliable (Ranchers Exploration), Santa Cruz (ASARCO et al.), Florence (BHP-Billiton), and Safford area (Kennecott Copper). Considerable expertise in copper oxide ISR mining is available in Arizona and elsewhere in the USA.

The Project envisages development in three production "stages" with capacities of 25 million pounds per annum (mppa) in Stage 1, 75 mppa in Stage 2, and 125 mppa in Stage 3. The stages to ramp up production were meant to minimize capital at risk until the in-situ recovery (ISR) process at the Gunnison Project is better understood. For Stage 1 operations, Excelsior will use the neighboring Johnson Camp Mine (JCM) that has a functional 25 mppa SX-EW plant north of the Gunnison Project wellfield on the north side of Interstate 10 that it purchased in 2015.

Stage 1 construction was completed in the fourth quarter (Q4) of 2020 and went into production using the JCM SX-EW plant. Once Stage 1 production has been reached, Stage 2 production will commence in Year 4 of the mine plan. A 50 mppa Gunnison SX-EW plant will be constructed on the south side of Interstate 10 next to the Gunnison wellfield to accommodate the increased production. Stage 3 production will commence in Year 7 of the mine life by doubling the size of the Gunnison SX-EW plant to 100 mppa, increasing production capacity to 125 mppa.

Excelsior selected M3 and other respected third-party consultants to prepare mine plans, resources/reserve estimates, process plant designs, and to complete environmental studies and cost estimates used for this report. All consultants have the capability to support the Project, as required and within the confines of their expertise. The costs are based on fourth-quarter 2021 U.S. dollars, except for the cost of acid and molten sulfur which were updated to align with the Johnson Camp Mine update presented in Section 24.

Stage 1 construction was completed in the fourth quarter (Q4) of 2020. The wellfield was installed and tested with groundwater circulating through the formation. Acidified leach solution began being circulated through the formation, representing the start of ISR operations. Challenges to the operation were encountered, most notably declining flow rates in the wellfield due to the buildup of carbon dioxide in the formation. A water treatment plant will be required to provide neutralized solutions to the wellfield to dissolve the carbon dioxide and restore flow rates in the formation. This will require further design, engineering and test work before construction should commence.

1.1 KEY DATA

The key results of this study are as follows:

- The average annual Stage 3 production is projected to be approximately 125 million pounds of copper. Total life of operation production is projected at approximately 2,154 million pounds of copper.
- The Project currently has 873 million short tons of measured and indicated oxide and transitional mineral resources (0.29% Total Copper Grade) at a 0.05% Total Copper cut-off grade, as well as 187 million short tons of inferred mineral resources (0.17% Total Copper Grade).
- The Project currently has a diluted mineral reserve of 782 million short tons of probable mineral reserves (0.29% Total Copper Grade).
- ISR is anticipated to recover approximately 48.4% of the total copper with an average “sweep efficiency” of 74%.
- The average life-of-mine direct operating cost estimated to be $0.945 per pound of copper for the Base Case, which includes building a sulfuric acid plant that commences operation in Year 7 (Stage 3). The average life-of-mine direct operating cost for the Alternate Case (No acid plant) is $1.354 per pound of copper.
- The estimated initial capital cost is $47.6 million which includes $9.2 million in capitalized pre-production costs.
- The total life-of-operation sustaining capital cost for the Base Case is estimated to be $1,033 million while the total life-of-operation sustaining capital cost for the Alternate Case is $880 million.
- The total cost for reclamation and closure is estimated to be $60 million and averages $0.028 per pound of copper recovered.
- The economic analysis for the Base Case before taxes indicates an Internal Rate of Return (IRR) of 40.6% and a payback period of 6.5 years. Based on a copper price of $3.75 per pound, the Net Present Value (“NPV”) before taxes is $1,435 million at a 7.5% discount rate.
- The economic analysis for the Base Case after taxes indicates that the Project has an IRR of 37.5% with a payback period of 6.7 years. The NPV after taxes is $1,166 million at a 7.5% discount rate.

1.2 PROPERTY DESCRIPTION AND LOCATION

The Project is located in Cochise County, Arizona, approximately 65 miles east of Tucson and 1.5 miles southeast of the historic Johnson Camp mining district. Figure 1-1 is a general location map and property location near the US Interstate 10 (I-10) freeway. Total area is approximately 11,802 acres (5,876 hectares).

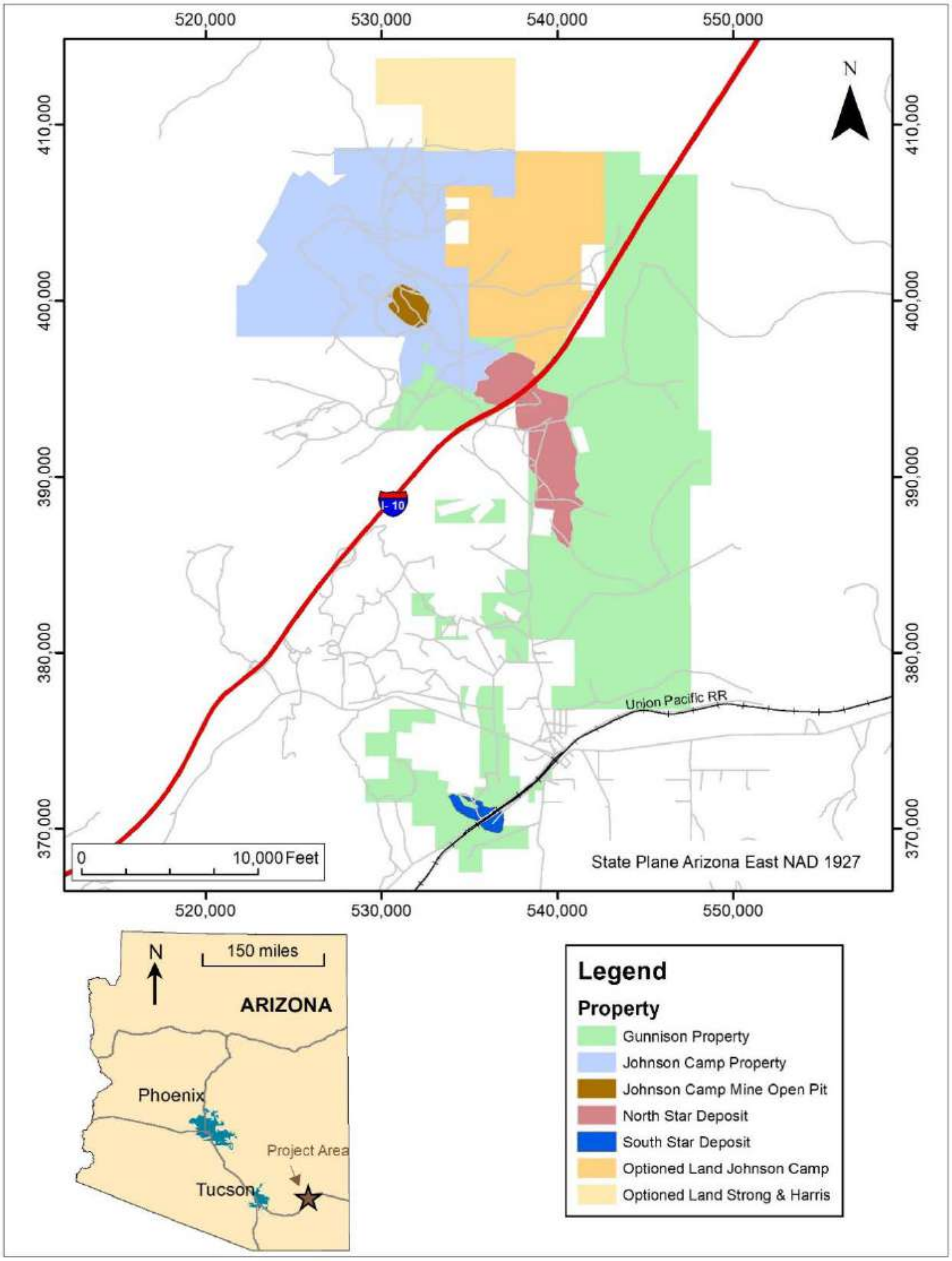


Figure 1-1: Project Location Map, North and South Star Deposits and Johnson Camp Mine

The Project is held by Excelsior through is wholly owned subsidiary Excelsior Mining Arizona, Inc. (Excelsior Arizona). Acquisition of all mineral interest from the James L. Sullivan Trust was completed in January of 2015. These assets represent, among other things, the mineral rights to the North Star and South Star Copper deposits (the Gunnison Project). Additionally, in December 2015 Excelsior purchased all assets of Nord Resources Corporation (Nord), as they relate to the JCM, through a court-appointed receiver.

1.3 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Project is located in a sparsely populated, flat to slightly undulating ranching and mining area about 65 road miles east of Tucson, Arizona. The Tucson metropolitan area is a major population center (approximately 1,000,000 persons) with a major airport and transportation hub and well-developed infrastructure and services that support the surrounding copper mining and processing industry. The towns of Benson and Willcox are nearby and combined with Tucson can supply sufficient skilled labor for the Project.

Access to the Project is via the I-10 freeway from Tucson and Benson to the west or Willcox to the east. The North Star deposit can be accessed via good quality dirt roads heading approximately 1 mile east from the south side of “The Thing” travel center and roadside attraction on the Johnson Road exit from I-10. Access to the Johnson Camp mine is via good quality dirt roads approximately 2.5 miles north of the Johnson Road exit from I-10.

The elevation on the property ranges from 4,600 to 4,900 feet above mean sea level in the eastern Basin and Range physiographic province of southeastern Arizona. The climate varies with elevation, but in general the summers are hot and dry, and winters are mild.

Vegetation on the property is typical of the upper Sonoran Desert and includes bunchgrasses, yucca, mesquite, and cacti.

1.4 HISTORY

There is no direct mining history of the North Star deposit; however, the district has seen considerable copper, zinc, silver, and tungsten mining beginning in the 1880’s and extending to the present day. Modern mining and leaching operations at the Johnson Camp Mine, began in the 1970s by Cyprus Minerals. Successor owners and operators include Arimetco, North Star, Summo Minerals, and Nord Resources Corporation. Nord mined fresh material until mid-2010 and maintained leaching operations until late 2015, when the property was purchased by Excelsior.

In 1970, a division of the Superior Oil Company (“Superior”) joint ventured into the northern half of the North Star deposit with Cyprus and the private owners (J. Sullivan, pers. com.). During the early 1970s, Superior did most of the drilling and limited metallurgical testing on North Star and by early 1974 had defined several million tons of low-grade acid-soluble copper mineralization.

1.5 GEOLOGICAL SETTING AND MINERALIZATION

There are several oxide copper deposits controlled by Excelsior, North Star, South Star and the Johnson Camp Mines, all situated in the Mexican Highland section of the Basin and Range physiographic province. The province is characterized by fault-bounded mountains, typically with large igneous intrusives at their cores, separated by deep basins filled with Tertiary and Quaternary gravels.

The Gunnison Project (North Star) lies on the eastern edge of the Little Dragoon Mountains. The ages of the rocks range from 1.4-billion-year-old Pinal Group schists to recent Holocene sediments. The southern portion of the Little Dragoon Mountains consists predominately of the Tertiary Texas Canyon Quartz Monzonite whereas the Pinal Group schists and the Paleozoic sediments that host the regional copper mineralization dominate the northern half.

Copper sulfide mineralization has formed preferentially in the proximal (higher metamorphic grade) skarn facies, particularly along stratigraphic units such as the Abrigo and Martin Formations near the contact with the quartz monzonite and within structurally complex zones. Primary mineralization occurs as stringers and veinlets of chalcopyrite and bornite. Primary (unoxidized) mineralization remains “open” (undetermined limits) at depth and to the north, south, and east.

Oxidation of the mineralization occurs to a depth of approximately 1,600 feet, resulting in the formation of dominantly chrysocolla and tenorite with minor copper oxides and secondary chalcocite. The bulk of the copper oxide mineralization occurs as chrysocolla, which has formed as coatings on rock fractures and as vein fill. The remainder of the oxide mineralization occurs as replacement patches and disseminations.

1.6 DEPOSIT TYPES

The North Star deposit is a classic copper-bearing, skarn-type deposit (Einaudi et al., 1980; Meinert et al., 2005). Skarn deposits range in size from a few million to 500 million tons and are globally significant, particularly in the American Cordillera. The North Star deposit is large, being at the upper end of the range of size for skarn deposits and is associated with a mineralized porphyry copper system that has been largely unexplored.

1.7 EXPLORATION

Since North Star's discovery, numerous companies have explored the area. During this time period, extensive drilling, and assaying, magnetic and IP geophysical surveys, metallurgical testing, hydrological studies, ISR tests, and preliminary mine designs and evaluations have occurred. The focus since the 1970's has been to utilize ISR or a combination of ISR and open pits as a potential mining strategy.

Mr. Stephen Twyerould first became involved with the Gunnison Project in mid-2005 and AzTech (Excelsior precursor) became involved in mid-2006. Since that time, significant work has been completed such as cataloguing, reviewing, and compiling high-quality historical data spanning over thirty years of investigations by Superior Oil and Gas, Cyprus, Quintana, CF&I, Magma Copper Corporation, Phelps Dodge Corporation, and James Sullivan. Excelsior conducted detailed ground magnetics over the exploration targets in June 2011.

Excelsior initiated a re-logging program in December 2010 that was completed in the third quarter of 2011. In addition, a re-assaying program began in March 2011 during which all of the Magma holes were re-assayed. In May 2011, a re-assay program was initiated for the Quintana Minerals holes (DC, S, and T series) to include sequential copper analyses for acid-soluble copper (ASCu). Previous results only included total copper (TCu) assays.

1.8 DRILLING

The North Star deposit drillhole database includes 88 historical drillholes that were completed by several companies. These holes extend to a depth of approximately 2,450 ft below the surface at North Star and cover an area of approximately 310 acres, with additional drilling extending beyond this area. There is a slightly higher density of drilling along the central axis of the North Star deposit. The 88 holes drilled by previous owners include 5,585 assays for total copper (TCu) and 2,754 assays for acid soluble copper as well as other assays for molybdenum, gold, silver, and tungsten.

Between 2010 and 2015, 54 diamond core holes were drilled by Excelsior for a total of 78,615 feet of drilling. Fifteen of these holes were for metallurgical samples and the rest were drilled for resource definition or exploration purposes (Table 10-6; Figure 10-2).

1.9 SAMPLE PREPARATION, ANALYSIS AND SECURITY

The laboratory sample preparation and analysis procedures used by the previous owners of the deposits are unknown; however, major commercial laboratories using best practices at the time completed the majority of analyses.

The data, information, samples, and core from the deposits have been under the control and security of AzTech Minerals since November 2006 and then Excelsior since October 2010. The original Information and samples are stored at a core storage facility in Casa Grande, with numerous copies held by Excelsior at its Phoenix, Arizona office.

It is the opinion of RESPEC Company LLC (RESPEC), the reviewer of the assay data for this report, that the sample procedures, processes, and security are reasonable and adequate.

1.10 DATA VERIFICATION

The verification of location and assay data in the drillhole database covers historic drilling and the verification of the data collected by Excelsior. No significant issues have been identified with respect to the data provided by Excelsior's quality assurance/quality control ("QA/QC") programs. QA/QC data are not available for the historical drilling programs at North Star, but Excelsior analyses dominate the assays used directly in the estimation of the mineral resources. Additionally, most of the historical data were generated by well-known mining companies, and the Excelsior drill data are generally consistent with the results generated by the historical companies.

Assaying and QA/QC procedures were industry standard. The TCu and ASCu assays used to estimate grades in the North Star model are acceptable for estimating mineral resources, based on RESPEC's review of the available data for repeat, check, duplicate, standard and blank assays, and on paired comparisons of assay data from different drilling campaigns.

1.11 MINERAL PROCESSING AND METALLURGICAL TESTING

There are two fundamental parameters to estimate overall copper recovery and acid consumption for a commercial-scale ISR operation: metallurgical recovery and sweep efficiency. In essence:

- Metallurgical recovery determines the amount and rate at which the copper dissolves from, and acid is consumed by, the rocks when contacted by the leach solution.
- Sweep efficiency determines how much of the copper in the ground will be effectively contacted by leach solution during the mining process.

In addition to historic testing, Excelsior has commissioned several rounds of varied metallurgical testing from as early as 2011 through 2015 that were intended to demonstrate the copper recovery and acid consumption which could be expected in an ISR operation for the Gunnison Project. The most recent testing was conducted at Mineral Advisory Group Research & Development, LLC (MAG) in Tucson, Arizona under the direction and control of Dr. Ronald J. Roman, P.E. of Leach, Inc., Tucson, Arizona. The primary objectives of this most recent group of tests were to:

- Determine the amount of copper that could be leached from the different ore types,
- Determine the relationship between the percentage of copper leached and the acid consumption for the different ore types, and
- Establish ISR metallurgical parameters at a feasibility level of confidence.

In addition to these tests, several rinsing tests were conducted for the purpose of determining a rinsing protocol to be employed after a block of ore had been leached by the ISR technique.

1.11.1 New Column Testwork

Since the 2014 PFS, two additional test programs have been completed. In the first of these, 19 modified column tests were run. The purpose of the new column testing was to determine how different ore samples would respond to the same leaching parameters to determine the variability of the ore with respect to the leachability.

Column tests were run on 51 to 52 kg of material crushed to minus 1 inch using 15 g/l sulfuric acid solution for up to 80 days. Separate columns were run for Lower Abrigo, Middle Abrigo, Upper Abrigo, and combined Martin and

Escabrosa formations. The results show that the recovery of acid soluble copper ranges from 65% to +90% but was dependent on rock type with Lower Abrigo formation having the highest and shortest duration leach cycle and the Martin-Escabrosa column tests having the lowest recovery over the longest period. Nearly all of the column leach plots of recovery vs time had positive slopes at the end of leaching, indicating the leaching process had not been completed in 80 days. As with prior test work, additional copper was recovered from the solubilization of minerals which do not report to the traditional ambient acid-soluble copper assay. These minerals include slowly soluble oxide copper minerals and transitional sulfides. Therefore, the conventional “acid-soluble copper assay” gives a good, if not conservative, approximation of the amount of copper which can be leached from the ore in the presence of a weak sulfuric acid solution.

1.11.2 Core Tray Tests

The second new test program termed “Core Tray” tests was intended to more closely simulate the in-situ recovery process than the modified column tests. In the Core Tray test pieces of core were mounted in epoxy in a tray with only the natural fracture surface exposed to the leach solution flowing across the top through the core tray.

Initially, the leach solution contained approximately 1.0 gram per liter (gpl) free acid. The free acid was increased in steps with time until it reached 15 gpl free acid. The data collected were recorded and an estimate of the following information about the response of the sample to leaching made:

- Incremental and cumulative recoverable copper, lbs/100 ft^2 of fracture surface
- Incremental and cumulative recoverable copper, wt%
- Incremental and cumulative gangue acid consumption, lbs/100 ft^2 of fracture surface
- Incremental and cumulative net acid consumption, grams of acid/gram of copper leached
- From these results the following were determined:
    - Recovery/time relationship
    - Acid Consumption/recovery relationship

The results of the Core Tray tests were stratified by rock type. Figure 1-2 is an example of the results for the Upper Abrigo formation. For all formations, the time vs recovery curves still have positive slopes during the test times of up to 200 days. Figure 1-3 is the Core Tray acid consumption data for the Upper Abrigo formation that indicates that the acid consumption curve steepens with recovery as expected.

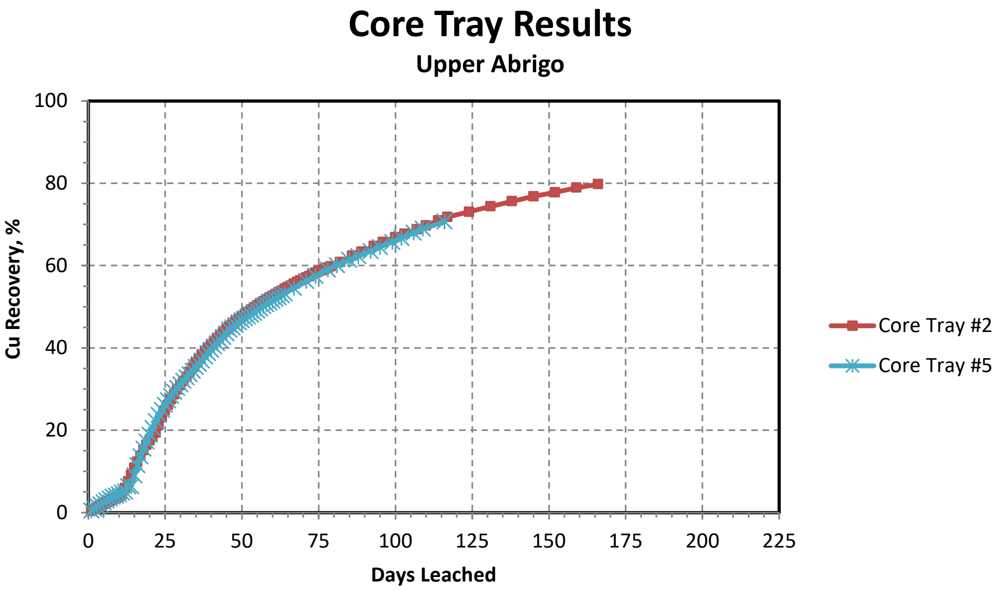


Figure 1-2: Core Tray Time vs Copper Recovery Results for Upper Abrigo Formation

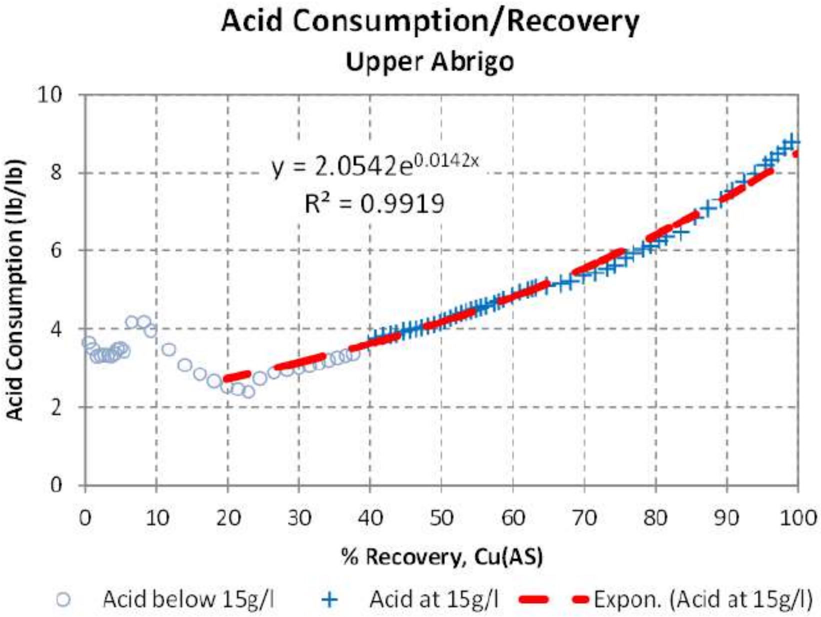


Figure 1-3: Core Tray Copper Recovery vs Acid Consumption Results for Upper Abrigo Formation

Sweep efficiency (or mining efficiency) for the North Star deposit is considered a function of fracture intensity. The most highly fractured rocks where most pieces of core are 4” or less are considered to have a sweep efficiency of 100%. In contrast, rocks that exhibit very weak fracturing are considered to have a low sweep efficiency of approximately 20%. The rocks at North Star exhibit a continuum of fracture intensities from very low (Fracture Intensity value of 1), to very high (Fracture Intensity value of 5), as determined by geological logging, geophysics and three-dimensional interpretation and modeling. To reflect this continuum, a polynomial algorithm was used to derive a predictive relationship between sweep efficiency and fracture intensity of the rocks.

Combining sweep efficiency with metallurgical test results and modelling of copper recovery it is possible to estimate cumulative copper recovery and acid consumption over a period of time for a 5-spot well pattern. The results of such calculations are shown in Table 1-1 below. The overall effect is for a weighted average total copper recovery of approximately 48% (acid soluble recovery of 74%).

Table 1-1: Predictive Model for Sweep Efficiency Factored, Cumulative Acid Soluble Copper Recovery and Acid Consumption for a 5-Spot Well Field Pattern

Cumulative Acid Soluble Cu Recovery (%)	Year 1	Year 2	Year 3	Year 4
Martin	40.2	55.8	65.9	72.8
Upper Abrigo	43.5	58.7	68.2	75.0
Middle Abrigo	42.0	57.6	67.6	74.9
Lower Abrigo	43.6	58.8	67.3	74.5
Bolsa, TQM, other*	43.6	58.7	67.2	74.4
Weighted average	41.9	57.3	67.0	74.0
Cumulative Acid Consumption (lb/lb)	**Year 1**	**Year 2**	**Year 3**	**Year 4**
Martin	5.2	6.8	8.6	10.1
Upper Abrigo	4.7	6.0	7.5	8.9
Middle Abrigo	5.1	6.9	8.6	10.2
Lower Abrigo	3.7	5.0	5.8	6.9
Bolsa, TQM, other*	4.5	4.6	4.9	5.2
Weighted average	4.8	6.4	7.9	9.3

*The Bolsa Quartzite, TQM and other minor host rocks make up less than 2% of the Probable Reserve and were not tested but are expected to perform similar to or better than the Lower Abrigo.

1.11.3 Wellfield Issues

Operation of the Gunnison wellfield has revealed that solution injection flowrates diminish with time, but that substitution of water for injected acidified raffinate restores the flowrate. The interpretation of this behavior is that CO_2 gas is accumulating in flow channels, impeding solution flow through the formation. Flow in the field was improved by flushing with neutral water under pressure, indicating that pH is highly likely to control gas solubility. After flushing with low solute water, flow improves substantially. Repeated acid leaching then repeated this cycle of leaching followed by loss of flow and the need to flush with fresh water again. The sustained improvement of flow rates dues to the cycling of water and acid injection and recovery clearly indicates that the blocking mechanism is remediated on water injection but exacerbated by acid injection. Given the CO_2 comes from the calcite in the fracture system, then once this calcite has been dissolved or removed from a particular fluid pathway, CO_2 gas will no longer form along that pathway and restrict acid injection flows. In general, the data indicate flow rates can be improved with repeated cycling of freshwater injection, acid injection and recovery.

It has been documented experimentally that multiple immiscible phases flowing intermingled through a porous medium will generally do so at lower effective rates than either phase flowing on its own. According to relative permeability theory, the higher the saturation of one immiscible flowing phase (as a fraction of the connected pore space), the lower the effective permeability of the other phase. The magnitude of this reduction is generally larger for the wetting phase, which is acid/water for the Gunnison case. Each phase will generally establish an “immobile” saturation below which it cannot flow due to capillary pressure and interfacial tension effects. Typically, the phase with the greater affinity for the solid surface (called the wetting phase) will have a higher immobile saturation than that of the non-wetting phase.

Geochemical modelling and literature regarding CO_2 sequestration in saline waters indicates neutralized raffinate would have a similar albeit reduced capacity to dissolve/remove CO_2 gas as does freshwater. For each new block of wells, Excelsior plans to use acidified leach solution to dissolve calcite creating CO_2 gas and then cycle it with neutralized solution to dissolve the CO_2 and restore flow to the formation. Excelsior's proposal to use neutralized raffinate to flush out CO_2 and dissolve calcite on a cyclical basis over a 12-to-15-month period is supported by the limited wellfield data available to date and is supported by the chemistry of raffinate versus water's ability to sequester CO_2.

1.12 MINERAL RESOURCE ESTIMATE

The North Star deposit mineral resource reported by RESPEC (M3, 2017) have been updated to include resources on lands newly acquired by Excelsior with the purchase of the Johnson Camp property. Table 1-2 is a summary of the oxide, transitional, and sulfide mineral resource tabulated at a total copper cut-off of 0.05% for oxide and transitional and 0.30% for sulfide. Table 1-3 is a summary of the sulfide portion of the deposit at a 0.50% TCu cut-off. Measured and indicated oxide and transition mineral resources are inclusive of mineral reserves.

Table 1-2: North Star Oxide, Transition, and Sulfide Mineral Resource Summary

Effective October 1, 2016

Resource Category	Short Tons (millions)	Total Cu (%)	Contained Copper (million pounds)
Measured	200.7	0.36	1,439
Indicated	710.8	0.27	3,875
Measured + Indicated	911.6	0.29	5,315
Inferred	240.9	0.22	1,070
0.05% TCu cut-off for oxide and transitional, 0.30% TCu cut-off for sulfide			

Table 1-3: North Star Deposit Total – Copper Resources

Effective October 1, 2016

Oxide Resources @ 0.05% TCu Cut-off

Resource Class	Short Tons (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	157.2	0.38	1.201
Indicated	502.1	0.28	2.782
Measured + Indicated	659.3	0.30	3.983
Inferred	108.0	0.16	0.351

Transitional Resources @ 0.05% TCu Cut-off

Resource Class	Short Ton (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	41.9	0.27	0.227
Indicated	172.0	0.23	0.785
Measured + Indicated	213.9	0.24	1.02
Inferred	79.2	0.18	0.279

Oxide + Transitional Resources @ 0.05% TCu Cut-off

Resource Class	Short Tons (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	199.1	0.36	1.427
Indicated	674.0	0.27	3.567
Measured + Indicated	873.2	0.29	4.995
Inferred	187.2	0.17	0.630

Sulfide Resources @ 0.30% TCu Cut-off

Resource Class	Short Tons (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	1.6	0.39	0.012
Indicated	36.8	0.42	0.308
Measured + Indicated	38.4	0.42	0.32
Inferred	53.7	0.41	0.44

Notes:

1. Mineral Resources are inclusive of Mineral Reserves.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Oxidized + Transitional Mineral Resources are reported at a 0.05% total-copper cut-off in consideration of potential mining by in-situ recovery.
4. Sulfide Mineral Resources are reported at a 0.30% total-copper cut-off in consideration of potential mining by open-pit extraction.
5. Rounding may result in apparent discrepancies between tons, grade, and contained metal content.
6. The Effective Date of the mineral resource estimate is October 1, 2016.

1.13 MINERAL RESERVE ESTIMATE

The mineral resource estimate discussed in Section 14 is used to estimate the probable mineral reserve estimate for the North Star deposit. Table 1-4 shows the diluted Probable mineral reserve estimate as defined for the PFS. The mineral reserves are in the Probable category. The estimate includes material from the measured and indicated categories of the mineral resource and excludes inferred mineral resources. It does not include material from the sulfide zone.

Table 1-4: Probable Diluted Reserve Estimate (October 2016)

Short Tons (millions)	782.2
TCu Grade (%)	0.29
TCu Contained Copper (million lbs)	4,505
Average Total Copper Recovery (%)	48
Recoverable Copper** (million lbs)	2,155
*Probable reserves were defined from measured and indicated resources. Inferred resources were not converted into reserves. **Total includes copper losses to water treatment.	

The Probable mineral reserve estimate summary prepared for the PFS was created using data and input from RESPEC and Excelsior. It is based on RESPEC's resource estimate detailed in Section 14. It assumes the use of ISR as a mining method, which requires a wellfield (injection and recovery wells) and pumps pregnant leach solution to an SX-EW plant to recover the copper. The boundaries of the Probable mineral reserve were defined using economic parameters and then further modified to consider lost production under the freeway and along some lease boundaries. Excelsior developed a wellfield / production schedule for the Project, and the mineral reserve estimate is the sum of the production schedule, which is discussed in Section 16.

1.14 MINING METHOD

Excelsior proposes to use the ISR method to extract copper from oxide mineralization located within the North Star Deposit (see location map on Figure 1-1). The ISR mining method was based on the fractured nature of the host rock, the presence of water-saturated joints and fractures within the ore body, copper mineralization that preferentially occurs along fracture surfaces, the ability to operate in the vicinity of Interstate 10, and to avoid the challenges of open pit mining in an area with alluvium overburden thickness ranging from approximately 300 feet to 800 feet.

The forecasted copper production for the Gunnison Project commences with an initial stage of 25 million pounds per annum (mppa) from Years 1 through 3, followed by a second stage of production of 75 mppa in Years 4 through 6, and followed a third stage reaching 125 mppa from Year 7 through Year 20 with a decline in production beginning in Year 21 through the end of the mine life in Year 24. The total amount of copper production forecasted over the 24-year LoM is approximately 2,165 million pounds. The following inputs and assumptions were used to generate the copper extraction forecast:

- Key physical parameters from RESPEC's 100-foot x 50-foot resource block model such as rock type, specific gravity of each rock type, total copper percentage and acid soluble copper percentage, fracture intensity, ore thickness, water table elevation, ore greater than 0.05% total copper, and lease boundaries (see Section 14 for details).
- Incremental acid soluble copper recovery curves over a 4-year recovery period and recovery factor (as discussed in Section 13.3); and
- Recovery well production rates described in Section 16.4.3.

ISR process injects a barren leach solution (lixiviant) with weak sulfuric acid into the ore body using a series of injection wells. The acidified solution dissolves oxide copper minerals as it migrates through the joints and fractures within the mineralized bedrock. Recovery wells surrounding each injection well extract copper-bearing pregnant leach solution (PLS) and combine to form the feed solution for the SX-EW process.

New blocks of wells require conditioning before leach solutions can be effective in removing copper due to the generation of CO_2 blocking the fluid flow paths, as presented in Section 1.11.3. Excelsior plans to alternate the circulation of acidified leach solutions to dissolve calcite with neutralized solutions to dissolve the CO_2 that is created the calcite.

Acidified raffinate will be introduced into half of the injection wells. The extracted solutions from the recovery wells will be pumped to the PLS pond and on to SX-EW. A portion of the raffinate equal to the flow rate of acidified raffinate solution will be diverted to the water treatment plant (Section 1.15.1) to be neutralized and pumped to the other half of the injection wells. Thus, half the wells will be receiving acid and half neutralized solution to flush out the CO_2. Approximately every month, the wells receiving acidified raffinate will be switched to neutralized raffinate and vice versa. The process will continue until the flow rates of the injection wells stabilize at the “pre-acid” flow rates signaling that the calcite has been removed and copper leaching can begin.

The wellhead design enables each of the wells to be operated as either a recovery well or an injection well. The change is facilitated by adjustments at the valve skids and connecting or disconnecting power to the well pump. This enables the operators of change and reverse the flow paths of solutions in the formation to resolve flow problems, reduce downhole scaling, and improve copper removal by varying the flow and direction of solution movement.

An additional one year of conditioning is projected to resolve the flow problems due to CO_2 generation in the fractures. Once the new wells are able to sustain pre-condition (before acid introduction) flow rates, the wells can be brought into full production. Copper and acid losses to neutralization in the water treatment plant have been estimated and are included in the financial model.

The SX-EW facility is designed to recover copper from PLS at a copper feed grade of approximately 1.6 gram per liter (gpl) (1.5 gpl net copper grade) to produce cathode-quality copper with 99.999% purity. The anticipated PLS flow rates are 3,800 gallons per minute (gpm) for Stage 1, 11,500 gpm for Stage 2, and 19,500 gpm for Stage 3. The process solutions are piped to and from the SX-EW plants in high density polyethylene (HDPE) piping. The process consists of the following elements (schematic representation in Figure 1-4):

- ISR wellfield
- Wellfield and drilling services building
- Lined PLS and raffinate ponds
- Solvent Extraction (SX) plant
- Tank Farm for handling process liquids
- Electrowinning (EW) Tankhouse equipped with an Automatic Stripping Machine
- Electrical substation
- Sulfuric Acid Receiving/Storage
- Administration offices, Security Building, and a Change House
- Plant Warehouse, Laboratory, and Plant Maintenance buildings
- Water treatment plant with a Clean Water Pond, Evaporation Ponds, and Solids Impoundments

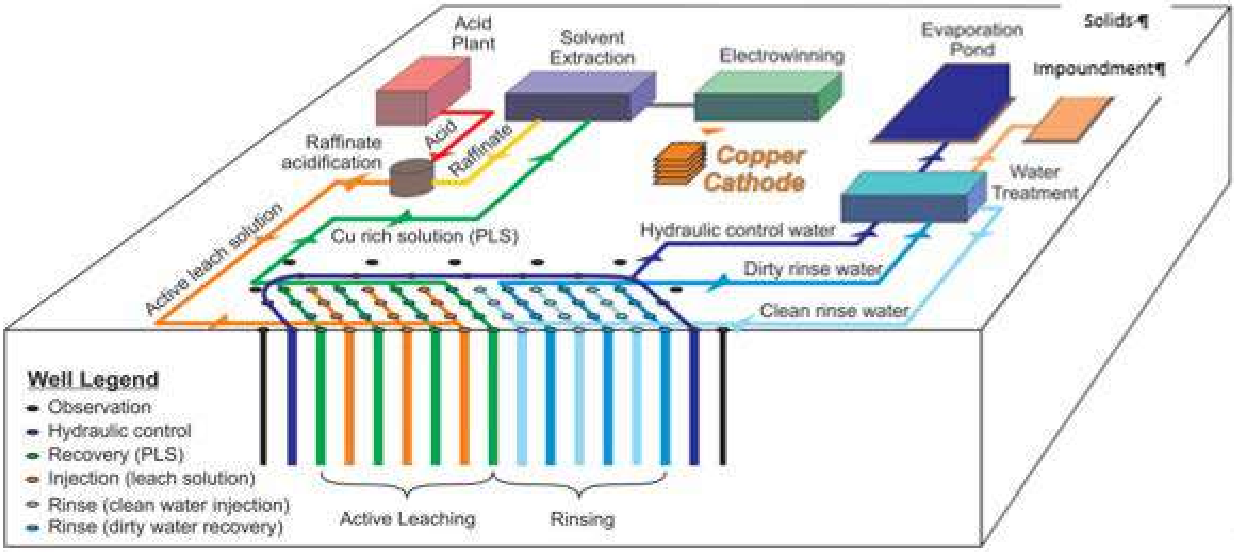


Figure 1-4: Recovery Process

Depleted portions of the mineralized zone are rinsed by injecting non-acidic (clean) water to flush out the leach solution and reduce the metals and other constituents to acceptable concentrations. A block of mineralization is considered depleted when the copper grade of the recovered PLS falls below an economic cutoff. The rinsing process consists of a three-stage process consisting of an early rinse, rest period, and late rinse. Early rinsing flushes and dilutes the PLS remaining in the formation.

At a certain level of dilution, typically 90 percent, the wellfield is shut in allowing the intrinsic neutralization capacity of the formation to neutralize the acid in the diluted solution. The final stage of rinsing flushes out the neutralized solution until all regulated constituents are below stipulated concentrations. Injection and recovery wells are abandoned by grout injection from the bottom of the well when wellfield closure criteria have been satisfied.

Production wells will be designed to meet Underground Injection Control Class III requirements and will be constructed in accordance with the guidelines of ADEQ's Mining BADCT Guidance Manual (2004). Boreholes will be drilled using air rotary, direct mud rotary, reverse circulation mud rotary, or casing advance drilling methods. Borehole diameters will be sufficient to allow for installation of casing that will accommodate the pumps. The cased portions of the boreholes will be 12-inch nominal (small diameter injection/recovery wells and hydraulic control wells), 15-inch nominal (large diameter injection/recovery wells), and 10-inch nominal (observation and POC wells). The open borehole sections within bedrock will be 5 and 7 inches in nominal diameter. Well screen may be used if the borehole is unstable. The outer annulus of the cased portions of Class III wells will be grouted to 100 feet above the basin fill/bedrock contact (or static groundwater level, whichever is shallower). The ISR operations do not require hydraulic fracturing of the mineralized formation.

1.15 PROJECT INFRASTRUCTURE

The primary access to the site will be from Interstate 10 via the Johnson Road exit between Benson and Willcox, Arizona. The mine access road to the Johnson Camp side of the property is approximately one mile long to the north. A new, asphalt paved access road to the Gunnison wellfield and plant site will head south and east from the Interstate exit for a distance of one mile.

The Johnson Camp mine has existing plant facilities, ponds and infrastructure and is currently in use for Stage 1 production. The JCM facilities will continue to be used for production at its rated capacity of 25 mppa for Stages 1, 2, and 3 of the mine plan.

The Gunnison SX-EW plant will be constructed for Stage 2 production in Year 3 for operation in Year 4 at an initial rate of 50 mppa. The electrowinning building (tankhouse) will be a steel building with corrugated metal roofing and siding. It will contain 80 electrowinning cells on one end of the building and the Automatic Stripping Machine, and the cathode handling equipment are on the other, with a paved cathode storage area outdoors. For Stage 3 production, 80 EW cells will be added to the opposite side of the building, mirroring the first 80 cells.

The Gunnison Tank Farm will be built for Stage 2 and have tankage added in Stage 3. It is uncovered and located downhill from the SX area and the tankhouse to facilitate gravity drainage of solutions to the Tank Farm. The Tank Farm has a concrete containment that drains to a sump with an oil-water separator to return spilled liquid to the proper location for recycling. There is a Plant Runoff Pond located downstream of the Tank Farm to capture any surface flows in the event of an upset condition at the plant.

Ancillary facilities needed to support the Gunnison Project include buildings, ponds, tanks, and trenches. Ancillary buildings include an Administration Building, Warehouse, Plant Maintenance building, Change House, Security Building (gatehouse), Wellfield Maintenance Building, Water Treatment Plant, and Sulfuric Acid Plant-Cogeneration complex. Other facilities will include ponds, and tanks. The Gunnison Project will use the existing assay lab located at the Johnson Camp mine.

Power for the facility will be taken from an existing 69 kilovolt (kV) power line feeding the existing Johnson Camp Mine located on the north side of I-10. The power line approaches the plant site along the eastern boundary of Section 31 shown on Figure 4-2. The existing power line is owned by the Sulfur Springs Valley Electric Cooperative Inc. located in Willcox, Arizona. A 69kV power line was constructed from the JCM substation and followed the pipeline route to the Gunnison project area. A substation and transformer near the PLS pond are used to power the Stage 1 operation. This powerline will be replaced for Stage 2 operation. A tap will be taken from the power line on the eastern boundary and connected to the plant main electrical substation located near the EW building.

Fresh well water will be taken from existing wells and mine shafts on the Johnson Camp property and pumped to an existing 500,000-gallon fresh water/fire water storage tank located on Water Tank Hill at the JCM site. The lower 300,000 gallons in the storage tank will be reserved for fire water. Process water for plant use will be taken from the storage tank above this reserve level for fire suppression. The JCM site has an existing potable water system. The Gunnison site will be served by an additional 7,000-gallon potable water tank and chlorination system, which will use a water supply well to be constructed east of the operation during Stage 2 development.

1.15.1 Water Treatment Plant

Water treatment is required for two primary purposes, neutralizing raffinate for dissolution of carbon dioxide during wellfield conditioning and removing acid, metals, and sulfate from solutions to rinse the formation after it is depleted of copper. The neutralization process requires raising the pH to near neutral (~7). The removal of metals and sulfate requires nanofiltration in addition to the neutralization. Rinsing of the formation is not scheduled to begin until Year 8 of the mine plan, so the water treatment plant (WTP) is planned for construction in phases.

Since wellfield conditioning is required to prepare the ore blocks for copper production, Phase 1 of the WTP (Train A) will be constructed in Year -1. Phase 2 is required to increase the capacity of the Train A neutralization system in advance of the Stage 2 ISR production expansion to 75 mppa. The Phase 3 WTP expansion adds a second train (Train B) that includes two stages of pH adjustment, clarification, filtration, nanofiltration, and desaturation to produce low-sulfate water for rinsing. Phase 4 adds additional capacity to Train B to produce a higher flow rate of low-sulfate water for rinsing.

Solids produced by the Phase 1 WTP will be discharged to the Evaporation Pond. Starting with Phase 2, all of the solids from the various clarifiers are discharged to a solids impoundment for dewatering and final solids disposal. Water drained from the solids impoundment or pumped from the supernatant pool in the impoundment is returned to the WTP as influent to Train A.

1.15.2 Acid Generation Plant

A sulfur-burning sulfuric acid plant is scheduled to be constructed for use in Stage 3. A PFS-level design and cost estimate were produced for this study by NORAM Engineering (2022). The plant is designed to produce 1,650 tonnes of concentrated sulfuric acid per day. Sulfuric acid generation uses molten sulfur to make sulfuric acid through the process of oxidation, which produces heat. Waste heat from the acid making process produces steam as a by-product to generate 9 MW of electrical power, which reduces operating costs from $150/short ton to $52/short ton of acid. The facility includes molten sulfur day tanks, sulfur burner and waste-heat boiler, drying and adsorption tower area, cogeneration building, water treatment building, power distribution building and substation, cooling towers, office building, sulfuric acid storage area, and a rail yard for unloading molten sulfur and sulfuric acid.

Molten sulfur is received at the plant in rail tank cars with a payload capacity of approximately 100 tons. The rail cars can be heated by steam to re-liquify the sulfur that may have solidified in transit. The molten sulfur is discharged to a receiving pit and pumped into heated storage tanks. Molten sulfur is oxidized with high-pressure air and converted to 98.5% sulfuric acid through a series of Adsorption, Interpass, and Final towers and sent to storage tanks.

Steam produced in the Waste Heat Boiler from cooling the sulfur burner is superheated and used to create electrical power in the steam turbine generator (STG). Steam production is proportional to the acid production: approximately 1.25 tons of steam per ton of acid. The Start-up/Emergency Boiler creates low-pressure steam needed to start up the sulfur burner and provide low-pressure steam when the process is down. Some low-pressure steam is extracted from the STG and used in the deaerator and molten sulfur heating system during the acid-making process. Condensate from the STG system is collected and polished (treated) to be reused as waste heat boiler feed water.

1.16 MARKET STUDIES AND CONTRACTS

The Company has an offtake agreement for the copper cathodes produced by the Project that is negotiated annually. The current agreement is for payment at the average monthly HG Copper COMEX settlement price.

The use of consensus prices obtained by collating the prices used by peers or as provided by industry observers and analysts is recognized by the Canadian Institute of Mining and Metallurgy (CIM) for technical reports and has the advantage of providing prices that are acceptable to a wide body of industry professionals (peers). These prices are generally acceptable for most common commodities, major industrial minerals, and some minor minerals.

The PFS has selected $3.75/lb copper for all years.

Market studies indicate that the long-term prices for the major reagents are as follows.

Sulfuric Acid	$150/st
Molten Sulfur	$130/st
Lime	$170/st

The price for sulfuric acid is predicted to be $150/st by truck and $130/st by rail after Year 6 for the remainder of the life of the project. The price of lime is $170/st based on quotes for supply and estimates for the transportation costs.

1.17 ENVIRONMENTAL AND PERMITTING

1.17.1 Environmental Studies

Anthropological and floral and faunal studies were carried out by Excelsior in 2010 over the wellfield area. There is no potential for U.S. Fish and Wildlife Service endangered, threatened, proposed, and candidate species (special-status species) to occur in the study area.

An archaeological study was conducted that showed no cultural resource sites in the mining area. Further archeological and floral/faunal studies were conducted by WestLand Resources (2014) for areas covered by infrastructure such as the SX-EW plant, evaporation ponds, sulfuric acid plant and railway facilities. No cultural resource sites were identified.

1.17.2 Groundwater Modeling

A groundwater model was constructed by Clear Creek Associates (CCA) to cover the greater Gunnison Project area of 87.8 square miles in support of the Aquifer Protection Permit (APP) and Underground Injection Control Permit (UIC) applications. The model was constructed using a number of extensive datasets created by Excelsior, including a detailed mapping of fracture intensity, which is key to groundwater flow in the Project area.

The model demonstrates that control of mining solutions can be maintained with hydraulic control wells located around the wellfield. Predicted pumping rates for hydraulic control presently range from a total of 15 gpm to approximately 200 gpm in later years. Water produced during hydraulic control will be used in the process, recycled, or evaporated.

1.17.3 Water Management

The Project's water management plan was designed to make the most efficient use of water resources and eliminate discharges. During Stage 1 of the Gunnison Project, existing lined ponds at JCM will be used. As production increases and Stage 2 and Stage 3 facilities are constructed south of Interstate 10, new solution and water management ponds will be constructed to support the Project. These include: the PLS pond, Raffinate pond, Plant Runoff Pond, Clean Water pond, Recycled Water pond, Evaporation ponds, and Solids Impoundments, which contain the precipitate from the Water Treatment Plant. With the exception of the Plant Runoff and Clean Water ponds, the ponds will be constructed with a double liner and a leak detection and recovery system between the liners according to prescriptive BADCT design.

Excess solutions will initially be routed to evaporation ponds where mechanical evaporators will be installed. During later stages of the Project, when the Water Treatment Plant is in operation, approximately 80% of the influent will be treated for reuse in the process or for rinsing, and it will report to the Clean Water Pond. The solids from the WTP process will be pumped to the Solids Impoundments as precipitated solids and the concentrate brine and filter backwash from the WTP will be pumped to the evaporation ponds. Groundwater produced from hydraulic control pumping will be conveyed to the Clean Water Pond or, if impacted by PLS, to the Evaporation Pond.

1.17.4 Geochemical Modeling

Geochemical modeling of raffinate and rinsing solutions indicates that the following 3-step closure strategy will result in concentrations of regulated constituents below Aquifer Water Quality Standards:

- Step 1: Rinsing 3 pore volumes
- Step 2: A rest phase (approximately 200 days or more) until near neutral pH conditions are attained
- Step 3: Rinsing at least 2 additional pore volumes

- Hydraulic control is maintained during rinsing

1.17.5 Community Relations

Excelsior has developed a broad-based community relations and stakeholder outreach program in support of the Gunnison Project. Elements of this program include:

- Targeted stakeholder outreach to government, community, business, non-profit and special interest groups, and leaders at the local, county and state level.
- Development of community relation and communication tools and resources (e.g., Project website, Project e-newsletter, and presentation materials).
- Public open houses and technical briefings when appropriate.

Crucial elements of Excelsior's community relations efforts will involve ensuring consistent and ongoing communication with all stakeholders and providing opportunities for meaningful two-way dialogue and active public involvement. Excelsior will focus on ensuring the public benefits related to the Gunnison Project, such as employment opportunities, supplier services, infrastructure development and community investment are optimized for the local communities.

1.17.6 Economic Benefits

Excelsior commissioned an Economic Impact Study through Arizona State University's W. P. Carey School of Business which forecasts the increase in economic activity within Arizona during the construction phase and life of the mine. The economic impact of mine development to surrounding communities and the State in general:

- Over 800 direct and indirect new jobs.
- Employment benefits are distributed in mining, construction, professional & technical services, and government sectors as well as other sectors.
- The annual average value added to Arizona's Gross State Product (GSP) during the entire Project life – pre-production, production, and closure – is approximately $109 million with approximately $28 million added within Cochise County. The total addition to the GSP is $2.9 billion, with $757 million locally within Cochise County.
- Economically modeling predicts the Project will have an average annual impact on state revenues of $10.9 million for a total impact of $295 million.

1.17.7 Permitting

The Gunnison Copper Project is permitted to the rate of 125 mppa of copper production and has been in early-stage operation since December 30, 2019. The Project is in compliance with all existing permits. There have been no new environmental related studies since the issuance of the various permits, therefore, the discussions on plans have been removed from this updated report.

Key federal, state, and local government environmental permits are listed in Table 1-5 along with permits that may be required when the Project expands onto BLM lands or additional processing facilities are planned at the wellfield.

Table 1-5: Environmental Permits

Agency	Permit	Description	Citation	When Required/ Permit No.
Federal				
Bureau of Land Management	Mining	1. Notice Level Operations may not exceed 5 acres. 2. All operations on public lands that disturb the surface require a Plan of Operations will require an environmental assessment or environmental impact statement and posting of a reclamation bond.	43 CFR §3809	Applicable only when mining on BLM lands
US Environmental Protection Agency (EPA)	Underground Injection Control	Establishes an Area of Review (AOR), beyond which mining related solutions shall not pass. Covers all subsurface well activities, i.e., monitor wells and injection/recovery wells located within the AOR. Will require amendment for life-of-mine production.	40 CFR §§124, 144, 146, 147 and 148	R9UIC-AZ3-FY16-1
US Fish & Wildlife Service	Incidental Take Permit	Mining activities that may affect species listed as endangered or threatened need to conduct studies to identify any targeted species and to apply for a permit to conduct their activities. Any identified threatened or endangered species identified in pre-mining surveys would need to be mitigated before mining could proceed.	50 CFR Sections 7 and 10	Non previously identified. New studies required prior to disturbing new ground
Nation Historic Preservation Act	Consultation and Mitigation	Requires Federal agencies to take into account the effects of their undertakings, such as construction projects, on properties covered by the NHPA.	42 CFR §137.88	None previously identified. New studies required prior disturbing new ground.
Section 404 of the Clean Water Act	Jurisdictional Waters of the US	Regulates the discharge of dredged or fill material into waters of the United States,	33 CFR §323	No jurisdictional waters identified
State of Arizona				
Arizona Department of Environmental Quality (ADEQ)				
Air Quality Division	Air Quality Control Permit	Ensures air pollutants from any source does not exceed the National Ambient Air Quality Standards. Will require amendment to incorporate for the Acid Plant option.	ARS §49-402	AQP-71633
Groundwater Section	Aquifer Protection Permit	Covers surface impoundments, solid waste disposal facilities, mine tailings piles and ponds, heap leaching operations. This permit requires designs for the proper management of process facilities, ponds, tailings impoundments, and includes monitoring requirements to ensure compliance with the permit. Will require amendment for life-of-mine production.	AAC R18-9 Articles 1 - 4	P-511633
	Reclamation & Closure Plan for	Reclamation plan; estimated cost of executing reclamation plan and surety bond. The reclamation plan includes reclamation	AAC R18-9 Articles 1 - 4	P-511633

Agency	Permit	Description	Citation	When Required/ Permit No.
	Facilities covered by APP	activities and post-closure monitoring, and bonding estimate must be approved by the agencies and the bond must be posted prior to commencement of construction. Will require amendment for life-of-mine production.		
Waste Management Division	EPA ID Number	Generators of hazardous waste must have an EPA ID prior to offering the waste for shipment.	ARS §49-922	Currently covered under Johnson Camp
	Pollution Prevention Plan	Plan identifying opportunities to reduce waste.	ARS §49-961 thru 973	Annually
	Toxic Release Inventory	Submit Form R for quantity of copper in waste rock.	40 CFR 372	Annually
Arizona Dept of Water Resources	Dam Safety Regulations	Obtain permit for qualifying dams and ponds	ARS §45-1201	Not Required
Arizona State Mine Inspector	Mined Land Reclamation Plan and Bond	Exploration and mining activities on private land with greater than 5 acres disturbance. Does not include facilities covered in Aquifer Protection Permit.	AAC R11-2-101 thru 822	Approved Oct 9, 2018
Arizona Department of Agriculture	Notice of Intent to Clear Land	Ensures enforcement of Arizona Native Plant Laws	ARS §3-904	60 days prior to disturbance
Arizona Game and Fish Department		Ascertain whether or not the mining operation would endanger fish and game habitat, etc.	AAC Title 12	No T&E Species identified
Arizona Department of Transportation	Encroachment Permit	Obtained to allow jack and bore installation of process solution pipelines under I-10.	AAC R17-3-502	Completed
State Historic Preservation Office		Submit a legal description with map of the area to be disturbed SHPO can inform applicants whether work will occur in a state designated historic district.	ARS §43-861	Only applies to public lands

1.18 CLOSURE AND RECLAMATION COSTS

All Project facilities governed by Arizona's Aquifer Protection Permit (APP) rules must be closed at the end of operations in accordance with the APP closure plan. Non-APP facilities, such as buildings and infrastructure, will be reclaimed in accordance with the Mined Land Reclamation Program overseen by the Arizona State Mine Inspector's Office. This program requires the development of reclamation plans that will ensure safe and stable post-mining land use. The plans must include cost estimates and financial assurance for implementing the reclamation plans.

APP-regulated impoundments, including the PLS, Raffinate, Recycled Water, and Evaporation Ponds will be closed in accordance with the approved closure plan. The solution ponds containing liquids (PLS, raffinate, pipeline draindown, etc.) will be emptied and cleaned. Liners will be inspected for signs of leakage. The soils beneath prospective defects will be investigated and remediated as necessary. After clearance, the liner materials will be folded into the bottom of the pond for burial in place. Perimeter berms above the natural land surface will be pushed into the pond to cover the liner, contoured, and revegetated to shed surface runoff and minimize infiltration. The impoundments containing solids (Evaporation and Solids Impoundments) will be closed in place and covered to minimize infiltration. The edges of the liner will be folded inward and covered with a low permeability cap. The cap will be contoured and revegetated to shed surface runoff and minimize infiltration.

1.19 CAPITAL AND OPERATING COSTS

Capital and operating costs for the Gunnison Copper Project were estimated on the basis of the prefeasibility design, estimates of materials and labor based on that design, analysis of the process flowsheets and predicted consumption of power and supplies, budgetary quotes for major equipment, and estimates from consultants and potential suppliers to the Project.

1.19.1 Capital Cost

Capital cost (CAPEX) is divided into initial and sustaining capital costs. Stage 1 of the original Gunnison Project was constructed in 2020 with acid injection commencing in December of 2020.

For this study, Pre-Stage 2 initial capital is defined as improvements to Stage 1 in Years -2 and -1 of the Gunnison wellfield, mainly the addition of the Phase 1 water treatment plant. This plant is needed to neutralize raffinate to dissolve CO_2 from the subsurface.

Sustaining capital costs include the ongoing year-by-year additions to wellfield drilling and development, construction of the Stage 2 SX-EW and Stage 3 SX-EW plants on the Gunnison side of the property, each adding 50 million pounds per annum (mppa) of copper cathode capacity, the addition of a new 69 kV to 24.9 kV Gunnison substation, three expansions of the water treatment plant, the addition of water ponds and solids ponds to support plant operation and water treatment, the construction of a sulfur burning sulfuric acid and cogeneration plant, and the addition of a railroad siding and railcar unloading facility.

Table 1-6: Summary of Capital Cost Spending Over the Life-of-Project

Stage	Copper Production	Description	Total ($000)
Initial Capital	25 mppa	Pre-production wellfield drilling, development & operations; Installation & operation of Phase 1 Water Treatment Plant	$47,621
Phase 2 WTP (Year 2)		First expansion of water treatment; Installation of Feed Water Pond, Recycled Water Pond; & Solids Ponds 1A & 1B	$7,629
Stage 2 (Years 2& 3)	75 mppa	Gunnison 50 mppa SX-EW; 80 EW cells; New Raffinate pond; new Gunnison substation, Gunnison ancillary bldgs. to support drilling and ISR mining, and the Railyard	$178,043
Stage 3 (Year 5 & 6)	125 mppa	Wellfield Expansion; Gunnison 50 mppa SX-EW; 80 EW cells; Water Treatment Plant (WTP); Wellfield expansion; Railroad Siding & Railcar Unloading	$104,263
Acid Plant (Years 5 & 6)		Sulfuric Acid Plant, Molten Sulfur Handling, Cogen Plant; Boiler Water Treatment (Optional)	$159,860
Phase 3 WTP (Year 7)		Second expansion of water treatment plant for membrane filtration	$47,435
Wellfield Development Sustaining Capital (All years)		All wellfield drilling costs, wellfield capital equipment and wellfield infrastructure development, Solids Ponds	$526,990
Phase 4 WTP (Year 17)		Third expansion of water treatment plant for additional membrane filtration capacity	$8,968
Total		**Initial & Sustaining Capital Cost**	$1,080,808

- The capital cost estimates on which this prefeasibility study is based were prepared from a level of engineering commensurate with a +/- 20% level of accuracy except where noted. Indirect capital costs were factored from the direct field cost.
- Indirect field mobilization is 1.5% of the direct field cost without mobile equipment.
- Temporary construction facilities is 0.5% of direct cost less mobile equipment.

- Construction power is 0.1% of direct cost less mobile equipment.
- Engineering Procurement and Construction management is 16.8% of the direct cost plus the indirect cost listed above.
- EPCM temporary facilities and utility setup were estimated as 0.5% of total constructed cost.
- Commissioning was estimated to cost 1% of plant equipment less mobile equipment.
- Vendor supervision is estimated as 1.5% of plant equipment costs during construction and 0.5% of plant equipment costs, each, for pre-commissioning and commissioning.
- Capital spare parts are estimated as 2.0% of plant equipment and commissioning spares are 0.5% of plant equipment.

Sustaining capital costs commences in Year 1 of the mine schedule and includes all capital expenditures that occur after pre-conditioning of the existing well block is completed by the end of Year -1. Starting in Year 1 Excelsior expects that Stage 1 production of PLS production will ramp up to a rate of 4000 gallons per minute (gpm). Stage 1 production will proceed for a period three years during which various wellfield installations will be made.

Sustaining capital costs include all capital expenditures that occur after production begins. For the Gunnison Project, major sustaining capital expenditures are planned for Year 3 when Stage 2 of the Project is constructed and Year 6 with Stage 3 of Project construction. Stage 2 includes construction of a 50 mppa SX-EW plant at the Gunnison site. Major facilities include a SX Facility with two extraction and one strip settlers; an 80-cell EW Tankhouse with an Automatic Cathode Stripping Machine; a Tank Farm to receive, store, process, and transfer process solutions; PLS and Raffinate Ponds, Sulfuric Acid Storage Tanks, a new Electrical Substation; and ancillary buildings including a Security Building with truck scale, Administration Building, Change House, Plant Warehouse, Plant Maintenance Building, and Wellfield Maintenance Building.

Stage 3 construction includes an 80 EW-cell expansion of the Gunnison SX-EW plant for an additional 50 mppa copper production (125 mppa total). Stage 3 also includes the installation of a Sulfuric Acid Plant with railroad siding/railcar unloading. Train B of the Water Treatment Plant will be added in Year 7. Separate capital cost build-ups were constructed for the Stage 2 and Stage 3 SX-EW plants, and the sulfuric acid plant. The Water Treatment Plant expansion CAPEX was included in the Stage 3 expansion CAPEX.

Sustaining capital beyond Year 7 is primarily related to wellfield development, the installation of additional evaporation ponds and solids impoundments for water management, wellfield rinsing and abandonment, and the expansion of the Water Treatment Plant.

The following costs and quantity estimates used by M3 were provided by others:

- Hatch (February 2022) provided phased design, capital cost for equipment and reagent consumption and of the Water Treatment Plant. The new design of the water treatment plant treats two streams of water: raffinate to neutralize for use in flushing the wellfield and water returned from the wellfield for rinsing operations in areas that have been depleted (in-situ leached) of economically recoverable copper.
- Kinley Exploration LLC (Kinley) (December 2021) provided update cost estimates for installation and development of extraction, injection, and hydraulic control wells, as well as well abandonment costs for existing wells and core holes and production wells that have been rinsed and are out of service.
- NORAM Engineering & Constructors of Vancouver, B.C. (January 2022) prepared a new PFS study for the sulfuric acid plant. They provided capital and operating cost for the sulfuric acid plant which will be constructed in Years 5 & 6 for operation in Year 7. The new study designs and costs equipment for a plant producing 1650

tons per day of concentrated sulfuric acid. The previous NORAM study (2013) was based on a plant that produced 1350 tons per day.

- MHF Services (2016), a railroad consulting company, estimated the capital costs to install a railroad siding off of the Union Pacific Southern Pacific railroad and rail transfer and unloading yard for deliveries of acid and/or sulfur. This study was escalated for the current study and the MTO provided in the MHF study was re-estimated.

1.19.2 Operating Cost

Table 1-7 gives example years within Stages 1, 2, and 3 showing the breakdown of SX-EW operating cost by operating labor, reagents, power, maintenance labor and spare parts, and operating supplies.

Table 1-7: Summary SX-EW Operating Cost ($000)

Cost Element	Stage 1 (Year 3) Annual Cost ($000)	Stage 1 (Year 3) $/lb Copper	Stage 2 (Year 6) Annual Cost ($000)	Stage 2 (Year 6) $/lb Copper	Stage 3 (Year 9) Annual Cost ($000)	Stage 3 (Year 9) $/lb Copper
SX-EW Labor	$1,657	$0.07	$3,020	$0.04	$3,111	$0.02
Electrical Power	$3,659	$0.15	$8,824	$0.12	$13,817	$0.11
Reagents	$833	$0.03	$2,479	$0.03	$2,948	$0.02
Maintenance Parts & Services	$1,752	$0.07	$6,642	$0.09	$6,780	$0.05
Supplies & Services	$197	$0.01	$508	$0.01	$799	$0.01
Total SX-EW Operating Costs	$8,098	$0.33	$21,473	$0.29	$27,454	$0.22

1.19.2.1 General and Administrative Operating Costs

General and Administrative (G&A) costs include labor and fringe benefits for administration and support personnel and other support expenses. G&A expenses are projected to increase slightly with Stages 2 and 3 but decrease in cost per pound of copper produced as shown in Table 1-8.

Table 1-8: Summary General and Administrative Operating Cost

	Year 3		Year 6		Year 9	
Copper Cathode Produced	24,851,262		73,906,508		125,697,500	
Cost Item	Annual Cost	$/ lb Copper	Annual Cost	$/ lb Copper	Annual Cost	$/ lb Copper
Labor & Fringes	$3,667,054	$0.148	$4,124,423	$0.056	$4,124,423	$0.033
Accounting (excluding labor)	$25,000	$0.001	$25,000	$0.000	$25,000	$0.000
Safety & Environmental (excluding labor)	$25,000	$0.001	$25,000	$0.000	$25,000	$0.000
Human & Resources (excluding labor)	$25,000	$0.001	$25,000	$0.000	$25,000	$0.000
Security (excluding labor)	$25,000	$0.001	$25,000	$0.000	$25,000	$0.000
Assay Lab (excluding labor)	$300,000	$0.012	$300,000	$0.004	$300,000	$0.002
Office Operating Supplies and Postage	$40,000	$0.002	$40,000	$0.001	$40,000	$0.000
Maintenance Supplies	$306,516	$0.012	$306,516	$0.004	$306,516	$0.002
Propane Power	$36,183	$0.001	$47,337	$0.001	$47,501	$0.000
Communications	$70,000	$0.003	$70,000	$0.001	$70,000	$0.001
Small Vehicles	$125,000	$0.005	$125,000	$0.002	$125,000	$0.001
Claims Assessment	$10,000	$0.000	$10,000	$0.000	$10,000	$0.000
Legal & Audit	$300,000	$0.012	$300,000	$0.004	$300,000	$0.002
Consultants	$150,000	$0.006	$150,000	$0.002	$150,000	$0.001
Janitorial Services	$50,000	$0.002	$50,000	$0.001	$50,000	$0.000
Insurances	$2,000,000	$0.080	$2,000,000	$0.027	$2,000,000	$0.016
Subs, Dues, PR, and Donations	$60,000	$0.002	$60,000	$0.001	$60,000	$0.000
Travel, Lodging, and Meals	$150,000	$0.006	$150,000	$0.002	$150,000	$0.001
Recruiting/Relocation	$125,000	$0.005	$125,000	$0.002	$125,000	$0.001
Total General & Administrative Cost	$7,489,753	$0.301	$7,958,276	$0.108	$7,958,440	$0.063

1.19.2.2 Water Treatment Plant Operating Costs

An estimate of annual OPEX has also been developed based on vendor data, previous estimates for similar treatment systems and plant operating experience (Hatch, 2022). Major OPEX categories include labor, utility power, chemical reagents, process consumables, waste disposal and compliance sampling, analysis, and reporting. Annual wages for operators and electrical power cost are site specific and were provided by M3. A summary of operating costs for the Water Treatment Plant is provided in Table 1-9.

Table 1-9: Water Treatment Plant Operating Cost Summary

Cost Element	Minimum (Year 4) ($000)	Maximum (Year 20) ($000)	LoM Costs ($000)
Labor	$906	$906	$25,371
Power	$94	$4,289	$38,484
Reagents	$1,853	$47,223	$469,686
Maintenance	$8	$319	$5,499
Total WTP Operating Costs	$2,861	$52,738	$539,040

1.19.2.3 Sulfuric Acid Plant

The annual operating costs for the sulfuric acid plant, power plant, and associated facilities is $34.4 million or $58.29 per ton sulfuric acid and $0.24 per pound of copper produced. The acid plant operating costs are summarized in Table 1-10.

Table 1-10: Sulfuric Acid Plant Operating Costs

Annual Sulfuric Acid Production	589,475	short tons / year	
Annual Average Copper Production	124,672,205	lbs / year	
Cost Element	**Annual Cost ($000)**	**$ / Short ton Acid**	**$ / lb Copper**
Labor	$5,114	$8.68	$0.04
Reagents	$28,902	$49.03	$0.20
Fuel (propane)	$631	$1.07	$0.01
Power (Credit)	($6,385)	($10.83)	-$0.05
Maintenance	$3,232	$5.48	$0.03
Supplies	$2,865	$4.86	$0.02
Total Acid Plant Operating Costs	$34,359	$58.29	$0.24

1.19.3 Reclamation and Closure Cost

The reclamation and closure costs for the Project include reclamation and closure activities at both JCM and Gunnison plant sites, reclamation of legacy heaps and stockpiles at JCM, well abandonment and closure of the ISR wellfield, and bonding costs. ISR rinsing and water treatment activities are not included in this category. Much of the well abandonment will be conducted concurrently with production. Table 1-11 summarizes the total reclamation and closure costs for the Project. Details of the activities included in reclamation and closure are provided in Section 21.6. Approximately 45% ($26.9 million) of these expenses are projected to be made prior to the end of production.

Table 1-11: Summary of Reclamation and Closure Costs

Area	Reclamation & Closure Costs ($000)
JCM Buildings, Ponds, Waste Dump & Heap	$5,084
Well Abandonment	$17,708
Gunnison Plant, Ponds	$24,647
Bond Fees	$12,444
Total Reclamation & Closure	**$59,884**

1.20 ECONOMIC ANALYSIS

The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the Project. Annual cash flow projections were estimated over the life of the operation based on the estimates of capital expenditures and production cost and sales revenue. The sales revenue is based on the production of a copper cathode.

The economic analysis was conducted on two cases: 1) a base case that includes the construction of a sulfuric acid plant in Year 7 of operation, lowering the price of acid from $150/ton to $52/ton (Base Case) and 2) an alternate case that uses purchased sulfuric acid for the life of the operation (Alternate Case). Both cases use a copper price of $3.75/lb.

Table 1-12 compares the financial indicators for both the Base Case and the Alternate Case. The payback period does not represent the payback solely for initial CAPEX. Rather, it includes the accumulation of initial capital to start the Project using the existing Johnson Camp SX-EW plant and sustaining capital from two successive stages of construction for the Gunnison SX-EW plant, sulfuric acid plant, the rail spur, and water treatment plant.

Table 1-12: Financial Indicators

	Base Case	Alternate Case
Years of Commercial Production	24	24
Total Copper Produced (million lbs)	2,154	2,154
LoM Copper Price (avg $/lb)*	$3.75	$3.75
Initial Capital Cost ($M)	$47.6	$47.6
Sustaining Capital Cost ($M)	$1,033	$880
Payback of Capital (pre-tax / after-tax)	6.5 / 6.7	5.9 / 6.0
Internal Rate of Return (pre-tax / after-tax)	40.6 % / 37.5%	41.0% / 38.1%
LoM Direct Operating Cost ($/lb Copper recovered)	$0.95	$1.35
LoM Total Production Cost ($/lb Copper recovered)	$1.22	$1.63
Pre-Tax NPV at 7.5% discount rate ($M)	$1,435	$1,178
After-Tax NPV at 7.5% discount rate ($M)	$1,167	$976

*Price provided by Excelsior

Table 1-13 provides a sensitivity analysis for the Base Case project financial indicators with the financial indicators when other different variables are applied. The results indicate that Project economics are impacted the most by fluctuation in the copper price. Fluctuation in the initial capital cost has the least impact on Project economic indicators.

Table 1-13: Base Case After – Tax Sensitivities ($millions)

	Copper Price		
	NPV @ 7.5% ($M)	IRR%	Payback
Base Case	$1,167	37.5%	6.7
20%	$1,697	50.4%	4.3
10%	$1,433	44.0%	6.2
-10%	$898	30.8%	7.3
-20%	$627	24.2%	8.0
	Operating Cost		
	NPV @ 7.5% ($M)	IRR%	Payback
Base Case	$1,167	37.5%	6.7
20%	$1,031	33.2%	7.1
10%	$1,099	35.3%	6.9
-10%	$1,233	39.7%	6.5
-20%	$1,299	41.9%	6.3
	Initial Capital		
	NPV @ 7.5% ($M)	IRR%	Payback
Base Case	$1,167	37.5%	6.7
20%	$1,160	36.1%	6.7
10%	$1,163	36.8%	6.7
-10%	$1,170	38.2%	6.7
-20%	$1,173	39.0%	6.6

The Alternate Case economic after-tax sensitivities are shown in Table 1-14.

Table 1-14: Alternate Case After – Tax Sensitives ($millions)

	Copper Price		
	NPV @ 7.5% ($M)	IRR%	Payback
Base Case	$976	38.1%	6.0
20%	$1,505	51.7%	4.3
10%	$1,241	45.0%	4.8
-10%	$706	30.8%	6.7
-20%	$432	23.0%	7.5
	Operating Cost		
	NPV @ 7.5% ($M)	IRR%	Payback
Base Case	$976	38.1%	6.0
20%	$790	32.7%	6.5
10%	$883	35.4%	6.2
-10%	$1,066	40.8%	5.4
-20%	$1,157	43.4%	5.0
	Initial Capital		
	NPV @ 7.5% ($M)	IRR%	Payback
Base Case	$976	38.1%	6.0
20%	$969	36.6%	6.1
10%	$972	37.4%	6.1
-10%	$979	38.9%	6.0
-20%	$982	39.8%	6.0

1.21 ADJACENT PROPERTIES

The Gunnison Project lies within the porphyry copper metallogenic province of the southwestern United States. It is located in the Cochise Mining District, which is dominated by Cu-Zn skarns. With the acquisition of the Johnson Camp Mine, Excelsior now controls a majority of historical producing properties in the district. Tungsten and minor lead-silver-gold have been produced in adjacent properties in the district (Cooper and Silver, 1964). In particular, tungsten has been historically produced in the area west of the Gunnison Project in the northern half of the Texas Canyon quartz monzonite stock before and during World War I. Lead-silver was also historically produced from Paleozoic limestones in the Gunnison Hills east of the Gunnison Project in the early 1900s (Cooper and Silver, 1964). Mineralization on adjacent properties is not necessarily indicative of the mineralization on the Gunnison Project. The author has relied on reports by others (as referenced) for the information presented in this section and has been unable to verify the information.

1.22 JOHNSON CAMP MINE HEAP LEACH PEA

Excelsior and its consultants, RESPEC Company LLC (RESPEC), T.P. McNulty & Associates, M3 Engineering & Technology Corporation (M3), and Clear Creek Associates (CCA) have prepared a preliminary economic analysis (PEA) for a copper heap leaching operation using the existing Burro and Copper Chief open pits on the Johnson Camp Mine (JCM) property. These deposits have been mined episodically since the mid-1970s and were last mined in 2010. The PLS from the proposed leach heap would be processed in the existing JCM SX-EW that in currently in operation.

Section 24 of this report contains the full PEA for the JCM heap leach option. A new mineral resource estimate was prepared by RESPEC for this study. JCM has a mineral resource of 20.8 million short tons of measured, 87.1 million short tons of indicated, and 51.0 million short tons of inferred mineral resources with respective total copper grades of 0.31% measured, 0.32% indicated, and 0.32% inferred.

A review of past metallurgical testwork including several rounds of column testing and literature on sulfide leaching suggests that a recovery of 95% of the acid soluble and cyanide soluble copper and 70% of the sulfide copper is reasonable. The mined copper is modeled to release 80% of the recovery in the first year on the leach pad and 20% in the second year. Overall life-of-mine recovery of total copper averages 77%.

RESPEC prepared a mining cost and conceptual mine plan for the remining of the JCM deposits. The mine plan includes 69.7 million tons of M&I and 15.6 million tons of Inferred with an average grade of 0.37% TCu. The waste tonnage is 110.8 million tons, and the stripping ratio is 1.3:1 (waste to mineralized material). The conceptual mine plan is spread over 20 years with leaching. The first three quarters of Year -1 will be used to pre-stripping the pits for mining.

The new leach pad, Pad 5, will need to be constructed. Half of Pad 5 design will be constructed to handle an initial 25.7 million tons of leach material. The remainder of the current Pad 5 design will be built out in Year 4, and another addition (35 percent of Pad 5 capacity) will need to be added in Year 14. M3 has prepared a capital cost estimate for Pad 5 including earthworks, piping, and electrical installations.

The JCM plant is a fully operational solvent extraction-electrowinning facility. It was upgraded for operation of the Stage 1 Gunnison ISR wellfield production in 2019 and 2020. Solution from the new leach pad will be pumped to the existing solution ponds.

The financial indicators for the JCM heap leach operation are shown in Table 1-15.

Table 1-15: Financial Indicators for JCM Heap Leach PEA

Item	LoM
Years of Commercial Production	20
Total Copper Produced (klbs)	491,754
LoM Copper Price (avg $/lb)	$3.75
Initial Capital Cost ($M)	$58.9
Sustaining Capital Cost ($M)	$36.1
Payback of Capital (pre-tax / after-tax)	4.01 / 4.04
Internal Rate of Return (pre-tax / after-tax)	32.2% / 30.4%
LoM Direct Operating Cost ($/lb Copper recovered)	$1.95
LoM Total Production Cost ($/lb Copper recovered)	$2.24
Pre-Tax NPV at 7.5% discount rate ($M)	$212.5
After-Tax NPV at 7.5% discount rate ($M)	$180.0

The cost of reclamation of the JCM site including Pad 5, waste stockpiles, and existing leach pads with demolition of piping on the JCM property and bonding costs is estimated at approximately $15.8 million.

1.23 INTERPRETATIONS AND CONCLUSIONS

A Gunnison production schedule has been developed using input from independent consultants and existing Project data. The production schedule anticipates recovery of 48.4% of the mineral reserves resulting in production of 2,154 million pounds of cathode copper over a mine life of 24 years.

The base-case economic analysis indicates an after-tax NPV of $1,167 million at a 7.5% discount rate with a projected IRR at 37.3%. The Base Case includes a sulfuric acid plant constructed in Year 6 to supply the acid for ISR copper extraction. If the sulfuric acid plant is replaced by purchased sulfuric acid supplied by rail, the after-tax NPV at a 7.5% discount rate is $976 million with projected IRR of 38.1%. Payback is anticipated in 6.7 years of production for the acid plant case and in 6.0 years in the case using purchased sulfuric acid.

The economics are based on a $3.75/lb copper price, a staged production schedule of 25 mppa for Years 1-3, 75 mppa for Years 4-6 and a full production design copper production rate of 125 mppa for Years 7-16, decreasing in the final 8 years of the mine life. Direct operating costs are estimated at $0.95/lb of copper in the acid plant case and $1.35/lb of copper using purchased acid. Initial capital costs are estimated at $47.6 million. Sustaining capital costs of $1,033 million are projected in the sulfuric acid plant case and $879.7 million using purchased sulfuric acid.

1.24 PROJECT RISKS

Initial operations commencing in 2020 highlighted a number of challenges related to flow attenuation, lower than expected flow rates, and hence copper production. Project-specific risks are identified in Section 25.2 along with the measures that Excelsior envisages to mitigate these risk. The risks identified are in the categories of copper recovery, wellfield flow attenuation, reagent consumption and cost, wellfield design and spacing, gypsum formation and rinsing, and permitting difficulties. Flow attenuation (reduced flow rate and thus sweep efficiency) are believed to be due to CO_2 gas bubbles forming in the flow paths and restricting or blocking further flow along that flow path. It is possible other mechanism are also contributing to flow attenuation, or that CO_2 related attenuation is masking other problems within the wellfield that could result in reduced copper production, lower sweep efficiency or poor performance. Recommendations are provided to investigate potential risk items or advance mitigation strategies.

Mitigation of copper recovery challenges are based on adaptive management using data collected during operations. Mitigation of the flow attenuation due to CO_2 requires approximately 15 months of wellfield pre-conditioning using injection of neutralized raffinate cycled between periods of acidified raffinate injection and recovery. It may take longer than 15 months to clean out the CO_2 from effected wells. Availability of neutralized raffinate is addressed by the water treatment plant, which is supported by geochemical modelling with CO_2 dissolution using neutralized raffinate. Actual results may differ from the modelled results. Reagent consumption and cost can be mitigated by the addition of the acid plant, use of limestone from onsite sources, and obtaining lime from a closer source or the revitalization of a dormant lime kiln in Cochise County, Arizona. Mitigating well design and spacing issues and gypsum formation and rinsing problems are based on experience and adaptive management. Permitting difficulties associated with future additions and expansions will be managed by maintaining strong relationships with regulators, the local community, government officials, mining support groups, and other identified stakeholders.

1.25 PROJECT OPPORTUNITIES

Several opportunities have been identified which could enhance the viability and economic attractiveness of the Project. Opportunities, detailed in Section 25.3, include higher copper recoveries than predicted, increases in the price of copper, identification of additional resources, wellfield optimization, and reductions to capital costs, particularly in the initial stage of operation. Other opportunities for reducing costs and schedule include exploring the use of portable water treatment equipment for the first three years, use of onsite limestone either directly in water treatment or as a source for making lime at the site, and the possibility of restarting the dormant lime plant in Cochise County to supply lime at a reduced price for water treatment at the site.

1.26 RECOMMENDATIONS

Based on the results of this Prefeasibility Study, it is recommended that Excelsior proceed with the Project through the engineering, procurement, and construction necessary to restart active production once financing is secured. The engineering for the water treatment infrastructure needs to be advanced in accord with the project development schedule. The drilling, mineral resource estimation, wellfield mine planning, wellfield drilling, and infrastructure development and the staged SX-EW plant have all been adequately defined. The initial wellfield is drilled, and solution is being pumped for processing, but the addition of raffinate neutralization capability is considered necessary to resolve the wellfield circulation and production difficulties. The following sections discuss areas for potential investigation and risk reduction.

There are four recommendations for investigating in-situ leaching with different lixiviants: sulfurous acid, ammonium carbonate, ammonium sulfate with oxygen, and glycine leaching. These techniques have received some attention as opportunities to leach metals without the formation of gypsum. A program of laboratory testwork should be undertaken to determine if any of these lixiviants are worth pursuing.

Flow attenuation associated with the addition of acidified leach solutions to the wellfield has been attributed to the buildup of CO_2 in the formation due to the dissolution of calcite and other carbonates. Continued research into the causes of the flow rate attenuation and buildup in the formation should be continued. Flow profiling and changes in wellfield operational parameters should be considered to learn more about the conditions which lead to reductions in flow and the methods which can be used to enhance flow.

Laboratory experimentation is recommended to ensure that neutralized raffinate is effective in dissolving CO_2 in the subsurface while the engineering, procurement, and construction is at an early stage to enhance the water treatment design criteria. Those experiments should also address the solubility of gypsum in mixed acidified and neutralized raffinate solutions to avoid conditions which might result in damage to the formation.

Well stimulation trials should be investigated as a mechanism to alleviate CO_2 blocking, improve connectiveness, increase flow rates and sweep efficiency. If well stimulation is effective, it has the potential to negate or reduce the amount of raffinate neutralization which would impact the design criteria for the neutralization plant. Well stimulation is allowed under Class III Underground Injection Control permits but requires EPA approval of the stimulation programs.

A scope of work and bid package should be assembled to select a water treatment vendor to design the water treatment system. Vendors should be screened and selected to advance the engineering process to shrink the implementation schedule. Selection criteria should favor rapid, low-cost solutions to demonstrate that the technology is effective in solving the wellfield challenges.

Excelsior has proposed a list and budget for additional work that will support the feasibility study. Table 1-16 defines the cost of the technical activities.

Table 1-16: Feasibility Budget for the Gunnison Project

Detail	Cost US$
Metallurgical Testwork	
Sulfurous acid leaching	$50,000
Ammonium carbonate leaching	$40,000
Ammonium sulfate leaching with oxygen	$40,000
Glycine leaching investigation	$65,000
Subtotal metallurgical testwork	$190,000
Wellfield Studies	
Flow attenuation	$150,000
CO_2 dissolution in neutralized raffinate testwork	$100,000
Well Stimulation Trials	$1,500,000
Flow profiling (mapping)	$500,000
Subtotal wellfield studies	$2,250,000
Water Treatment Testwork	
Raffinate neutralization testwork	$50,000
Solids management and densification testwork	$50,000
Solid liquid separation and filtration studies	$75,000
Subtotal Water Treatment Studies	$175,000
Total	$2,615,000

2 INTRODUCTION

Excelsior Mining Corp. commissioned M3 Engineering & Technology Corporation (M3) to prepare an updated Prefeasibility Study (PFS) covering the process and infrastructure design, capital cost, operating cost, and an independent Technical Report prepared in accordance with the Canadian National Instrument 43-101 ("NI 43-101") standards for reporting mineral properties, for the Gunnison Copper Project (the "Project") – North Star Deposit in Cochise County, Arizona, USA. The updated report also includes a report at a PEA level on restarting heap leaching of materials from the Burro and Copper Chief pits at the Johnson Camp mine.

The Gunnison Project is a staged in-situ copper leaching operation whose first stage of construction was commenced in 2019 and completed in 2020. The first stage of production of the Gunnison wellfield was projected to accept up to 4,000 gallons per minute (gpm) of raffinate solution which would circulate through the rock formations containing copper oxide mineralization to ramp up to produce 25 mppa of cathode copper. Stage 1 of the Gunnison Project uses the existing Johnson Camp Mine solvent extraction-electrowinning (SX-EW) plant that was refurbished and upgraded for Stage 1 operation of the Gunnison wellfield.

Operating difficulties began after Excelsior commenced circulating acidified raffinate solution into the wellfield in December of 2019. Precipitation of copper hydroxides on well pumps and piping restricted flows of pregnant leach solution (PLS). The wellfield pumping configuration originally consisted of discrete injection wells and recovery wells. Excelsior reconfigured the wellhead piping so each well in the wellfield could be operated as either an injection well or a recovery well by adding valves at the wellhead. This modification mitigated the copper hydroxide precipitation issues, however, the inflows of raffinate to the wellfield were still well below the budgeted flowrate.

Investigation and analysis of the flows from individual wells indicate that the source of the problem is the build-up of carbon dioxide (CO_2) gas bubbles that are formed by the dissolution of calcite in the fractures when the calcite is contacted by sulfuric acid in raffinate. The CO_2 bubbles are thought to blind off the fractures that support the flow paths between the injection and recovery wells thus restricting flow. CO_2 bubbles will dissolve in solution with a neutral pH, but acidic solutions will dissolve calcite and create more bubbles.

To mitigate the CO_2 blinding of fractures, Excelsior plans to alternate pulses of acidified raffinate with neutralized raffinate solution to mine out the calcite in fractures that provide porosity for flow that are also the site of leachable copper oxide mineralization. To be conservative of water resources and because fresh water supply at the Gunnison site (including Johnson Camp) is limited, Excelsior proposes to neutralize raffinate to create a neutral pH solution to dissolve the CO_2 bubbles. The alternation of acidified raffinate and neutralized raffinate will theoretically remove the CO_2 and restore the porosity needed to maintain in-situ leaching of copper oxides in the formation. This PFS describes the new proposed process and updates the capital and operating cost as well as the new timeline to extract copper from the Gunnison deposit.

Excelsior plans to build a Water Treatment Plant (WTP) to neutralize raffinate in sufficient volumes to support this new unit operation to dissolve and remove CO_2. The WTP was originally planned to treat solutions needed to rinse the exhausted formation later in the mine life. The WTP will consist of a raffinate neutralization step using lime to bring the raffinate back to a pH of 7.

Another section of the WTP will be added later in the mine schedule to rinse the exhausted formation and will consist of membrane filtration to remove metals and other cations from rinse solutions. The membrane filtration also requires the lime neutralization step and additional conditioning so that both sections of the plant will be operated simultaneously once rinsing begins.

The updated PFS is based on Excelsior's "staged" development of the Project but includes the addition of a water treatment unit operation for the new mine plan. When operating to design, the wellfield will produce 25 million pounds of cathode copper per year in Stage 1. Once production is stabilized at 25 million lbs/yr, a new Stage 2 SX-EW plant will be built with a capacity of 50 million lbs/yr, ramping the total Gunnison JCM copper capacity to 75 million lbs/yr. When that capacity is reached and stabilized, the Stage 2 SX-EW plant will be expanded by 50 million lbs/yr to reach the full Stage 3 combined plant capacity of 125 million lbs/yr.

The staging of copper production impacts the project cash flow as well as the capital requirement to develop the Gunnison SX-EW hydrometallurgical plant. Excelsior purchased the Johnson Camp Mine which includes a complete SX-EW plant that is capable of producing 25 million pounds of copper per year as it is currently configured. Stage 1 facilities have been constructed so the current effort is to bring Stage 1 production up to its full potential.

This PFS focuses on the engineering design, capital improvements, and costs/financial model to improve Stage 1 production. It describes the Gunnison wellfield installation and the proposed improvements needed to mitigate the CO_2 blinding problem. The remainder of this report focuses on re-estimating Stage 2 and 3 additions and establishing a new discounted cashflow model for the overall project based on new timelines. None of the Stage 2 and Stage 3 process methodology and infrastructure requirements have changed, but the capital and operating costs and financial analysis for Stages 2 and 3 have been updated. Two areas which have been revisited are the Water Treatment Plant which is planned to be first operated in Year -1 and the sulfuric acid plant which will commence operation in Year 8.

No additional engineering has been done for the railroad spur and siding which will support the acid plant. They are introduced in Stage 3 of the project development, and their impact to the discounted cash flow is minor.

The Gunnison Copper Project contains copper oxide and sulfide mineralization with associated molybdenum, in potentially economic concentrations. The North Star (formerly known as I-10) deposit is the mineral deposit within the Project area that is the source for copper for this mining operation.

On October 15, 2010, Excelsior Mining Corp. (the “Company” or “Excelsior”), completed a reverse takeover (“RTO”) by acquiring all of the issued and outstanding common shares of AzTech Minerals, Inc. (“AzTech”) through a plan of merger with Excelsior Mining Arizona, Inc. (“Excelsior Arizona”). Excelsior Arizona was the surviving corporation in the plan of merger and acquired all assets of AzTech, including the Gunnison Copper Project. The Company is listed on the TSX under the symbol “MIN”.

Legally, the Company is the parent of AzTech, however, as a result of the share exchange described above, control of the combined companies passed to the former shareholders of AzTech. This type of share exchange is referred to as a "reverse takeover.” The executive management of AzTech continued on as the executive management of Excelsior.

Excelsior Mining Corp. retained several consultants, including M3, to provide a review of prior work on the Project and to prepare technical and cost information to support a FS and this Technical Report compliant with the Canadian NI 43-101 reporting standards. Mr. Richard Zimmerman, SME-RM of M3 is the principal author and Qualified Person responsible for the preparation of this report. Mr. Zimmerman visited the Project site on numerous occasions between 2013 and 2016 in support of the 2014 PFS, 2016 PFS Update, 2016 FS, and most recently September 28, 2021. Other contributing authors and Qualified Persons responsible for preparing sections of this report include Dr. Terence McNulty, metallurgical consultant; Robert Bowell, geochemist, on carbon dioxide blocking in the wellfield; Doug Bartlett and Alison Jones of Clear Creek Associates (CCA) for hydrology; mining methods, and environmental/social/permitting topics, Colin Kinley of Kinley Exploration for well design and well field development; and Neil Prenn and Jeffrey Bickel of RESPEC Company LLC (RESPEC) for mineral resources and mineral reserves.

Jeffrey Bickel of RESPEC is the Qualified Person responsible for geology, mineralization, sample preparation, security, data verification, and mineral resources taken from or updated since the *“Gunnison Copper Project, Cochise County, Arizona, USA, Mineral Resource of the North Star Deposit”* Technical Report dated August 2011 and revised October 2011 prepared by Herb Welhener of IMC for Excelsior. Jeffrey Bickel of RESPEC visited the Project site and the core storage facility on May 3rd, 2021. Neil Prenn of RESPEC is the Qualified Person responsible for mineral reserves. The mineral resource in support of this PFS has been updated from the mineral resource published in the 2016 FS Update. The mineral reserve from the 2016 PFS Update has also been updated.

R. Douglas Bartlett, CPG, of Clear Creek Associates (CCA), is responsible for the preparation of Section 16 – Mining Methods and Section 20 and 24.20 – Environmental Studies, Permitting, and Social Impact. Mr. Bartlett visited the site most recently on May 15, 2019. Alison Jones of CCA visited the Project site in 2015.

Dr. Terence P. McNulty is responsible for review of the historical metallurgical testing programs for both the Gunnison in-situ program and the Johnson Camp heap leach evaluation. Dr. McNulty is responsible for the preparation of Section 13 – Mineral Processing and Metallurgical Testing except for Section 13.2.3.1. Dr. McNulty visited the Johnson Camp Site in 1990s. Dr. McNulty has worked extensively on copper hydrometallurgical projects in the US and elsewhere. He replaces Dr. Ron Roman, whose previous metallurgical testwork was used at the Johnson Camp Mine for its previous Owner's, Nord Resources Corporation.

Rob Bowell of SRK Consulting is responsible for the preparation of Section 13.2.3.1. Dr. Bowell visited the Gunnison property in September 2021.

Hatch designed the water treatment plant described in Section 18. Hatch prepared the equipment list, general arrangement of facilities, capital and operating cost estimate for the water treatment plant that is constructed in four phases.

M3 was responsible for developing process design criteria, process flowsheets, an equipment list, general arrangements of the site plan and process facilities, process ponds, infrastructure, capital cost, operating cost, prefeasibility-level financial assessment, and integrating the work by other consultants into a final Technical Report prepared in accordance with Canadian NI 43-101 standards.

NORAM Engineering and Constructors of Vancouver, British Columbia prepared prefeasibility-level engineering design and a capital cost estimate for the sulfur burning sulfuric acid plant. The sulfuric acid plant has been designed to produce 1,650 tons This estimate was augmented to include molten sulfur and unloading and storage, sulfuric acid storage, and steam turbine cogeneration using waste heat from the sulfur burning process.

2.1 LIST OF QUALIFIED PERSONS

Site visits and areas of responsibility are summarized in Table 2-1 for the QPs.

Table 2-1: Dates of Site Visits and Areas of Responsibility

Author	Company	Designation	Site Visit Date	Section Responsibility
Richard Zimmerman	M3 Engineering & Technology Corp. – Tucson, AZ	SME-RM	September 28, 2021	Sections 1, 2, 3, 4, 5, 17, 18, 19, 21, 22, 23, 24.1, 24.2, 24.3, 24.4, 24.5, 24.17, 24.18, 24.19, 24.21 (except 24.21.1, 24.21.2), 24.22, 24.23, 24.24, 24.25, 24.26, 24.27, 25, 26, 27
Jeffrey Bickel	RESPEC Company LLC	CPG	May 3, 2021	Sections 6, 7, 8, 9, 10, 11, 12, 14, 24.6, 24.7, 24.8, 24.9, 24.10, 24.11, 24.12, 24.14
Thomas L. Dyer	RESPEC Company LLC	SME-RM	March 18, 2021	Sections 24.16 (except 24.16.5), 24.21.1, 24.21.2
Neil Prenn	RESPEC Company LLC	MMSA-QPM	November 10, 2021	Section 15
Robert J. Bowell	SRK Consulting, Cardiff, UK	PhD C.Chem. C.Geol	September 28, 2021	Section 13.2.3.1
Dr. Terence P. McNulty	T.P. McNulty & Associates	PE, DSc	Johnson Camp Site in 1990s*	Sections 13 (except 13.2.3.1), 24.13
R. Douglas Bartlett	Clear Creek Associates	CPG	May 15, 2019	Sections 16, 20, 24.16.5, 24.20

*visits to JCM plant which is now used for Gunnison ISR Project

2.2 DEFINITIONS OF TERMS USED IN THIS REPORT

- **In-Situ Recovery:** A closed loop mining system, where ground water from the aquifer is utilized as the transport medium of the lixiviant and minerals or metals are dissolved in-situ within the host formation using an appropriate lixiviant.

- **Lixiviant:** Aqueous media, in this case, sulfuric acid, to extract copper from the oxide copper mineralization.

- **Pregnant Leach Solution (PLS):** Lixiviant after it is loaded with dissolved copper. PLS is stripped of copper in the solvent extraction process.

- **Raffinate:** Lixiviant after has been stripped of copper in the solvent extraction process. Raffinate is re-acidified and pumped back to the wellfield to dissolve more copper.

- **Diluent:** Organic medium in which solvent extract takes place in the SX settlers.

- **Extractant:** Organic chemical that is used to extract copper from PLS into the diluent and then transfer the copper from the diluent to the electrolyte.

- **Electrolyte:** The aqueous solution carrying concentrated copper in solution which is pumped into the EW Tankhouse to electroplate copper onto steel blank sheets. The depleted electrolyte is recirculated to the SX circuit to load more copper.

- **Sulfuric acid:** A dense, colorless liquid chemical (H_2SO_4) used extensively to leach oxide copper ores.

- **Sulfurous acid:** The chemical species, H_2SO_3, which is formed by dissolving sulfur dioxide, SO_2, in water was used briefly as a lixiviant for copper in the 1920's.

2.3 UNITS AND ABBREVIATIONS

This report is in English units. Tons are short tons and ktons mean 1,000 short tons. Copper grades are in percentage by weight. All tonnages reported in this document are in dry tons. Lengths are in feet (except where noted) and currency is in U.S. dollars (except if noted otherwise).

Table 2-2: Units, Terms and Abbreviations

Abbreviation	Unit or Term
%	percent
°	degree (degrees)
°C	degrees Centigrade
$M	million dollars
µ	micron or microns, micrometer, or micrometers
A	Ampere
A/m^2	amperes per square meter
AA	atomic absorption
ADEQ	Arizona Department of Environmental Quality
ADWR	Arizona Department of Water Resources
APP	Aquifer Protection Permit
AQL	Aquifer Quality Limit
ASCu or CuAs	Acid-soluble copper
AzTech	AzTech Minerals, Inc.
BADCT	Best Available Demonstrated Control Technology
BLM	US Department of the Interior, Bureau of Land Management
cfm	cubic feet per minute
cm	Centimeter
cm^2	square centimeter
cm^3	cubic centimeter
CNCu or CuCN	Cyanide soluble copper
CoG	cut-off grade
Crec	core recovery
Cu	Copper
dia.	Diameter
EA	Environmental Assessment
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
FA	fire assay
famsl	feet above mean sea level
FS	Feasibility Study
ft	foot (feet)
ft^2	square foot (feet)
ft^3	cubic foot (feet)
ft^3/st	cubic foot (feet) per short ton
g	gram
g/L	gram per liter

Abbreviation	Unit or Term
g/st	grams per short ton
GA	General Arrangement
gal	Gallon
g-mol	gram-mole
gpl	gram per liter
gpm	gallons per minute
Ha	hectares
HDPE	High Density Polyethylene
hp	horsepower
IMC	Independent Mining Consultants
in	inch
IRR	Internal Rate of Return
ISR	In-Situ Recovery
JCM	Johnson Camp Mine
kg	kilograms
km	kilometer
km^2	square kilometer
ktons	thousand short tons/ kilotons
kst/d	thousand short tons per day
kst/y	thousand short tons per year
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/st	kilowatt-hour per short ton
L	liter
L/sec	liters per second
lb	pound
LHD	Load-Haul-Dump truck
LoM	Life-of-Mine
M	meter
m.y.	million years
m^2	square meter
m^3	cubic meter
M3	M3 Engineering & Technology Corp.
Ma	million years ago
mg/L	milligrams/liter
mi	mile
mi^2	square mile
MIW	Mine-influenced water
MM lb	million pounds
mm	millimeter
mm^2	square millimeter
mm^3	cubic millimeter
mppa	million pounds per annum (year)

Abbreviation	Unit or Term
Mst	million short tons
Mst/y	million short tons per year
MVA	megavolt ampere
MW	million watts
NI 43-101	Canadian National Instrument 43-101
NPV	Net Present Value
PAST	Professional Archeological Services of Tucson
PFS	Pre-Feasibility Study
PLS	Pregnant Leach Solution
PMF	probable maximum flood
POO	Plan of Operations
ppb	parts per billion
ppm	parts per million
psi	pounds per square inch
QA/QC	Quality Assurance/Quality Control
RC	reverse circulation drilling
RQD	Rock Quality Description
RT	Reverse takeover
SEC	U.S. Securities & Exchange Commission
sec	second
SG	specific gravity
st	short ton (2,000 pounds)
st/d	short tons per day
st/h	short tons per hour
st/y	short tons per year
SX-EW	Solvent Extraction (SX) - Electrowinning (EW)
t	tonne (metric ton) (2,204.6 pounds)
TCu or CuT	total copper
TSF	tailings storage facility
TSP	total suspended particulates
UIC	Underground Injection Control
USEPA	United States Environmental Protection Agency
V	volts
VFD	variable frequency drive
W	watt
WTP	water treatment plant
XRD	x-ray diffraction
yd^2	square yard
yd^3	cubic yard
yr	year

3 RELIANCE ON OTHER EXPERTS

The authors, as Qualified Persons, relied upon historical data for the Gunnison Copper Project provided by Excelsior Mining Corp. In the opinion of the authors, the Gunnison historical data, in conjunction with borehole assays conducted by Excelsior, are present in sufficient detail to prepare this report and are generally correlative, credible, and verifiable. The Project data are a reasonable representation of the Gunnison Copper Project. Any statements in this report related to deficiency of information are directed at information that, in opinion of the authors, is recommended by the authors to be acquired.

Jerry L. Haggard, P.C. of Phoenix, AZ, was relied upon to provide legal determinations of the lands and claims on the Gunnison side of the property. Excelsior relied on reports by John C. Lacy of the law firm, DeConcini, McDonald, Yetwin, & Lacy, for legal determination of lands on the Johnson Camp side of the property. Excelsior also obtained an ALTA Title Insurance Policy from First American Title Insurance Company for the patented mining claims and fee lands of the Johnson Camp property.

4 PROPERTY DESCRIPTION AND LOCATION

The Project is held by Excelsior through its wholly owned subsidiary Excelsior Mining Arizona, Inc. (Excelsior Arizona). JCM is also held by Excelsior Arizona. Acquisition of all mineral interests comprising the Gunnison Project from the James L. Sullivan Trust was completed in January of 2015. These assets represent, among other things, the mineral rights to the North Star and South Star Copper deposits of the Gunnison Project (the Project). Additionally, in December 2015 Excelsior purchased all assets of Nord Resources Corporation, as they relate to the Johnson Camp property, through a court-appointed receiver. Further certain unpatented mining claims adjacent to the Project are held by Excelsior Mining Holdings, Inc. (Excelsior Holdings), which is a wholly-owned subsidiary of Excelsior.

The Project and JCM are located in Cochise County, Arizona, approximately 65 miles east of Tucson. Figure 4-1 is a general location map and property location near the I-10 freeway. The Project and JCM include portions of Section 22, 23, 24, 25, 26, 27, 34, 35 and 36 T15S R22E, Sections 1, 2, 11, 12, 13, 23, 24, 25, and 26 T16S R22E, Sections 5, 6, 7, 8, 17, 18 and 19 T16S R23E, and Sections 19, 20, 29, 30, 31 and 32 T15S R23E and is centered at 32º 04’ 55” N latitude and 110º 02’ 40” W longitude. Total area of the Project and JCM is approximately 11,802 acres (4,776 Ha), approximately 3,092 of which was added with the Johnson Camp property acquisition.

Figure 4-2 shows the claim status for the Gunnison Project and JCM as of January 2023. Table 4-1 contains a summary of the land packages that constitute the Gunnison Project. Table 4-2 contains a summary of the land packages that constitute the Johnson Camp property. Following the tables are brief descriptions of the claims, permits, deeds and land holdings. Appendix B contains a detailed list of all the mining claims and land packages.

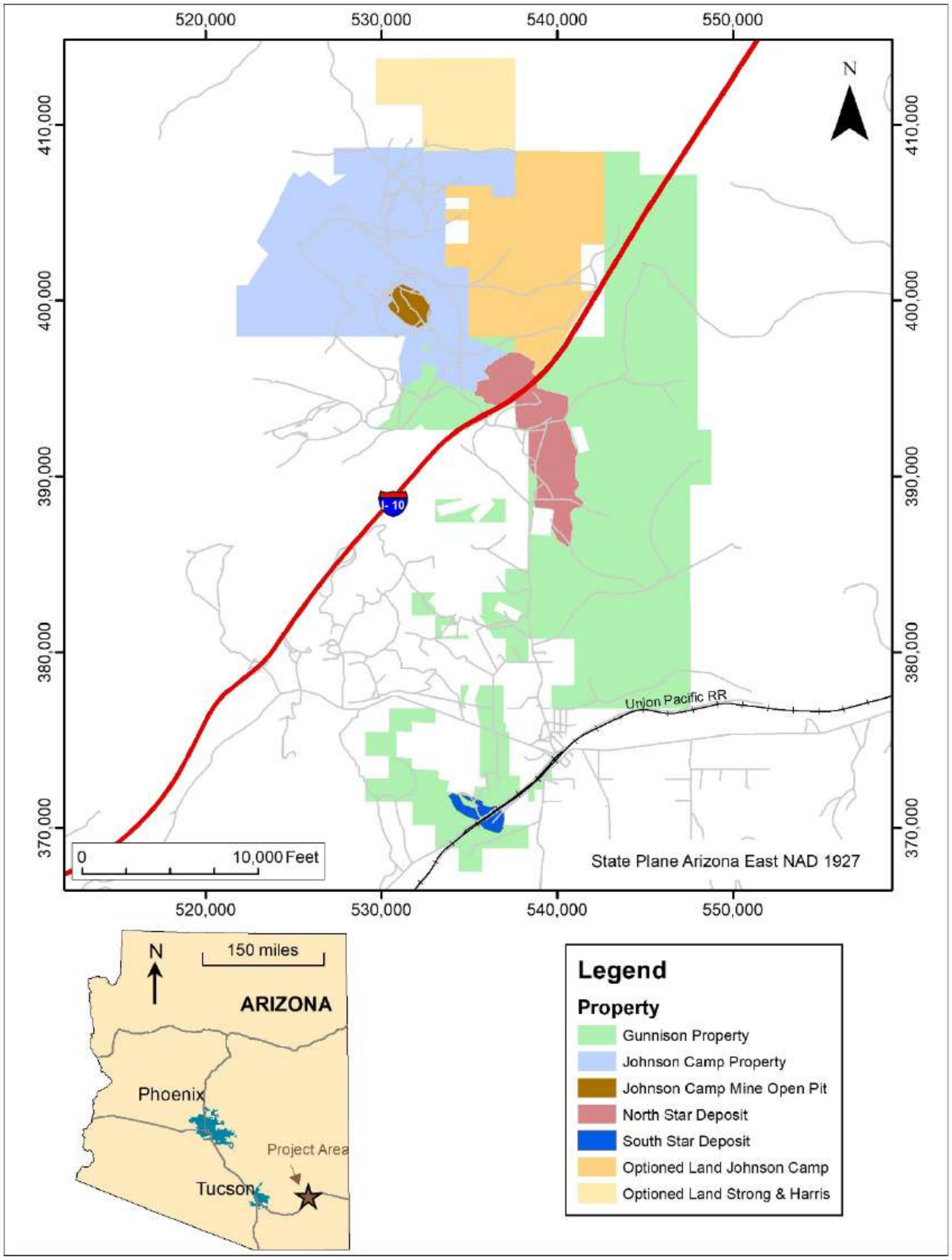


Figure 4-1: Location of the Gunnison Project, North Star and South Star Deposits and Johnson Camp Property – January 2023

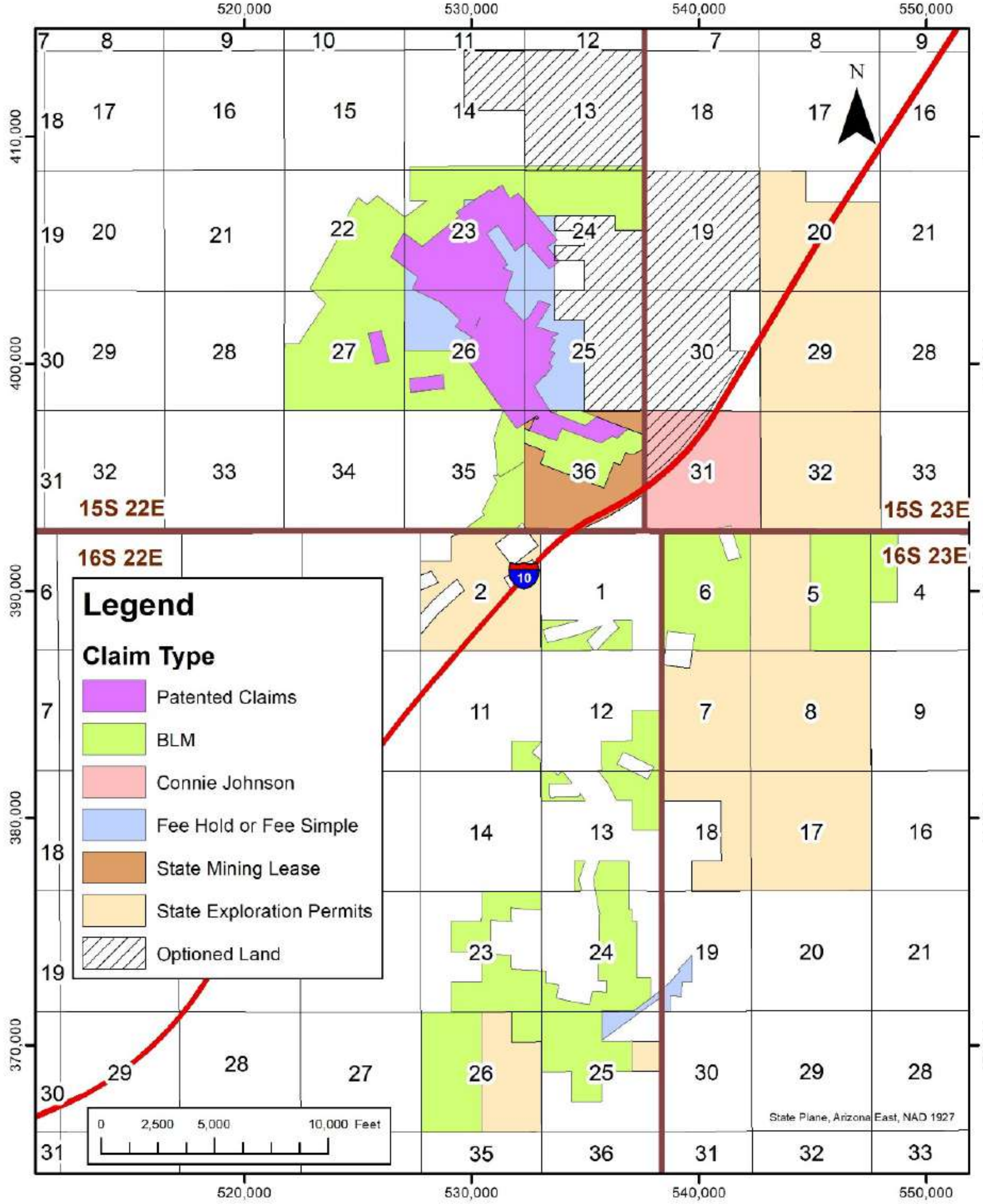


Figure 4-2: Project Mineral Rights by Claim Type – January 2023

Table 4-1: Summary of Land Packages that Constitute the Gunnison Project

Claim Type	# of Claims	Approximate Area	Approximate Holding Costs	Surface Rights
Federal Unpatented Mining Claims	172	2.473 acres 1001 hectares	Annual $28,380.00	Subject to US mining law
Arizona State Mineral Lease	1	319 acres 129 hectares	Annual $18,345.95	Subject to AZ state laws
Arizona State Exploration Permits	11	4,952 acres 2,004 hectares	Annual up to $109,375.2	Subject to AZ state laws
North Star Freehold Mineral Rights via "Connie Johnson" Deed	1	616 acres 249 acres	Nil	Subject to deed of trust (see below)
South Star Freehold Land and Mineral Rights	4	62 acres 25 hectares	Annual $32.00	Controlled by Excelsior
Total	189	8,422 acres 3,408 hectares	Annual $156,113.15	

Table 4-2: Summary of Land Packages that Constitute the Johnson Camp Property

Claim Type	# of Claims	Approximate Area	Approximate Holding Costs	Surface Rights
Federal Patented Lode Mining Claims	59	871 acres 352 hectares	Annual $1,589.94	Controlled by Excelsior
Federal Unpatented Mining Claims	133	1,892 acres 766 hectares	Annual $22,275.00	Subject to US mining law
Fee Simple Lands	4	617 acres 250 hectares	Annual $658.47	Controlled by Excelsior
Total	196	3,380 acres 1,368 hectares	Annual $24,523.41	

Ownership of the unpatented mining claims is in the name of the holder (locator), subject to the paramount title of the United States of America, under the administration of the U.S. Bureau of Land Management ("BLM"). Under the Mining Law of 1872, which governs the location of unpatented mining claims on federal lands, the locator has the right to explore, develop, and mine minerals on unpatented mining claims without payments of production royalties to the U.S. government, subject to the surface management regulation of the BLM. As of the Effective Date, annual claim-maintenance fees are the only federal payments related to unpatented mining claims, and Excelsior represents these fees have been paid in full to September 1, 2022. The current annual holding costs for Gunnison and JCM are estimated at $50,655, including the county recording fees.

Excelsior has rights to use the surface of the Project that is in the form of federal patented lode mining claims and fee land parcels. The federal unpatented claims grant surface access but do not provide for surface ownership. Unpatented mining claims give the owner the right to develop and exploit valuable minerals contained within the claim, so long as the claim is properly located and validly maintained.

4.1 PATENTED MINING CLAIMS

There are 59 patented mining claims held in the name of Excelsior Arizona totaling 871 acres (352 ha). A completed list of the claims is provided in Appendix B. The claims include all surface and mineral rights. The claims are located on the ground and have no expiration dates.

4.2 UNPATENTED MINING CLAIMS

There are 172 unpatented mining claims held by Gunnison in the name of Excelsior Arizona and Excelsior Holdings totaling 2,473 acres (1001 ha) and 133 unpatented mining claims held by JCM in the name of Excelsior Arizona and Excelsior Holdings totaling 1,892 acres (766 ha). Collectively, Excelsior controls 305 unpatented mining claims totaling 4,365 acres (1,766 ha). A completed list of the claims is provided in Appendix B. The unpatented claims are for minerals only, with no surface ownership. The BLM requires that all unpatented claims use a rental year from September 1 through August 31; claims for which fees are not paid by August 31st are automatically forfeit. The claims otherwise have no expiration dates and under current mining law can be held indefinitely if properly maintained. The claims are located on the ground and the location descriptions are filed with the BLM.

4.3 STATE MINERAL LEASE AND PROSPECTING PERMITS

Excelsior Arizona holds the Arizona State Mineral Lease and Prospecting Permits. The tenements are administered by the Arizona State Land Department and are for minerals only. Rents, fees, and expenditure commitments are due each year and all payments and expenditure commitments are current. The 2023 expenditure commitment will be up to $99,442.5 with fees of up to $29,178.1. A detailed list of these fees and the due dates is supplied in Appendix B. A state royalty is payable on state leases for copper that is produced and sold. The amount is set by the Arizona State Land Department using a sliding scale royalty. The sliding scale royalty uses an upper and lower limit based on copper price and has the highest possible royalty rate of 8% and the lowest possible royalty rate of 2%. Excelsior is required to pay a minimum annual royalty regardless of production. The minimum annual royalty is $6,381.20 and is due each year on or before the anniversary of the commencement date of the lease and shall be a credit for Excelsior, fully recoupable against production royalties. Mineral lease and prospecting permit boundaries are described by the Arizona State Land Department. Surface rights include the right to use the surface for exploration, mining, mineral processing, and related activities subject to a state approved Mineral Development Report or Exploration Plan as the case may be. The mineral lease was renewed by the Arizona State Land Department June 16, 2014, and expires on June 15, 2034. The individual expiration dates of the Prospecting Permits are shown in Appendix B and range from February 23, 2026, to November 3, 2026. There are provisions in the Arizona State mining law to retain the area held by the permits, subject to meeting certain state requirements, by converting the permits to mineral leases or by applying for new exploration permits.

4.4 "CONNIE JOHNSON" DEED

Excelsior owns the mineral rights in Section 31, T15S., R23E, that were subject to the provisions of a Deed of Trust dated January 22, 1998, between Excelsior Arizona and the seller of the mineral rights. The Deed of Trust was released, and the mineral rights transferred, to Excelsior Arizona through a Beneficiary Deed of Full Release and Full Reconveyance that was recorded on February 6, 2015. The area (approximately 616 acres or 249 ha) covers about 1/3 of the North Star deposit, is for the minerals only and is defined by the boundaries of Section 31, T15S.and R23E. Surface and mineral rights are defined by the Deed of Trust and include "All mines and minerals in and under Section 31, Township 15 South, Range 23 East, Gila and Salt River Base and Meridian, containing 615.62 acres, more or less, together with the power to take all usual, necessary or convenient means for working, getting, laying up, dressing, making merchantable, and taking away the said mines and minerals, and also for the above purposes, or for any other purposes whatsoever, to make and repair tunnels and sewers, and to lay and repair pipes for conveying water to and from any manufactory or other building…".

4.5 FEE SIMPLE LAND

Mineral and in some cases mineral and surface rights to a small portion of the South Star deposit are held directly by Excelsior Arizona. Mineral rights only pertain to Parcel F (approximately 15.3 acres), Section 25 T16S., R22E and Parcel A (approximately 39 acres), Section 19, T16S., R23E., Union Pacific Railroad that covers an easement along the Union Pacific Railroad. Surface and Mineral rights are held via Parcel D (approximately 14.24 acres), Section 19 T16S., R22E., and Parcel E (approximately 4.28 acres), Section 19 T16S., R23E. Holding costs for the fee simple land amount to approximately $32 per year in property taxes. Property boundaries are defined by the property descriptions on public record.

The Johnson Camp property acquired by Excelsior Arizona includes additional Fee Simple Lands to those listed above at the Gunnison Project. There are four parcels of Fee Simple Lands all situated in Township 15S, Range 22E. Parcel 1 is situated on Section 26 and covers approximately 139 acres. Parcel 2 is situated on Section 26 and covers approximately 1 acre. Parcel 3 is situated on Sections 24 and 25 and covers approximately 53.44 acres. Parcel 4 is situated on Sections 23, 24, 25, and 26 and covers approximately 116.27 acres.

Excelsior Arizona has entered into an option agreement with certain landowners that provide Excelsior Arizona the right to acquire approximately 2563.05 acres of Fee Simple Lands that are referred to as the “Optioned Land Johnson Camp” and “Optioned Land Strong & Harris.” The terms of the option agreement require an upfront fee of $40,000 and an annual fee of $30,000 to maintain the option in good standing. The term of the option is seven years and a final option payment to acquire the lands is required in the event Excelsior Arizona chooses to exercise the option.

4.6 ADDITIONAL ROYALTIES

4.6.1 Gunnison Project and Johnson Camp Property

Greenstone Royalty: Greenstone Excelsior Holdings L.P. (“Greenstone”) holds a 3.0% gross revenue royalty over the Gunnison Project and the Johnson Camp Property. The gross revenue royalty is defined as royalty percentage times receipts, which is the sum of physical product receipts and deemed receipts. The Greenstone royalty applies to the entirety of the Gunnison Project and the Johnson Camp Property and production therefrom.

The Gunnison Project and the Johnson Camp Property is also subject to a Metal Stream Agreement with Triple Flag Mining Finance Bermuda Ltd. (“Triple Flag”) that is applicable to all oxide minerals production from the parts of the Project located in the “Stream Area”. The Metal Stream Agreement is summarized in Table 4-3.

Table 4-3: Triple Flag Metal Stream Agreement for Gunnison Project and the Johnson Camp Property

<table>
<tr><td>Stream Deliveries</td><td>Excelsior Mining Arizona Inc. (“Seller”) is required to deliver Grade A Copper Cathodes in an amount equal to the “Payable Copper”. The amount of Payable Copper is calculated based on a percentage of the amount of copper that is sold and delivered to Offtakers under the terms of Offtake Agreements (for percentages see heading – Payable Copper).</td></tr>
<tr><td>Payment</td><td>The Buyer pays to the Seller a price for copper equal to 25% of the daily official LME Grade A Settlement quotation for copper quoted in U.S. Dollars, as published in the Metal Bulletin.</td></tr>
<tr><td>Payable Copper</td><td>“Payable Copper” means a percentage of the Reference Copper equal to:
<table>
<tr><th>Scenario</th><th>Stage 1
(25 mppa)</th><th>Stage 2
(75 mppa)</th><th>Stage 3
(125 mppa)</th></tr>
<tr><td>Upfront Deposit</td><td>16.5%</td><td>5.75%</td><td>3.5%</td></tr>
<tr><td>Upfront Deposit + Expansion Option</td><td>16.5%</td><td>11.0%</td><td>6.0%</td></tr>
</table>
At the current stage of the Project, the Buyer has made the initial Upfront Deposit ($65 million) and the Seller is ramping up to 25 mppa.
The “Expansion Option” provides Buyer the option to invest an additional $65 million in the event Seller approves an expansion to at least 50 mppa.</td></tr>
</table>

4.6.2 Gunnison Project Only

Callinan Royalties Corporation (now a wholly owned subsidiary of Altius Minerals Corporation) holds a gross revenue royalty over the Gunnison Project. The gross revenue royalty is defined as royalty percentage times receipts which is the sum of physical product receipts and deemed receipts. The royalty rate is 1.625% while the plant capacity is less than 75 million pounds per annum and 1.5% once plant capacity is greater than or equal to 75 million pounds per annum.

4.6.3 Johnson Camp Property only

Royal Crescent Valley, Inc. ("Royal Crescent") holds a 2.5% net smelter returns ("NSR") royalty interest in minerals produced and sold from the 15 patented claims. These 15 patented claims are also subject to the terms of a "Royalty Deed and Assignment of Royalty," recorded with the Cochise County Recorder's Office on June 19, 2009, at No. 2009-14847, and the "Grant of Production Payment" recorded with the Cochise County Recorder's Office on June 10, 1999, at No. 1999-18419, as modified by a certain "Assignment of Production Payment" between Arimetco, Inc. and Styx Partners, L.P. (collectively, the "Production Payment Agreements"). The Production Payment Agreements provide for a non-participating payment of $0.02 per pound out of production during the calendar month in which copper produced from the 15 patented claims. The production payment is only payable when copper prices are in excess of $1.00 per pound and is capped at an aggregate of $1,000,000, of which $428,675 has been paid and or accrued as of February 2, 2022.

4.7 ADDITIONAL PROPERTY TAXES

The Johnson Camp property is subject to an annual property tax from Cochise County based on the full cash value of the deposit. The total property taxes for 2021 were $123,961.

4.8 ENVIRONMENT AND PERMITTING

4.8.1 Gunnison Project

The Gunnison Project operates under an Aquifer Protection Permit (APP), Air Quality Permit (AQP), a Resource Conservation and Recovery Act (RCRA) site specific ID number. All of these permits are issued and administered by the Arizona Department of Environmental Quality (ADEQ). In addition, Gunnison operates under an Underground Injection Control Permit (UIC) administered by the EPA. Gunnison also has a site wide Reclamation Plan approved by Arizona State Mine Inspector (ASMI).

Existing closure liabilities at Gunnison are covered under the APP, UIC and ASMI. These include closure of the existing ponds, wellfield, and disturbed grounds. There are existing bonds in place to cover all reclamation and closure obligations.

4.8.2 Johnson Camp Mine

The Johnson Camp Mine (JCM) operates under an Aquifer Protection Permit (APP), Air Quality Permit (AQP), a Resource Conservation and Recovery Act (RCRA) site specific ID number. All of these permits are issued and administered by the Arizona Department of Environmental Quality (ADEQ). The on-site septic system is grandfathered under the APP regulations and therefore, does not require a permit. These permits will be amended as required to address restart of open pit mining and construction of a new heap leach pad. JCM has a site wide Reclamation Plan approved by Arizona State Mine Inspector (ASMI).

Existing closure liabilities at the JCM are covered under the APP and the ASMI Reclamation Plan. These include closure of the existing ponds, the leach pad, and all other disturbed grounds. There are existing bonds in place to cover all closure obligations. The amended APP is expected to include a compliance schedule item for updating closure costs and subsequent bonding of the leach pad closure in ten years from issuance of the amended APP.

4.9 OTHER SIGNIFICANT RISK FACTORS

There are no other known significant factors or risks that may affect access, title, or the right or ability to perform work on the property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Gunnison Copper Project (the Project) and the Johnson Camp Mine (JCM) are located in a sparsely populated, flat to slightly undulating ranching and mining area about 65 road miles east of Tucson, Arizona. The Tucson metropolitan area is a major population center (approximately 1,000,000 persons) with a major airport and transportation hub including well developed infrastructure (highways and rail) and services that support the surrounding copper mining industry. The towns of Benson and Wilcox are nearby and combined with Tucson can supply sufficient skilled labor for the Project.

Access to the Gunnison Project and JCM is via the Interstate10 (I-10) freeway from Tucson and Benson in the west or Wilcox in the east. The North Star deposit can be accessed via a short, improved dirt road heading approximately 1 mile east from the south side of the “Thing” roadhouse on the Johnson Road exit from I-10. JCM can be accessed from the same Johnson Road exit but along 1.5 miles of improved dirt roads.

The Project area encompasses approximately 10 square miles within Cochise County, Arizona and includes unpatented mining claims, private land, Arizona State Prospecting Permits, a single Arizona State Mineral Lease, and direct ownership of mineral rights. Unpatented mining claims give the Owner exclusive right to possess the ground (surface rights) covered by the claim, as well as the right to develop and exploit valuable minerals contained within the claim, so long as the claim is properly located and validly maintained.

For the Fee Simple lands (private), both the land and mineral rights are owned by Excelsior. The Connie Johnson Deed grants the mineral right to Excelsior as well as access to mining. The Arizona State Prospecting Permits gives lessee the right to explore and convert mineral discoveries to Arizona State Mineral Leases so long as the claim is validly maintained. Surface rights for the various land packages appear sufficient for Excelsior to conduct its mining operations, subject to applicable laws and permits. The proposed mining technique, In-Situ Recovery, is comprised of a wellfield developed over the deposit, requiring minimal surface disturbance. The SX-EW plants and ponds have a relatively small surface footprint compared with conventional mines requiring waste dumps, open pits, and tailings impoundments.

The Project has existing, well-developed infrastructure sufficient for copper exploitation. The Excelsior-owned Johnson Camp Mine is located 1.5 miles north of the Johnson Road exit. JCM has an existing complete SX-EW plant, process ponds, 69 kV powerline, fresh water supply wells, a complete road network, and an assortment of ancillary buildings that can be used administration, maintenance, laboratory, warehousing, and safety. These buildings can be minimally modified and improved to support the Gunnison wellfield copper exploitation.

The Gunnison SX-EW plants (Stages 2 and 3) and ancillary facilities have been located on hill tops that will be accessed by existing roads on State lands covered by State permits northeast and south of the North Star deposit.

The main Union Pacific Southern Pacific railway runs 3 to 5 miles south of the North Star deposit. Engineering plans, cost estimates, and preliminary discussions have been made to construct a siding and rail spur to the Gunnison Project to supply lime and tanker cars of sulfuric acid and/or molten sulfur for the production of sulfuric acid onsite after the initial years of copper production. A railroad spur could also be used to ship cathode copper, as well as other non-metallic products that could be produced on the Johnson Camp site such as crushed rock for railroad ballast.

The existing 69 kV electrical power line skirts the eastern border of the Gunnison Project and lands at the main Johnson Camp Mine substation. In Stage 2 development of the Gunnison Project, a tap will be taken off the existing power line to provide 69 kV power on the south side of I-10 to the new Gunnison substation where it will be stepped down to 24.9 kV for electrical distribution to the wellfield, the SX-EW facilities, the water treatment plant, and ultimately, the sulfuric acid plant in Stage 3. Once the sulfuric acid plant for the Gunnison plant is built in Stage 3 of the Project, approximately 9 MW of cogenerated power will be available from waste heat from the sulfur burning process. This power will be placed back into the local power grid and credited to the Project.

If the sulfuric acid plant, for whatever reason, is not built, it may be necessary to build a powerline and substation to tap a higher voltage transmission line (115 kV or 230 kV) to augment power available from AEPCO's Apache power plant southeast of Willcox. AEPCO's distribution subsidiary, Sulphur Springs Valley Electrical Cooperative (SSVEC) has mentioned that the capacity of the Apache power plant may not be enough to supply power to the Project by itself in Stage 3. SSVEC has given a scoping level estimate of $6.4 million to tie into another transmission power line for the Project.

Freshwater supply will be provided from existing wells and mine adits located on or near the Johnson Camp property. There are sufficient water resources on the Johnson Camp side of the property to satisfy freshwater make-up for the wellfield, tankhouse operations, and Water Treatment Plant reagent mixing as well as potable water supply for human consumption. The addition of the water treatment plant will give extra freshwater availability in Year 8 when membrane filtration is instituted for rinsing the wellfield of sulfuric acid. Because the Gunnison deposit is saturated, and nearly all the water pumped from the Wellfield is recycled back to the Wellfield, net water consumption is minimized.

The elevation on the property ranges from 4,600 to 4,900 feet above mean sea level in terrain of the eastern Basin and Range physiographic province of southeastern Arizona. The climate varies with elevation, but in general the summers are hot and dry, and winters are mild.

The area experiences two rain seasons in general, one during the winter months of December to March and a second summer season from July through mid-September. The summer rains are typical afternoon thunderstorms that can be locally heavy. Average annual rainfall for Dragoon is 13.2 inches and the average highs range from 58ºF in January to 94ºF in June. Occasional light snow falls at higher elevations in the winter months. Exploration programs and mining activities operate year around in the region.

Vegetation on the property is typical of the upper Sonoran Desert and includes bunchgrasses, yucca, mesquite, and cacti.

Figure 5-1: Typical Vegetation and Topography of the Gunnison Project

6 HISTORY

Prior to Excelsior involvement, there was no direct mining history of the North Star deposit, but the adjacent Cochise district has seen considerable copper, zinc, silver, and tungsten mining beginning in the 1880s and extending to the present day. Between 1882 and 1981, the district produced 12 million tons of ore containing 146 million pounds of copper, 94 million pounds of zinc, 1.3 million pounds of lead, 720 thousand ounces of silver, and minor quantities of gold (Keith et. al., 1983). Much of the historical production came from small-scale underground copper-zinc mines located on what is now the Johnson Camp property controlled by Excelsior. The most significant of these producers were the Republic and Moore mines. From 1904-1940, the ore from these mines reportedly contained 4 to 4.5 percent copper and 0.5-0.75 ounces of silver per ton (Cooper, 1964). The zinc content for this period was not reported. Post 1940, the ore contained 1.5 to 3 percent copper, 5 to 10 percent zinc, and about 0.3 ounces of silver per ton. The Republic mine was the site of the historic concentrating plant in the district. Smaller underground mines in the area, such as the Peabody, reportedly yielded very high-grade ore which averaged 7.5 percent copper, 4 ounces of silver per ton, and contained as much as 44 percent zinc (Cooper, 1964).

Copper-oxide mineralization has been mined 1.5 miles northwest of North Star at the Johnson Camp open-pit operation since 1975, most recently by Nord Resources Corporation from 2008 until 2010. This property is now controlled by Excelsior. Overall, approximately 39 million tons of ore and 187 million pounds of copper have been produced out of the Johnson Camp open pits.

In the 1960s, it was recognized that potentially economic copper-skarn mineralization could be identified remotely by magnetic highs related to the magnetite content of these mineralized bodies. As a result, a magnetic high lying southeast of the now nonexistent town of Johnson was drilled in the 1960s and the North Star deposit was discovered.

Since North Star's discovery, several companies have explored the area. During this time period, extensive drilling and assaying, magnetic and IP surveys, metallurgical testing, hydrological studies, In-situ Recovery (ISR) tests, and preliminary mine design and evaluations have been undertaken.

By the late 1960s, the North Star deposit was partly controlled by Cyprus and partly by private owners. In 1970, a division of the Superior Oil Company (Superior) joint ventured into the northern half of the North Star deposit with Cyprus and the private owners. During the early 1970's, Superior did most of the drilling and limited metallurgical testing of the North Star deposit, and by early 1974 had defined several million tons of low-grade, acid-soluble copper mineralization. During this time, the southern portion of the North Star deposit was controlled by Quintana Minerals Corporation, who drilled several diamond holes and conducted metallurgical testing.

By the late 1970s, Superior had relinquished its rights to North Star. Cyprus maintained the ground holdings on North Star for a time but did very little work. Cyprus handed most of the ground covering the North Star back to the private owners in 1977.

The focus since the 1970s has been to utilize in-situ recovery (ISR) or a combination of ISR and open pits as a potential mining strategy. By the early 1980s, Mr. James Sullivan had gained full control of Section 6 of the North Star deposit and by 1991 had gained control of Section 31 and Section 36 via the State Mineral Lease. Apparently, no work was done from the early 1980s through 1992.

6.1 1993 TO 1998: MAGMA COPPER AND PHELPS DODGE

Magma Copper Company (Magma) optioned North Star from Mr. Sullivan in 1983. Magma drilled 8 holes, completed several metallurgical tests (some on six-inch diameter core), undertook limited hydrological studies, and calculated a copper-oxide resource. Magma's interest in the Project was for ISR of the copper-oxide resource. Metallurgical test work completed by Magma indicated that greater than 70% recovery is possible with ISR. Shortly after being acquired by BHP-Billiton (BHP), Magma (BHP) relinquished the Project in 1997.

After BHP relinquished its option on North Star in 1997, Phelps Dodge Mining Company (Phelps Dodge) optioned the North Star deposit and, in conjunction with Mr. Sullivan, drilled several holes on the periphery of the deposit. In 1998, before Phelps Dodge finished their investigation of both deposits, the company decided to focus its exploration activities outside the continental U.S. and returned the Project to Mr. Sullivan.

6.2 1999 – 2006

No work was done at the Gunnison Project in 1999 through 2006.

6.3 2007 – 2010: AZTECH MINERALS

AzTech Minerals Inc. (AzTech) acquired an option for the Project in May 2007. Prior to this, Mr. Stephen Twyerould and AzTech had spent nearly two years compiling, summarizing, and digitizing historical project data from over thirty years of investigations by Superior, Cyprus, Quintana, CF&I, Magma, Phelps Dodge, and James Sullivan. This process involved building a digital database, verifying historical data, re-interpreting the geology in 3D, and calculating a copper mineral resource.

Biological surveys were conducted for AzTech by Darling Environmental & Surveying, Ltd (Darling). It was found that no federally listed, endangered, threatened species, or proposed and candidate species for listing were present in the survey area from their known distributions and ranges. In addition, the survey area does not contain suitable habitat necessary for survival or life-history requirements of such species. Anthropological surveys conducted for AzTech by Darling indicated only random artifacts were present and occasional clusters of artifacts scattered outside of the area of mineralization. No burial sites or significant cultural sites were identified. Nine lines of ground magnetic data were also obtained, and a water-table depth study was completed in June 2010.

In June 2010, AzTech and Excelsior announced their intent to merge. The merger was completed in October 2010 when AzTech merged with Excelsior Arizona, with Excelsior Arizona as the surviving corporation.

6.4 HISTORICAL RESOURCE ESTIMATES

Historical resource estimates for the North Star deposit were completed by Superior in 1974, Phelps Dodge in 1998, AzTech in 2010, and Excelsior in 2011 and 2014 (Table 6-1). The Superior and Phelps Dodge estimates were not prepared in accordance with the requirements of NI 43-101 and CIM definitions. Mr. Bickel has not done sufficient work to classify these historical estimates as current mineral resources or mineral reserves and Excelsior is not treating the historical estimate as current mineral resources or mineral reserves. All of these historical estimates are superseded by the mineral resource estimates presented in Section 14 of this report and are not to be relied upon; they are presented here only for ease of reference and historical completeness.

Table 6-1: Comparison of Previous Oxide Copper Resource Estimates to AzTech 2010 Estimate

Source	TCu Cut-off	North Star	
		Million Tons	TCu Grade
AzTech	0.1%	404	0.35%
Phelps Dodge	0.1%	440	0.39%
AzTech	0.3%	242	0.45%
Phelps Dodge	0.3%	300	0.47%
Superior Oil	0.3%	304	0.47%

TCu = Total Copper

7 GEOLOGICAL SETTING AND MINERALIZATION

The Gunnison Project including the North Star copper deposit is located in southeastern Arizona within the Mexican Highland section of the Basin and Range province (Figure 7-1). The province is characterized by fault-bounded mountains, typically with large intrusive cores, separated by deep basins filled with Tertiary and Quaternary gravels. Generalized stratigraphy of the Project area is shown in Table 7-1 below.

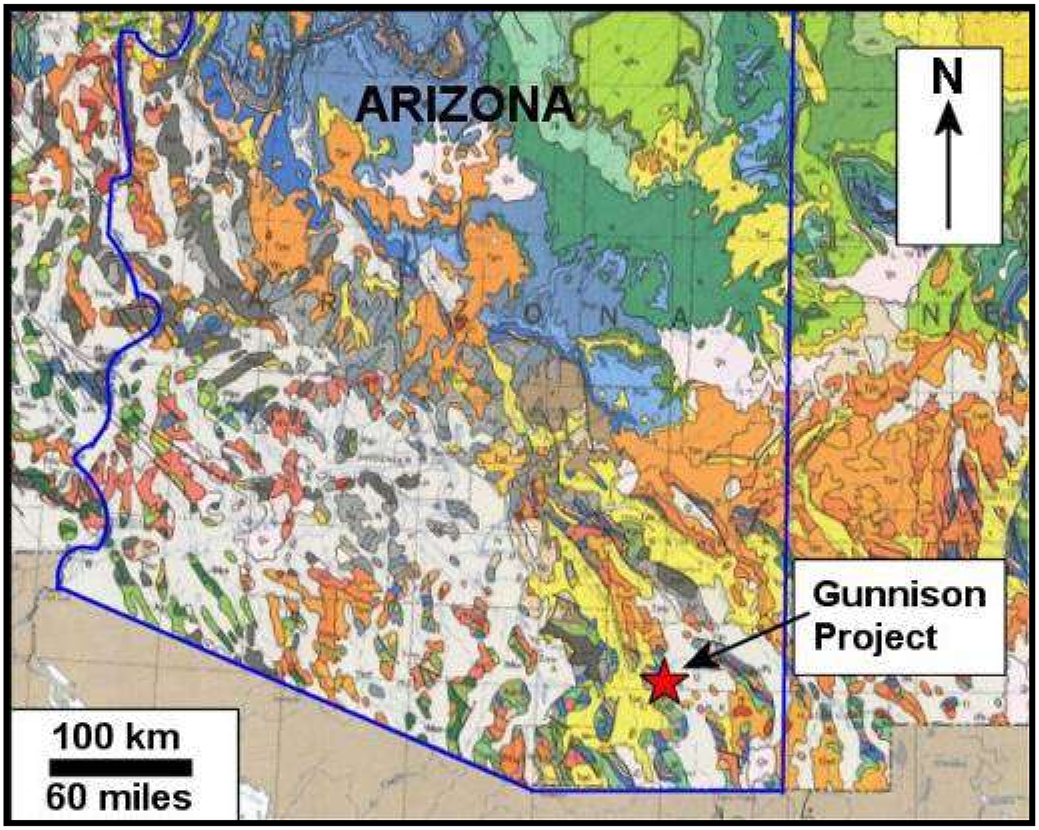


Figure 7-1: Regional Geologic Setting of the Gunnison Project (Modified from King and Beikman, 1974)

Table 7-1: Stratigraphy of the Gunnison Project Region
(Modified from Weitz, 1979; Clayton, 1978)

Rock Unit or Formation	Age	Gunnison Geology	Regional Geology
Basin Fill/Alluvium	Upper Tertiary and Quaternary	Unconsolidated boulders, sand, and gravel.	Stream laid gravels, sand and silt.
Texas Canyon Quartz Monzonite	Lower Tertiary	Quartz monzonite and related intrusions.	Intrusions important in mineralizing event.
Horquilla Limestone	Middle Pennsylvanian	Pyroxene rich calc-silicate hornfels and skarn, marble.	Limestone with abundant thin beds of shale.
Black Prince Limestone	Lower Pennsylvanian	Pyroxene rich calc-silicate hornfels and skarn, marble.	Limestone with thin shale at the base.
Escabrosa Limestone	Lower Mississippian	Garnet rich skarns and calc-silicate hornfels, marble.	Cliff forming limestone and dolomite. Copper skarns.
Martin Formation	Upper Devonian	Diopside-garnet skarns with diagnostic magnetite.	Dolomite with some shale and sandstone. Copper skarns.
Abrigo Formation	Upper Cambrian	Garnet-epidote-pyroxene-amphibole skarns and calc-silicate hornfels.	Shale, impure limestone and sandy dolomite. Copper skarns.
Bolsa Quartzite	Middle Cambrian	Red-brown to white quartzite and green hornfels.	Red-brown to white quartzite.
Apache Group (Pioneer Shale)	Upper Precambrian	Quartzite and metadiabase sills.	Basement rocks.
Pinal Schist	Lower Precambrian	Sericite schist.	Basement rocks.

7.1 REGIONAL GEOLOGY

The Gunnison Project including the North Star copper deposit is situated on the eastern edge of the Little Dragoon Mountains (Figure 7-2). The Little Dragoon Mountains are an isolated, fault bounded, up thrown block within the Basin and Range province in southeastern Arizona. The ages of the rocks range from the Proterozoic Pinal Schist to Holocene sediments. The southern portion of the Little Dragoon Mountains consists predominantly of the Eocene age Texas Canyon Quartz Monzonite, whereas the Pinal Schist and the Paleozoic sedimentary units that host the regional copper mineralization dominate the northern half.

The oldest rocks in the area, the Pinal Schist, are composed of Proterozoic sandstones, shales and volcanic flows that have been metamorphosed to greenschist and amphibolite facies. The Proterozoic Apache Group unconformably overlies the Pinal Schist and is composed of conglomerates, shales, and quartzite that were subsequently intruded by diabase sills. The Apache Group is then unconformably overlain by the Paleozoic rocks that host the mineralization including the Bolsa, Abrigo, Martin, and Escabrosa Formations. Overlying the mineralized rocks are the Black Prince and Horquilla Limestones. Tertiary/Quaternary basin fill has filled in the valleys.

The Texas Canyon Quartz Monzonite is thought to be the source of the copper mineralization at the North Star and South Star deposits, and is coarsely porphyritic, with potassium feldspar phenocrysts from 1 to 10 cm. Livingston *et al.* (1967) determined the age to be 50.3 ± 2.5 million years (Ma), which is uncorrected for current decay constants, and Reynolds *et al.* (1986) list eight determinations ranging from 49.5 to 55.0 Ma. The intrusion crops out to the west of the North Star deposit.

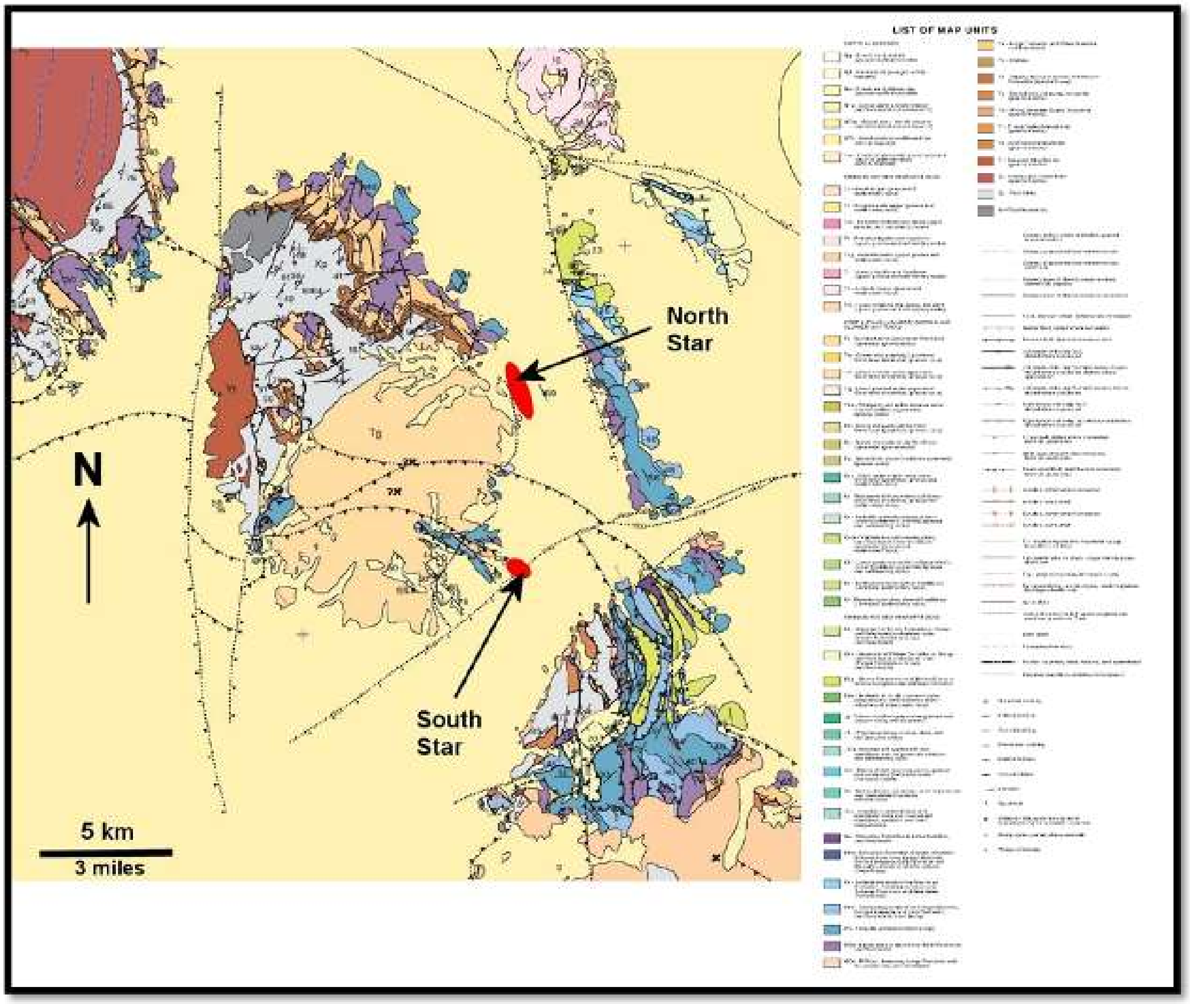


Figure 7-2: Geologic Map of the Little Dragoon Mountains (Modified from Drewes et al, 2001)

Several deformations have occurred in the Project area, with the most relevant being the Laramide Orogeny, to which the mineralization is related, and Basin and Range extension that has modified the topography to its current appearance. Much earlier, Pre-Apache Group deformation of the Pinal Schist included isoclinal folding with steep to overturned fold axes with a general northeastern structural trend. Minor deformation took place in the late Precambrian Era and between the end of the Paleozoic Era, but prior to the Cretaceous Period. The post Paleozoic, but pre-Cretaceous deformation produced steep northeast-to easterly-striking faults with offsets up to hundreds of feet.

The Laramide deformation was at right angles to the Pre-Apache Group deformation, with structures striking in a northwesterly direction. Older faults were reactivated and modified; folding and thrust faulting are common features of the Laramide deformation. The Centurion Fault of Laramide age is located south of the North Star deposit.

Structural trends at the regional scale include lithological units that strike approximately north-northwest and dip 20° to 45° NE; recurrent northeast-striking normal faults, and local north-northwest striking faults of variable slip directions. Regional geology and structure have been described extensively by Cooper and Silver (1964).

Two episodes of block faulting prior to the Quaternary Period have created the Basin and Range topography that dominates the current landscape and postdates the mineralization.

7.2 NORTH STAR GEOLOGY

The North Star deposit is covered by un-mineralized basin fill, varying between 300 and 800 feet in thickness. The mineralized Paleozoic host rocks below the basin fill strike approximately north-northwest and dip 20° to 45° east-northeast. Baker (1953) recognized three sets of faults in the Johnson Camp area and similar faults have been interpreted in the North Star area. These faults include the "Northeaster" (N10° to 30°E striking; 70° to 75° dip to the SE), "Easter" (N60° E to S60° E striking; 30° to 50° S and higher angle reverse faults dipping 75° S) and "Northwestern" orientations (N15° W strike; steep E or W dip). Only minor displacements are thought to have occurred in the North Star area; however numerous sheared and brecciated faults, generally filled with copper-oxide mineralization, cut through the deposit.

The Paleozoic host rocks have been intruded by the Texas Canyon quartz monzonite along the western margin of the deposit. The intrusion has formed wide zones of calc-silicate and hornfels alteration, as well as extensive low-grade copper sulfide mineralization within the Paleozoic rocks. Metamorphic alteration grading outward from the stock includes garnet-wollastonite-idocrase, diopside, tremolite and chlorite-talc (Kantor, 1977) (Figure 7-3). More specifically, the Martin Formation grades from a wollastonite-diopside-rich rock near the porphyry, to a distal diopside-tremolite-actinolite assemblage, and finally to dolomite. The Abrigo has garnet-actinolite-epidote-diopside alteration with some biotite hornfels near the porphyry, and this grades to a distal tremolite alteration leading into un-metamorphosed limey shale. Quartz-orthoclase-carbonate ± magnetite and chalcopyrite veins are characteristic of the lower Abrigo where it is mineralized.

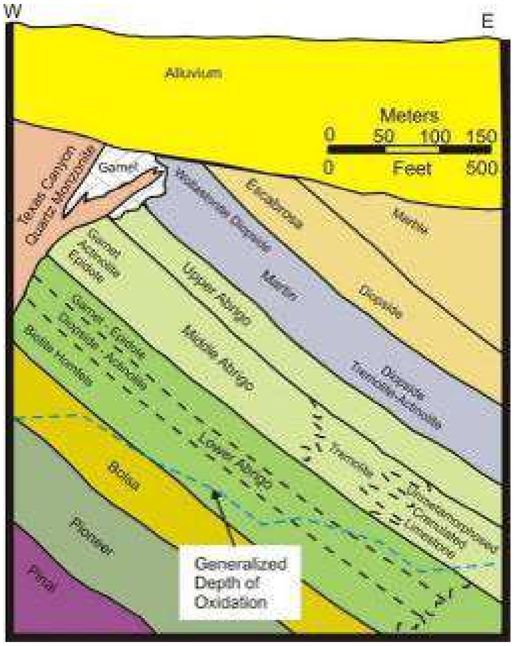


Figure 7-3: North Star Generalized Geological Cross Section (after Kantor, 1977)

7.2.1 Structural Framework of the North Star Deposit

At North Star, the mineralized formations strike approximately N10° to N40° W and dip from 30° to 45° NE. The strong regional trend of N10° to N30° E striking normal faults is overprinted by an abundance of N10° to N40° W striking reverse faults, joint sets, and normal faults which range in dip from 35° NE, sub-parallel to bedding, to 75° NE. The reverse faults strike parallel to the long axis of the deposit. Late-stage N70° E to S70° E striking vertical faults at the north end of the deposit contain local zones of high-grade copper-oxide mineralization. Porphyritic quartz monzonite intrusions occur along the western margin of the mineralization. At the southern end, the intrusion forms a sill between the Lower Abrigo Formation and the Bolsa Quartzite. At the northern end of the deposit, the intrusion commonly occurs as thin dikes and sills which cut the strata in numerous locations.

Excelsior has carried out on-going studies to model and understand the subsurface structural geology of the North Star deposit and its relation to mineralization and hydrology. Excelsior's methods and procedures for collecting and analysis of subsurface structural data, and the resulting interpretations and models are summarized in Section 9.

7.3 MINERALIZATION

Within the Project area the important mineralized host rocks include the Abrigo and Martin Formations and, to a lesser extent, the Horquilla Limestone, and the lower parts of the Escabrosa Limestone. Mineralization is also found in the Bolsa Quartzite and Precambrian basement rocks. Copper mineralization is related to calc-silicate skarns that have replaced these carbonate rocks adjacent to the Texas Canyon quartz monzonite (TQM).

Oxidation has occurred to a depth of approximately 1,600 feet and has resulted in the formation of dominantly chrysocolla with minor tenorite, copper oxides, and secondary chalcocite. Copper-oxide mineralization is present in the calc-silicate skarns as fracture coatings and vein fillings mainly in the form of chrysocolla. The remainder of the oxide mineralization occurs as replacement patches and disseminations. Copper-oxide mineralization extends over a strike length of 11,100 feet, has an aerial extent across strike of up to 3,000 feet and is more than 900 feet thick in places. Figure 7-4 shows the plan view geology of the deposit and Figure 7-5 and Figure 7-6 are east-west cross sections. Note the thickness and continuity of the mineralization. The north-south long-section view in Figure 7-7 also confirms the thickness and continuity of the mineralization.

Copper sulfide mineralization has formed preferentially in the proximal (higher metamorphic grade) skarn facies, particularly within stratigraphic units such as the Abrigo and Martin Formations, and within structurally complex zones. There are three types of sulfide mineralization within the skarns. In decreasing order of abundance these are fracture coatings and vein fillings, distinct quartz-orthoclase-carbonate ± magnetite and chalcopyrite veins 0.2 to 10 cm wide (Weitz, 1976), and disseminations. The veins have retrogressive haloes of chlorite, actinolite and epidote. Primary mineralization also occurs as stringers and veinlets of chalcopyrite and bornite.

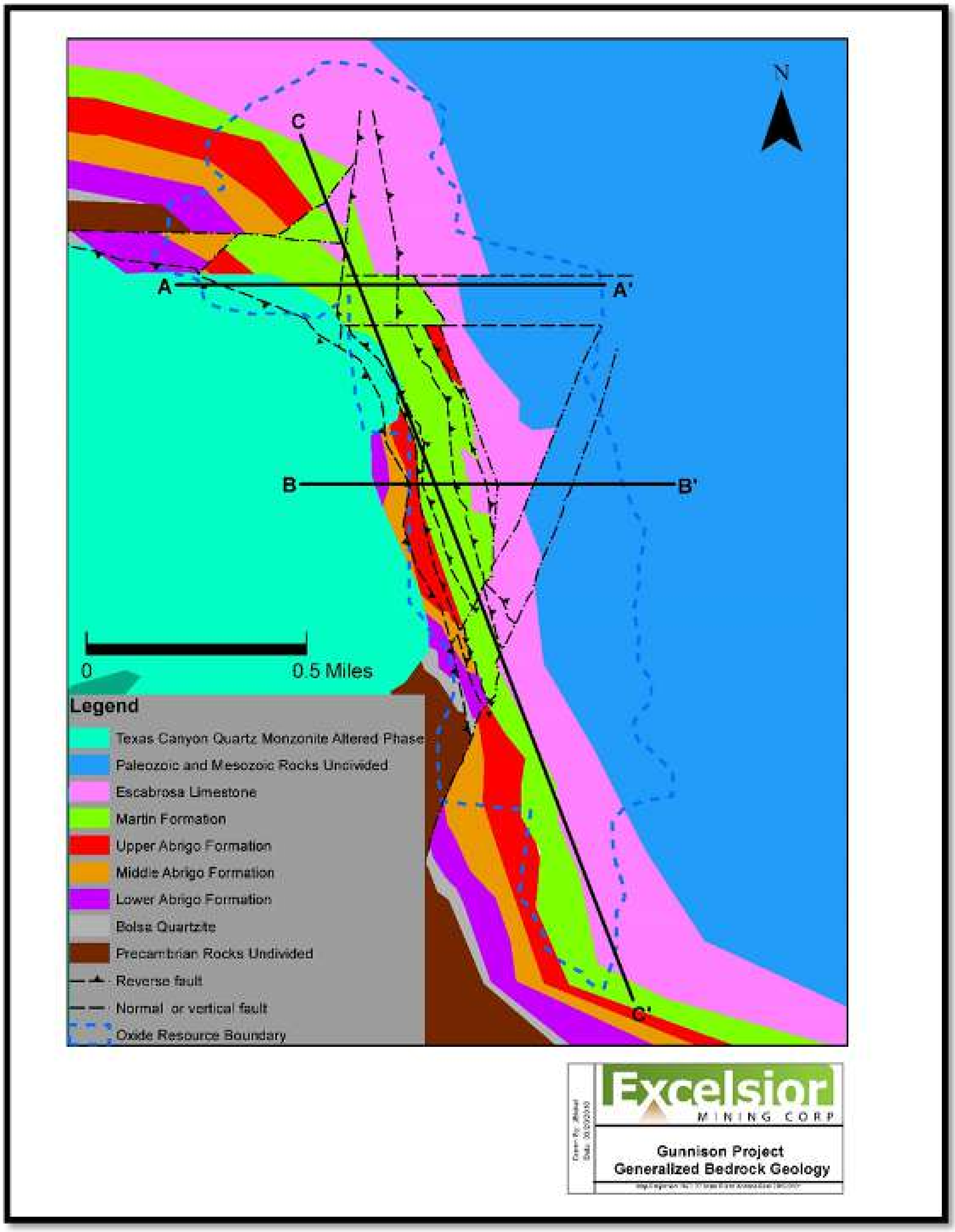


Figure 7-4: North Star Generalized Geology in Plan View, Below Basin Fill

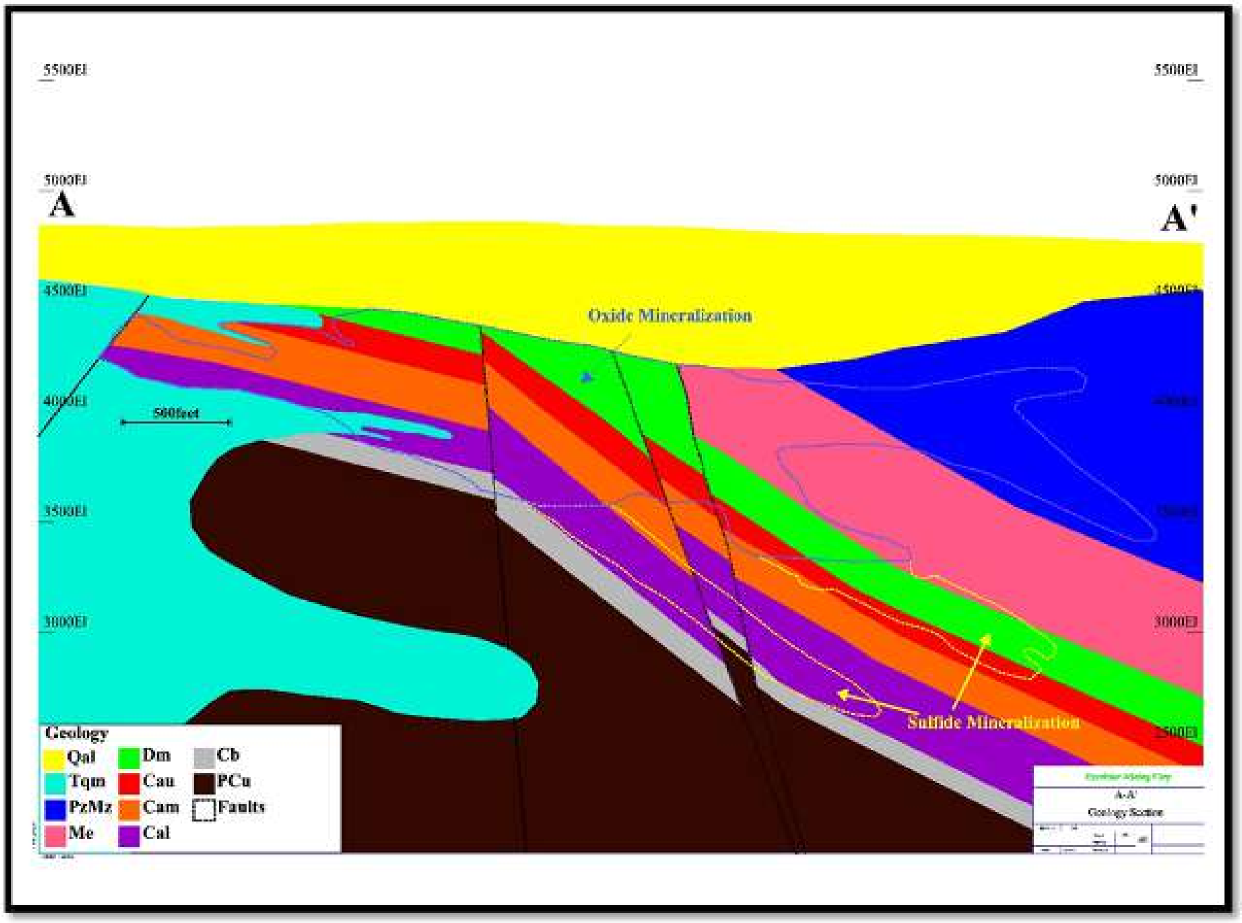


Figure 7-5: North Star East – West Geology Section at 394,400N Looking North

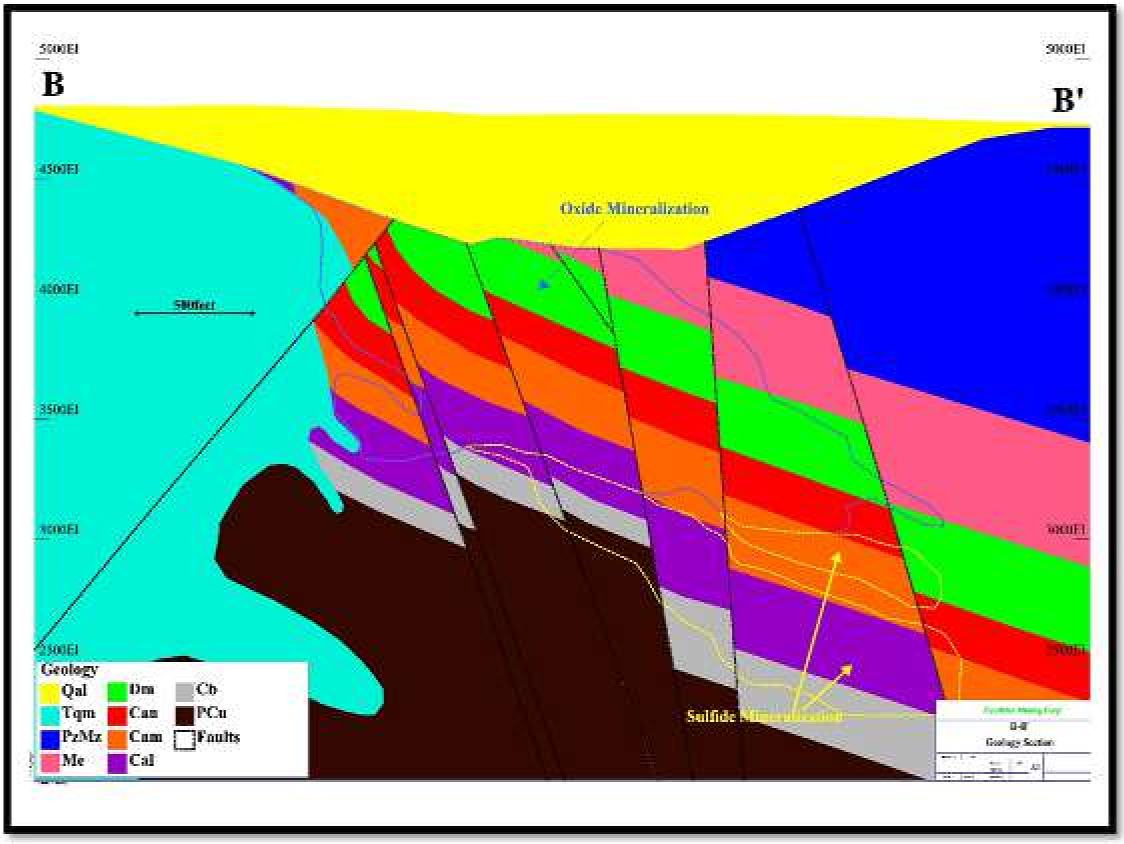


Figure 7-6: North Star East – West Geology Section at 392,000N, Looking North

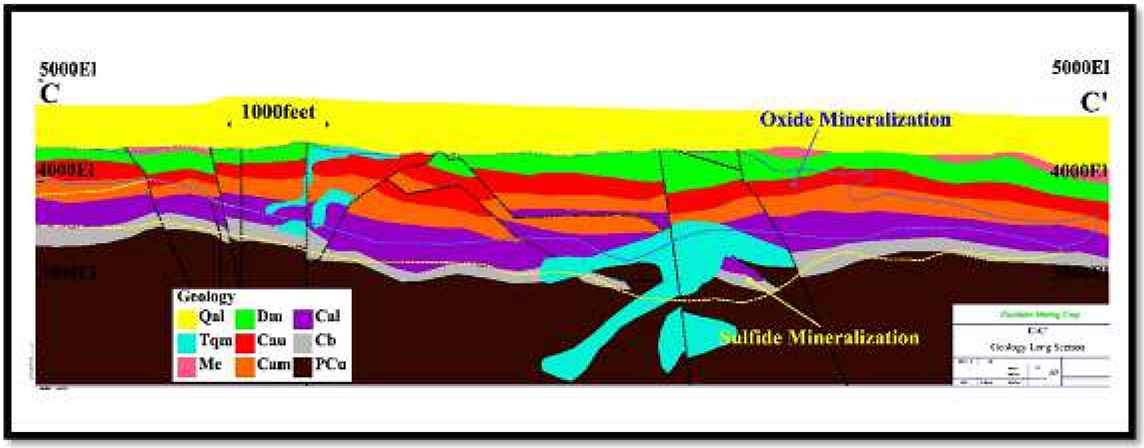


Figure 7-7: North Star North – South Geology Section, Looking East

Figure 7-8: Photograph of Typical Oxide Mineralization for North Star
Hole J-9: 780 to 806 feet

Texturally, pyrite and magnetite are later than, and replace, the skarn minerals, and chalcopyrite formed last. The magnetite occurs as disseminated 0.2 to 0.5 mm euhedral to anhedral grains and is closely associated with pyrite. Ninety percent of the magnetite is in the skarns and may compose up to five percent by volume of the rock. The disseminated magnetite and magnetite bearing veins are most likely what is giving the magnetic response for the deposit (Colburn and Perry, 1976).

Primary chalcopyrite-molybdenite disseminations and veins also occur in the mineralized porphyry below and to the west of the skarn mineralization at North Star. Only nine drillholes intersected the quartz monzonite over significant lengths (lengths > 100 feet). Most were mineralized with a best interval of 289 feet averaging 0.31% Cu and 0.028% Mo, including 30 feet at a grade of 1.35% Cu. This mineralization has never been fully assessed.

Both oxide and sulfide mineralization exhibit strong fracture control. This fracturing and faulting are best developed in terms of width and close spacing in a zone around the intrusive contact, and this decreases away from the intrusive contact in the less altered rocks to the east. The initial formation of the skarn created denser minerals and liberated CO_2 resulting in volume reduction, which created significant fracturing, and a consequent increase of porosity and permeability, allowing penetration by the later copper-bearing fluids. Weitz (1976) calculated a 30% volume reduction in the skarn-altered portions of the Abrigo and Martin formations at North Star.

Oxide copper also exists within the transition zone. It mainly occurs along fractures and in quartz vein selvages as chrysocolla. Secondary supergene copper sulfide minerals such as chalcocite are often associated with the oxide mineralization in the transition zone. The transition zone is typically 100 feet to 200 feet in thickness and is strongly fractured and broken, similar to the oxide zone.

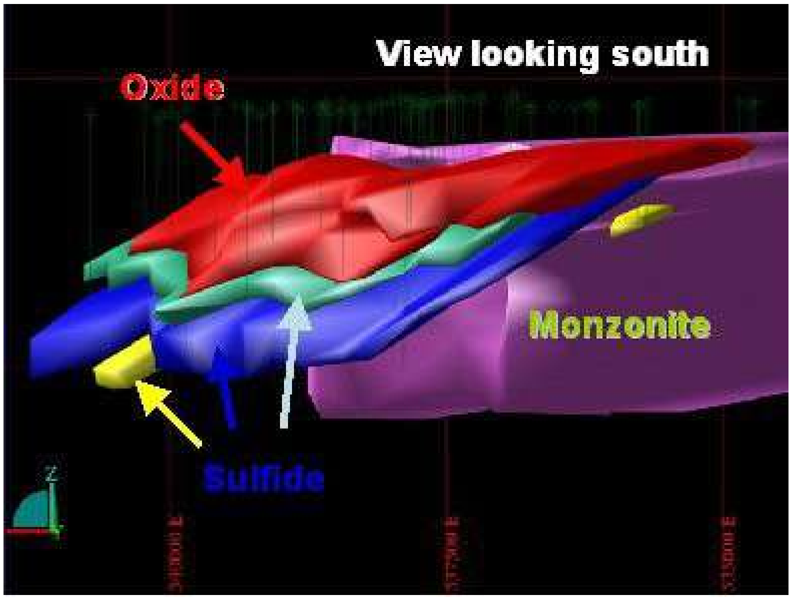


Figure 7-9: Generalized 3D View of Mineralization Looking South

7.4 ONGOING MODELING AND ANALYSIS

In 2021, Excelsior engaged RESPEC Company LLC (RESPEC) to begin reviewing data from the production wells completed in 2018-2019 and assisting with the construction of new geological models for the small area in and around the current wellfield for internal production purposes. The construction of the models is currently ongoing. Mr. Bickel has reviewed the data as a part of the modeling process and determined that any discrepancies between these new data and the modeled geology described in this report are immaterial to the mineral resources tabulated in Section 14. The reasoning behind this determination is discussed in Section 12.6.

8 DEPOSIT TYPES

The North Star copper deposit is a classic copper skarn (Einaudi et al, 1980 and Meinert et al, 2005). Skarn deposits range in size from a few million to 500 million tonnes and are globally significant, particularly in the American Cordillera. They can be stand-alone copper skarns, which are generally small, or can be associated with porphyry copper deposits and tend to be very large. The North Star deposit is large, at the upper end of the range of size for skarn deposits and is likely associated with a mineralized porphyry copper system that has not been discovered.

Copper skarns generally form in calcareous shales, dolomites, and limestones peripheral or adjacent to the mineralized porphyry. Copper mineralizing hydrothermal fluids are focused along structurally complex and fractured rocks and convert the calcareous shales and limestones to andradite rich garnet assemblages near the intrusive body, and to pyroxene and wollastonite rich assemblages at areas more distal to the stock. Retrograde hydrothermal fluids produce actinolite-tremolite-talc-silica-epidote-chlorite assemblages that overprint earlier garnet and pyroxene. The mineralization is typically pyrite-chalcopyrite-magnetite proximal to the mineralizing porphyry and chalcopyrite-bornite more distally from the body. The copper-gold porphyry and skarn model by Sillitoe (1989) (Figure 8-1) is being used as a conceptual exploration model for the North Star deposit. Application of the model entails testing magnetic highs (potential skarns) around magnetically quiet areas (copper porphyry).

Copper-zinc skarns are important in the region and have been historically mined from the Republic, Copper Chief, Moore, and Mammoth mines from underground operations (Baker, 1953). These copper and zinc rich skarns are probably more distal to the mineralized porphyry, whereas the North and South Star skarns contain only Cu and are proximal to the mineralizing porphyry system. Mineralization similar to that at the North Star deposit has been mined 1.5 miles to the north at Johnson Camp. Tungsten and minor lead-silver-gold have also been produced in the district (Cooper and Silver, 1964).

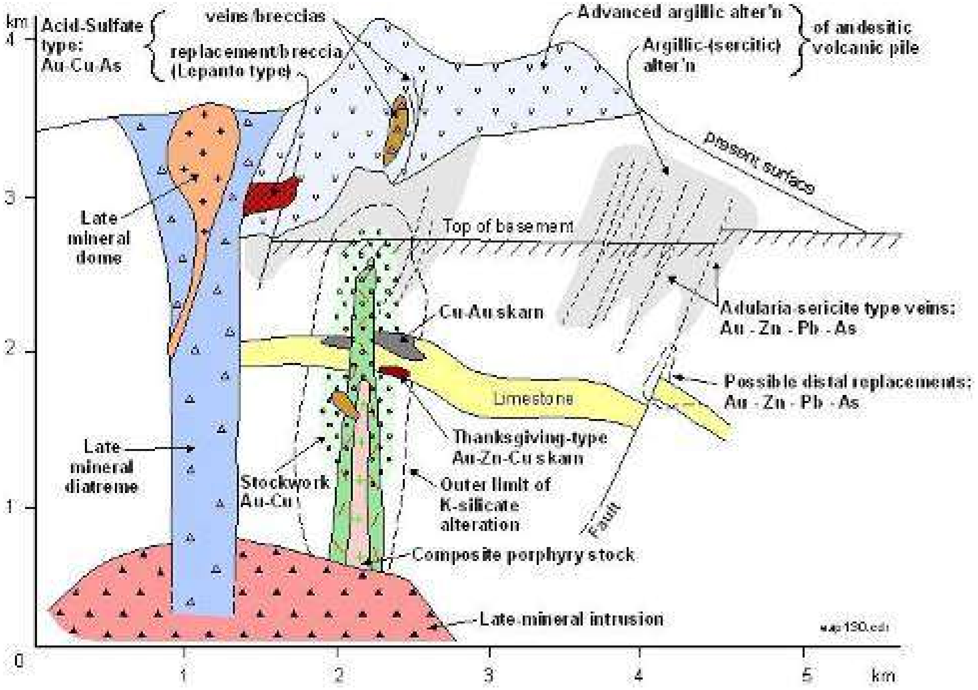


Figure 8-1: Porphyry Copper and Skarn Model (from Sillitoe 1989)

9 EXPLORATION

Excelsior initiated a re-logging program in December 2010 that was completed in 2011. In addition, a re-assaying program began in March 2011 during which all of the Magma Copper drillholes were re-assayed. Prior to the re-assay, historical Cyprus/Superior (CS) holes that had both total copper (TCu) and acid soluble (ASCu) results were re-split and check assayed at Skyline Labs in Tucson. The results are described in Section 12. In May 2011, a re-assay program was initiated for the Quintana Minerals holes (DC, S and T series) to include sequential Cu analysis. Previous results only included TCu assays.

From late in 2010 through early 2015, Excelsior has drilled 54 diamond drillholes, totaling 78,615 feet, for metallurgical samples and copper resource definition and expansion. Commencing in 2011, Excelsior also drilled 33,077ft in 32 rotary holes for hydrologic testing and observation in the North Star deposit area.

Southwest Exploration Services, LLC and COLOG were contracted by Excelsior to complete down-hole geophysical surveys during the 2011 to 2015 drill programs. Some holes were not surveyed due to bad ground conditions, and the surveys were shortened in others not reaching the total drilled depths. Altogether, down-hole geophysical data were obtained from a total of 66 drillholes in the deposit. Data collected included temperature, caliper log, sonic log, and acoustic televiewer. The down-hole geophysical data have been analyzed and evaluated as described in Section 9.2.1.

From late in 2018 through late 2019 Excelsior drilled 57 wells totaling 74,342 feet into and around a 400-foot by 400-foot area which is serving as the current ISR production wellfield. Southwest Exploration Services, LLC was contracted by Excelsior to complete down-hole geophysical surveys during the 2018 to 2019 drill program. All production wells were surveyed. Some surveys did not reach the total drilled depths due to bad ground conditions. The down-hole geophysical data have been analyzed and evaluated as described in Section 9.2.1.

9.1 EXCELSIOR STRUCTURAL GEOLOGIC METHODS

Excelsior's technical team has made a substantial effort to understand the structural geology of the North Star Deposit, particularly as it relates to controls on oxide copper mineralization and ground water hydrology. High-quality data collection and research regarding the structural nature of the subsurface has been fundamental to advancing the Project. This subsection summarizes how Excelsior has collected, interpreted, and modeled subsurface structural data as part of its exploration program to aid resource estimation, mine planning and extraction. Excelsior collects structural data by the following four main methods.

9.1.1 Structural Logging

As a part of the core logging process, Excelsior's geologist logged structure type (fault, shear, breccias, etc.), took angle to core axis measurements of the structures, and noted the mineralogy existing on the feature planes, infill, gouge, and selvages.

9.1.2 Down-hole Geophysical Surveys

For Excelsior's drilling programs since 2011, borehole geophysical tools including an acoustic borehole televiewer, were used to collect geophysical data down the holes. Images produced by the televiewer are used by Excelsior's geologist to identify and interpret structures by comparing the geophysical logs with the core, characterize structures by type and infill or gouge mineralogy, and obtain their true structural orientation using WellCad software. Other data collected from the surveys included caliper, sonic, and temperature logs.

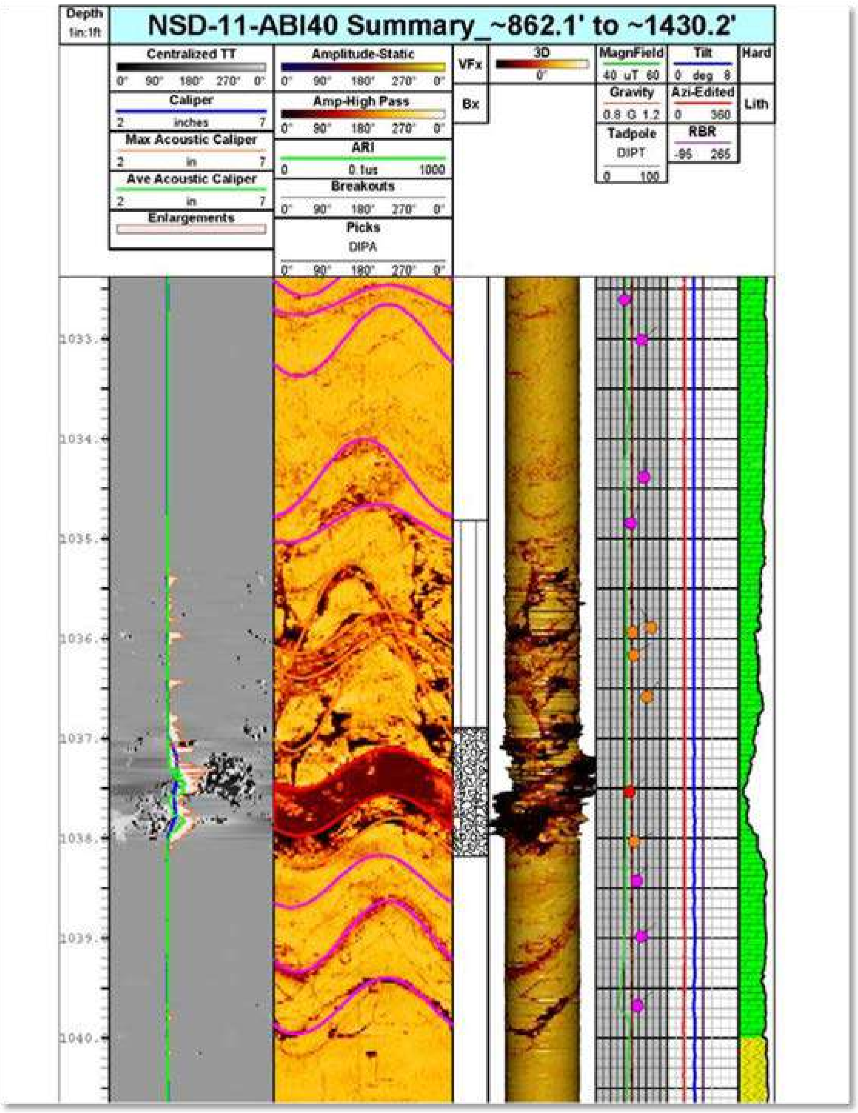


Figure 9-1: Graphical Example of Geophysical Log

9.1.3 Fracture Intensity

Fracture Intensity is defined as the relative brokenness, and hence permeability control, of the rock based on pieces of drill core that are less than or equal to 4 inches in length. Beginning in 2011, Excelsior geologists logged Fracture Intensity for each drillholes based on a scale of 1-5, with a value of 5 representing the most fractured rock. Definitions for the scale of Fracture Intensity are described in Table 9-1.

Table 9-1: Fracture Intensity Definitions

Fracture Intensity	Description
1	Very Weak (0-5% ≤4")
2	Weak (5-20% ≤4")
3	Moderate (20-50% ≤4”)
4	Strong (50-80% ≤4”)
5	Very Strong (80-100% ≤4”)

Examples of Fracture Intensity are shown below by rock unit. In general, the Fracture Intensity rankings are consistent regardless of formation (see Figure 9-2 and Figure 9-3). Higher Fracture Intensity levels tend to be characterized by large amounts of iron and copper-oxide minerals.

Intensity = 5

Intensity = 4

Intensity = 3

Intensity = 2

Intensity = 1

Figure 9-2: Fracture Intensity Examples from the Abrigo Formation

Intensity = 5

Intensity = 4

Intensity = 3

Intensity = 2

Intensity = 1

Figure 9-3: Fracture Intensity Examples from the Martian Formation

9.1.4 Fracture Mapping

For every assay sample (every 10ft unless truncated by a lithologic boundary), Excelsior’s geologist logged “Fracture Mapping”. This is the quantity of fractures per assay sample in the drill core, which can be used to calculate fractures per foot. The following categories were logged for Fracture Mapping:

- quantity of mineralized, open fractures per assay sample,
- quantity of mineralized closed fractures per assay sample,

- quantity of non-mineralized open fractures per assay sample; and
- quantity of non-mineralized closed fractures per assay sample.

9.2 EXCELSIOR STRUCTURAL DATA ANALYSIS, INTERPRETATION AND MODELING

The data collection described in Section 9.1.1, Section 9.1.2, Section 9.1.3, and Section 9.1.4 was used to create the following relevant outputs:

- Structural Analysis of the deposit;
- 3-D Wireframe Structural Model; and
- Structural Block Model.

9.2.1 Structural Analysis

Excelsior staff performed a Structural Analysis that examined all collected structural data outlined in Section 9.1 in detail and was the fundamental building block for all other structural interpretations. It was also used to aid the geology interpretation.

Figure 9-4 shows the major faults which displace stratigraphy in the deposit projected at the bedrock surface. Their spatial locations and orientations were defined in the Structural Analysis. The numerous parallel reverse faults which strike approximately N-NW cause repetition in stratigraphic section. All reverse faults dip steeply (70-80°) to the NE, except the westernmost reverse fault which dips approximately 60°SW. A subset of NE-striking normal faults, which dip steeply to the SE, is located on the margins of the deposit to the north and south. Also at the north end, E-W sub-vertical faults intersect the deposit along its short axis.

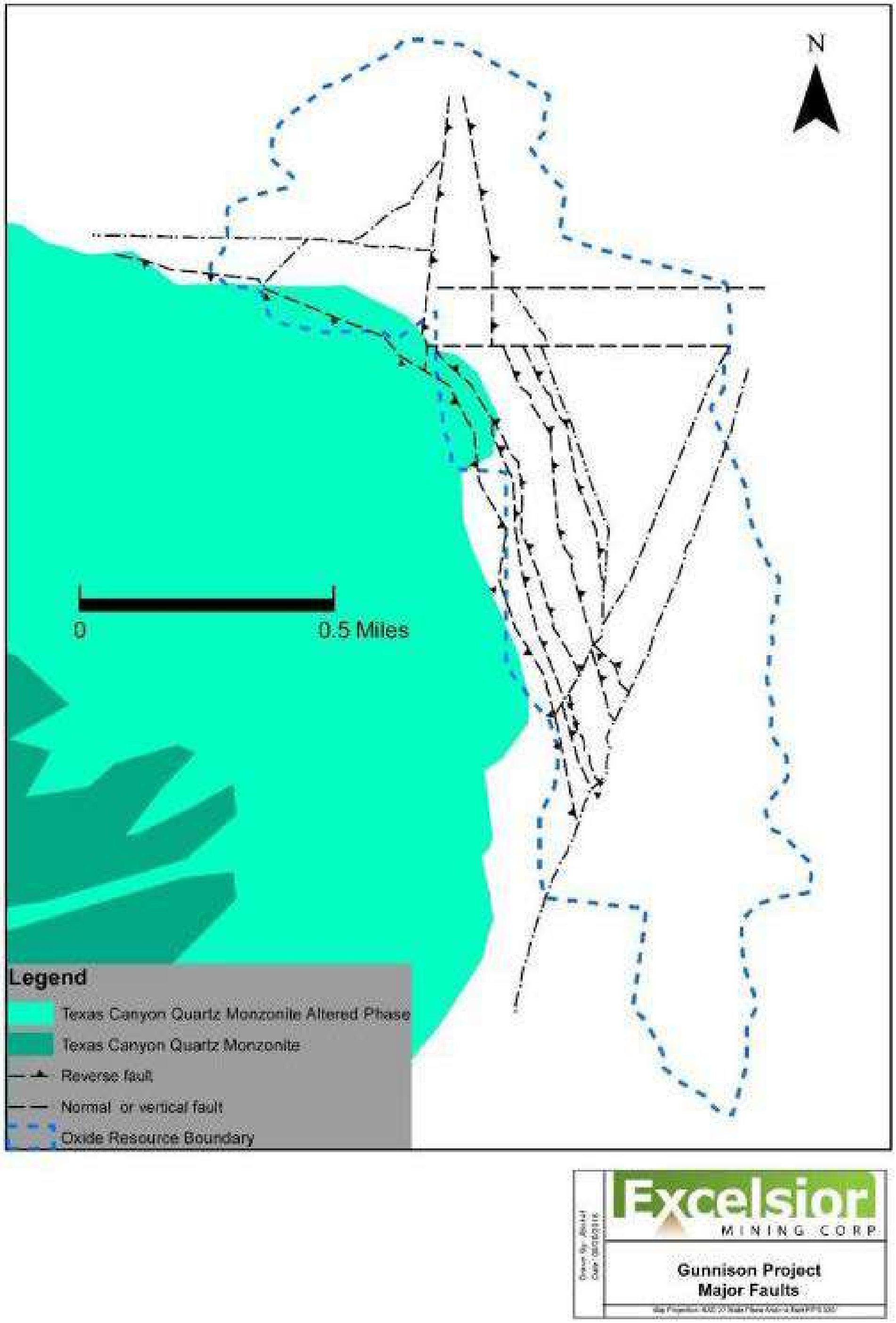


Figure 9-4: Plan View of Major Faults at Bedrock Surface which Displace Stratigraphy

The Structural Analysis also showed that, aside from the major faults which displace stratigraphy, the deposit is dominantly cut by faults, fractures, and joints which strike and dip sub-parallel to bedding. Figure 9-5 is a contour plot of structural data from the geophysical surveys. It contours the poles to dip directions for all structural features measured in the deposit (excluding bedding orientations). Note the strong presence of features which dip moderately to the NE and strike N-NW. These features are approximately sub-parallel to the strike and dip of the stratigraphic units at Gunnison.

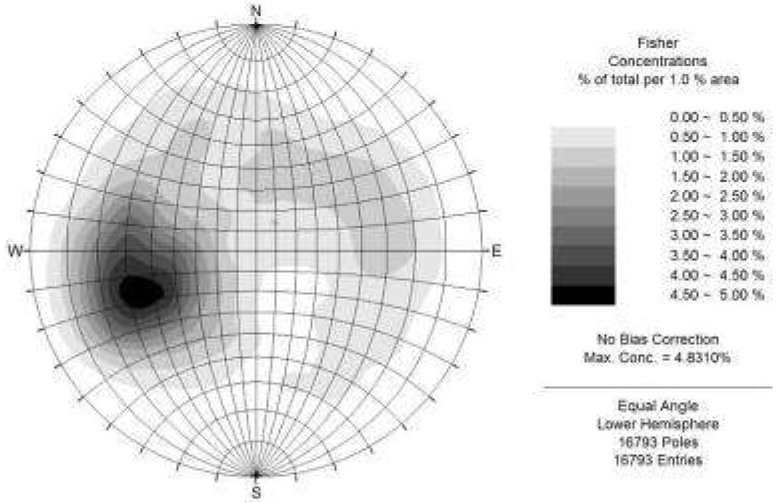


Figure 9-5: Contour Plot of Poles to Dip Directions for Structural Features, Excluding Bedding Orientations

The structural architecture of the subsurface resulting from the interpretations made in the Structural Analysis is a framework of high angle structures with numerous conjugate structures which are sub-parallel to bedding. Figure 9-6 is a schematic east-west cross section showing this framework. The cross section shows the approximate thickness of the structural zones as defined by the Structural Analysis.

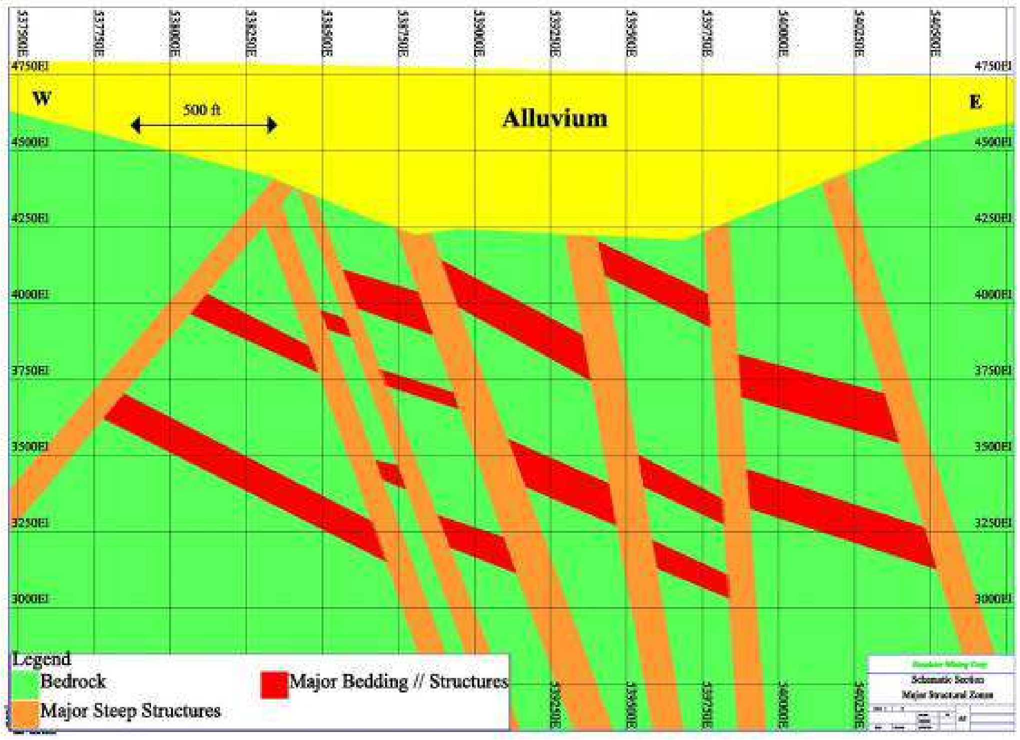


Figure 9-6: Schematic East – West Cross Section Showing the Structural Framework of the Deposit

Much of the copper-oxide mineralization in the North Star Deposit occurs on or proximal to fractures in the rock. Highly fractured zones are typically enriched in chrysocolla. The Structural Analysis validated this relationship through the examination of structural data. Figure 9-7 is a chart which shows a positive correlation between Fracture Intensity and the average total copper grade (TCu) for each Fracture Intensity ranking, at a 0.05% TCu cut-off in the approximate oxide zone of the deposit.

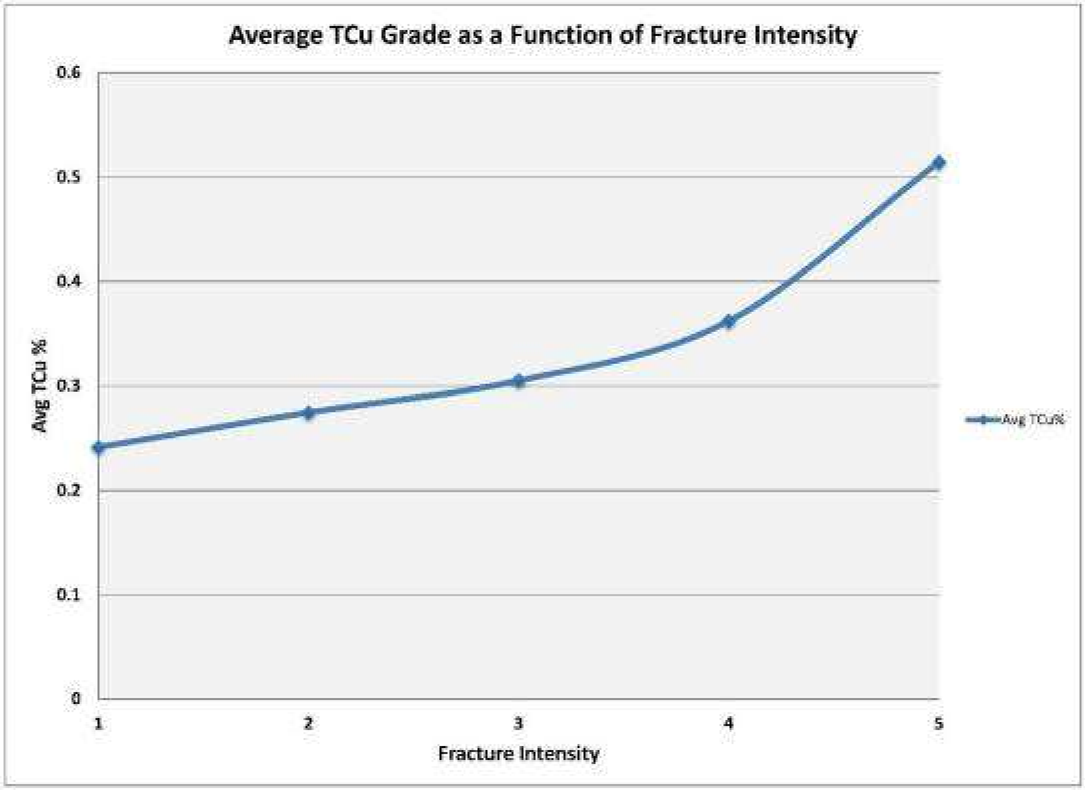


Figure 9-7: Correlation between Fracture Intensity and TCu Grade

Analysis of the Fracture Mapping data also yielded results which validated the relationship between fracturing and mineralization. Figure 9-8 shows the average number of fractures per foot as a function of the assay grade. Note that there are less data available on Fracture Mapping than Fracture Intensity because Fracture Mapping could not be performed on historical core.

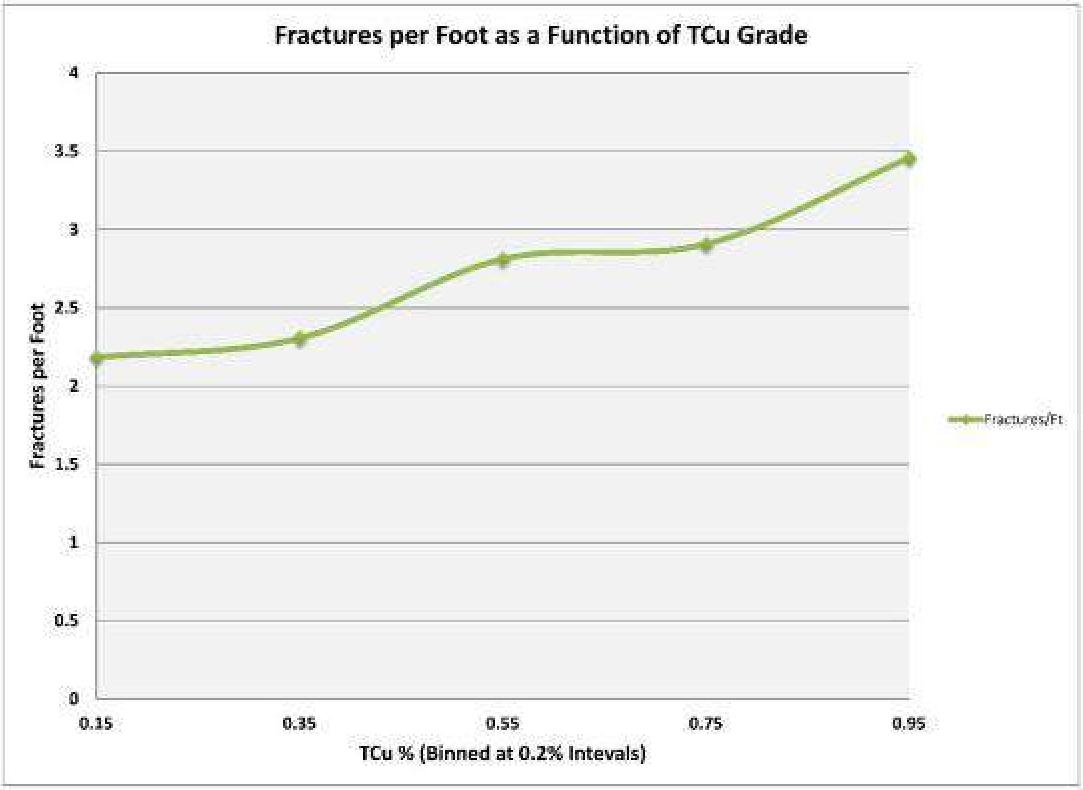


Figure 9-8: Relationship between Fractures per Foot and Assay Grade (TCu)

9.2.2 3-D Wireframe Structural Model

Excelsior geologists constructed a 3-D Wireframe structural model (or “Structural Domains” model) that consists of three-dimensional volumes that encapsulate significant structurally affected zones in the deposit. Their spatial locations and orientations were defined by the Structural Analysis. To be considered significant for the purposes of the model, these highly fractured and/or faulted zones were required to envelop drillhole intersections that have a minimum thickness of 30ft and a Fracture Intensity value of 3 or above. The outlines of the shapes were wire framed and subsequently used to triangulate volumes using Surpac software. This model is discussed further in Section 14.2.3.

9.2.3 Structural Block Model

Excelsior staff constructed a three-dimensional Structural Block Model, or “Fracture Intensity Model”, based on the logged Fracture Intensity data, the Structural Analysis, and the 3-D Wireframe Structural Model. The Structural Block Model blocks are coded with the Fracture Intensity value for each block and have dimensions of 100ft x 50ft x 25ft. Specific details regarding its generation are discussed in Section 14.2.5.

9.3 REGIONAL HYDROLOGY

A regional groundwater study was completed in April 2011 and updated in November 2015 by compiling available data for the region surrounding the North Star deposit. This compilation shows groundwater flows mainly to the east and southeast from the North Star deposit. Section 16 contains additional details regarding hydrology.

10 DRILLING

Excelsior's digital database for the North Star deposit mineral resource estimate includes 217 drillholes totaling 245,509 feet. A total of 122 core and RC holes were drilled in the deposit area, and 96 of these, totaling 140,034 feet, directly contributed assay data to the estimation of copper resources. Excelsior has also drilled 57 wells totaling 74,342 feet in their production wellfield area in 2018-2019. These wells are summarized in section 10.3. The data from these wells were not used in the mineral resource estimate tabulated in this report, which is discussed in Section 14.

Historical drilling was primarily conducted by diamond drilling methods, although a small amount was done by reverse circulation. The majority of drillholes have vertical orientations, which cross the predominant, generally shallow-dipping mineralized zones at North Star. A small number of angle holes were also completed by Excelsior, in attempts to intersect and validate interpreted geology and structure within the deposit.

The predominant sample length for the drill intervals in the Excelsior database is 10 feet (3.048 meters), with a relatively small percentage of shorter or longer intervals based on lithologic factors. RESPEC believes the drillhole sample intervals are appropriate for the style of mineralization at the North Star deposit. Furthermore, RESPEC is unaware of any sampling or recovery factors that may materially impact the accuracy and reliability of the results and believes that the drill samples are of sufficient quality for use in resource estimations.

Figure 10-1 is a plan map showing the North Star drillholes by company.

10.1 HISTORICAL DRILLING

The database includes 88 historical drillholes that were completed by several companies as shown in Table 10-1. These holes extend to a depth of approximately 2,450 ft below the surface at North Star and cover an area of approximately 310 acres, with additional drilling extending beyond this area. There is a slightly higher density of drilling along the central axis of the North Star deposit.

The historical drillholes are vertical and the mineralization ranges from flat lying to a 30° dip to the east, resulting in a ratio between sample length and true thickness of 1 to 0.87 depending on the true dip of the mineralization.

Table 10-1: Pre-Existing Drilling at North Star
(Diamond Drilling Includes Percussion Pre-Collar)

Company	Date	Type	Pre-fix	# of holes	Feet drilled
Cyprus	early 1970s	Diamond core	K	4	3,755
Cyprus/Superior	early 1970s	Diamond core	CS	36	45,786.6
Cyprus/Superior	early 1970s	Diamond core	CYS	1	887
Cyprus/Superior	early 1970s	Diamond core	J	10	12,167
Cyprus/Superior	early 1970s	Diamond core	K-20-X	1	983
James Sullivan	late 1980s	Diamond core	JS	3	1,665.5
Magma Copper	mid 1990s	Diamond core	MCC	6	8,099
Minerals Exploration	early 1970s	Diamond core	JD	4	2,206
Phelps Dodge	late 1990s	RC chip	Sullyl97	6	6,026
Quintana	early 1970s	Diamond core	DC	1	1,080
Quintana	early 1970s	Diamond core	S	3	3,394
Quintana	early 1970s	Diamond core	T	12	20,756
Superior	early 1970s	Diamond core	D	1	1,500
			Total	88	108,305.1

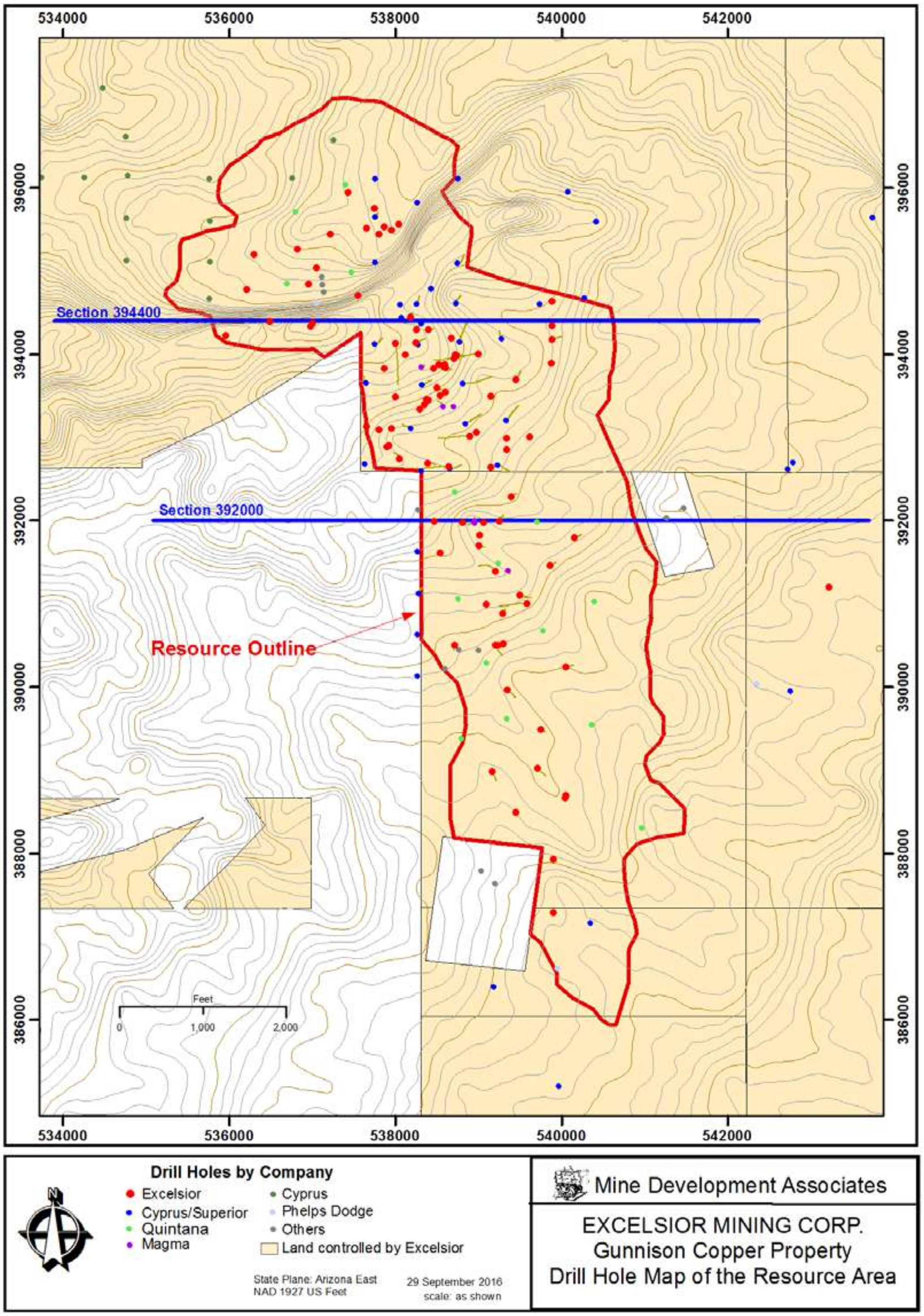


Figure 10-1: North Star Drillhole Collar Locations

Historical core drilled by Cyprus-Superior, Magma, and Quintana is NQ diameter with the exception of two Magma holes (MCC-7 and MCC-8), which were 6-inch metallurgy core holes. James Sullivan diamond drillholes were drilled with HQ-diameter core. The Cyprus-Superior holes used Joy Manufacture Co. as a drilling contractor. Magma drillholes were drilled by Christensen Boyles Corp. RESPEC has no further information on the drilling contractors, rig types, core sizes, and rotary or reverse-circulation drill-bit diameters used to perform the historical drilling.

Sampling of the drill core was on irregular downhole intervals based on geology using half-core splits. For the most part, the entire mineralized intersections have been sampled without any indication of sampling biases towards "high-grading". Individual down-hole sample intervals ranged from less than 2 feet to about 30 feet. Sample intervals larger than 25 feet generally represent intervals in the overburden (composite chip sampling). All historical drill core was split manually, divided in half, and placed in sample bags for transport to the assay laboratories. Samples have been assayed at commercial laboratories or in-house laboratories as listed in Table 10-2. All laboratories were located in Arizona.

Table 10-2: List of Assay Laboratories Used by Historical Operations

Company	Assay Laboratory	Comments
Superior	American Analytical and Research Laboratories	
Quintana	Southwest Assays and Chemists	
Phelps Dodge	Actlabs / Skyline Lab[1]	Some check assays at Morenci[2]
Magma	Magma's San Manuel mine laboratory[2]	

[1] Certified by American Association of Laboratory Accreditation
[2]-Denotes non-independent analytical lab

10.1.1 Historical Collar Position Surveys

Excelsior has located 46 historical drillhole collars and had them surveyed by Darling Geomatics using a Trimble Global Positioning System (GPS), which can be accurate to 0.05 ft horizontally and 0.2 ft vertically.

10.1.2 Historical Down – Hole Surveys

Historical borehole deviation data, where available, has been documented and added to the Excelsior database. Twenty-nine total historical holes have available survey data. The data came from either gyroscopic or down-hole camera surveys as a part of the initial procedures for the historical drillholes.

Table 10-3: Summary of Historical Borehole Deviation Surveys

Company	Hole Series	# of Holes Surveyed	Survey Types
Cyprus - Superior	CS	17	Gyroscopic
Cyprus - Superior	J	5	Gyroscopic
Magma	MCC	6	Survey Camera

10.2 EXCELSIOR DRILLING 2010 – 2015

Fifty-four diamond core holes have been drilled by Excelsior for a total of 78,615 feet of drilling. Fifteen of these holes were for metallurgical samples and the rest were drilled for resource definition or exploration purposes (Table 10-4; Figure 10-2). Twenty holes were completed from December 2010 to May 2011, eleven holes were drilled from March 2012 to May 2012, and an additional 23 diamond holes were drilled from September 2014 to January 2015. 6 ¼ inch pre-collars were drilled with rotary methods to the base of alluvium (100 to700 feet) and then cased with 4 ½ inch steel casing. HQ-size diamond core was drilled to a maximum depth of 2,000 feet, except where conditions required reduction to NQ size. Five metallurgy holes were drilled with PQ diameter core. Excelsior also completed diamond drilling through the entire section of alluvium for 2 holes in the 2012 program (NSM-001 and NSD-032). Of the 54 holes drilled, 44 have been assayed for inclusion into the mineral resource estimate described in Section 14. Excelsior has also drilled 32 rotary holes for hydrologic purposes between 2010 and 2015. Assays from these holes do not influence the mineral resource, but the rock chips collected from drilling were logged and used to aide in geologic interpretations of the deposit.

The Excelsior drillholes are mostly vertical. All Excelsior drillhole collars have been surveyed by Darling Geomatics using a Trimble GPS, which can be accurate to 0.05 ft horizontally and 0.2 ft vertically. Borehole deviation surveys were conducted for each drillholes using a Reflex down-hole camera survey for each Excelsior drillholes. Additionally, borehole geophysical logging was carried out on 84% of the Excelsior drillholes. Where available, the deviation surveys acquired from the geophysical logging supersede the camera surveys due to higher precision of the data.

Table 10-4: Listing of Excelsior Diamond Drilling 2010 – 2015

Hole ID	Northing (feet)	Easting (feet)	Elevation (feet)	Azimuth	Dip	Pre- Collar Depth (feet)	Diamond Depth (feet)	Total Depth (feet)	Purpose
NSD-001	393496.2	537998.1	4827.2	0	-90	460	1045.5	1505.5	Resource
NSD-002	392910	537923.6	4809.8	0	-90	580	1327	1907	Resource
NSD-003	392651.2	538646	4805	270	-70	565	1443	2008	Resource
NSD-004	391619.2	538540.6	4781.7	0	-90	510	799	1309	Resource
NSD-005	390510.7	538711.4	4740.2	0	-90	420	1488	1908	Resource
NSD-006	391109.8	539499.2	4753.6	0	-90	390	1610	2000	Resource
NSD-007	391470	539858.8	4737	0	-90	430	1370	1800	Resource
NSD-008	392291.2	539398.8	4783.4	0	-90	560	1212.5	1772.5	Resource
NSD-009	393007	539614.5	4788.2	0	-90	620	1173	1793	Resource
NSD-010	391983.3	538810.4	4768.2	0	-90	540	969	1509	Resource
NSD-011	393882.5	538523	4834.3	0	-90	650	788	1438	Metallurgy
NSD-012	390998.4	539093	4749	0	-90	400	1331.5	1731.5	Resource
NSD-013	391010.1	539587.2	4748.9	270	-70	480	1527	2007	Resource
NSD-014	390507	539202.9	4733.7	0	-90	400	1512.5	1912.5	Resource
NSD-015	389971.5	539349.6	4730.6	0	-90	400	1556	1956	Resource
NSD-016	389026	539713	4731.4	0	-90	420	1268.5	1688.5	Exploration & Resource
NSD-017	387936.5	539900.7	4695.4	0	-90	400	949	1349	Exploration & Resource
NSD-018	382749.3	538255.3	4688.2	210	-70	140	1264	1404	Exploration
NSD-019	393832.7	537871	4848.3	0	-90	620	834	1454	Resource
NSD-022	391700.4	539007.9	4759.5	0	-90	500	839	1339	Metallurgy
NSD-023	394132.1	538004.1	4857.3	180	-70	557	989	1546	Resource
NSD-024	394009.6	538994.7	4823.3	270	-70	672	1300	1972	Resource
NSD-025	393019.7	538893.5	4789.8	270	-70	637	1006.5	1643.5	Resource
NSD-026	394710.5	537551.9	4846.6	0	-90	466	702	1168	Resource
NSD-027	394377.4	537002.1	4883.3	0	-90	404	600.5	1004.5	Resource
NSD-028	394391.7	536487.3	4880.6	0	-90	396	359	755	Resource
NSD-030	394780.8	536207.8	4784.9	0	-90	240	527	767	Resource
NSD-031	395445.8	537220.3	4770.2	0	-90	416	592	1008	Resource
NSD-032	395280.2	536824	4786.4	0	-90	338	905	905	Resource
NSD-033	392745.5	538051.5	4809.1	0	-90	499	1080	1579	Resource
NSD-034	388494.6	539451.3	4708.7	0	-90	343	654	997	Resource
NSD-035A	388985.8	539165.4	4713.4	0	-90	321	838.9	1159.9	Resource
NSD-036	394225.5	535954.6	4888.2	0	-90	504	289.3	793.3	Resource
NSD-037	395565	538041.6	4751.3	0	-90	524	760.4	1284.4	Resource
NSD-038	388669.9	540044.5	4719.4	0	-90	402	1191	1593	Resource
NSD-039	389494	539748.4	4729.3	0	-90	383	1131.8	1514.8	Resource
NSD-040	390249.4	540050.5	4722.9	0	-90	222	1658	1880	Resource
NSD-041	391796.5	540151.9	4746.9	0	-90	383	1383	1766	Resource
NSD-042	391998.8	538464.7	4793.1	0	-90	460	1053	1513	Resource
NSD-043	393699.3	539451.7	4802.4	0	-90	628	1108	1736	Resource
NSD-044	387296.5	539902.7	4684.9	0	-90	322	445	767	Resource

Hole ID	Northing (feet)	Easting (feet)	Elevation (feet)	Azimuth	Dip	Pre- Collar Depth (feet)	Diamond Depth (feet)	Total Depth (feet)	Purpose
NSM-001	394139.3	538247.4	4850.5	0	-90	0	1150	1150	Metallurgy
NSM-002	392695.2	538391.1	4809.4	0	-90	507	493	1000	Metallurgy
NSM-003	392892.6	537897	4810.2	0	-90	608	420	1028	Metallurgy
NSM-004	393948.5	538702.4	4829.1	0	-90	596	518.3	1114.3	Metallurgy
NSM-005A	393065.2	538976.9	4786.9	0	-90	592	579.5	1171.5	Metallurgy
NSM-006	393997.1	538123.5	4847.5	0	-90	529	688	1217	Metallurgy
NSM-007	394447.2	538182.6	4844.2	0	-90	604	563.9	1167.9	Metallurgy
NSM-008	393344.9	538291.6	4815.6	0	-90	548	725	1273	Metallurgy
NSM-009	392647.8	539150.5	4794.1	0	-90	585	764.1	1349.1	Metallurgy
NSM-010A	390508.5	539236.6	4732.7	0	-90	424	414.9	838.9	Metallurgy
NSM-011	391996.1	539252.3	4774.9	0	-90	540	799.7	1339.7	Metallurgy
NSM-012	391397.3	539202.4	4765.4	270	-70	584	298	882	Metallurgy
NSM-013	394341	536980.7	4881.1	0	-90	404	549	953	Metallurgy

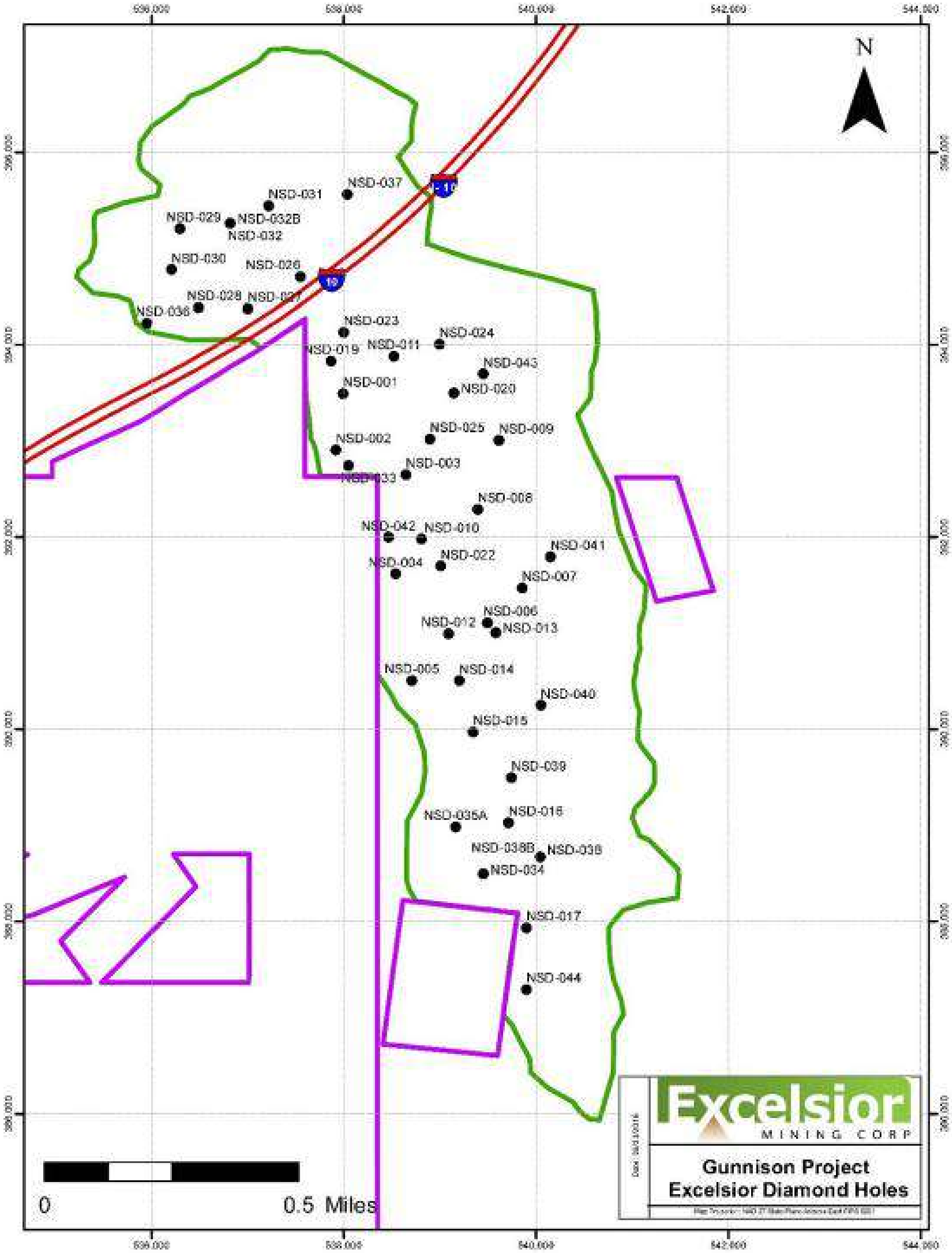


Figure 10-2: Excelsior Drillhole Collar Locations

10.2.1 Excelsior Drill Logging and Sampling Procedures

Following delivery of drill core from the drill sites to Excelsior's core storage facility in Casa Grande, Arizona, the core was laid out to check labeling, identify any missing intervals, and cleaning. Excelsior technicians measured and recorded core loss and RQD. The core was then logged digitally using customized AcQuire data-entry forms, which were then forwarded to the Excelsior database administrator. Additional logging of individual fractures from the borehole geophysical data was done in WellCad software.

The logging geologist marked up the core for sampling and splitting prior to photographing the core. Sample intervals were standardized at 10 feet; however, sample intervals were terminated at lithological boundaries. Other geological factors also led to shorter sample intervals at the discretion of the geologist. The core was then photographed wet and dry, and magnetic susceptibility was measured within each sample interval using a SM-30 handheld susceptibility meter.

Specific-gravity measurements were made using the water-displacement method for every assay sample in zones of mineralization, and every 10 feet outside of mineralized zones. The geologist made the determination on where SG measurements were taken in consideration of mineralized and un-mineralized materials, but measurement intervals most typically respected the assay intervals. The core was not wrapped or waxed for the density measurements. A quartz (SG = 2.65) and marble (SG = 2.71) standards were measured alternatively every 20 samples for quality control of the SG measurements. Readings outside of acceptable limits (three standard deviations) resulted in re-measurement of all samples back to the previous successful standard measurement. Duplicate SG measurements were made every 20 samples.

Samples were split using hydraulic splitters and bagged for shipment to the assay laboratory. Care was taken to ensure that no bias was introduced into the splitting by visually observing the mineralization in the core and splitting appropriately. The fines produced were also manually split and included in the sample.

10.2.2 Excelsior Core Recovery and RQD

Core recovery and RQD were measured for each drill run in every Excelsior diamond drillholes. Recovery was very high (average of 95%) with only rare occurrences of poor recovery due to discrete structures and/or narrow voids. RQD averaged 66%. Table 10-5 below defines RQD and Core Recovery as they relate to the total copper resource domains described in Section 14.2.7. The RQD and recovery values for geotechnical intervals lying within the modeled low-grade and high-grade domains are similar and are also close to the values for intervals lying outside of the modeled domains.

Table 10-5: Core Recovery and RQD for Excelsior Diamond Drilling 2010 – 2015

	All Excelsior Drilling	Inside Low-Grade Domain	Inside High-Grade Domain	Inside All Domains	Outside All Domains
% RQD	66%	63%	67%	65%	68%
% Recovery	95%	96%	96%	96%	95%
Intervals Measured	7,752	2,139	2,309	4,448	3,304

10.3 EXCELSIOR PRODUCTION WELL DRILLING 2018-2019

From late 2018 to late 2019, Excelsior drilled 57 wells totaling 74,342 feet for their first in-situ production wellfield. Of the 57 wells completed, 41 were Injection/Recovery production wells ("IR Wells") contained within a 400-foot by 400-foot area in the deposit and the remaining 16 were drilled and designed for various monitoring and compliance purposes exterior to the wellfield, in accordance with their operational permits. A map of the 2018-2019 drilling is shown in Figure 10-3. The monitoring and compliance well types include Hydraulic Control Wells ("HC Wells"), Intermediate Monitoring Wells ("IMW Wells"), Observation Wells ("OW Wells"), and Point of Compliance Wells ("POC Wells"). The 2018-2019 drilling campaign is summarized in Table 10-6.

Table 10-6: Summary of 2018-2019 Excelsior Drilling

Well Type	Count	Total Footage	Alluvium Footage	Bedrock Footage
IR Wells	41	53,387	24,925	28,462
HC Wells	9	11,790	5,160	6,630
IMW Wells	2	2,616	1,080	1,536
OW Wells	2	2,620	1,210	1,410
POC Wells	3	3,929	1,420	2,509
Total	57	74,342	33,795	40,547

The 41 IR Wells and 9 HC Wells were drilled by rotary methods with conventional circulation through the alluvium and into the bedrock to a specified footage, cased, and sealed in accordance with compliance requirements. The bottoms of the wells were then drilled through the rest of the ore zone with reverse-circulation and sampled per Excelsior's internal sampling procedures. All monitoring and compliance wells were generally drilled by rotary methods with conventional circulation in the alluvium and by mud-rotary methods with conventional circulation in the bedrock. The bedrock in the compliance wells was sampled per Excelsior's internal sampling procedures. Some exceptions made to the specific drilling methodologies described above were employed to address specific ground conditions.

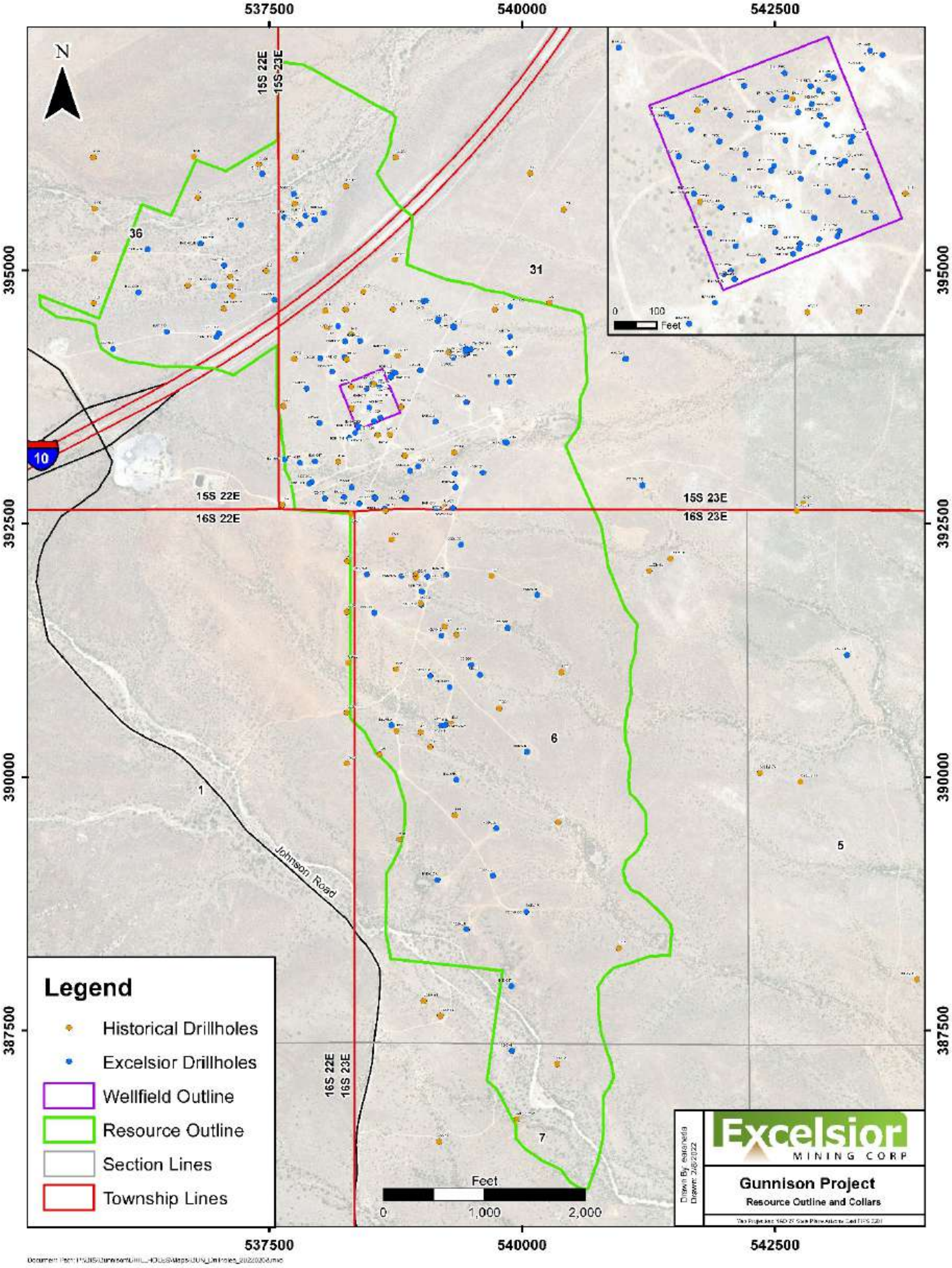


Figure 10-3: Collar Locations with 2018-2019 Drilling in Wellfield Area shown inside the insert

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following sections summarize the extent of Mr. Bickel's knowledge regarding the sample preparation, analysis, security, and quality assurance/quality control protocols used in the various drilling programs at North Star.

11.1 HISTORICAL SAMPLE PREPARATION, ANALYSIS AND SECURITY

The laboratory sample preparation and analytical procedures used by the previous owners of the deposits are unknown. However, the commercial analytical laboratories known to have been used by the historical operators at North Star are, or were at the time, well recognized and widely used in the minerals industry. In addition, all of the historical operators were reputable, well-known mining/exploration companies, and there is ample evidence that these companies and their chosen commercial laboratories followed accepted industry practices with respect to sample preparation, analytical procedures, and security.

For the most part, James Sullivan maintained security of the project information and drill samples since the early 1980's to 2006. Information and samples collected by Superior, Cyprus and Quintana in the 1970's to 1980's were handed over to James Sullivan and relocated to his core facility in Casa Grande, Arizona between 1980 and 1998. Magma Copper had security and control of its own information and samples from approximately 1993 to 1997, after which Magma relinquished control to James Sullivan who relocated all the Magma Copper information and samples to his core facility. Phelps Dodge maintained its information and samples until 1998, after which time they were transferred to James Sullivan and were relocated to his core facility.

From November 2006 until October 2010, the original information and samples were under the control of AzTech Minerals at the former James Sullivan core facility. Excelsior has maintained control of the core facility since October 2010.

11.2 EXCELSIOR SAMPLE PREPARATION, ANALYSES AND SECURITY

Excelsior's drill core sampling procedure is as follows:

- Assay tickets are placed at the start of the assay interval.
- Sample intervals are recorded within the AcQuire form as well as written within paper ticket books.
- All skarn and porphyry units are sampled. Additional sampling of rock types and/ or mineralization is left up to the discretion of the geologist, under the guidance with senior staff.
- Sample intervals are based on lithologic boundaries and are not taken across the boundary with the following exceptions:
    - short intervals (~<1 foot) can be included within a larger sample where isolating the unit would be problematic; and
    - thin lithologic units can be included within a larger sample when sampling such a unit is impractical.
- Sample length is 10 feet within all rock types. It is understood that irregular sample lengths may be needed at geological boundaries.
- In areas of poor ground conditions or poor recovery, sample lengths may extend up to 20 feet.
- Samples must be bracketed on either side by an additional sample (no isolated samples).

The core samples were manually split by an Excelsior technician using a hydraulic splitter, with one half placed in a numbered sample bag and the other half retained in the core box. Quality Assurance/Quality Control processes are discussed below in Section 12.3.

11.2.1 Excelsior Analytical Methods

Skyline Assayers and Laboratories (Skyline) in Tucson, AZ has been Excelsior's primary assay lab for drill samples since 2010. Skyline is accredited with international standard ISO/IEC 17025:2005 General Requirements for the Competence of Testing and Calibration Laboratories. Total copper (TCu), acid-soluble copper (ASCu) and cyanide-soluble copper (CNCu) were analyzed. Samples were also assayed for molybdenum in some cases at the discretion of the geologist. Excelsior has no relationship with Skyline other than Skyline being a service provider.

Upon receipt at Skyline, Excelsior's drill samples were lined up and coded into Skyline's lab information system. Any missing, illegible, or damaged samples were reported. Samples were crushed to 70-80% passing minus 10 mesh. The crushed samples were then split and recombined 3 times, and 250 to 280 grams of material were split and pulverized to 95% passing 150 mesh. Washed river rock was used to clean the crusher between samples.

The analytical methods for copper assays are as follows:

Total Cu (TCu) analysis: Samples are digested in a mixture of hydrochloric, nitric and perchloric acids. This solution is heated and taken to dryness. The contents are treated with concentrated hydrochloric acid and the solution is brought to a final volume of 200 mL with de-ionized water. This solution is read by Atomic Absorption using Standard Reference Materials made up in 5% hydrochloric acid.

Sequential Analysis of Acid-Soluble Cu (ASCu) and Cyanide-Soluble Cu (CNCu): Samples are digested in 5% sulfuric acid and supernatant solution is diluted to 100 mL with de-ionized water. The residue is digested in 10% sodium-cyanide solution and diluted to 100 mL. The ASCu samples are read on Atomic Absorption units using 0.5% H_2SO_4 calibration standards. The CNCu samples are read on Atomic Absorption units using 1% NaCN calibration standards.

11.2.2 Excelsior Sample Security

Drilling was carried out 24 hours a day, 7 days a week, during the drilling periods. Drill core was temporarily stored at the drill rig, supervised by both the driller and the site geologist. The drilling occurred on isolated ranch land behind a locked gate, limiting the access to authorized Excelsior and drilling personnel. The core was placed in closed core boxes on pallets and banded for pick up by a transport service. A transfer form was signed by both parties upon pickup and delivery of the core to Excelsior's core facility in Casa Grande. Once in Casa Grande, the core was stored in a locked facility. Core samples ready for assaying were transported from the core facility to the assay laboratory by Skyline personnel.

The sample preparation, analysis, and security protocols of Excelsior for the Gunnison Project meet current industry standards.

12 DATA VERIFICATION

The data verification reported herein is largely unchanged from that which was previously reported (M3, 2017). Jeffrey Bickel, C.P.G., of RESPEC Company LLC (RESPEC), is the Qualified Person responsible for the data verification in this report. Verification of data relevant to the mineral resources of the North Star deposit was originally completed under the supervision of Michael M. Gustin, also of RESPEC, unless otherwise noted. Mr. Bickel has reviewed the project data and the data verification completed by Mr. Gustin determined it to be good and adequate.

The major contributors to the current North Star deposit database include Excelsior and Cyprus-Superior, with smaller quantities of data from and Quintana and several other companies. The author experienced no limitations with respect to its activities related to the verification of the project data related to these companies.

No significant issues have been identified with respect to the data provided by Excelsior's quality assurance/quality control ("QA/QC") programs. QA/QC data are not available for the historical drilling programs at North Star, but Excelsior analyses dominate the assays used directly in the estimation of the mineral resources, most of the historical data were generated by well-known mining companies, and the Excelsior drill data are generally consistent with the results generated by the historical companies. Mr. Bickel believes the North Star data as a whole are acceptable as used in the estimation of the mineral resources presented in this report.

12.1 INTRODUCTION

In order to place the following discussions of database auditing and QA/QC into context, it is helpful to understand the origin of the most relevant project data. There are 122 holes in the project database that were drilled in the North Star deposit area; these holes have a total of 9,996 assayed sample intervals in the database. Of these sample intervals, 7,573 directly contribute data to the estimates of resource grades discussed in Section 14.

Table 12-1 lists the drillholes by company, as well as the percentages of the 7,573 sample intervals that are attributable to each company. Note that the percentages shown for all companies have been adjusted to reflect Excelsior's analyses of historical sample pulps and resampled historical core, as these Excelsior analyses replaced the historical assays in the project database where available.

Table 12-1: Drillhole Data by Company

Company	Hole Series	Number of Holes	Percent of Coded Assays Total Copper	 Acid-Soluble Copper
Excelsior	NSD, NSM	44	69%	70.70%
Cyprus - Superior	CS, CYS, J	43	24.00%	24.90%
Quintana	S, T, DC	15	5.50%	2.60%
Magma	MMC	8	0.00%	0.00%
Cyprus	K	2	0.40%	0.50%
Phelps Dodge	Sullyl97	2	0.30%	0.10%
Others	D, JS	8	1.20%	1.20%
Totals		122	100.00%	100.00%

12.2 DATABASE AUDITING

12.2.1 Collar Table

Excelsior provided RESPEC with two spreadsheets described as originating from Darling Environmental & Surveying, Ltd. of Tucson, Arizona – one spreadsheet with 2012 survey data and the other with 2015 surveys. RESPEC used this information to audit the locations of 71 Excelsior, 26 Cyprus-Superior, 13 Quintana, and 7 Magma drillholes. With the exception of one hole in which the survey location was based on an open hole in the ground, all of the locations of the historical holes were based on drill casing in the ground.

Out of the 117 holes audited, two discrepancies between the database and surveyed locations were identified, one of which was resolved by Excelsior. The other discrepancy involved a Cyprus-Superior hole, whereby the x, y, and z coordinates in the database differed from the survey coordinates provided to RESPEC by 0.2 to 1.5 feet. The database coordinates are correctly derived from a 2015 survey, while the audit records used by RESPEC have older coordinates.

In addition to RESPEC's auditing of the database, "M3, 2014" state that, "*During the author's site visit in 2007, a number of the drillholes locations were checked with a handheld GPS and found to reasonably match the recorded collar coordinated* [sic]."

12.2.2 Survey Table

RESPEC audited the down-hole deviation survey data for the Excelsior drillholes using both original digital files generated as part of the down-hole geophysical-survey data and scanned copies of original handwritten paper documentation of Reflex EZ-Shot measurements. The survey data for eight of the 45 Excelsior NSD-series core holes were audited, which includes 2,804 individual surveys out of the 10,233 surveys of the Excelsior holes. Six discrepancies between the database and the original records were identified, all in the azimuth readings. Two of the discrepancies exceed 0.1 degrees (0.4 and 0.6 degrees) and none are considered material.

RESPEC audited down-hole deviation data from three Magma holes and four Cyprus-Superior CS-series holes. No errors were found in the Magma deviations in the project data, which were audited using scans of original paper records from Eastman Whipstock, Inc. The depths of two out of the four CS-holes audited have discrepancies in the depths of the down-hole readings, whereby the down-hole back-up data have readings at depths of 200, 300, and 400 feet, for example, while the database has these same readings at depths of 300, 400, and 500 feet. Excelsior examined all of the data for these two holes and found that the information used by RESPEC in the audit is actually derived from averaged values of multiple readings over 100-foot intervals. The data used by RESPEC in the audit represent the "from" depth of each averaged interval, while the database has the same data at the "to" depth. Excelsior is investigating the deviation data for all CS holes in detail and will make corrections if warranted. However, all of the CS-series holes are vertical, and the dip changes for each 100-foot data point are usually small (the average dip change for each 100-foot interval in the four holes audited is less than 0.4 degrees), so any changes are very unlikely to materially affect the modeling of the project resources.

12.2.3 Assay Table

A total of 6,427 sample intervals were analyzed by Skyline for Excelsior, including intervals from Excelsior drillholes, as well as intervals from historical (pre-Excelsior) core holes and re-analyses of historical sample pulps. RESPEC obtained and compiled Excelsior's digital analytical data directly from Skyline and used a computer script to complete an automated audit of the database values. A total of 5,141 TCu values and 6,413 ASCu and CNCu (sequential leach) values were audited using the automated routine. A small number of discrepancies between the Skyline and database values were identified, all but one of which were found to be re-analyses in the database due to quality assurance/quality control issues. No errors were found in the ASCu and CNCu data.

RESPEC used historical paper records to audit the database values of five CS-series and two J-series holes drilled by the Cyprus-Superior joint venture. Out of the total of 1,858 CS-series sample intervals in the database that have historical analyses, 656 TCu and 650 ASCu values were audited using scanned copies of original American Analytical assay certificates. Five discrepancies were found in the TCu data (<1% of the audited data), only one of which was significant. Six discrepancies in the ASCu data were also identified (<1% of the audited data), with two of them being significant. One of the significant errors in the ASCu data is from the same sample interval as the single significant TCu error; these are the result of incorrect repeating of the analyses from the previous sample interval in the hole. Excelsior found that the other discrepancies are due to the derivation of the database values from handwritten geologic logs, as opposed to the copies of the original assay certificates used by RESPEC in the auditing; Excelsior corrected their database to match the values on the certificates.

In the J-series holes, 173 TCu values and 103 ASCu values in the project database were checked against typed Cyprus Mines assay sheets; there are 425 J-series sample intervals in the project database. No discrepancies were identified.

12.3 QUALITY ASSURANCE/QUALITY CONTROL PROGRAMS

12.3.1 Historical QA/QC

QA/QC data are not available for any of the historical drilling programs, if any ever existed. Excelsior has attempted to validate, and has partially replaced, the historical assay data through a resampling and re-assaying program.

12.3.2 Excelsior QA/QC

The QA/QC program instituted by Excelsior for the North Star 2011 to 2015 drilling programs included the systematic analyses of certified analytical standards, coarse blanks, and field duplicates. Skyline performed copper analyses on all of Excelsior's original drill samples and their related QA/QC samples. The QA/QC program was designed to ensure that at least one standard, blank, or field duplicate was inserted into the drill-sample stream for every 10 drill samples. The 2011 and 2012 drill programs also employed check assaying by ALS. All holes drilled by Excelsior at the North Star deposit have been subject to this QA/QC program.

12.3.3 Certified Standards

Certified standards were used to evaluate the analytical accuracy and precision of the Skyline analyses during the time the drill samples were analyzed. Two certified standards were purchased from African Mineral Standards ("AMIS"), located in Eastern Johannesburg, South Africa. These standards were chosen by Excelsior because they are derived from oxidized copper deposits. The certified values and standard deviations for these standards are listed in Table 12-2.

Table 12-2: Excelsior Certified Standards

Standard ID	Standard Source	Certified Value (TCu%)	Standard Deviation (%)	Standards Analyzed
AMIS0118	AMIS	0.4615	0.0135	419
AMIS0249	AMIS	0.3692	0.0072	42

Prior to each drilling campaign, Excelsior attempted to obtain certified standards for ASCu but could not locate any.

Of the standards listed in Table 12-2, only the 301 standards submitted with Excelsior NSD-series core holes were evaluated by RESPEC, all of which were the AMIS0118 standard. Standards submitted with samples from the NSH-series rotary holes were not reviewed; these drill data were not used in the resource estimation.

Excelsior assigned sample numbers for the standards in sequence with their accompanying drill samples, and the standards were inserted into the drill-sample stream submitted for analysis.

Figure 12-1 charts the Skyline analyses of standard AMIS0118.

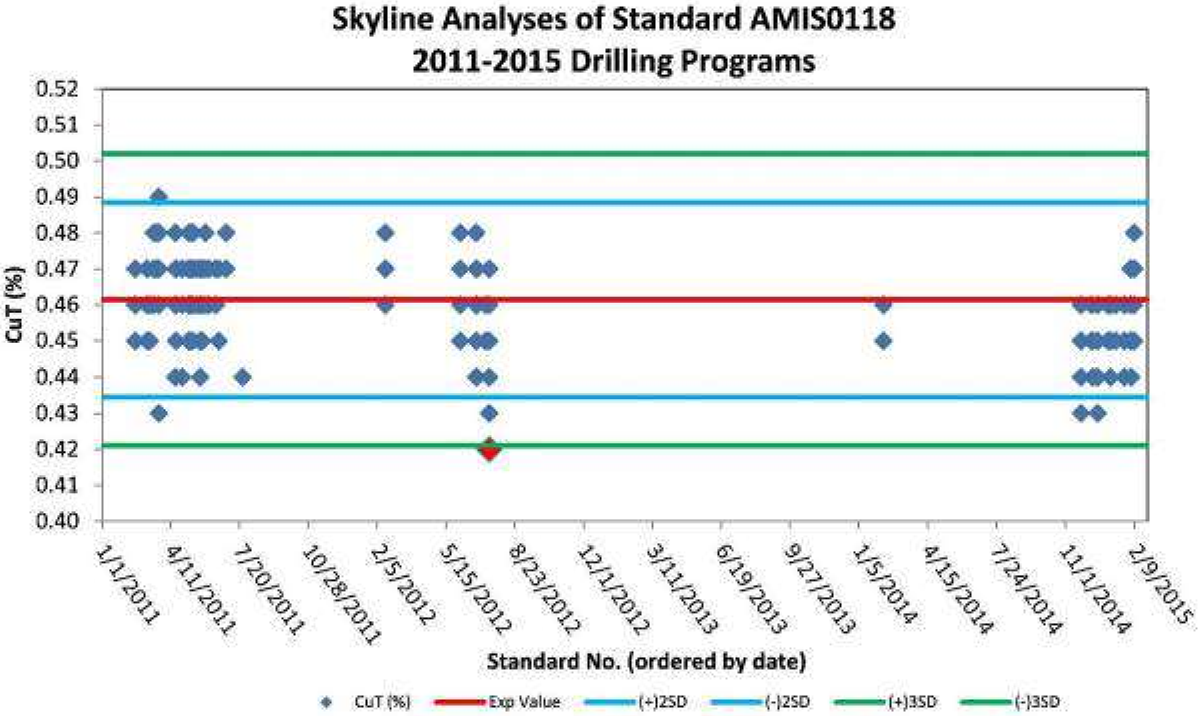


Figure 12-1: Plot of Certified Standard AMIS0118 Analysis

In the case of normally distributed data, 95% of the standard analyses are expected to lie within the two standard-deviation limits (shown as blue lines) of the certified value (shown as the red line), while only 0.3% of the analyses are expected to lie outside of the three standard-deviation limits (green lines). Samples outside of the three standard-deviation limits are therefore considered to be failures. As it is statistically unlikely that two consecutive analyses of standards would lie between the two and three standard-deviation limits, such samples could be considered failures, unless further investigation proves otherwise.

Only one sample of the 301 assays evaluated lies outside the three standard-deviation limits, and therefore could be considered as a failure (shown as a red diamond in Figure 12-1). However, the failure exceeds the limit by only 0.001% Cu. If the certified standard values and standard deviations were rounded to two decimal places, as the Skyline assays are, instead of three, this standard analysis would not be considered a failure.

There is one case of consecutive analyses that lie between the two and three standard-deviation limits, and these two analyses were performed in the same laboratory batch. However, one of the standards lies above the two standard-deviation limits while the other is below, an instance worth investigating further but that does not qualify as a 'failure'.

Table 12-3 compares the mean of Skylines analyses of the standard against its certified value.

Table 12-3: Skyline Analyses of Standard AMIS0118

Drill Program	Standard Analyses Mean	%Diff	Count
2011	0.47	1.00%	178
2012	0.46	-0.50%	43
2014 - 2015	0.45	-1.80%	80
All	0.46	0.10%	301

The data reviewed indicate no bias in the Skyline analyses of the standards inserted with the 2011 and 2012 drill samples, with a slight low bias of about 2% in Skyline's analyses of the standards associated with the 2014-2015 drill samples.

12.3.4 Coarse Blanks

Coarse blanks are samples of barren material that are used to detect possible laboratory contamination, which is most common during sample-preparation stages. Therefore, in order for analyses of blanks to be meaningful, they must be sufficiently coarse to require the same crushing and pulverizing stages as the drill samples. It is also important for blanks to be placed in the sample stream within a series of mineralized samples, which would be the source of most contamination issues.

Blank results that are greater than five times the lower detection limit of the analysis are typically considered failures that require further investigation and possible re-assay of associated drill samples (0.05% and 0.005% Cu for the Excelsior copper analyses, based on the 0.01% and 0.001% Cu detection limits, respectively).

Excelsior used landscape river rock purchased from a local home-improvement store as coarse blank material. These blanks were coarse enough to require the same primary and secondary crushing applied to the drill samples.

A total of 236 coarse-blank analyses were analyzed from the 2011 through 2015 drill programs. Of these, 47 were associated with drillholes not used in resource estimation (NSH-series holes), leaving a total of 189 blanks with TCu, ASCu, and/or CNCu analyses that were evaluated by RESPEC. Of these, 126 blanks were preceded by mineralized (above background) drill samples.

There were no failures in the TCu analyses of the blanks and no systematic contamination issues were found in the blank analyses. While two 0.007% ASCu analyses of blanks slightly exceeded the threshold limit of 0.005%, these clearly are not material to the resource modeling discussed in Section 14.

12.3.5 Field Duplicates

Field duplicates are secondary splits of drill samples. Field duplicates are mainly used to assess inherent geologic variability and subsampling variance. The field duplicate samples were submitted to Skyline with, and immediately following, their associated original drill samples. Only drillholes used in the resource estimate, all of them core holes, are considered in this discussion. Duplicate samples produced by other drilling methods, such as the NSH-series holes which employed conventional-rotary drilling, were not evaluated.

In the case of Excelsior's core drilling, field duplicates consisted of quarter core splits, with the paired originals being half core splits; (quarter)¼-core was left in the core library. The field duplicates were collected at regular intervals which resulted in a large percentage of duplicates being derived from original samples with values at or below the analytical detection limit.

A total of 107 core duplicates were collected by Excelsior and analyzed by Skyline. The core-duplicate data for TCu are presented in Figure 12-2; 17 pairs in which both the duplicate and original analyses are below the detection limit were removed from the dataset.

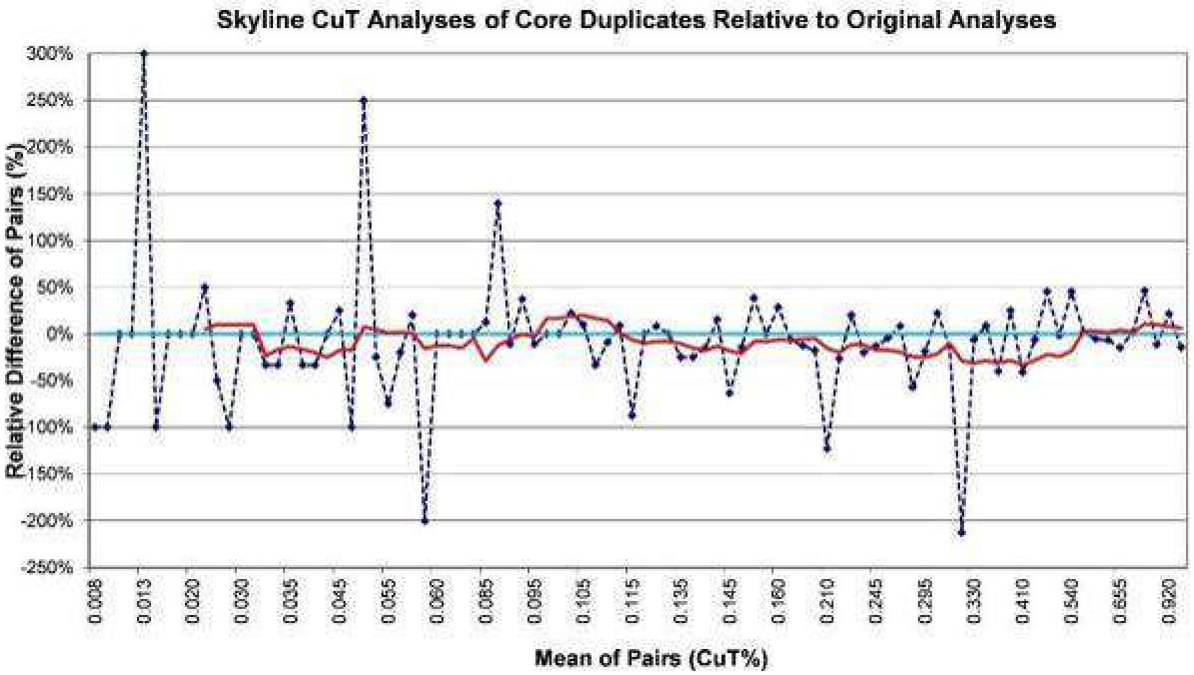


Figure 12-2: Core – Duplicate TCu Analyses Relative to Original Assays

Figure 12-2 is a relative-difference graph, which shows the percentage difference (plotted on the y-axis) of each duplicate assay relative to its paired original analysis. The x-axis of the graph plots the means of the TCu values of the paired data in a sequential, non-linear fashion. The red line is the moving average of the relative differences of the pairs and provides a visual guide to trends in the data. Positive relative-difference values indicate that the duplicate analysis is greater than the original. Relative-difference graphs are very useful in determining biases in the data that may not be evident using basic descriptive statistics.

The TCu mean of the core-duplicate analyses is 4% lower than the original samples, but this difference decreases to 2% if the single duplicate pair at a mean of the pair ("MOP") of 0.330% is removed from the dataset. While there is an indication of a low bias in the core duplicates relative to the originals in the MOP range of ~0.15 to ~0.4% TCu, there are insufficient data to make statistically meaningful conclusions. The average of the absolute values of the relative differences is 25% at a MOP cut-off of 0.1% TCu, indicating a moderate amount of variability between the original and core-duplicate assays.

Figure 12-3 shows the 80 core-sample field-duplicate pairs for ASCu in which both the originals and duplicates were above the lower detection limit; 27 pairs with both below the detection limit were excluded. The mean ASCu grade of the core duplicates is 3% lower than the mean of the original analyses, although the means are identical if five pairs with relative differences exceeding ±150% are removed. The average of the absolute values of the relative differences is 34% at a MOP cut-off of 0.1%, lowering to 23% if the five high relative-difference pairs are removed. As with the TCu data, there is a suggestion of a low bias in the ASCu analyses of the core duplicates in the MOP range of ~0.08 to 0.2%, but no statistically valid conclusions can be drawn due to insufficient data in this grade range.

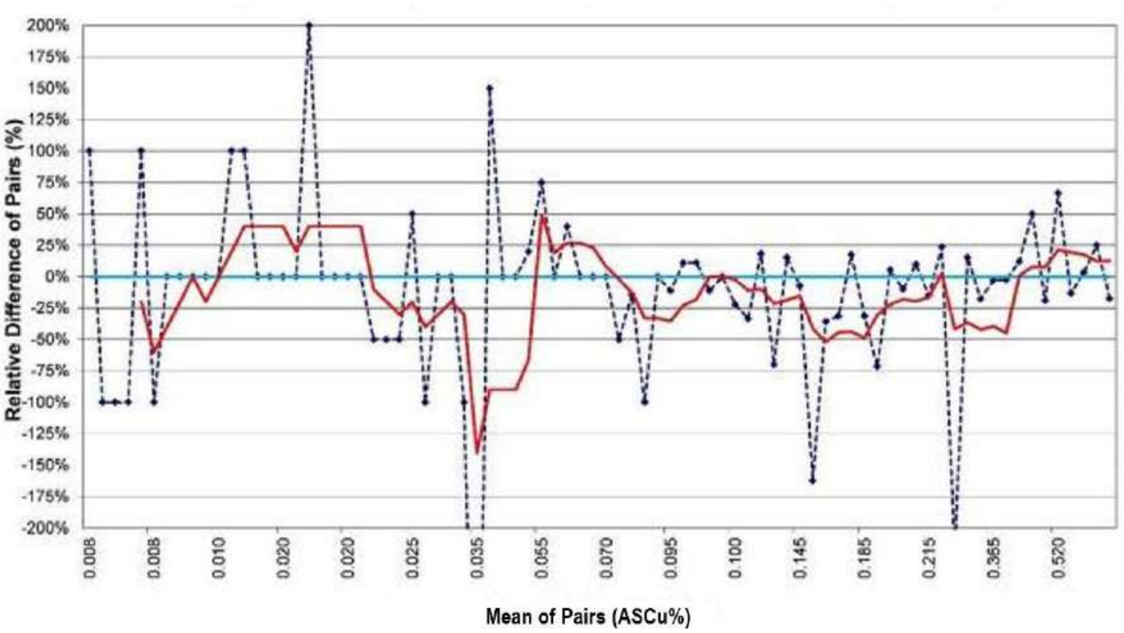


Figure 12-3: Core – Duplicate Analyses Relative to Original ASCu Assays

The mean of the ASCu/TCu ratios derived from the core duplicates is identical to that of the original analyses for all pairs where both TCu analyses exceed 0.03% (low TCu values can lead to meaningless ratios).

While these statistical analyses of the core duplicate TCu and ASCu show no statistically significant issues, more data are needed to properly evaluate Excelsior's subsampling of the core.

12.3.6 Replicate Analyses

Replicate analyses are secondary splits of the original sample pulps that are analyzed by the original laboratory in the same assay batch as the original analysis. These are mainly used to assess variability instilled by the subsampling of the pulp and the analysis itself.

The replicate analyses were analyzed regularly by Skyline as part of its internal QA/QC program. Only the 814 replicates of samples derived from Excelsior's NSD core holes are evaluated in this discussion.

The TCu replicate data are presented in Figure 12-4; 138 pairs in which both the duplicate and original analyses are below the detection limit were removed from the dataset.

No bias is evident in the replicate data, and the means of the replicates and the originals are identical at a range of MOP cut-offs. Removal of extreme relative-difference pairs does not affect the means of the datasets because they occur on both sides of the 0% line. Variability of the replicate data is low, as measured by the average of the absolute values of the relative differences, which is 4% for all of the data and 2% at a MOP cut-off of 0.1%.

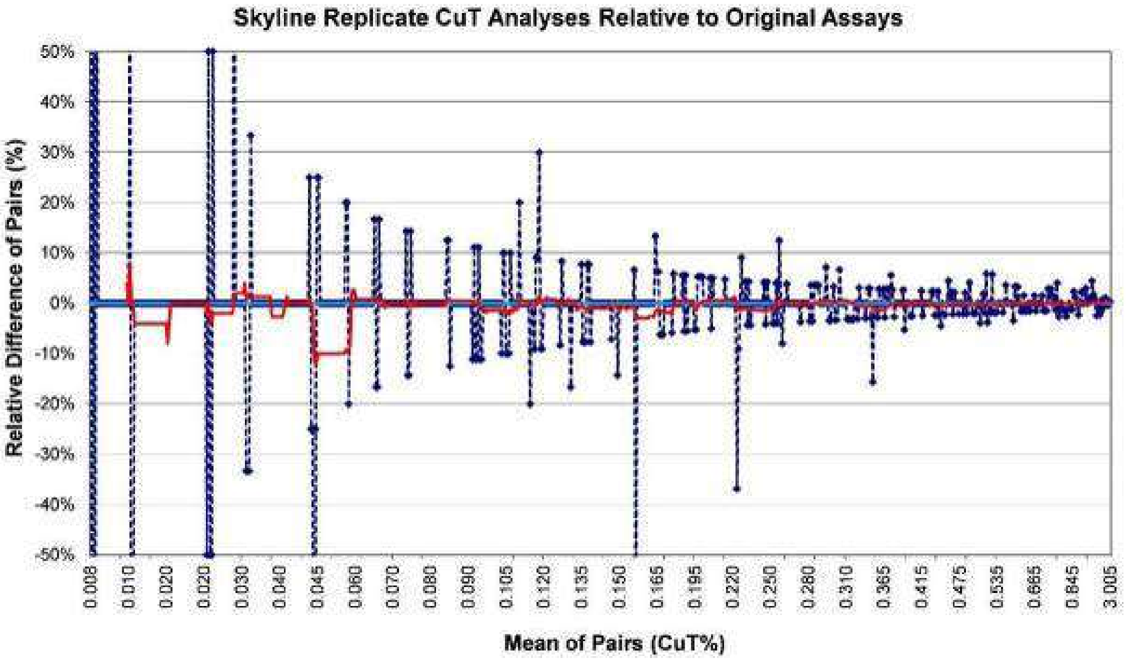


Figure 12-4: Replicate TCu Analyses Relative to Original Assays

Figure 12-5 shows the 975 of 1,289 total replicate-original pairs for ASCu in which both the originals and duplicates were above the lower detection limit; 314 pairs in which both analyses are below the detection limit were excluded. The relative differences displayed at the lowest-grade portion of the ASCu chart are an artifact of variable detection limits that cause extreme, but artificial, variability.

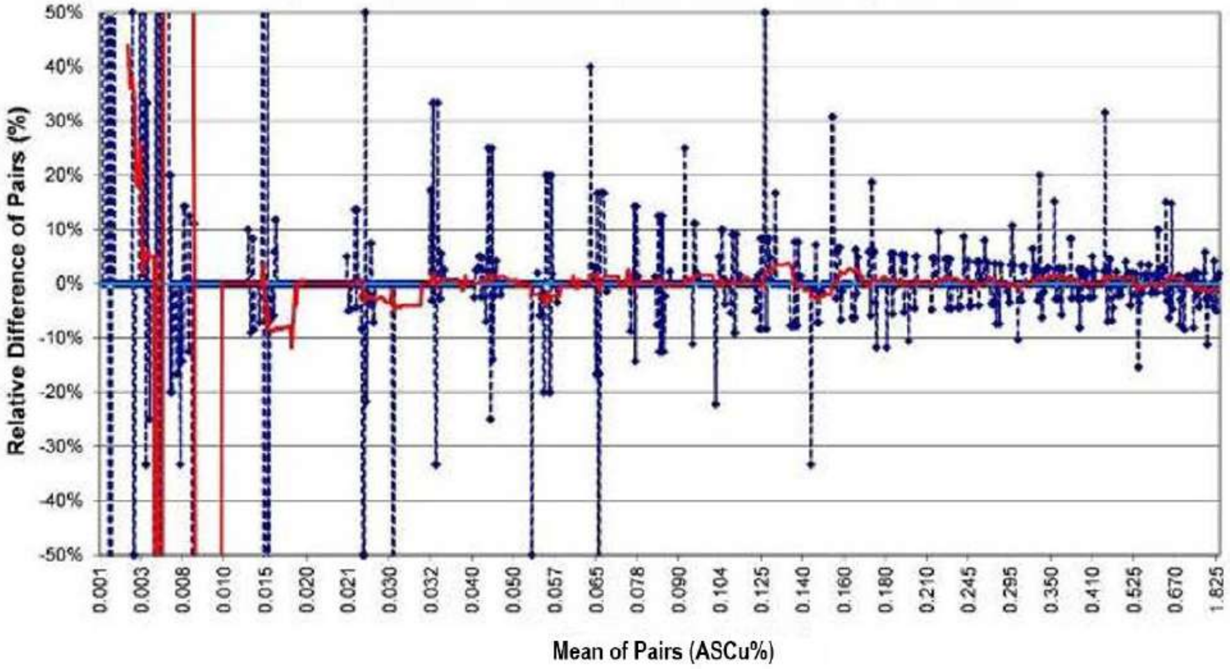


Figure 12-5: Replicate ASCu Analyses Relative to Original Assays

As with TCu, no bias is evident in the ASCu replicate data, and the means of the replicate and original analyses are identical. The average of the absolute values of the relative differences is ~3% at a MOP cut-off of 0.1% ASCu, which indicates slightly higher variability than for TCu.

The mean of the ASCu/TCu ratios derived from the replicate analyses (0.52) is slightly lower than that of the original analyses (0.53) in pairs where both TCu analyses exceed 0.03%.

12.3.7 Check Assays

As a further check on analytical accuracy, Excelsior selected a portion of the original sample pulps from each yearly drill program and sent these to ALS for re-assaying of the original Skyline pulps. Roughly every 20th sample, or approximately 5% of the total sample data, was selected for re-assay. A total of 220 pulps from the 2011, 2012 and 2014/2015 programs were sent to ALS for check assaying.

Figure 12-6 compares the ALS check TCu assays to the original Skyline assays from NSD-series core holes from the 2011 and 2012 drilling programs, the ALS check-assay results from which are very consistent. Eight data pairs in which both the check and original analyses are less than the detection limit are removed. Their graph shows a very consistent low bias of about 5% in the ALS check assays relative to the Skyline original analyses at MOP greater than about 0.07% TCu. The variability of the duplicate TCu analyses above this MOP is ~7%.

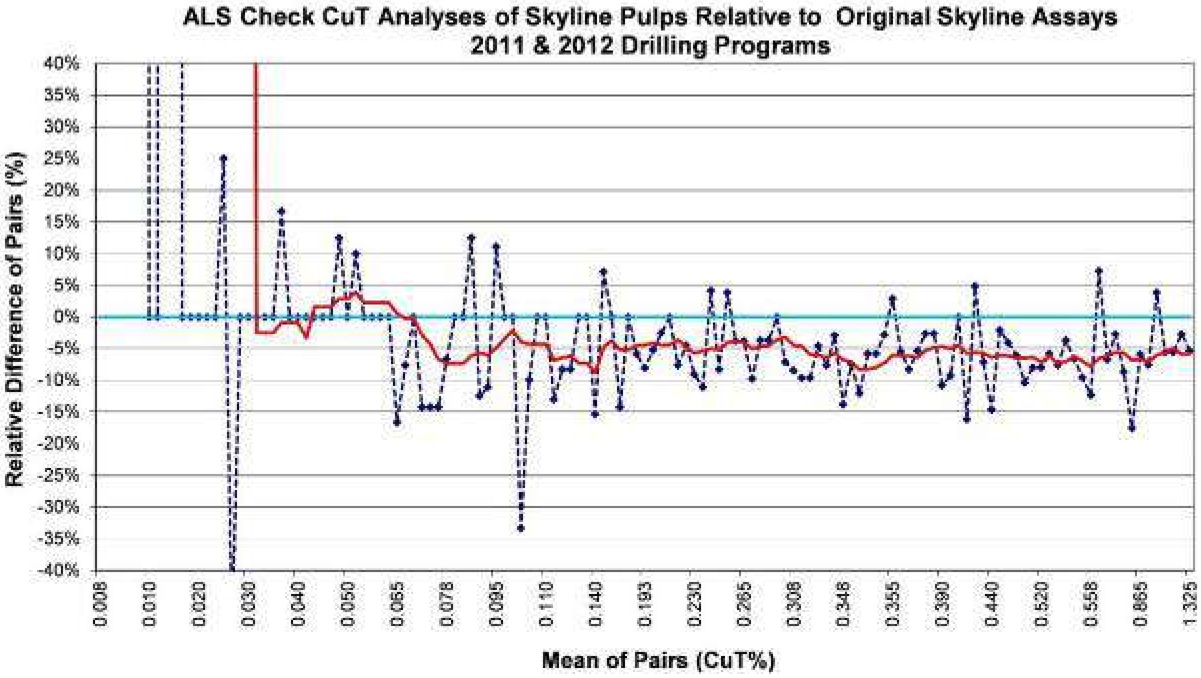


Figure 12-6: ALS Check TCu Assays Relative to Original Skyline Analyses

The check-assay data for ASCu for the 2011 and 2012 drilling programs are very similar to the TCu data, with a consistent low bias in the ALS analyses in this case of about 8% to 10% relative to the original Skyline assays. The variability of the ASCu duplicate pairs is ~10%.

The check-assay data for the 2014-2015 drilling program yield different results. While there are fewer duplicate pairs, with only 29 pairs above a MOP cut-off of 0.1% TCu, the ALS check TCu assays of the samples from the 2014-2015 program are higher than the original assays up to a MOP grade of ~0.3% TCu, although the extent of the high bias continually decreases over this grade range; in the range MOP range of 0.1 to 0.3% TCu, the ALS analyses are ~5%

higher than the original assays. It is worth noting that the Skyline standard analyses for the 2014-2015 drilling program are also biased low. The limited data above the 0.3% TCu cut-off shows reasonably close agreement between the check and original analyses. The variability in the paired data is ~5% above a MOP cut-off of 0.1% TCu.

The ASCu paired data is again similar to the TCu data for the 2014-2015 drilling program, with a high bias in the check assays of about 10% up to a MOP of ~0.07% ASCu. At higher grades, the check analyses are close to the original analyses. Variability is ~4% above a MOP cut-off of 0.05% ASCu.

Excelsior included standard pulps with the submissions of Skyline drill-sample pulps to ALS for check assaying at the end of each drill program. ALS analyzed a total of 28 AMIS0118 standard pulps in the check assaying of pulps from the 2011 and 2012 drilling programs and three AMIS0249 standards with the 2014-2015 sample pulps. Figure 12-7 charts the results of the ALS analyses of standard AMIS0118 from the 2011 and 2012 drilling programs, and Table 12-4 summarizes the results for all ALS analyses of the standards.

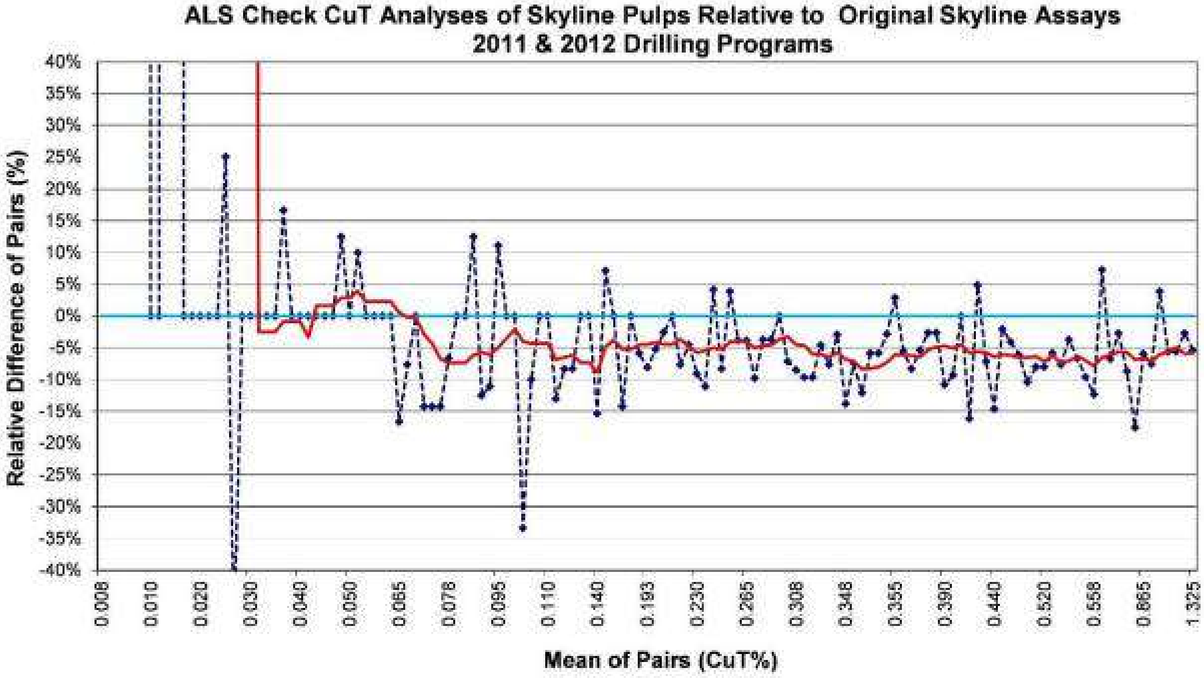


Figure 12-7: Plot of ALS Check Assay Analyses of Standard AMIS0118

Table 12-4: Summary of ALS Analyses of Standards from Check – Assaying Programs

Drill Program	ALS Mean	Certified Value	%Diff	Count
2011 & 2012	0.44	0.4615	-4.70%	28
2014 & 2015	0.36	0.3692	-3.00%	3

The 2011 and 2012 ALS TCu analyses of AMIS0118 are systematically biased low at a level consistent with the low bias in the ALS analyses of 2011 and 2012 Skyline drill-sample pulps relative to the original Skyline analyses. While the 2014-2015 ALS analyses of the three AMIS0249 are slightly low, there are insufficient standard analyses to determine a definitive bias. Excelsior does not have ASCu standards.

In the evaluation of the TCu and ASCu check-assay data, it is important to note that the analytical procedures employed by ALS differed significantly from those used by Skyline. Skyline analyzed TCu by atomic absorption following multiacid digestion. A sequential-leach procedure was performed separately for the ASCu and CNCu analyses. In the case of ALS, ASCu and CNCu were also obtained from sequential-leach analyses. A third analysis was then run on the residua of the sequential-leach analyses. TCu is indirectly determined by adding the three values (ASCu + CNCu + residual Cu). This means that an error in any of the three analyses will similarly affect the calculated TCu value as well. It is unfortunate that ALS did not complete TCu analyses directly on the sample pulps in addition to the sequential-leach analyses.

12.3.8 Excelsior Inter – Laboratory Check Program

In light of the discrepancies between the original Skyline and ALS check assays, “M3, 2014” recommended additional inter-laboratory check-assaying programs. Following these recommendations, Excelsior selected 30 coarse rejects from the original Skyline drill samples at the end of each of the 2011, 2012, and 2014-2015 drill programs and sent them to ALS. ALS prepared and analyzed the 30 pulps at the end of each program and then sent the pulps to Skyline for check assaying.

RESPEC completed a detailed analysis of this inter-laboratory program using techniques described above for the other duplicate datasets. The data were examined as a whole as well as by drill program. Consistent TCu and ASCu biases were found between the ALS analyses and both the Skyline check assays of the ALS pulps and the original Skyline analyses of the drill core for all three drill programs. In all cases, the Skyline TCu and ASCu analyses are biased high relative to those of ALS. These biases are consistent with those identified in the original check-assaying of the 2011 and 2012 drilling programs discussed above but are not consistent with the check-assay data from the 2014-2015 program.

Table 12-5 compares the means of the original Skyline analyses of core, the ALS analyses of the Skyline coarse rejects, and the Skyline check analyses of the ALS pulps. One of the 90 samples was removed from due to an extreme outlier in a 2011 ASCu analysis. Based on detailed reviews, as well as the exclusion of this single outlier sample, RESPEC believes the means shown in Table 12-5 provide a reasonable summary of the results of the inter-laboratory check program.

Table 12-5: Summary of the Inter – Laboratory Check Program

	2011			2012			2014-2015			All Data					
	Skyline		ALS	Skyline		ALS	Skyline		ALS	Skyline		ALS	Skyline	ALS vs Skyline	
	Core	ALS Chk	Cse Rej	Core	ALS Chk	Cse Rej	Core	ALS Chk	Cse Rej	Core	ALS Chk	Cse Rej	Chk vs Core	vs. Core	vs Chk
TCu	0.52	0.53	0.5	0.79	0.79	0.77	0.35	0.35	0.34	0.55	0.56	0.54	0.40%	-3.20%	-3.60%
ASCu	0.4	0.42	0.39	0.66	0.64	0.63	0.21	0.23	0.2	0.43	0.43	0.41	0.90%	-4.50%	-5.30%
ASCu/TCu	0.76	0.79	0.77	0.84	0.8	0.8	0.6	0.65	0.59	0.73	0.75	0.72	2.30%	-1.20%	-3.50%
CNCu	0.007	0.008	0.009	0.026	0.037	0.017	0.048	0.04	0.032	0.27	0.022	0.022	7.40%	18.50%	24.10%

Note: one 2011 sample removed due to spurious ASCu analysis

In contrast to the Skyline – ALS biases, comparisons between the original Skyline analyses and the Skyline check analyses of the ALS pulps (which are derived from Skyline coarse rejects) show no biases. The close correspondence between the two sets of Skyline analyses suggests that laboratory sample preparation is not the cause of the Skyline – ALS biases. This leads to the conclusion that the biases are probably rooted in either the subsampling of pulps to obtain aliquots for analysis or in the analyses themselves; RESPEC believes the former explanation is very unlikely.

Excelsior inserted standards with the coarse rejects analyzed by ALS and the ALS pulps analyzed by Skyline (Table 12-6). All of the ALS and Skyline analyses of these standards yielded values within two standard-deviations of the certified standard grades. While the data are not sufficient to derive definitive conclusions, the Skyline analyses of the standards tend to be higher than the certified values in the 2011 and 2014-2015 data, while the ALS analyses are generally lower. Note that Skyline's much more numerous analyses of the same standard inserted with the original drill samples show no high bias whatsoever.

Table 12-6: Skyline and ALS TCu Analyses of Standards – Inter – Laboratory Program

Drilling Program	Standard		ALS		Skyline	
	Certified Value	Std Dev	Analysis	%Diff	Analysis	%Diff
2011	0.4615	0.0135	0.45	-2.50%	0.48	4.00%
			0.49	6.50%	0.48	4.00%
			0.45	-2.50%	0.47	1.80%
2012	0.4615	0.0135	0.47	1.80%	0.46	-0.30%
			0.46	-0.30%	0.46	-0.30%
			0.45	-2.5	0.45	-2.50%
2014-2015	0.3692	0.0072	0.365	-1.10%	0.38	2.90%
			0.355	-3.8	0.38	2.90%
			0.355	-3.80%	0.38	2.90%

12.3.9 Summary of Excelsior QA/QC Results

No significant issues were identified in the results of Skyline's TCu and ASCu analyses of the certified standards, coarse blanks, and replicates. While the TCu and ASCu analyses of the core duplicates are slightly lower than the original analyses over certain TCu and ASCu grade ranges, there are insufficient data at these grades to allow for definitive conclusions.

The check-assay data indicate that Skyline TCu and ASCu analyses are systematically higher (~5% for TCu and ~8% for ASCu) than ALS at relevant grades for the two copper species. ALS analyses of standards inserted with the drill-sample pulps for check assaying are systematically ~5% lower than the certified values, however, while Skyline analyses of the same standards submitted with the original drill samples show no biases or other issues. The inter-laboratory program undertaken to further examine the ALS versus Skyline discrepancies accomplished little more than largely confirming the biases identified in the check-assaying program. Based on all of these data taken as a whole, as well as the differences between the analytical methods employed by Skyline and ALS, RESPEC concludes that there are no significant issues with the Skyline TCu analyses of the original Excelsior drill samples, although there may be a slight low bias in the 2014-2015 data.

The accuracy of the ASCu analyses in the project database cannot be directly assessed. An ASCu analysis only measures a portion of a sample's copper content, and this portion will vary laboratory to laboratory based on the specifics of the analytical methodologies. Key variables include the leaching time, the temperature of the leach solution, the strength of the leach solution, and the degree of agitation. In other words, there is no 'correct' value for ASCu in any particular sample. What is important to any particular project, however, is the consistency in the ASCu analyses, which in the case of North Star can be evaluated by examining the ASCu/TCu ratios of the core duplicates and the replicate analyses. In both cases, the differences between the duplicate and original ratios are very close (less than one percent). RESPEC finds no issues with the ASCu analyses in the project database.

The core-duplicate data are useful in estimating variability in the copper analyses that is attributable to geological heterogeneity, subsampling by Excelsior and the laboratory, and analytical precision. At a cut-off of about 0.1% for both TCu and ASCu, the variability in the core duplicates is about 20%. Since the core duplicates are comprised of ¼-core samples, and the original drill samples are ½-core samples, this variability probably overstates the variability inherent in the original ½-core samples. The data therefore suggest that the total uncertainty in any single TCu or ASCu analysis in the existing North Star data is less than ± 20%. Approximately 3% of this total is attributable to analytical precision, as evidenced by the replicate data.

12.3.10 QA/QC Recommendations

RESPEC recommends that Excelsior consider the following changes to their QAQC protocols:

- The addition of two certified TCu standards, one at a grade lower than the standard presently in use and the other at a higher grade;
- The addition of preparation duplicates to the QA/QC protocols. Preparation duplicates are analyses by the primary assay laboratory of second pulps prepared from the original coarse rejects. These duplicates monitor the subsampling undertaken by the primary lab;
- The TCu and ASCu analytical procedures used by the check-assay laboratory should be identical to those used by Excelsior's primary lab; and
- The use of the present inter-laboratory check program should be terminated. In the event that discrepancies between check assays and the original analyses cannot be resolved by the laboratories' analyses of certified standards, the check-assay pulps should be sent to a third 'umpire' lab along with the same standards analyzed by the primary and check-assay labs.

12.4 EXCELSIOR RESAMPLING AND RE-ASSAYING OF HISTORICAL CORE AND SAMPLE PULPS

12.4.1 Resampling of Cyprus – Superior Drill Core

Core Duplicates: Excelsior resampled selected intervals of Cyprus-Superior core from holes CS-02 and CS-06 and sent the 40 core-duplicates to Skyline for preparation and analysis. "M3, 2014" state that the mean of the Skyline TCu analyses is 12% lower than the mean of the original analyses (0.37 *vs*. 0.42%, respectively), and the mean of the Skyline ASCu analyses is 8% lower (0.21 versus 0.23%). "M3, 2014" concluded that "*these results indicate that* [the original analyses] *may be biased high relative to Skyline for* [TCu and ASCu]*, but the number of pairs is too small and the scatter of points too large to confirm that a systematic bias is present.*" RESPEC independently analyzed the data and agrees with this conclusion. While the potential bias is more evident in the TCu data than ASCu, the mean of the ASCu/TCu ratios of the original and core-duplicate datasets are very close, which suggests any bias in the Skyline TCu data is mirrored in their ASCu analyses. Approximately 25% of the data directly used in the estimation of TCu and ASCu resource grades are derived from the original Cyprus-Superior analyses.

12.4.2 Pulp – Check Analyses and Resampling of Quintana Drill Core

Core Duplicates: The core from 101 sample intervals in holes T-01 and T-05 was resampled by Excelsior and analyzed for TCu by Skyline). A systematic low bias in the Skyline analyses is evident at mean grades of the pairs greater than ~0.08% TCu, and the mean of the Skyline analyses is 10% lower than the mean of the original assays (Figure 12-8).

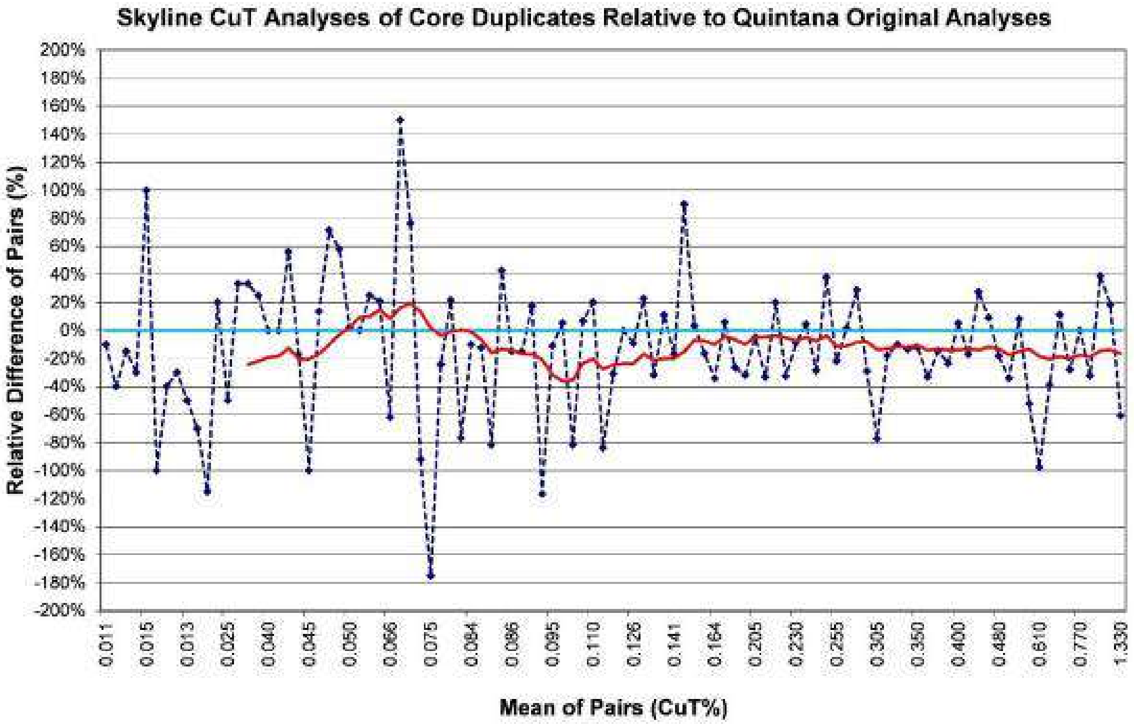


Figure 12-8: Skyline TCu Analyses of Core Duplicates Relative to Original Quintana Assays

Skyline also completed ASCu analyses on 274 core duplicates from seven T-series holes and holes S-3 and DC-09 (Figure 12-9).

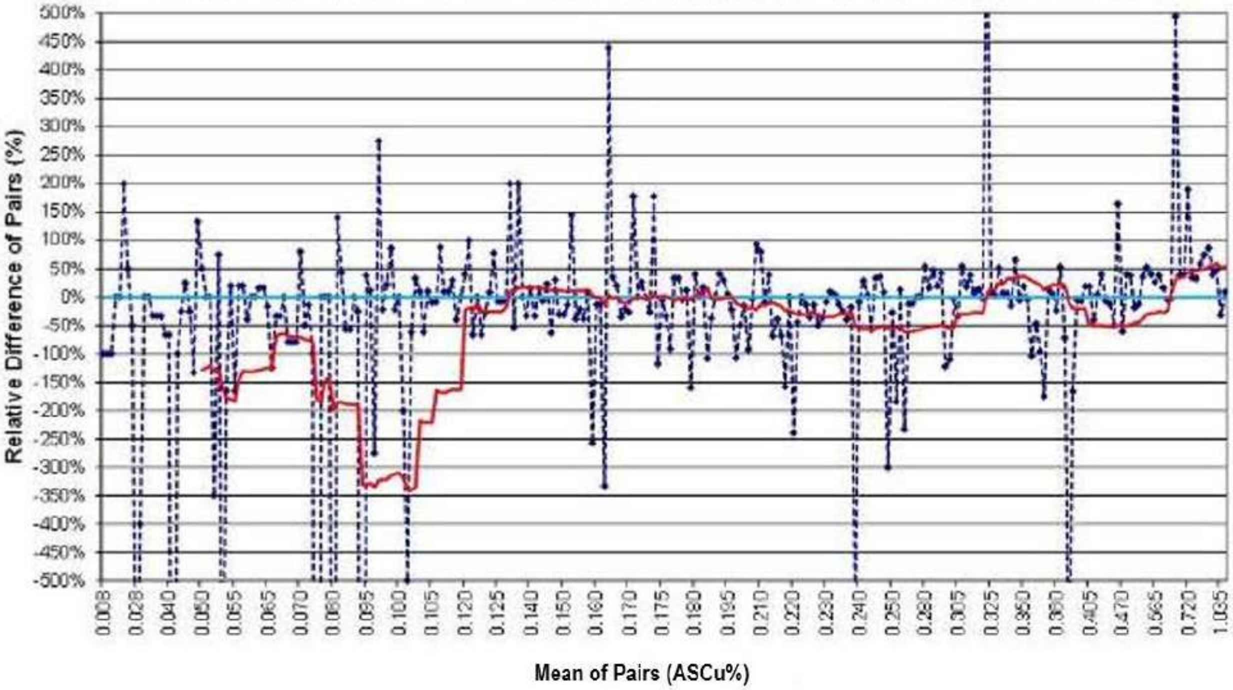


Figure 12-9: Skyline ASCu Analyses of Core Duplicates Relative to Original Quintana Assays

The mean of the Skyline data is close to the mean of the original analyses (1% higher). However, the Skyline mean includes an anomalous number of instances in which the Skyline analyses are significantly lower than the originals (as seen in pairs with relative differences >~-150 to -200%). These pairs, in part, lead to an apparent low bias in the Skyline analyses for pairs with means up to about 0.1% ASCu, as well as in the range of ~0.2 to 0.3% ASCu.

RESPEC also investigated the core-duplicate data for the 58 sample intervals within the dataset in Figure 12-9 for which paired ASCu analyses are available. These pairs also show a low bias in the Skyline analyses within a similar range of the MOP of ~0.15 to ~0.3% ASCu, although there are not enough data to make definitive conclusions. The mean of the Skyline ASCu/TCu ratios is identical to the mean of the ratios of the original analyses.

Re-Assays of Original Pulps: Skyline completed ASCu analyses on original Quintana sample pulps from seven T-series holes (S-01, S-04, and DC-09). A total of 331 of these pairs are compared in Figure 12-10.

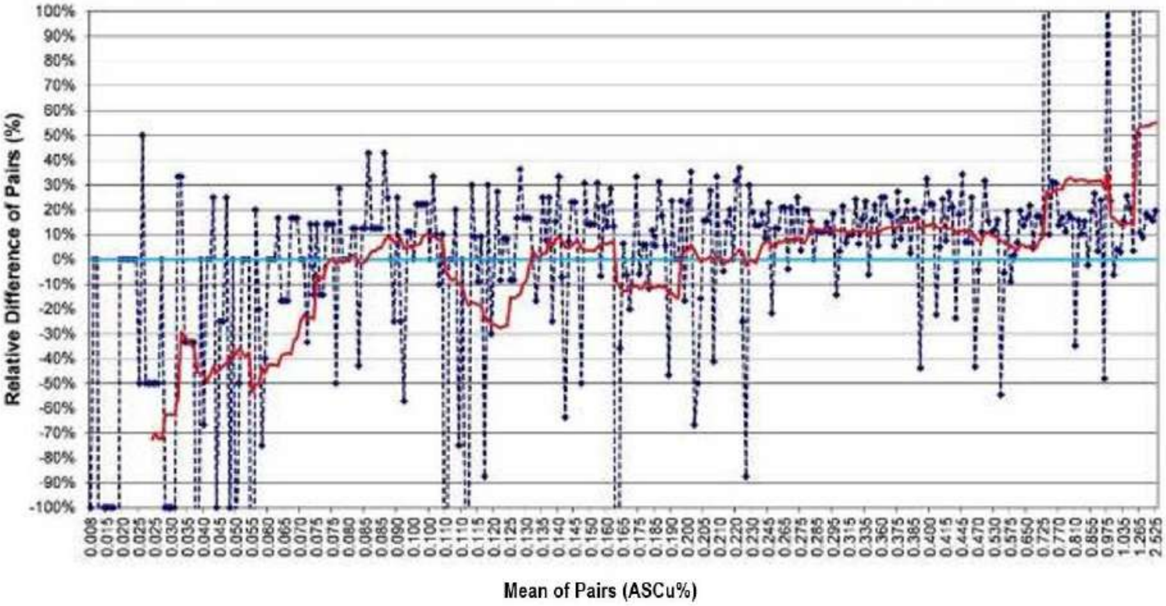


Figure 12-10: Skyline ASCu Analyses of Pulp Relative to Original Quintana Assays

The mean of the Skyline analyses of the original pulps is 12% higher than the original assays (0.30 *vs*. 0.27% ASCu, respectively); removal of all pairs with relative differences >100% decreases the difference to 9%. A strong and systematic high bias in the Skyline analyses is seen at MOP's greater than about 0.25% ASCu.

There is a distinct bias in the pairs with relative differences in excess of about 40%, and relative differences of this magnitude are high for check analyses of pulps. Instances in which the Skyline analyses are significantly lower than the originals dominate these pairs. If not for these high relative-difference pairs, the high bias in the Skyline analyses would be exacerbated, and it would extend the bias to MOP's greater than ~0.12% ASCu. There are no Skyline TCu analyses that accompany this ASCu dataset.

Quintana TCu and ASCu analyses in the project database represent 5.5% and 2.6%, respectively, of the data directly used in the estimation of the project resources.

12.4.3 Resampling of Magma Copper Drill Core

Core Duplicates: Excelsior resampled historical Magma drill core and sent the 519 core-duplicate samples to Skyline for preparation and analysis of both TCu and ASCu. Skyline's core-duplicate results differ significantly from the original

Magma analyses, which led Excelsior to completely replace the Magma analytical data with analyses of resampled core. The Skyline TCu analyses of the core duplicates are compared to the original analyses in Figure 12-10. While the ASCu comparison is similar that shown in Figure 12-11, the magnitude to the differences in the two datasets is less. This leads to the Skyline mean of the ASCu/TCu ratios (0.74) in the duplicate analyses being significantly higher than the mean of the ratios of the original analyses (0.62).

Note that the pairs with extreme relative differences are highly biased towards those in which the Skyline analyses are lower than the originals.

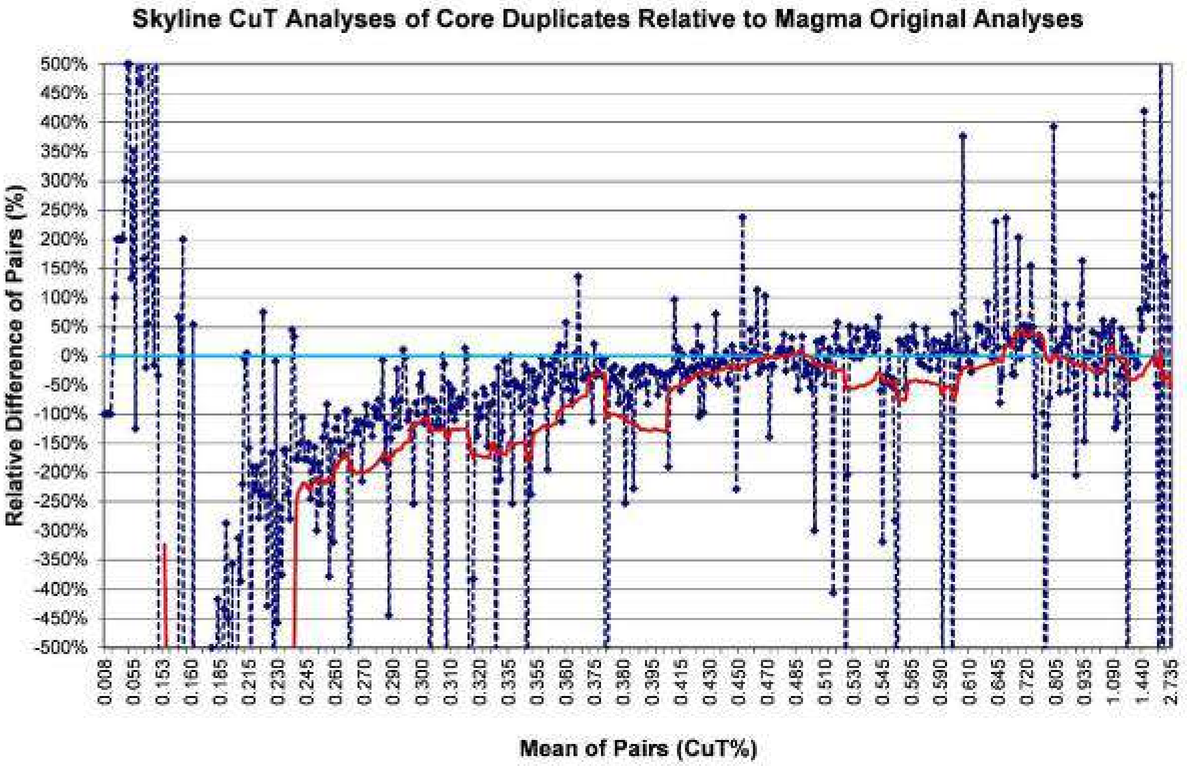


Figure 12-11: Skyline TCu Analyses of Core Duplicates Relative to Original Magma Assays

Excelsior completely replaced all original Magma assays in the project database with Skyline's duplicate-core analyses.

12.5 INDEPENDENT VERIFICATION OF MINERALIZATION

Mr. Bickel visited the Gunnison Project site on May 3rd, 2021. Drill cuttings from several production wells drilled at the North Star deposit were examined, and data such as core photos, cross sections, and geophysical well logs were reviewed. Prior to this site visit, Mr. Bickel has inspected the Gunnison drill core in March of 2021 at the Casa Grande core shack and has reviewed Excelsior procedures for logging, sampling, sample handling, and SG determinations.

Mr. Bickel did not collect samples of core for the purposes of verifying the presence of copper mineralization at North Star. Outcrops a short distance to the east of the deposit with visible copper-oxide mineralization were inspected and significant copper mineralization in long intervals of Excelsior drill cuttings visually confirmed by Mr. Bickel during the site visit. The existence of the North Star deposit has been known widely in the industry for many years prior to Excelsior's involvement, based on the results of drilling programs conducted by major copper-mining and exploration companies (*e.g.*, Magma, Cyprus, and Superior).

12.6 DISCUSSION OF 2018-2019 PRODUCTION WELLFIELD DRILLING DATA

The author has reviewed the data collected from the 2018-2019 production wellfield drilling. In Mr. Bickel's opinion, the production wellfield data is immaterial to the mineral resources estimated herein. This conclusion is based on the scale of the wellfield compared to that of the mineral resource, and the general agreement of the existing model with the data collected in the production wellfield drilling. Mr. Bickel found that these data, through a combination of visual and statistical reviews and checks, reasonably match the modeled copper grades and geology from the existing mineral resource estimate. Mr. Bickel sliced 50-foot cross-sections through the existing block model and reviewed the production well drilling data to check its accuracy against the model and found the results to be acceptable. Mr. Bickel also compared the copper assays from the production wells against the modeled grades statistically and found that the production well assays were on average slightly higher (4% higher for acid-soluble copper, and 2% higher for total copper) than the predicted grades in the model throughout the wellfield area. This difference is well within an acceptable margin of error, especially when considering that the drilling and sampling methods to collect the data for the wellfield are different from those dominantly used to create the model. Mr. Bickel also considers that above all else, the scale of the wellfield, which only represents approximately 3% of the metal contained reported copper resources, is not material to the entire resource. Therefore, any meaningful update to the existing mineral resources would require new drilling with greater spatial coverage than that which currently exists.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Core samples from the Gunnison copper mineralization have been evaluated by various entities since the early-1970s and many leaching techniques have been considered, at least in a cursory manner. However, laboratory-scale simulation of in-situ recovery of copper was pioneered by Excelsior, with the commencement of an extensive program in 2010. This work was conducted in two commercial laboratories in Tucson, AZ, and included conventional column leaching, supplemented by novel approaches toward mimicking the behavior of leaching solutions passing horizontally through naturally fractured copper-mineralized host rocks.

These studies by Excelsior culminated in 2016 with modelling of the advance of a dilute aqueous sulfuric acid front through natural fractures in groundwater-saturated rock from an injection well to an extraction well. Predictions were made of the extraction of copper from minerals that were determined by classical wet chemistry to be, not only acid-soluble (“ASCu), but also both very slowly soluble in dilute sulfuric acid, and essentially acid-insoluble. The modelling included estimates of sweep efficiency, expressed as the probability of copper mineral contact over an extended time by leaching solution.

The general outcome of these investigations was estimation of sulfuric acid consumption per unit of copper dissolved, expressed as lb H_2SO_4/lb Cu, and percent extraction of both acid-soluble and total contained copper versus time. Predicted acid consumptions were generally in the range 7-10 lb/lb with essentially complete extraction of potentially leachable copper in 30 to 50 months. Owing to the continuing reaction of gangue minerals with prolonged exposure to acid, the gangue component of acid consumption naturally increased with time.

With regard to overall acid consumption, the Gunnison project has a unique advantage through ownership by Excelsior of the nearby Johnson Camp Mine (“JCM”). The JCM is an open-pit heap leaching facility, equipped with a solvent extraction/electrowinning plant, and has been operated intermittently on ore from the North Star by at least four owners since the 1970s. The orebody supplying the JCM has essentially the same geology and copper mineralization as the Gunnison resource, and raffinate from JCM was used in all of the Excelsior testwork, as well as being the injection fluid for in-situ leaching. Owing to decades of solution stacking and forced evaporation of excess water, the JCM raffinate is mature and near saturation with the major gangue constituents, thereby minimizing the gangue-related component of acid consumption.

13.2 LABORATORY METALLURGICAL TESTS

In this section, metallurgical investigations are presented chronologically and the final sub-section, Recommendations, will refer back to some of the earlier work, as it provides useful insights.

13.2.1 Early Laboratory Test Programs Pre-2006

Between 1972 and 2012, samples from the Gunnison resource, earlier named the “I-10 Project” by Phelps Dodge, were tested and evaluated by Superior Oil, Quintana Minerals, Phelps Dodge, Magma Copper, Nord Resources, and Excelsior Mining Corp. Unfortunately, though, the usefulness of those tests was often impaired by the absence of sample locations, descriptions, and/or mineralogical characterization, or by unrealistic or inappropriate test conditions and parameters. Salient features of the metallurgical reports are summarized below in chronological order with document titles and laboratories, or property owners given in the footnotes.

Metcon conducted some agitated sulfuric acid leaching tests on crushed samples of mined Martin and Upper Abrigo formations with heads of 0.61% TCu and 0.57% ASCu, yielding PLS grades of 1.3-1.7 gpl Cu (Metcon, 1972). Mountain States Research and Development, Inc. (MSRDI) performed a variety of tests on coarse core rejects from drill hole T-2 at different depth intervals (MSRDI, 1973a). Laszlo Dudas, a renowned consulting mineralogist, observed that 60 percent of the copper was present as true chrysocolla, but that the remainder was a semi-refractory form of dilute copper silicate impregnating a layer silicate lattice. Both sulfuric acid and aqueous ammonium carbonate were used in agitated leaches, but acid was more effective. The deepest core interval consumed 9-14 pounds acid per pound of copper leached ("lb/lb") and MSRDI concluded that a sufficiently high acid dosage should readily dissolve 70-80 percent of the total copper. Actual extractions by MSRDI were in the range 72.3-81.1%.

MSRDI also performed tests that included heat treating followed by ammonium carbonate leaching, calcite flotation prior to leaching with sulfuric acid, and simulated vat leaching with sulfuric acid (MSRDI, 1973b). None of these methods produced results that were sufficiently encouraging to justify further evaluation.

Magma carried out a series of bottle roll tests on minus 10-mesh samples of unspecified origin (Magma, 1992). An average of 62.8 percent of the total copper dissolved at pH 1.5, producing pregnant leach solution ("PLS") grades of 0.46-1.2 gpl Cu, essentially proportional to the ASCu assay of the samples. Magma then published an addendum presenting head and "tailing" (leach residue) assays that illustrated leaching of calcium and magnesium minerals and precipitation of gypsum (Magma, 1993). Because of gypsum precipitation, residue assays as high as 12% S were produced from samples containing <0.1% S.

Magma conducted subsequent bottle roll tests with sulfuric acid on two minus 10-mesh composites and obtained 50.7 and 84.9 percent ASCu extractions, but the residue from the former test still contained 0.28% ASCu, casting doubt on the validity of the test (Magma, 1995). The residue from the latter test assayed only 0.05% ASCu, as one would expect from the high ASCu extraction.

Magma then ran "mini-column" acid leaches with epoxy-coated core fragments (to seal fractures created by drilling and core splitting) (Magma, 1996). Total copper extraction was very poor at only 16.9 percent, but it is worth noting that recirculation ("stacking") of the leaching solution produced a PLS grade of 0.72 gpl Cu. The tests were run at only 1 gpl free acid, which probably limited copper extraction.

HRI loaded clear PVC columns 6 inches in diameter by 10 feet high with fragments of 6-inch core and smaller pieces and leached the columns with sulfurous acid (H_2SO_3) at concentrations of 20 gpl and 40 gpl aqueous SO_2 (HRI, 1996). After 5 months of operation, 70 percent of the copper had dissolved from the column with the stronger lixiviant, and 48 percent had dissolved from the other. Equivalent sulfuric acid consumptions were 9 lb/lb from the more acidic column and 8 lb/lb from the other. These results were very encouraging, and the use of sulfurous acid deserves further consideration. Digestion with sulfurous acid is, of course, the preferred analytical procedure for assaying ASCu. Although sulfurous acid will attack calcium carbonate, it probably forms calcium sulfite, not gypsum, and calcium sulfite, may be more soluble than gypsum in dilute sulfuric acid. The stronger lixiviant produced an initial PLS grade of 2.88 gpl Cu that eventually equilibrated at about 0.3 gpl Cu.

Phelps Dodge subjected six samples to ammonia leaching, sulfidization and flotation, and dilute sulfuric acid leaching in bottle rolls (Phelps Dodge, 1996). The first two techniques did not yield promising results, but bottle roll copper extractions with dilute sulfuric acid were in the range 74-98 percent with five of the six above 92 percent. Heads of 0.43 to 0.88% TCu produced residues that generally contained 0.01-0.06% TCu, with one at 0.14% TCu.

Although a significant number of metallurgical tests were conducted by five laboratories for four property owners between 1972 and 1996, the results were variable and do not allow derivation of reliable interpretations or projections of copper extraction by ISR technology. Faced with this uncertainty, Excelsior Mining Corp. commissioned Hazen in early 2011 to conduct column leaching tests that were intended to predict ISR copper response and acid consumption (HRI, 2011).

The samples used by Hazen were designated Met 1, Met 2, and Met 3, and can be described as follows:

- Met 1 was made up of 2-foot to 7-foot intervals totaling 17 feet between depths of 667 feet and 714 feet in Hole NSD-22. This represented the Martin formation, a high acid consumer.
- Met 2 comprised 3-foot to 10-foot intervals totaling 18 feet between 1182 feet and 1215 feet in Hole NSD-22. This came from the Lower Abrigo formation, a low acid consumer.
- Met 3 was made up of the 20 feet of Hole NSD-11 between 910 feet and 930 feet in the Upper Abrigo formation, a variable acid consumer.

Most of the core fragments had been epoxy-coated prior to arrival at Hazen to seal surfaces and fractures created by drilling and core handling.

The samples were loaded into 6-inch diameter clear PVC pipe and leached with dilute sulfuric acid solutions for 164 days, followed by a 14-day water rinse. Unfortunately, several aspects of Hazen's test procedure precluded direct and reliable prediction of the rate of dissolution of copper, the ultimate amount of copper dissolved from the samples, and both maximum and average PLS grades.

1) Rather than adding acid to the lixiviants at a continuous rate in order to quickly achieve and then maintain a target PLS acidity of about pH 1.5, the lixiviants was made up in batches with different free acid concentrations ranging from 1.91 to 4.68 gpl free acid. The technique that was employed is a reasonable one for samples that consume little acid. However, when applied to samples that consume significant amounts of acid, the gangue minerals tend to deplete the free acid and raise the pH, impairing the solution's ability to dissolve copper.

   This deficiency was especially serious with the Met 1 sample because the PLS was nearly always less acidic than pH 2.0-2.5 with one exception of a 7-day period when it was pH 1.6. Examination of the raw data reveals that the rate at which copper dissolved was 1.7 percent per day at pH 1.6, but only 0.2-0.3 percent per day above pH 2.5. This problem was less pronounced with Met 2 where the PLS was between pH 1.5 and 2.0 and Met 3, where it was mainly in the range pH 1.7-1.3 after 22 days. Nonetheless, the rate at which copper dissolves from chrysocolla diminishes rapidly in solutions less acidic than pH 1.5, so all three columns were operated under very non-ideal conditions.

   Given that the non-sulfide component of the copper mineralization is essentially all chrysocolla and other silicates, and that it occurs on fracture planes and surfaces, it is reasonable to assume from testwork done to date that approximately 65 to 80 percent of the copper will dissolve if contacted by dilute sulfuric acid solutions at nominally pH 1.5.

2) Hazen estimated total acid consumption based on a material balance accounting for acid added to each column, free acid at various time intervals, and the cumulative mass of copper dissolved up to that time. By this procedure, their estimates are as follows in pounds of acid per pound of copper dissolved (lb/lb), Met 1, 8.9 lb/lb; Met 2, 5.0 lb/lb; and Met 3, 3.9 lb/lb. Here, it is important to point out that all acid consumptions that have been reported are total acid consumed. In practice, electrowinning creates 1.54 pounds of 100% equivalent sulfuric acid, so gangue acid consumption equals the total minus 1.54.

3) The columns were loaded with coarse core fragments whose dimensions were sometimes as large as half of the column diameter. However, the voids were not filled with inert solids such as sand, as is sometimes done in such experiments. This resulted in calculated void volumes of 67 percent, 80 percent, and 73 percent, respectively for Met 1, Met 2, and Met 3. Yet the calculated natural pore volume was only 5.4 percent. As a consequence, the solution inventory in the columns was roughly 12 to 14 times the pore volume and some of the solution may not have contacted copper mineralization. This had a profound impact on PLS copper grade.

Although the Met 3 column produced a peak PLS grade of about 0.94 gpl Cu after 35 days, Met 1 peaked at 0.47 gpl Cu and Met 2 at 0.30 gpl Cu. All PLS grades eventually declined with time.

Because of the excessive void volume, the Hazen tests used 135 to 210 pore volumes of solution during leaching and 18 to 24 pore volumes during the 14-day rinse or "sweep" cycle. As an illustration, assume that ISR will require 25 pore volumes for leaching and 5 pore volumes for rinsing. At a porosity of 5.4 percent, this would equate to 12.97 gallons of pores per ton of rock, and 30 pore volumes would total 389 gallons of solution per ton. At a grade of 0.4% Cu with 65 percent dissolved, there would be 5.2 pounds of copper dissolved per ton of formation. This would yield an average PLS grade of 1.61 gpl Cu. Application of more pore volumes will proportionately reduce the PLS grade.

4) Rinsing of the columns with water for 14 days with 18-24 pore volumes only reduced acidity to the following terminal values: Met 1, pH 3.33; Met 2, pH 3.47; and Met 3, pH 3.22. However, ineffective rinsing was due to the large void volume and, possibly, to "dead space" that was not swept efficiently by the advancing water.

13.2.1.1 Conclusions from Pre-2006 Metallurgical Testing

Bottle roll and column leaching tests have long been industry standards and will provide reliable process design criteria in performed correctly on samples that faithfully represent major fractions of the resources that will be commercially exploited. If the tests are conducted with crushed and screened, but still relatively coarse ore fragments, nominally minus 10-mesh, they yield reliable estimates of maximum acid consumption and maximum metal extraction. For heap leaching of oxidized gold ores and copper ores containing oxide copper or secondary sulfides, these tests are very predictive because adequate ore/solution contact can be designed into a heap. They also are applicable to ISR for the extraction of uranium from roll-front type deposits in paleochannels because the sand grains usually are barely consolidated and the lixiviant flows freely through the deposit, except when clay lenses are encountered. Moreover, the uranium ISR lixiviant is only oxygenated groundwater with enough dissolved CO_2 to form the uranyl carbonate complex, so the gangue is not attacked. Given these qualifications, it is reasonable to conclude from the early test programs that total sulfuric acid consumption (before the electrowinning credit) will be approximately 9 lb H_2SO_4/lb of copper dissolved, that average PLS grade will be as high as 1.5 gpl Cu, and that about 65 percent of the total copper will dissolve.

13.2.2 Recent Laboratory Metallurgical Testing

During late-2012 through mid-2013, extensive testing was done at the direction of Dr. Ronald J. Roman. This work was conducted by the Mineral Advisory Group Research & Development LLC laboratory in Tucson, AZ, supervised by James Minnow. Sample descriptions, details of the laboratory procedures, interpretations of the test results, and modelling of acid consumption and copper extraction were provided in the original NI 43-101 report ("TR-FS") issued on January 16, 2017 (R. Zimmerman, et al., 2017).

Since much of the earlier testing was done in conventional vertical columns like those that have customarily been used to simulate heap leaching, the new program was designed specifically to mimic conditions that characterize lixiviant/mineral contact in an in-situ leaching regime. This design philosophy recognized the potential importance of creating horizontal solution flow and led to a series of configurations in addition to vertical columns.

Laboratory simulation of in-situ extraction has customarily led to development of alternatives to columns, not so much with the expectation of different outcomes, but rather as confirmation that column results are valid irrespective of flow regime. For at least the last 60 years, flooded upflow columns have been used to develop data for designing vat leaching plants, and downflow columns have been used to develop design criteria for heap leaching. The result is a vast body of data with generally close correlation between laboratory and commercial plant.

13.2.2.1 Box Tests

A limited number of tests were conducted in a horizontal plastic box with a removable lid into which sections of core could be arranged horizontally and aligned with the long axes perpendicular to solution flow. Interstices were packed with acid-washed sand to minimize void volume and solution short-circuiting. This way of simulating horizontal solution flow was used briefly, but mechanical problems necessitated adoption of conventional columns as a more convenient way of obtaining basic data. As discussed below, the column series was then followed by another experimental design embodying a novel apparatus called a “Core Tray”. Finally, a “Bucket Test” was devised with the objective of correcting the apparent leaching behavior of core that had been influenced by the presence of unnatural (“man-made”) fractures.

13.2.2.2 Series 3 Column Tests

In this series, 24 core samples were crushed through a 1-inch screen, split into five smaller weighed screen fractions, and the fractions were assayed for TCu and ASCu, enabling calculation of the aggregate head grade for the entire sample. The individual samples varied in fracture intensity and solubility index, i.e., the ratio of ASCu to TCu. Geological formations that were represented included Upper Abrigo (low intensity), Middle Abrigo (low intensity fractures), Lower Abrigo (low intensity), Undivided Abrigo (high intensity), Martin/Escabrosa (high intensity), Martin/Escabrosa (low intensity), and Undivided Abrigo with transition mineralogy (low intensity).

Each sample was then reconstituted, blended, and charged into a separate column for testing, Six of the 24 columns plugged, and were abandoned. Dr. Stephen Twyerould has provided the Excel database describing all of the samples that were tested and has explained the nature of the plugging that was experienced (S. Twyerould, 2022). All of the failed columns had been charged with crushed material from the Martin/Escabrosa formation. The core samples were all logged as having a low fracture intensity. Dr. Twyerould’s recollection of a 2014 conversation with Dr. Roman at the MAG laboratory has clarified the cause of failure. The samples all contained high volume percentages of primary carbonates that were present in the matrix, rather than existing as fracture fillings.

Dr. Roman’s examination of the failed columns revealed several very likely causes for the plugging: (1) the high concentrations of relatively soft and friable primary calcite released large amounts of fines during crushing; (2) the columns had been charged incorrectly, leading to accumulations of fines at the bottoms of the columns, rather than even distribution throughout the column height; and (3) conversion of the calcite to gypsum was accelerated initially by incorrect solution flowrates and free acid concentration. The result, at least in one instance, was creation of a solid mass of gypsum-cemented carbonate minerals at the column bottom.

Although we depend on standardized tests such as packed columns to predict process design criteria, copper extraction, and typical acid consumption, we also rely on those tests to warn of abnormal behavior that may presage project failure. Dr. Roman’s observations at the time of the test program (2014) were good examples of the qualitative, but insightful, observations that should always accompany laboratory investigations.

Loss of permeability in a column is typically caused by an accumulation of alteration products such as clays. This is usually evident at a standardized irrigation rate of 0.005 US gallons per minute per square foot of charge surface and is reflected by ponding of applied solution on the surface of the column charge. However, standard column tests are conducted by dripping solution onto the upper surface of the charge at a flowrate that does not normally lead to full submergence (“flooding”) of the column’s contents. In contrast, the Series 3 columns were irrigated by dripping solution onto the upper surface, as in conventional downflow contact, but the sample was flooded by elevating the solution discharge line to the level of the top surface of the column charge. This irrigation method is depicted in Figure 13-1 of the TR-FS.

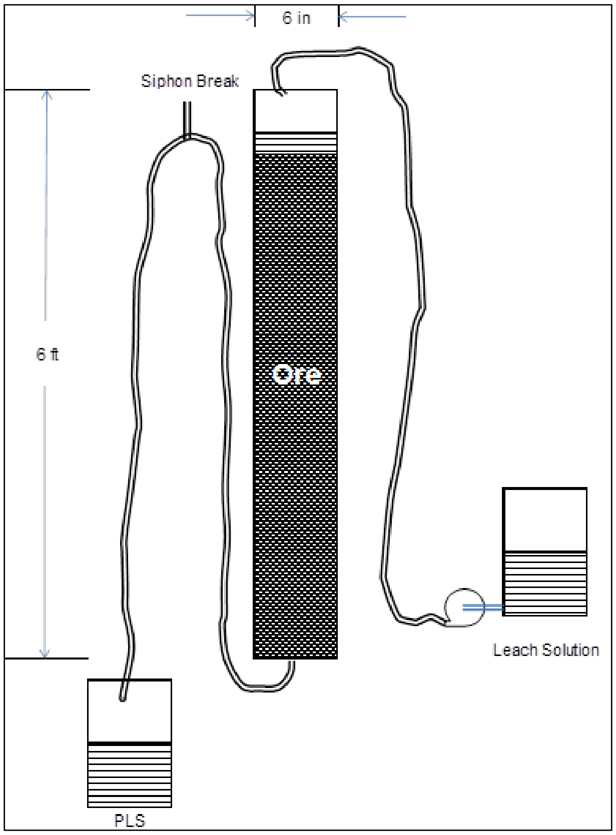


Figure 13-1: Column Test Set-Up

Operation of the Series 3 columns in this manner was designed to simulate in-situ leaching of a mineralized zone that is submerged in solution. In principle, this approach is quite reasonable, but it negates a direct comparison with conventional column testing that has been applied historically to prediction of heap leaching performance. This mode of column operation is essentially anaerobic, whereas unflooded columns allow moderate oxygenation of the percolating lixiviant solution. Qualitatively, this method of irrigation would be expected to promote less oxidation and dissolution of secondary sulfides than a conventional unflooded column test.

The standard irrigation rate for conventional column tests and for many gold and copper heap leaching operations is 0.005 gallons per minute per square foot of charge surface (12.2 liters per minute per square meter). At this rate of solution application, agglomerated heap feed or granular ore fragments without agglomeration will accept the leaching solution without a pond or “perched aquifer” forming atop or within the column or heap. Some column tests were conducted with application rates as high as 22 lpm/m^2 (Martin formation) and 37 lpm/m^2 (Upper and Lower Abrigo), so at least 75 percent of the Series 3 columns were sufficiently permeable to accommodate nearly twice to three times the standard application rate.

The TR-FS on pages 61-63 contains photographs of drill core boxes arranged according to fracture intensity ranging from very weak (denoted 1) to very strong (denoted 5). A core interval was classified as weak if less than 5% of the interval comprised fragments shorter than 4 inches, whereas very strong intensity was characterized by 80-100% of the fragments being shorter than 4 inches.

All of the column feed was crushed to minus 1-inch, irrespective of fracture intensity. Crushing rock with a low fracture intensity exposed more of the massive matrix, thereby liberating carbonates that might not be contacted by lixiviant solutions during in-situ leaching. A likely consequence of this is that the column tests overstated gangue acid consumption. Also, rock with a high initial fracture intensity contained a greater abundance of natural fines, whereas crushing of rock with low fracture intensity created excessive man-made fines. Given the relative softness and friability, those man-made fines would be expected to contain a disproportionate concentration of carbonates.

Superficially, it appears that intervals logged as 1 and 2 contained essentially no fragments smaller than nominally 1-inch. Even the intervals logged as 3 and 4 contained minimal true fines. In contrast, intervals logged as 5 (very strong fracture intensity) contained predominantly small chips, interspersed with fragments several inches long, but more importantly approximately 20 to 30% of the recovered material was extremely fine.

A possible consequence of the finely divided fracture fillings in the most intensely fractured portions of the resource is that they may become mobilized by solution movement, thereby behaving somewhat like clays during leaching, and creating resistance to solution flow.

13.2.2.3 Core Tray Tests

Concerns about potential overstatement of copper extraction, rate of extraction, and gangue acid consumption led to development of the “Core Tray Test”, embodying 9-foot-long plastic boxes in which vertical sections of uncrushed core were embedded in epoxy with just a naturally fractured surface exposed to solution flowing across the fracture surfaces. Apparently, fractures normal to the upper surface were minimal or nonexistent. These tests yielded significantly lower projections of gangue acid consumption.

13.2.2.4 Bucket Tests

In an effort to illuminate the behavior of gypsum as it forms during leaching, sections of massive core with copper mineralization, as well as barren calcite-filled seams, were leached in buckets to allow contact with lixiviant solutions. Cracks that had contained carbonates developed rims of gypsum on the edges, and this was interpreted by Dr. Roman as evidence that gypsum remains at the site of creation, rather than mobilizing into the moving solution or being transported to other solid surfaces. Projected gangue acid consumptions from these tests were 50 to 90 percent lower than projections from column tests.

Based on the QP’s experience in a number of gypsum-forming processing environments, the immobility argument can only be confirmed if the experiment includes a micro-pore filter to remove suspended gypsum crystals from solution. As a general rule, some gypsum may remain at its origin, but only because it precipitated from a supersaturated liquid boundary layer. As the solution moves on, it remains saturated until its remaining free acid encounters more unreacted carbonate and re-saturates. Otherwise, precipitation within the moving body of solution may proceed until a change in temperature or composition triggers another precipitation event.

Another way of developing a clearer perspective on the potential role of gypsum on formation permeability is to consider the volume of gypsum created during reaction of calcium carbonate with sulfuric acid. Since the reaction takes place at essentially ambient temperature in a water-saturated environment, it is safe to assume that the calcium sulfate product hydrates fully to gypsum, $CaSO_4 \cdot 2H_2O$, with molecular weight of 172.14. Commercial sulfuric acid containing 98% H_2SO_4 has an effective molecular weight of 0.98 x 98 = 96. Therefore, 1 pound of sulfuric acid will create **1.79 pounds** of gypsum, neglecting the reaction products from other gangue minerals such as orthoclase.

According to the TR-FS, the average porosity of the Gunnison deposit is 3 percent by volume and the in-place specific gravity is 2.60, or 162 lb/ft^2 or 12.35 ft^3/ton. Therefore, the void volume, including cracks, is about. **0.37 ft^3/ton.** Massive crystalline gypsum has a specific gravity of 2.32, or 145 lb/ft^3 or 13.79 ft^3/ton, but it is likely that freshly precipitated gypsum is relatively fluffy with a much lower effective bulk density (dry mass divided by volume occupied). If one assumes that the bulk density is somewhere between 75 and 145 lb/ft^3. If so, the mass of gypsum needed to fully occupy the natural voids is between 0.37 x 75 = **28 pounds** and 0.37 x 145 = **54 pounds per ton of mineralized rock**.

However, it is necessary to consider the voids created by dissolving carbonates. If the total gangue acid consumption is 50 lb/ton of rock, and carbonates account for half of the total consumption, then 25 pounds of sulfuric acid will be consumed in leaching carbonates. If calcite is the dominant carbonate, then the amount of calcite dissolved will be about 25 pounds. The molecular weight of calcite is 100.1 versus 172.1 for gypsum. Therefore, approximately 25 x (172.1/100.1) = **43 pounds** of gypsum will be created. With a maximum specific gravity of 2.73, or 170 lb/ft^3, 25 pounds of calcite would occupy 0.147 = approximately **0.15 ft^3** per ton of rock.

These assumptions suggest that the sum of natural voids and the voids formed by loss of calcite could be approximately 0.37 + 0.15 = **0.52 ft^3 per ton of rock**, and 43 pounds of gypsum or fluffy gypsum would occupy somewhere in the range of (43/145) to (43/75) = 0.30 to 0.57 ft^3 respectively. Therefore, gypsum or fluffy gypsum could potentially occupy (0.30/0.52) x100 to (0.57/0.52) x100 = 58 to 110 percent, respectively, of total void volume. Obviously, this rationale for estimating void filling by gypsum is strongly dependent on the relative abundances and crystal morphologies of primary carbonates, secondary calcite, and gypsum. (Some of the carbonate is dolomite, which has only half of the gypsum exchange volume that calcite has.) If the secondary calcite, for instance, is in the form of porous acicular crystal masses, the bulk density of the calcite will be lower, and less gypsum will form per mass unit of calcite.

13.2.3 Current Plan for Overcoming Wellfield Issues

Since the Gunnison host rock is a skarn with high calcium carbonate content, it has always been recognized that the consumption of a dilute sulfuric acid lixiviant will inevitably be elevated in parts of the deposit. Although the net gangue acid consumption (after the electrowinning credit) determined during various test programs has usually been moderate at 7-10 pounds H_2SO_4 per pound of copper dissolved, the potential exists for localized consumptions that are higher. Calcium carbonate is present mainly as the mineral, calcite, which dissolves, forming gypsum and carbon dioxide according to reaction (1).

$$(1)\quad CaCO_3 + H_2SO_4 + H_2O = CaSO_4{\bullet}2H_2O + CO_2$$

Ideally, carbon dioxide is dissolved in an aqueous system like an in-situ wellfield, but the equilibrium constant for this reaction is rather small, so de-carbonation of the water product allows formation of gaseous CO_2 at a relatively low hydrostatic head around 60-120 psig (approximately 4-8 bar). Technically, reaction (1) is reversible, according to the most common example of Le Chatelier's principle, but it may be kinetically hindered, an effect that is not predictable by classical equilibrium thermodynamic calculations.

Operation of the Gunnison wellfield has revealed that solution injection flowrates diminish with time, but that substitution of water for injected acidified raffinate restores the flowrate. One interpretation of this behavior is that CO_2 gas is accumulating at entrances to flow channels and impeding solution flow through the formation. Water injection then re-dissolves or flushes out the CO_2, thus restoring flows until acid is re-injected. A program that is now being engineered will test this hypothesis.

It is possible that gypsum, not CO_2, is impeding flow. If so, alternative interpretation is that gypsum or “fluffy gypsum” is inhibiting acidified raffinate injection flow and that this gypsum is removed, remobilized, or washed away under water injection, thus explaining the improvement in flow rates on water injection. According to Excelsior data, there is experimental evidence suggesting that gypsum would tend to remain at the site of calcite attack and would therefore not cause the type of formation blockage that is reversible by water injection, primarily because gypsum is only moderately soluble in water. However, the flow improvement due to water injection could be explained by poorly crystallized or “fluffy gypsum” being soluble in water or being mobilized and flushed out of the formation during water injection.

A modelling study by *OLI systems, inc.* has concluded that gaseous CO_2 will not form if the injected acidity is less than about 5 gpl free acid (H^+ ion), but SRK has been engaged to examine this phenomenon in greater detail and is discussed in the section below (A.J. 2021).

13.2.3.1 Introduction to the Problem of CO_2

The wellfield commenced operations with water recirculation in December 2019, designed to commission wells and equipment. Commissioning on water was successful.

Acid injection commenced on December 31, 2019, via 16 injections wells (5000 series wells), and recovered via 25 production wells (7000 series wells). The 41 wells were operated in compliance with the EPA and ADEQ permits.

Initial acid injection resulted in strong flow improvement compared to the commissioning period on water, however this was not sustainable due to the precipitation of copper hydroxides that were negatively impacting recovery pumps. This challenge was resolved by re-configuring the well heads to allow for injection and recovery operations in each well, which allowed the backflushing with weak acid solution to redissolve the precipitates. This reconfiguration also allows for greater wellfield operational flexibility.

Following a period of care-and-maintenance due to the COVID-19 pandemic, operations were re-started at a smaller scale in late July 2020. Shortly after restarting, injection wells showed declines in injection flow rates. Given large amounts of gas were observed coming from nearby wells (bubbling in recovered water and measured as CO_2 with CO_2 meters), it was suspected these declining flow rates were the result of CO_2 gas bubbles clogging the formation (porosity) adjacent to the injection wells. The CO_2 was believed to be coming from the reaction between the injected acid solution and secondary calcite in the fracture system.

To test this hypothesis fresh water was injected into some of the affected wells, the concept being that the fresh water would; a) not form additional CO_2 gas and b) redissolve the CO_2 gas bubbles that were potentially being trapped in the formation and restricting injection flow.

These experiments were successful in rapidly restoring injection flow rates. However, returning to acid injection caused an equally rapid re-deterioration of injection flow rates (Figure 13-2). Figure 13-2 shows Well 5474 showing a decline in flow rate on acid injection (red bars), and rapid improvement in flow rate on water injection. In both cases of water injection, flow rates were still improving when switched back to acid injection that generates declines in flow rates.

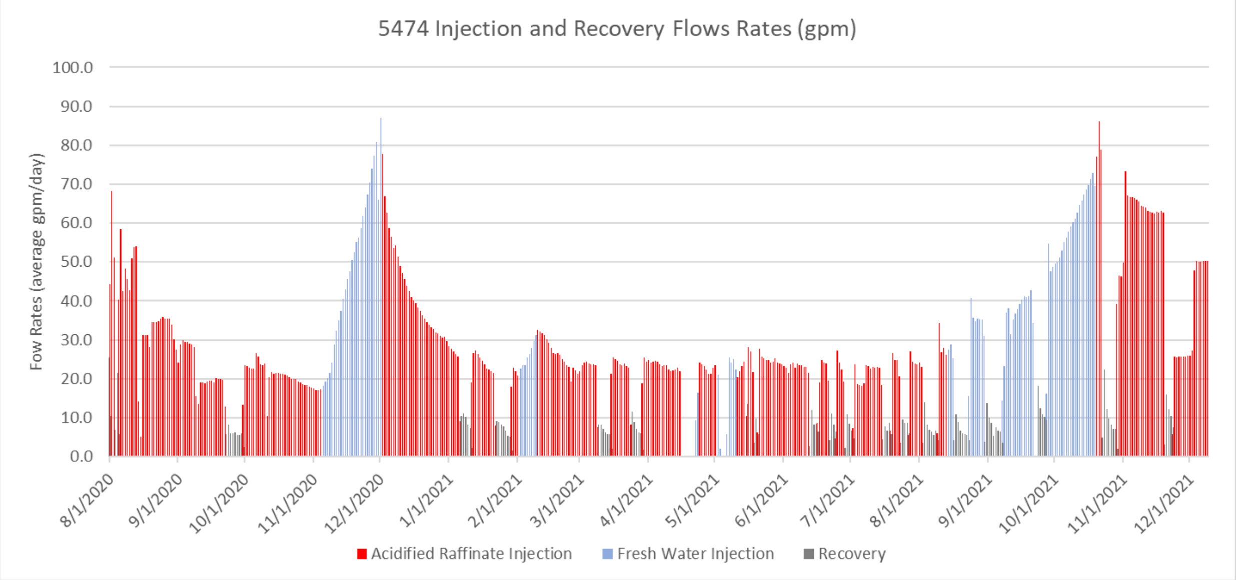


Figure 13-2: Well 5474 Injection and Recovery Flow Rates

13.2.3.1.1 Mineral Precipitates

Mineral precipitates or swelling clays were also considered as possible causes of the declining injection flow rates. Candidates for mineral precipitates include gypsum, bassanite ($CaSO_4$• $0.5H_2O$, also called hemihydrate of $CaSO_4$), basaluminite, (hydrous aluminum sulfate), jarosite, copper hydroxides or other mineral hydroxides or sulfates.

Except for, Gypsum whose precipitation or dissolution is not significantly pH dependent, the other mineral precipitates dissolve in low pH (acidic conditions) and precipitate as the pH rises. As the acid front moves from injection well to recovery well, the solution front transitions from low pH to higher pH, representing the neutralization front of the acid solution. Mineral phases likely precipitate at or near this neutralization front and the precipitation of these minerals could reduce the permeability and flow path volume.

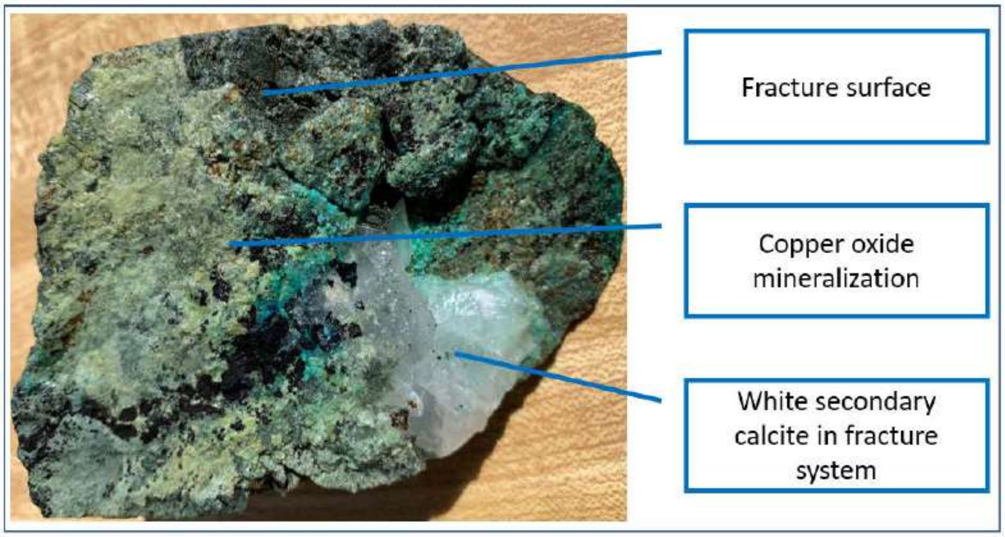
However, in these examples, injecting fresh water at pH of approximately 7 would significantly exacerbate the problem by further and more rapidly increasing the pH, thus increasing the amount and rate of mineral precipitation, which would in turn more rapidly reduce permeability and flow rates.

Switching back to acid injection would redissolve the mineral precipitates and thus improve flow. However, the opposite is observed in the wellfield. Flows increase on water injection and decrease on acid injection, indicating the bulk of the problem is not mineral precipitates, at least not to the extent causing the rapid flow increases and decreases observed. Other than possibly Gypsum, mineral precipitates are considered unlikely to be the causing of the rapid decline on acid and increase on water injection.

Swelling clays are rare in calc-silicate skarn environments and have not been observed or recorded other than in trace amounts in the mineralized fractures at the Gunnison deposit (J. Bickel, RESPEC, pers comm). In addition, the degree of swelling in common swelling clays (e.g., smectite, nontronite, bentonite, chlorite, montmorillonite, beidellite, attapulgite, illite and vermiculite), is not a function of pH, and swelling is typically larger in low salinity waters compared to high salinity water. Both properties are inconsistent with the changes in flow observed as water or acid are injected. Therefore, swelling clays are not considered to be a significant source of the flow restrictions.

13.2.3.1.2 CO_2 Reactions and Chemistry

Calcite is the most abundant and stable polymorph of calcium carbonate in the Gunnison deposit and typically occurs as late-stage secondary calcite withing the fracture system or deeper within the matrix of the rock as residual carbonate. It is the secondary calcite within the fracture system that is most readily exposed to acid solutions (Figure 13-3).



Source: Excelsior

Figure 13-3: Typical Example of Gunnison Oxide Copper Ore Showing Secondary Calcite

Sulfuric acid solution reacts almost instantly with calcite to generate carbon dioxide by a mechanism such as;

Equation 1

$$CaCO_3 + H_2SO_4 = Ca^{2+} + SO_4^{2-} + CO_2 + H_2O$$

Under normal operating conditions with the presence of excess hydrogen ions (acidic conditions), the reaction above is driven to the right, thus producing CO_2, and severely limiting the amount of CO_2 gas that can dissolve into solution, even under higher pressures.

The equation can also be written in the form below:

Equation 2

$$2H^+(aq) + CaCO_3\ (s) \rightleftharpoons Ca^{2+}(aq) + H_2O\ (l) + CO_2\ (g)$$

In narrow fractures the surface area to volume ratio is high and this generates surfaces for gas bubbles to nucleate. Carbon dioxide gas occupies considerably larger volume than the aqueous form and, as such, these bubbles have potential to limit solution transport by reducing permeability.

The persistence of CO_2 gas in the leach field may be abetted by two other factors: the relatively slow kinetics of gas dissolution and the buoyancy of CO_2 gas. The former factor means there is a lag time between the rapid formation of CO_2 in the acid-calcite reaction and the ability of any CO_2 gas produced to be dissolved in the Pregnant Leach Solution. The effects of this asymmetry may be exaggerated by the high buoyancy of gaseous CO_2 relative to the surrounding liquid which mobilizes the gas allowing it to “escape” to various regions of the fracture system. If individual gas bubbles can congregate and coalesce into a continuous phase across meaningful length scales, it may gain the ability to suppress liquid flows. This phenomenon would be fully consistent with experimental studies.

In the absence of excess hydrogen ion (above pH 6.1) and under sufficient pressure and at thermodynamic equilibrium, the produced carbon dioxide remains in solution as bicarbonate ion (see Equation 3).

Equation 3

$$H_2O\ (l) + CO_2\ (g) \rightleftharpoons H^+(aq) + HCO_3^-\ (aq)$$

Carbon dioxide should dissolve in the water at a pH below 6.1 to form carbonic acid, H_2CO_3 (Figure 13-4).

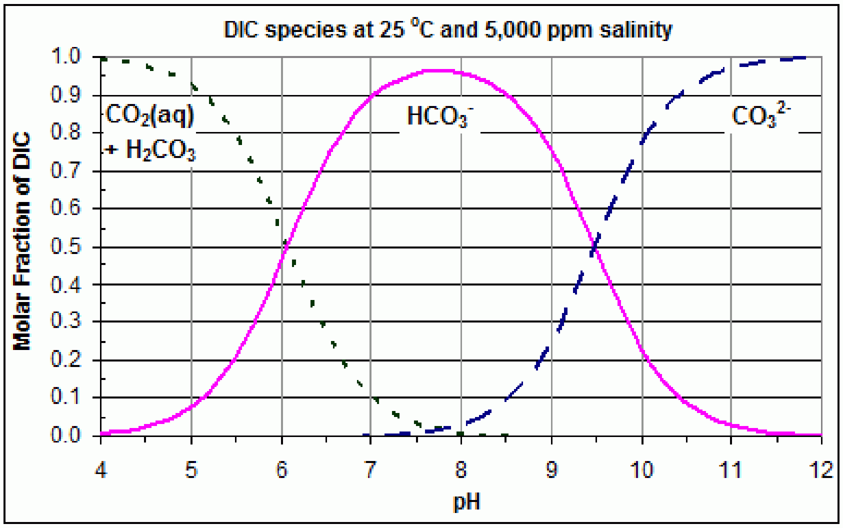


Figure 13-4: Carbon Aqueous Speciation Versus pH at Different Molalities

The predominance diagram, Figure 13-4, shows the predicted speciation of dissolved inorganic carbon ("DIC") in water at 298°K or 25˚C. The diagram indicates that at a pH below 6.1, CO_2 (aq) or carbonic acid is the dominant form of inorganic carbon in water. Above pH 6.1 the presence of more hydroxyl ions consumes some of the free protons and supports the dominance of bicarbonate with no predicted CO_2 (aq) above pH 7.6. Above pH 9.4, the free carbonate ion is predicted to be the dominant form. Under thermodynamic equilibrium it is predicted that any CO_2 present should be present as an aqueous species as carbonic acid.

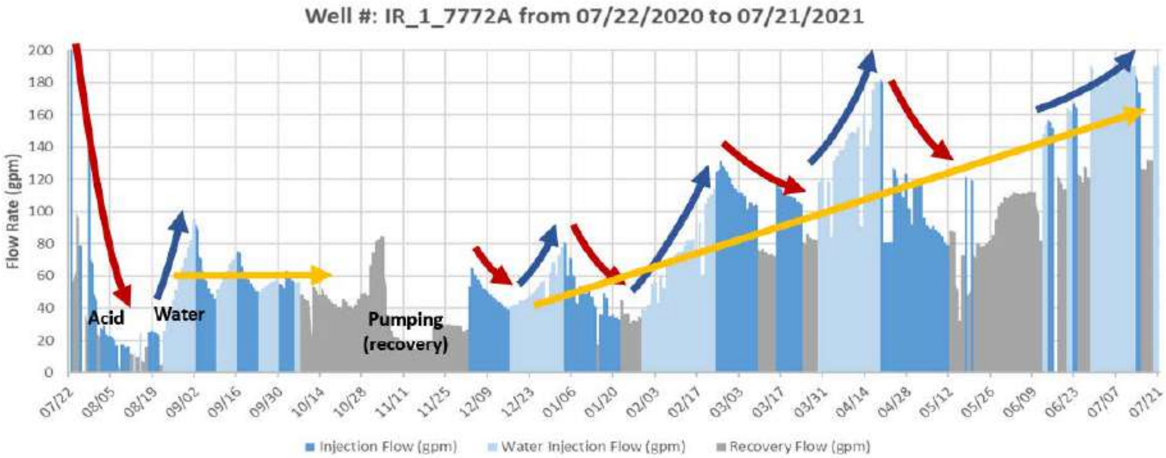
At Gunnison, there are two important factors to consider:

- At low pH (excess hydrogen ion) Equation 3 is driven to the left, the dominate species is aqueous CO_2 and the solubility of CO_2 into the acid solution is very low. This is compounded by the second point.
- The mass of CO_2 generation is high due to high calcite in the upper part of the leach field (up to 20-30% in the Martin formation), so this over-saturates the system with CO_2, exceeding the limit for solubility in raffinate.

The consequence of this is that the reaction of one mole of sulfuric acid (98 g) with one mole of calcite (100 g) would generate one mole of carbon dioxide that can occupy a volume of 5.9 gallons equivalent space. On this basis it is reasonable to assume that with 20-30% calcite in places and flows of 2000 gpm at concentration of 10 g/L sulfuric acid available, permeability space could easily and quickly be consumed.

13.2.3.1.3 Wellfield Experiments and Water Flushing

Flow in the field was improved by flushing with neutral water under pressure, indicating that pH is highly likely to control gas solubility. After flushing with low solute water, flow improves substantially. Repeated acid leaching then repeated this cycle of leaching followed by loss of flow and the need to flush with fresh water again. The nature and impact of this cyclic flushing and rinsing is evident in the copper recovery in the wells (as an example data from 7772A in Figure 13-5).



Note: Well 7772A showing a decline in flow rate on acid injection (dark blue bars and red downward pointing arrows), versus an improvement in flow rates on water injection (light blue bars and blue upward facing arrow). Source: Excelsior

Figure 13-5: Well 7772A Flow Rate Data

Initial injection of acid in well 7772A around July 22, 2020, was at approximately 200 gpm. This rapidly declined to rates of approximately 20 gpm. Repeated cycles of water injection, acid injection and recovery (grey bars), commencing around December 16, 2020, and extending through July 21, 2021, resulted in a sustained flow improvement from a low around 40 gpm to a high of around 180 gpm. This period included approximately 4 weeks during May 2021 where the cycling of water and acid was limited due to the availability of water.

The sustained improvement of flow rates due to the cycling of water and acid injection and recovery clearly indicates that the blocking mechanism is remediated on water injection but exacerbated by acid injection. This is consistent with the problem being primarily caused by something that forms quickly on acid injection, but dissolves or is removed quickly by water. CO_2 gas is considered the best candidate for such a phenomenon and is consistent with the chemistry and mineralogy of the rocks in the fracture system.

Given the CO_2 comes from the calcite in the fracture system, then once this calcite has been dissolved or removed from a particular fluid pathway, CO_2 gas will no longer form along that pathway and restrict acid injection flows. Subsequent flushing on well 7772A appears to have demonstrated the finite nature of calcite and CO_2 along those flow paths responsible for the flow in this well (Figure 13-6 below). Post July 2021, flow rates in this well no longer declined on acid, suggesting the calcite has been removed. It is also worth noting that recovery rates (grey bars on Figure 13-6) also improved with injection flow rate improvements. In this example recovery flow rates improved from around 30 gpm to 150 gpm over a similar approximately 6-month period.

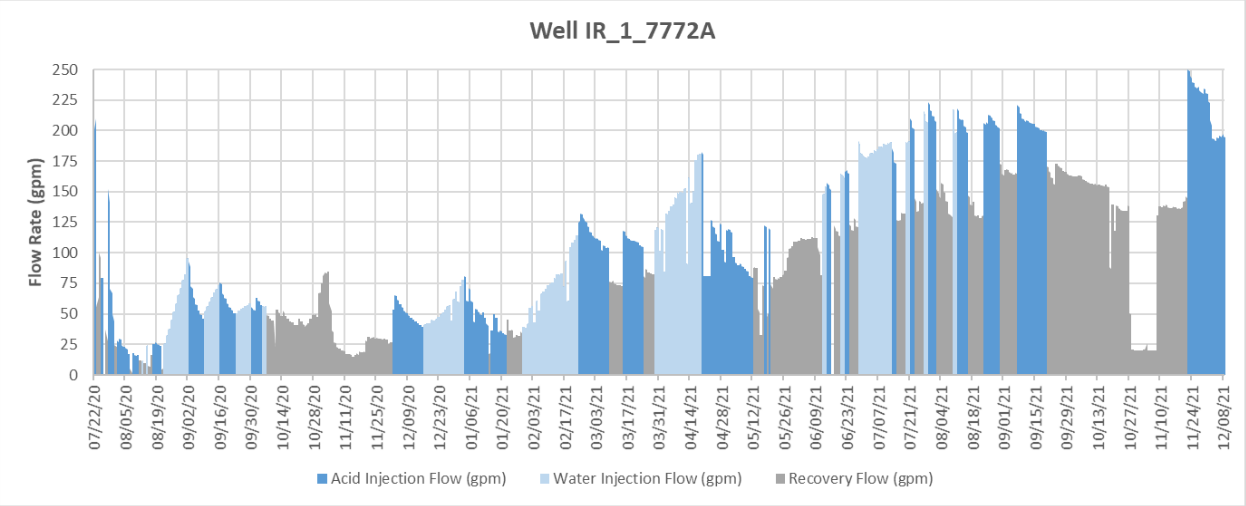


Figure 13-6: Well 7772A Showing Sustained Acid Injection Flow Rates Post July 2021

On average, it took approximately 6 months of cycled acid and water injection with intermittent recovery to clean this well of calcite along the active flow paths up to approximately 200 gpm. Further improvements were considered possible with this well however it had exceeded the approximately 200 gpm water and evaporation capacity (availability) of the wellfield and related infrastructure.

Two wells, IR_2_5270 and IR_3_5269, underwent a similar experiment with water flushing (Figure 13-7 below). In these examples, the water flushing cycle commencing around August/September 2021 did not include acid injection cycled with the water injection (just water injection separated by recovery). Flow improved substantially; however, the graphs below highlight some of the challenges with water flushing. During periods of water flushing, there is no active acid and therefore no related copper production or calcite removal. In both cases water dilutes the copper grade in solution and, although water flushing improves flow rates (including recovery flows), unless there are intermittent periods of acid injection, water flushing will not ultimately solve the problem because some acid is needed to eventually dissolve and remove the calcite.

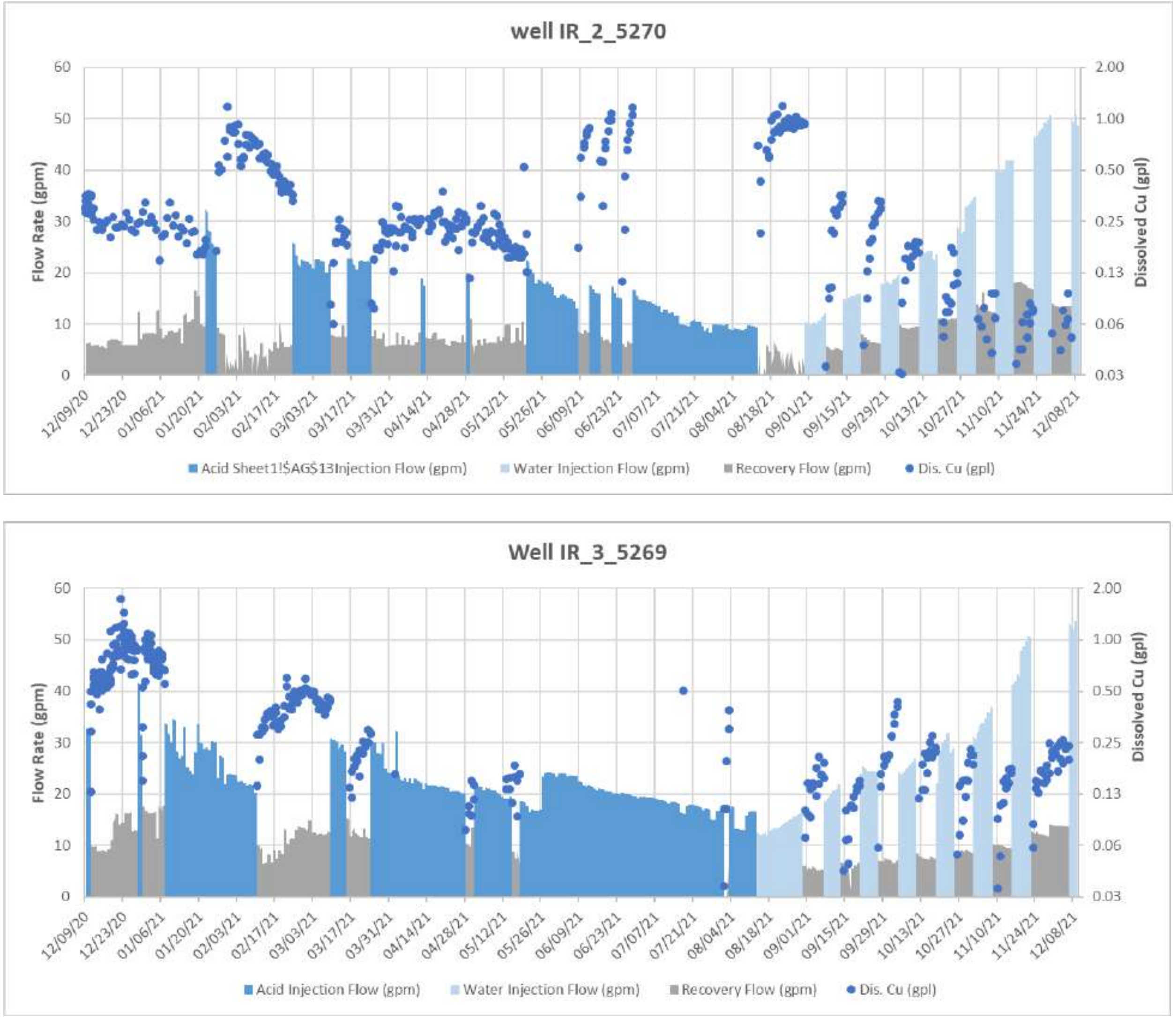
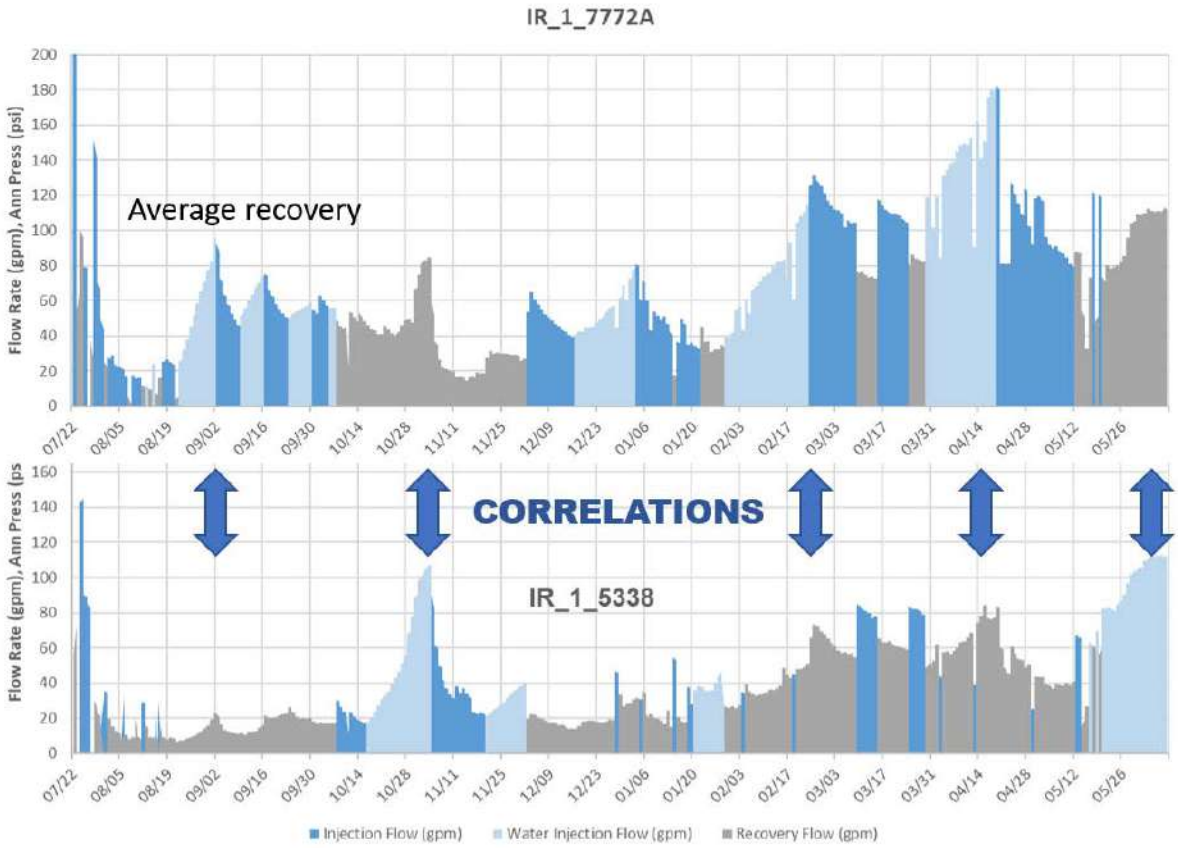


Figure 13-7: Wells 5269 and 5270 Showing Water Flushing to Improve Flow Rates in the Latter Part of 2021

In continued evaluation of the wellfield, Excelsior have demonstrated that recovery of flow also improves if adjacent wells are similarly flushed, indicating connectivity in the flow paths and the need to maintain this to expel CO_2 (Figure 13-8).



Source: Excelsior

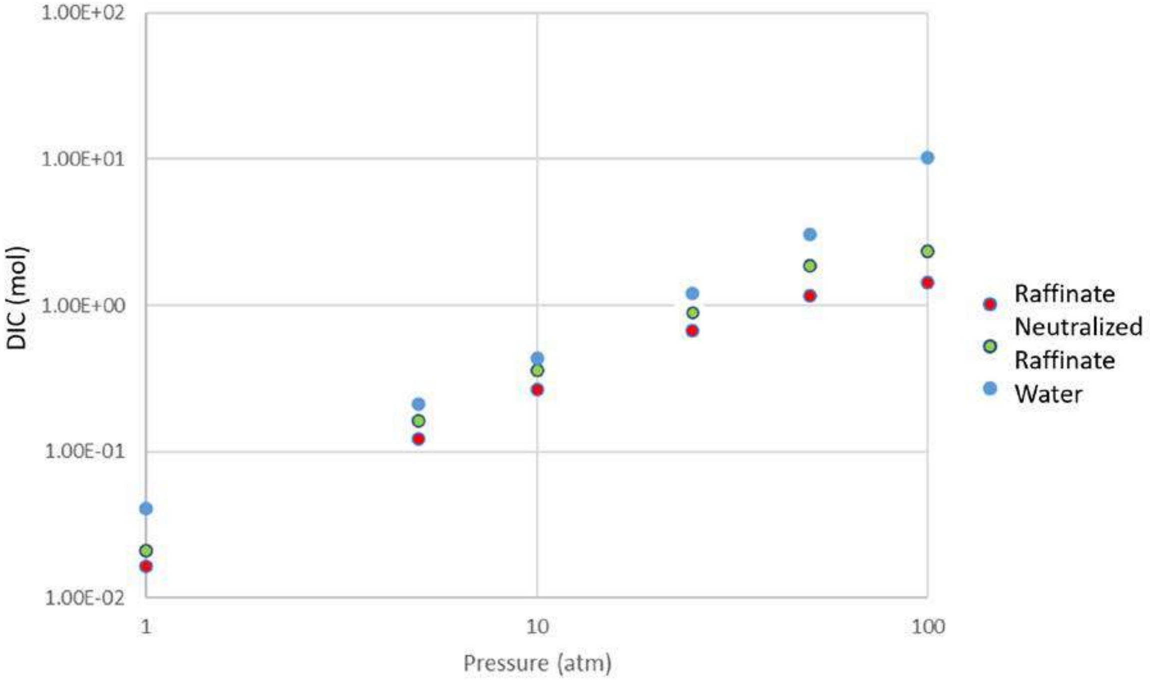
Figure 13-8: Recovery Relationship in Connected Wells

In general, the data indicate flow rates can be improved with repeated cycling of freshwater injection, acid injection and recovery. During periods of freshwater injection, copper grades in solution decline due to dilution by the fresh water. Periods of acid injection, particularly at higher flow rates, then dissolve calcite and, over time, the calcite within the fracture system will be removed and CO_2 flow restrictions will cease along those flow paths.

13.2.3.1.4 Neutralized Raffinate as an Alternative to Water Flushing

At high pressure, some dissolution of CO_2 will occur, and this increases rapidly, both with pH and pressure (Figure 13-9). In Figure 13-9, the dissolved concentration (in mols) of dissolved inorganic carbon has been calculated for three solutions.

- Raffinate (in red) at pH 1.2
- Raffinate neutralized to pH 7 (predicted calculation in PHREEQC, an equilibrium thermodynamic software)
- Make-up water at pH 7



Calculated dissolved inorganic carbon concentration (DIC), in mols, at varying pressure as a function of water pH for make-up water and raffinate at pH 1.2 and at pH 7 (raffinate neutralized). Source: SRK calculation in PHREEQC

Figure 13-9: Calculated Dissolved Inorganic Carbon Concentration (DIC), in mols, at Varying Pressure as a Function of Water pH

Results of laboratory experiments when multiple crop species are irrigated with modified rainwater are presented. The solubility of CO_2 in rainwater is highly sensitive to the pH of rainwater as indicated in Figure 13-10 (Yunge, 1965). The figure indicates that CO_2 solubility decreases rapidly and non-linearly when water becomes acidic. It is assumed that a similar relationship occurs in groundwater and at extremely the low pH expected in the ISR wellfields.

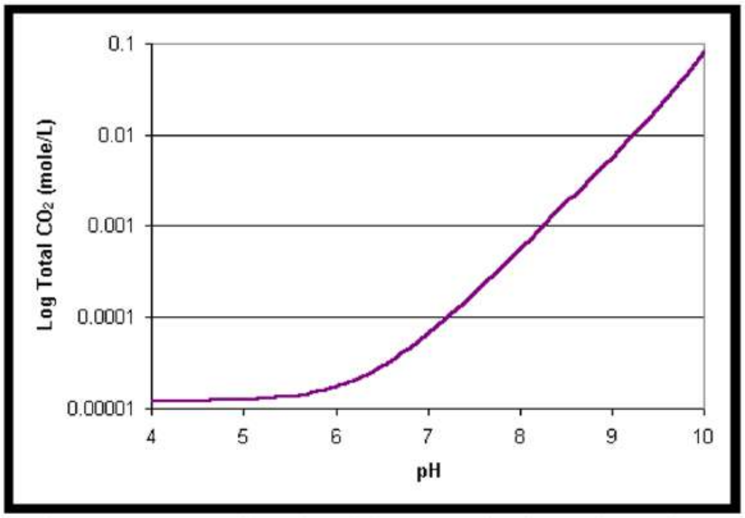


Figure 13-10: CO_2 Dissolved in Water in Function of pH (Yunge, 1965)

13.2.3.1.5 Physical Aspects of CO_2 Gas Blocking

Basics of Two-Phase Flow in Porous Media

It has been well documented through theory and experiment that multiple immiscible phases flowing intermingled through a porous medium will generally do so at lower effective rates than either phase flowing on its own. According to relative permeability theory, the higher the saturation of one immiscible flowing phase (as a fraction of the connected pore space), the lower the effective permeability of the other phase. The magnitude of this reduction is generally larger for the wetting phase, which is acid/water for the Gunnison case. The relative viscosities of the phases, the interfacial tension between them, and their relative affinities for contact with the solid surface of the porous medium (capillarity and wettability) all play significant roles in the multi-phase flow process.

In the case of a non-zero interfacial tension (miscibility), each phase will generally establish an “immobile” saturation below which it cannot flow due to capillary pressure and interfacial tension effects. Typically, the phase with the greater affinity for the solid surface (called the wetting phase) will have a higher immobile saturation than that of the non-wetting phase.

Also, the wetting phase will tend to occupy the smaller pores in the system. The non-wetting phase, CO_2 in the case of Gunnison, will have a reduced ability to enter smaller pores and fracture apertures pores (up to the limit of a large aperture) for a given displacement pressure. The pressure required to cause a volume of non-wetting fluid to flow into a pore of a specific cross-sectional area is called the “entry pressure”. These phenomena are meaningful in natural fractures as well as intergranular pores in a rock system within a confined aquifer. As aperture size increases, or if the aquifer is unconfined, then that allows the fluids to separate and to flow independently.

Multi-phase flow effects could manifest themselves in several ways that could be detrimental to recovery of copper in an ISR process wherein calcite and acid can react to produce gaseous CO_2. These include the change in relative permeability where local, high concentrations of CO_2 gas can impede the flow of lixiviant as the gas occupies more of the cross-sectional area available for liquid flow.

Phase trapping of CO_2 can occur when certain volumes of CO_2 gas can become trapped in the micro-fractures and consume available pore spaces surrounded by liquid. These non-wetting phase volumes, called ganglia, can become trapped due to interfacial tension effects with the net effect of impeding acid-rock contact for the rock surfaces they cover.

Observations of Asymmetric Injectivity Between Acid and Freshwater Cycles

It has long been observed by Excelsior operations staff that the injectivity (injection rate per unit pressure) of acidic lixiviant is significantly lower than that for fresh water. This pattern is seen consistently across many wells and in the repeated cycling between acid and freshwater injection in a singular well. The Excelsior SCADA system collects rate and pressure data from all wells in the wellfield, allowing clear-cut documentation of the phenomenon by Excelsior technical staff. Regarding multiple cycles of acid and freshwater in a single well, the details of the injection asymmetry are also very repeatable.

At the onset of acid injection following a freshwater cycle, the flushing rate approaches 2,500 gpm. In fact, it can, for a very short period, exceed that of the tail end of the water cycle. This may be attributable to the acid dissolving away scale that has formed on the wellbore rockface during water injection. However, in a very short time, this rate drops-off to about a third of the initial injection rate. When the absolute injection rate of acid reaches a point of limited economic benefit, the well is generally switched to freshwater injection. Water injectivity starts out low but quickly increases roughly linearly with cumulative volume injected. In many cases, virtually all of the well-formation injection capacity lost during acid injection is fully restored. In cases wherein restored injectivity falls short of that level, some of the two-phase flow effects discussed earlier could be active.

Because the pH environment in the mixing zone between fresh water and the previously injected acid is unknown and likely varied, it may be difficult to describe all eventualities for formation of precipitates in a chemically complex, intermediate pH system. Additional flow and reaction studies might answer this question.

Contrary to the low solubility, hard precipitate scenarios described above, freshwater injection is consistent with restored injectivity if the “damage” mechanism is CO_2 gas blocking. Fresh water, equilibrated with air at the surface, will contain little dissolved carbon dioxide and, therefore, has some capacity to redissolve gaseous CO_2 when injected into the orebody under pressure. Thus, one mechanism of CO_2 gas blockage removal is simple solubilization. Fresh water has a favourable mobility ratio with CO_2 owing to the former’s higher viscosity. The leading edge of the injected waterfront should have the ability to displace gaseous CO_2 in a piston-like manor. The similar viscosity of acid may be negated in this same regard because acid would continue to produce CO_2 from calcite in quantities that inhibit flow.

Finally, gravity segregation of CO_2 and water in the fracture network (again because no new CO_2 is produced) may allow water to “under run” CO_2 as it migrates to the top of the orebody, adding some additional restoration of trans-pattern flow. This phenomenon is often seen in CO_2 floods and other gas floods performed in the energy industry to enhance the recovery of oil.

In summary, CO_2 dissolution, frontal displacement, and gravity segregation are all mechanisms consistent with removal of CO_2 from the orebody and restoration of injectivity via freshwater flushing.

Quantification of Calcite Removal from the Wellfield

Acids are frequently used in oil and gas operations, both as a means of removing near-wellbore formation damage and for creating etched tunnels into the surrounding rock. Both effects increase well-to-formation flow capacity. Typically, hydrochloric acid (HCl) is used in these operations. When acidizing is carried out in carbonate-bearing formations, CO_2 is created as a reaction by-product just as it is in sulfuric acid- based ISR projects when calcite is present. In this case, however, the other reaction by-product is water-soluble calcium chloride.

Acid stimulation of carbonates aims to create acid dissolution tunnels (termed wormholes) that penetrate radially from a wellbore to the furthest possible distance and with a minimum of unproductive spending of acid. In this sense, the process is analogous to the ISR process goal of efficiently penetrating the copper-bearing formation with low-pH lixiviant while avoiding the excess spending of lixiviant on low-grade formation minerals.

While it is impossible to directly observe the “worm holing” process in a commercial ISR field operation, substantial experimental work has been performed to optimize acid job performance parameters, including study of the evolved CO_2 (Qiu et al., 2014). Carbonate core flood acidizing experiments have shown that excessive CO_2 formation can lead to “gas blocking” i.e., restricting flow of fresh acid to the tip of the wormhole tunnel with an associated slowing of tunnel length growth. Instead, the blocked acid reacts with the carbonate walls of the tunnel. This “spending” of the acid enlarges wormhole diameters but does so unproductively, as beyond a certain diameter, further increases do not substantially enhance wellbore performance. The observed gas blocking effect was most pronounced at lower pressures at which free gas volumes are expected to be larger. It has been hypothesized that the blocking effect is produced largely by CO_2 gas bubbles that have been able to coalesce into a continuous gas phase which is displaced en-masse to the tunnel tip.

As in the oil and gas industry’s field acidizing treatments, the evolved CO_2 blocking effect has not been observed directly in the Excelsior lixiviant injection process. However, the quantifiable reduction in injectivity of low pH lixiviant in the Gunnison project is highly consistent with a CO_2 gas blocking process owing to both the known presence of calcite in the formation and the relatively low pressures in the orebody system.

In mid-2021, Excelsior embarked on a program to test the feasibility of “mining” calcite out of the major flow pathways between wells in an area at the center of the wellfield. The concept was to determine if a “push-pull” operating strategy would ultimately remove enough calcite to allow stronger acid solutions to be injected on a continuous basis with less loss of injectivity and improved copper recovery efficiency.

The “push-pull” procedure cycled between acid injection and a period of back-production, which in turn, was followed by fresh water flushing before returning to acid injection. The volumes of fluid injected per cycle were relatively small. These were varied in size/duration in response to changes in injectivity trends.

Conceptually, the early acid injection cycles would encounter calcite closer to the wellbore, dissolve some of it, creating some CO_2. Putting the well on production for a period would back-produce calcite in the form of dissolved CO_2, gaseous CO_2, and dissolved calcium minerals. It is recognized that some of the dissolved calcium might convert to gypsum and deposit on calcite mineral surfaces. This latter process is considered beneficial to the effort, as gypsum deposition tends to “armour” calcite against rapid dissolution by subsequent volumes of acid. It is also conjectured that some gaseous CO_2 remains in-situ following flow-back.

The freshwater injection cycles would, in principle, remove additional CO_2 and restore injectivity via the mechanisms discussed above. If the calcite mining approach is operative as envisioned, several trends in the injection/production cycles should emerge:

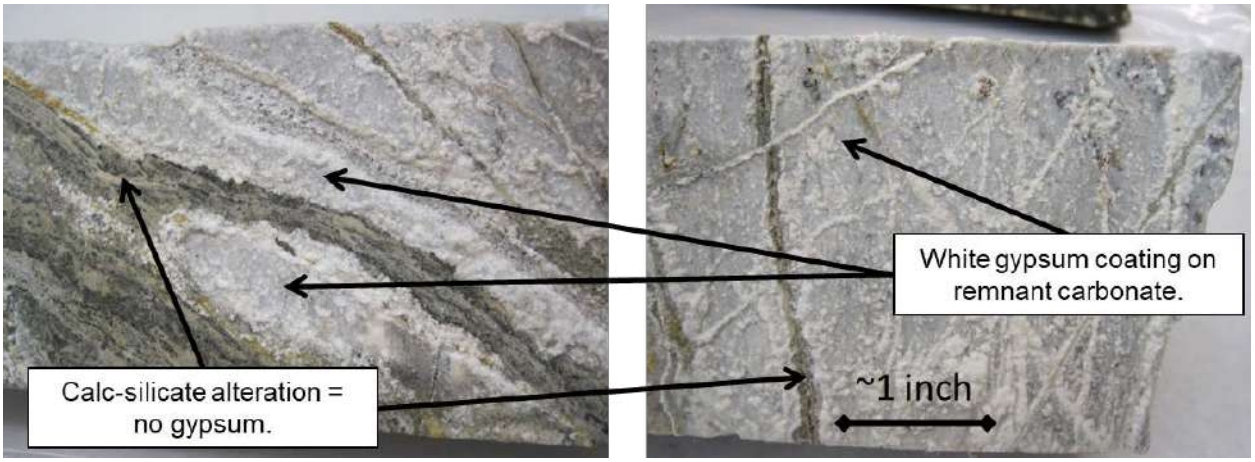
- As acid injection cycles progress, there should be a trend towards higher injectivity as near-wellbore calcite becomes depleted and the mass of calcite in mid-pattern flow paths is diminished.
- Fresh water injection cycles should evidence progressively higher initial rates. As calcite removal proceeds, there should be more pathways open to allow removal of CO_2 as well as requiring a larger volume ratio of fresh water to CO_2 in the fracture system.
- Eventually, the injectivity of the acid and freshwater cycles should approach the same value.
- Certain trends in the back-produced mineral content and pH might also appear.

The calcite mining project for the initial set of wells was completed in September 2021. Many of the projected trends in injection cycle behaviour were, in fact, observed. Injection of acid in that area is now possible without occurrence of the historically rapid losses of flow capacity.

13.2.3.1.6 Gypsum

Gypsum ($CaSO_4.2H_2O$) is a product of the reaction between calcite and acid (Equation 1). As the calcite dissolves, it liberates calcium ions that quickly join with the excess sulfate ions to form gypsum. Once formed, gypsum is mostly insoluble and its limited solubility is not pH dependent, meaning acidic solutions wont re-dissolve it like most other mineral precipitates.

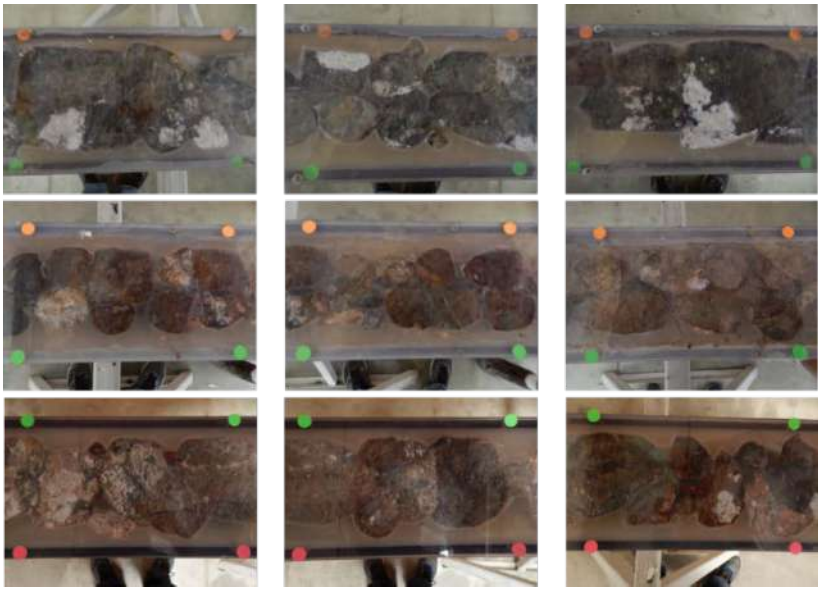
Excelsior completed certain metallurgical tests under the supervision of Dr. Ron Roman that were designed to semi-quantitatively examine acid consumptions and copper recovery of epoxy coated drill core as compared to non-epoxy coated drill core (see section 13.2.2.4 Bucket Tests). Photographs of the non-epoxy coated drill core showed that gypsum pseudomorphs calcite when the calcite is exposed to acid (Figure 13-11).



Samples from laboratory testing (see Section 13.2.3.4) completed by Excelsior show the coating and replacement of remnant calcite or dolomite by gypsum (white mineral in the photographs). Replacement is highly selective at a scale of less than 1 mm. Source: Excelsior

Figure 13-11: Samples from Laboratory Testing Completed by Excelsior

In other testing (see Section 13.2.2.3 Core Tray Tests), Excelsior observed the formation of gypsum on the core samples that were imbedded in the acid resistant epoxy (Figure 13-12).



Typical occurrence and distribution of gypsum (white or light-colored minerals) in post leach photographs from core-box leach tests. Samples are from Martin, Upper Abrigo and Lower Abrigo formations. Dots of the same color are approximately 12 inches apart. Source: Excelsior

Figure 13-12: Typical Occurrence and Distribution of Gypsum (white or light-colored minerals) in Post Leach Photographs from Core-Box Leach Tests

Post-leaching the lid was removed from these boxes and all decrepitated rock and precipitates were scraped off (Figure 13-13) and sent away for quantitative electron microscope scanning (QEMScan) at the Colorado School of Mines to identify the chemical and mineral phases.

Source: SRK

Figure 13-13: Image of the surface of a sample from the box leach tests after the scraping and removal for sampling of decrepitated rock and precipitates. Field of view is approximately 10 cm (4 inches).

The amount of gypsum formed was a function of the amount of calcite on the fracture surfaces or in exposed veinlets. This is variable and somewhat dependent on the rock types, with typically more calcite and hence gypsum in the Martin and Upper Abrigo formations.

The quantitative electron microscope scanning measured and calculated the weight percent of gypsum in the scraped off leach reject from the core box tests. The amount (moles) of gypsum formed (precipitated) was then stoichiometrically compared to the amount (moles) of acid consumed to make that gypsum (assumes all the sulfate in the gypsum came from the sulfuric acid). The average amount of acid consumed (grams or moles) to make gypsum can be expressed as a percentage of the total amount of acid consumed (grams or moles) for each major rock type used in the core box tests (Table 13-1).

Table 13-1: Percentage of Total Acid Consumed that is Converted to Gypsum for the Main Rock Types in the Core Box Tests

Formation (Rock Type)	Approximate Percentage of total acid consumed that used to make gypsum
Martin	Formation (Rock Type)
Upper Abrigo	Martin
Middle Abrigo	Upper Abrigo
Lower Abrigo	Middle Abrigo

Table 13-1 shows that around 3% to 20% of all acid consumed during the core box leaching was converted to gypsum. This is consistent with the geological estimates of around 3% to 20% calcite in the fracture system and is consistent with the qualitative visual distribution of gypsum post-leach in the box tests (e. g., Figure 13-11). Other gangue minerals containing magnesium, sodium, potassium, and aluminum would have dissolved and formed the water-soluble sulfates of those alkali elements.

As calcite dissolves it creates additional void space (increasing porosity and aperture)., However, as gypsum forms, it reduces porosity and aperture. The volume of one mole of gypsum (density 2.36 g/cc) is about two times the volume of one mole of calcite (density 2.71 g/cc), so the net effect on volume occurs in a ratio of about 2 to 1. As an example, if the porosity of a volume of rock is about 3% (approximate average of the Gunnison deposit), and if 20% of that rock was calcite, then due to the replacement of that calcite by gypsum, the porosity would be reduced by about 20%. The spatial distribution of calcite and porosity can be quite variable and there are likely situations where the porosity reduction is much greater than (or less than) 20%. Nevertheless, on a purely mass, mole and volumetric analysis, the porosity reduction over time by gypsum (e.g., 3% porosity converting to 2.4% porosity) would not appear to be a significant cause of the flow reductions observed at Gunnison when compared with the potential for flow restrictions that would be caused by the much larger volume of CO_2 gas. Additionally, gypsum is not readily soluble in water, so the observed flow improvements when injecting with water are not the result of dissolution of gypsum. Ergo, the flow reductions that are reversable with water injection cannot be due to the formation and dissolution of gypsum.

13.2.3.1.7 Summary and Conclusion

Excelsior has observed rapidly declining flow rates in wells when injected with acidic solution. Switching injection to fresh water restores the lost flow at approximately the same rate at which it declined. Several possibilities have been considered for the primary cause of this behavior, including mineral precipitates, swelling clays, and gas formation.

Based on the following points, it is considered that CO_2 gas bubbles are the primary cause of decline during acid injection.

- Excessive amounts of CO_2 gas bubbles coming off the wells
- Chemistry is consistent with the reaction of calcite with the acid forming CO_2 gas
- Calcite is commonly observed in the fracture system
- The reactions are supported by geochemical modelling
- Reversibility on water injection supports CO_2 dissolution and does not support gypsum dissolution (mostly insoluble in water), or other typical minerals precipitates that dissolve in low pH (acid) and precipitate in high pH (water)
- Behavior is consistent with gas blocking observed in the oil & gas industry and is supported by the physical processes of two-phase flow or blockage (gas and liquid)

The CO_2 gas is coming from the reaction of secondary calcite in the fracture system with the acidic solution being injected. On this basis, once the calcite has been completely dissolved or removed, no more CO_2 gas should form. This has occurred along the flow-paths of at least one well within the wellfield (7772A), which no longer declines on acid injection. It took approximately 6 to 9 months of cycling fresh water and acid injection to achieve this outcome.

Wells that have had some freshwater flushing have exhibited a similar response of significant improvement in flow rate on water and a declining flow rate on acid. Excelsior has shown cycling water, acid and recovery at regular intervals generates a long-term, positive, and sustainable, improvement in flow rates.

Geochemical modelling and literature regarding CO_2 sequestration in saline waters indicate that neutralized raffinate has a similar, although somewhat reduced, capacity to dissolve/remove CO_2 gas as does freshwater. Excelsior's proposal to use neutralized raffinate to flush out CO_2 and dissolve calcite on a cyclical basis over a 12-to-15-month period is supported by the limited wellfield data available to date and is supported by the chemistry of raffinate versus water's ability to sequester CO_2.

13.2.4 Recommendations for Future Process Development

If the current plan of alternately flushing the mineralized and partially leached formation with water and acidified raffinate is not effective, laboratory investigation of the following alternative lixiviants is recommended:

1) Sulfurous acid, H_2SO_3, is standard reagent for assaying ASCu, and it can be synthesized by dissolving sulfur dioxide in water. It was tested by Hazen Research and was found to be effective in dissolving non-sulfide copper from crushed core at ambient temperature and pressure. It should not react very aggressively with calcite, but if there is a reaction, the product is calcium sulfite, $CaSO_3$, which is fairly soluble in water, but also is likely to be readily dissolved at low free acid concentrations.
2) Ammonium carbonate, $(NH_4)_2CO_3$, with excess aqueous ammonia, NH_4OH, will dissolve non-sulfide copper minerals, including chrysocolla, but this lixiviant reacts slowly, forming the cupric amine complex, $Cu(NH_3)_4CO_3$. There is no reaction with calcite. A reducing roast accelerated the leaching kinetics of the Twin Buttes mineralization, but applicability of direct leaching of the Gunnison mineralization would have to be examined in the laboratory. As mentioned earlier, MSRDI did not obtain encouraging results, but a few tests may be justified, either to rule out this option or to reveal a potentially fruitful avenue. For instance, increased pressure to simulate wellfield conditions might improve kinetics. Anaconda conducted an extensive pilot plant campaign on Twin Buttes, AZ, "oxide" ore in the early-1970s, and intended to commercialize the process. However, cheap sulfuric acid from the new San Manuel smelter acid plant became available, so a 10,000 ton/day fine grind/acid leach/CCD-SX-EW plant was built instead. It typically consumed 500-1,000 tons of concentrated sulfuric acid daily.

   Ammonium carbonate leaching yields a pregnant solution that can either be concentrated by SX or treated with carbon dioxide, typically produced with a natural gas-fired submerged combustion evaporator, to precipitate copper carbonate, $CuCO_3$, that can either be dried and sold or dissolved in sulfuric acid and reduced to cathode copper by conventional SX-EW.

3) Ammonium sulfate with oxygen-enriched air will dissolve all copper minerals, both non-sulfide and sulfide, although chalcopyrite is leached more slowly than secondary sulfides such as chalcocite. Pregnant leach solutions are readily treatable with reagents like LIX 65-N that do not transfer ammonia to the electrolyte.

4) Researchers in a joint venture of Curtin University with MPS in Perth have developed the amino acid, *glycine*, as a lixiviant for both precious and base non-ferrous metals. Glycine has the formula, $C_2H_5NO_2$, with the organic structure, NH_2-CH_2-COOH. Since glycine is widely used as a human dietary supplement, it offers obvious advantages from a permitting standpoint. Preliminary test work using glycine should be carried out on Gunnison mineralization.

In addition to a laboratory investigation of alternative lixiviants, a comprehensive study could be launched into potential crystal habit modifiers for gypsum. Crystallization is a complex phenomenon and many compounds exhibit a wide range of crystal morphologies, depending on crystallization conditions. Physical variables include degree of saturation, temperature, solution composition (free acid, for instance), and agitation intensity (flow velocity in the case of in-situ leaching). However, crystal habit modifiers may also be important, and they can include both inorganic and organic additives. The objective might, for instance, be encouragement of gypsum precipitation as fine dense crystallites, rather than fluffy deposits, thereby encouraging removal by entrainment and settling/filtration from the PLS.

Testing and/or modelling should be carried out to determine the effects of weaker acid strengths on CO_2 formation and/or declining flow rate declines.

14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

The mineral resources reported herein are unchanged from those previously reported (M3, 2017). Jeffrey Bickel, C.P.G., of RESPEC, is the Qualified Person responsible for the mineral resources reported in this report. Mr. Bickel is independent of Excelsior by the definitions and criteria set forth in the Canadian National Instrument 43-101 ("NI 43-101"); there is no affiliation between Mr. Bickel and Excelsior except that of an independent consultant/client relationship. Modeling and estimation of the mineral resources of the North Star deposit were originally completed under the supervision of Michael M. Gustin, also of RESPEC. Mr. Bickel has reviewed the mineral resource estimate completed by Mr. Gustin, pertinent data and inputs to the model, and determined them to be good and adequate.

Although Mr. Bickel is not an expert with respect to any of the following topics, as the date of this report, he is not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the North Star mineral resources and that are not otherwise discussed in this report.

The North Star resources are classified in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories in accordance with the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" and therefore Canadian National Instrument 43-101. CIM mineral resource definitions are given below, with CIM's explanatory text shown in italics:

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.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, *application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.*

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.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drillholes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Prefeasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

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.

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

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. Measured mineral resources may be converted to a Proven Mineral Reserve by designing and scheduling the Measured mineral resources into the mine plan and having positive economics.

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

14.2 RESOURCE MODELING

14.2.1 Data

The North Star copper resources were modeled and estimated using data generated primarily by Excelsior, including information derived from core, reverse circulation, and conventional rotary drillholes. Additional drill data used in the modeling were derived from historical exploration programs completed by Cyprus Minerals, Superior Minerals, Quintana Minerals, Magma Copper, Phelps Dodge, Minerals Exploration, and James Sullivan. No holes were drilled subsequent to the previously reported resources (M3, 2016). These data, as well as digital topography of the project area, were provided to RESPEC by Excelsior in a digital database in Arizona State Plane, East Zone coordinates in US Survey feet using the NAD27 datum. This database is summarized in more detail in Section 10.

All modeling of the North Star deposit resources was performed using proprietary software developed at RESPEC as well as GEOVIA Surpac™ mining software. The North Star resource block-model extents and block dimensions are provided in Table 14-1.

Table 14-1: Block Model Summary

In Feet (ft.)	x	y	z
Min Coordinates	529,000	384,750	0
Max Coordinates	549,450	398,250	5,200
Block Size	50	100	25
Rotation	0	0	0

The project database includes drillhole information from holes drilled immediately adjacent to lands controlled by Excelsior. The modeling of the project resources incorporated the results from these holes, but the reported project mineral resources include only modeled mineralization that lies within Excelsior-controlled lands.

14.2.2 Deposit Geology Pertinent to Resource Modeling

The North Star copper mineralization occurs primarily in Paleozoic sedimentary units adjacent to the Texas Canyon Quartz Monzonite, although the quartz monzonite and Precambrian rocks host minor quantities of mineralization as well. The primary controls on the North Star mineralization include: (i) proximity to the Texas Canyon Quartz Monzonite; (ii) carbonate-bearing stratigraphic units altered to various calc-silicate/skarn mineral assemblages; and (iii) the degree of fracturing. The development of primary copper-sulfide skarn mineralization is related to the proximity to the intrusion. The skarn mineralization preferentially developed in carbonate-bearing units, with the combination of this and proximity to the intrusion leading to the Martin and Abrigo Formations being the primary host units. Fracture intensity is controlled by two factors: fracturing related to volume loss during skarn development and fracturing related to pre- and post-mineral faulting. The effects of oxidation overprint the primary copper mineralization to depths of approximately 1,600 feet.

Geologic factors critical to the modeling of the North Star copper mineralization therefore include lithology, structure, and oxidation.

14.2.3 Modeling of Geology

Excelsior completed stratigraphic interpretations on a set of east-west vertical cross sections that were used for all modeling of the North Star deposit. These sections are spaced at 100-foot intervals over a north-south extent of 9,000 feet, which covers the resource area, with four 500-foot spaced sections appended to the north and south of the 100-foot sections. The stratigraphic units modeled on the cross sections include the Naco Group, Escabrosa Limestone, Martin Formation, Abrigo Formation (subdivided into the upper, middle, and lower units), Bolsa Quartzite, undivided Precambrian rocks (including the Pinal Schist and Apache Group), Texas Canyon Quartz Monzonite, and Tertiary/Quaternary basin fill. The Excelsior stratigraphic cross sections were used to assign a single lithologic code to each block in the model (Figure 14-1 and Figure 14-2).

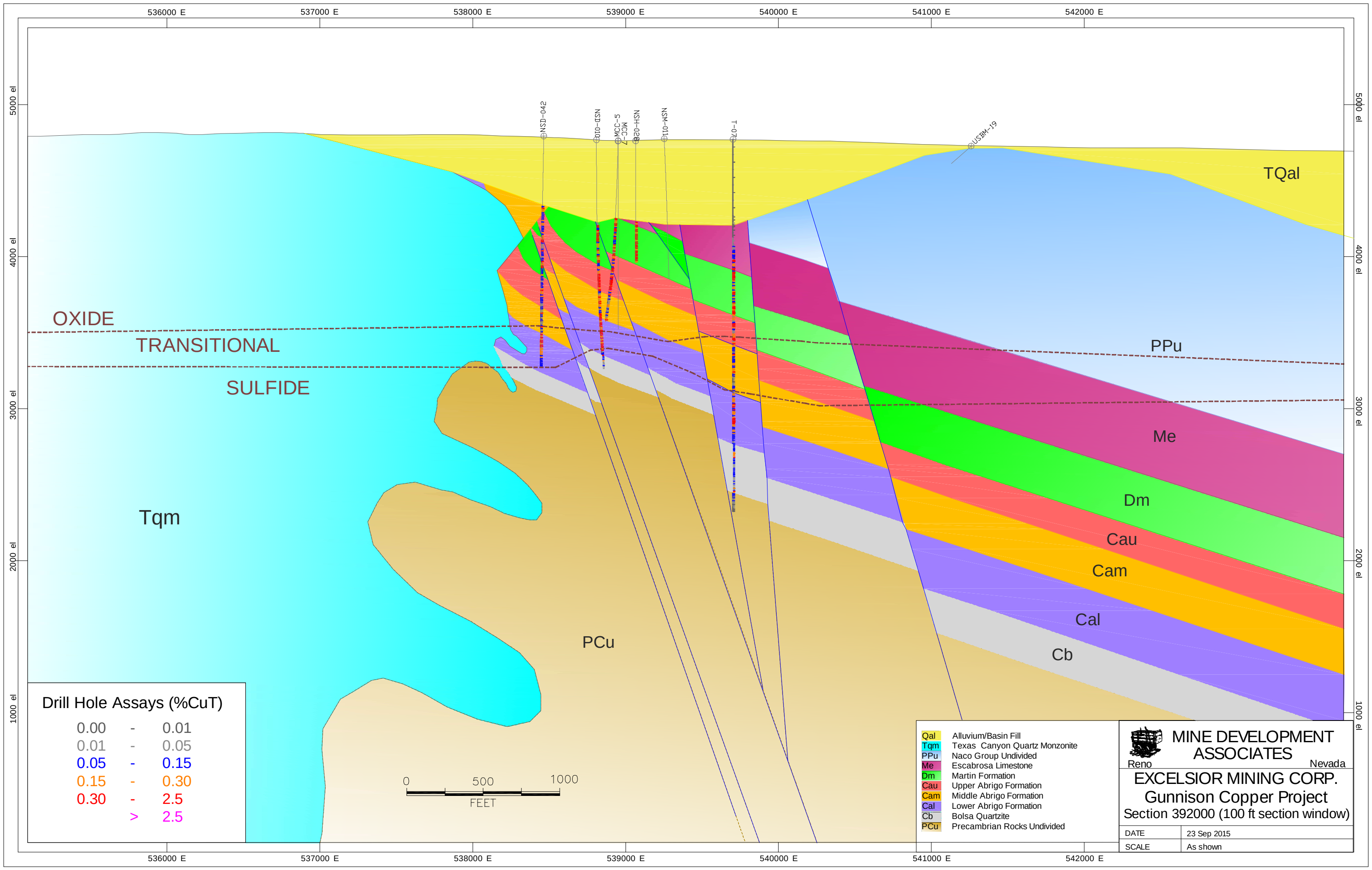


Figure 14-1: Cross Section 392000N Showing North Star Geologic Model

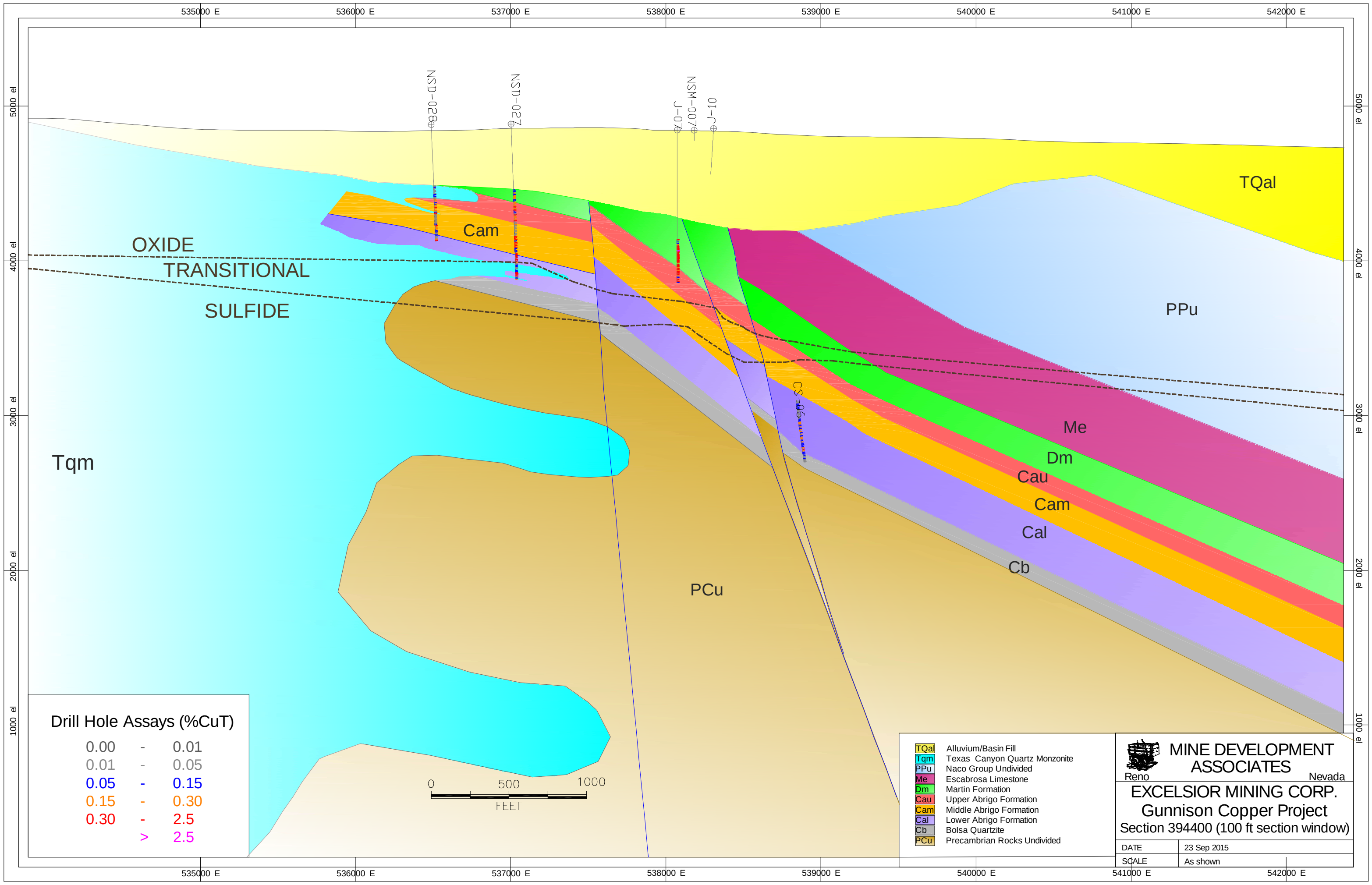


Figure 14-2: Cross Section 394400N Showing North Star Geologic Model

As part of the geologic modeling, Excelsior also completed detailed structural interpretations. A total of 61 individual structural domains were modeled as three-dimensional wire-framed solids (Figure 14-3). These solids were used to code model blocks to each of the 61 modeled structural domains. A block that encompasses any volume of one of the structural domains was assigned the code of that domain, which effectively expands the volumes of the structural domains from those represented by the structural solids.

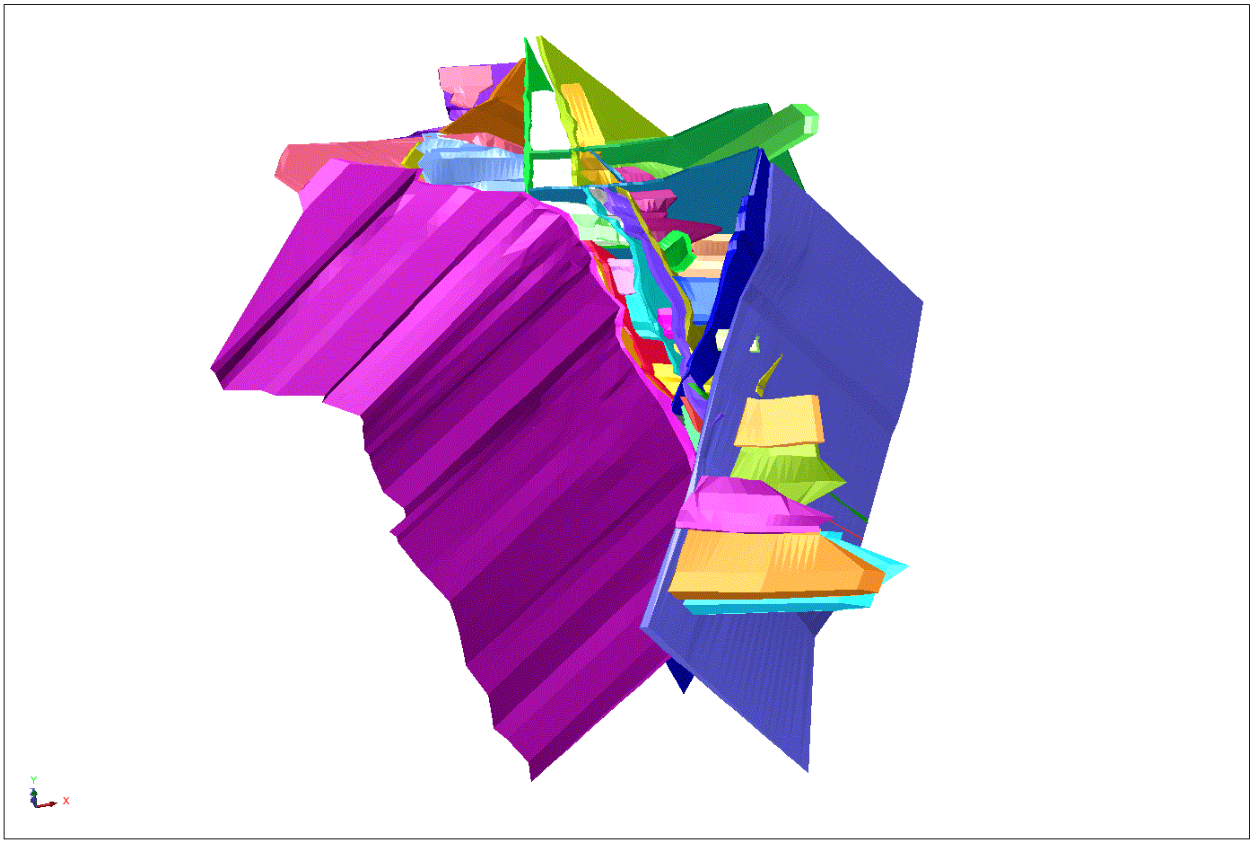


Figure 14-3: Oblique Northerly View of Structural – Domain Wire – Frame Solids

14.2.4 Oxidation Modeling

Using drillhole logging and copper sequential-leach data total copper ("TCu"), acid-soluble copper ("ASCu"), and cyanide-soluble copper ("CNCu"), Excelsior modeled both the base of more-or-less complete oxidation and the bottom of oxidation/top of unoxidized materials on a set of 100-foot spaced, east-west vertical sections. In general, if the ASCu to TCu ratio was greater than or equal to 50%, the mineralization was assigned to oxide. If the ASCu to TCu ratio ranged between 49% to 20%, the mineralization was assigned as transitional material. These oxidation ratio rules were modified primarily by geological common sense.

The outcome of the modeling was to interpret three dimensional surfaces between oxide, transitional, and sulfide portions of the North Star deposit. The surfaces were then used to code each model block to one of the three oxidation zones.

14.2.5 Fracture – Intensity Modeling

Fracture intensity at the North Star deposit is defined based on geological logging and down-hole geophysical data. A relative fracture-intensity value was assigned to each logged interval in the project database on a scale of one to five, irrespective of the rock unit, with a value of "5" representing the most fractured rock (Table 14-2).

Table 14-2: Fracture – Intensity Scale

Intensity Code	Description **(% of Core ≤ 4 inches)**
1	Very Weak (0-5%)
2	Weak (5-20%)
3	Moderate (20-50%)
4	Strong (50-80%)
5	Very Strong (80-100%)

The wireframe solids discussed in Section 14.2.3 were used to code the fracture-intensity intervals in the project database to the structural domains. Fracture-intensity intervals lying outside of the structural domains were also assigned a code, leading to a total of 3,485 coded fracture-intensity intervals in the database, 26% of the intervals inside of the solids and the remainder outside. The intervals inside and outside of the structural domains have length-weighted mean fracture intensity values of 3.4 and 2.3, respectively.

The coded fracture-intensity values were composited to 25-foot lengths for use in inverse-distance-to-the-fifth-power interpolations of the fracture intensity into the resource-model blocks. All composites coded to the 61 structural domains were used for the interpolation of values into each of the structural domains coded into the model, and outside-domain composites were used to estimate the values in the remainder of the model. The inside-domain estimations used one of eight search-ellipse orientations to match the average strike and dip of each modeled structural domain. Fracture intensity values of the Paleozoic sedimentary units and Precambrian rocks outside of the structural domains were estimated using an ellipse that is consistent with the average strike and dip of the sedimentary units, while the Texas Canyon Quartz Monzonite was estimated using an isotropic search ellipse (Table 14-3). These search ellipses for fracture intensity were also used in the estimation of TCu grades and ASCu to TCu ratios ("ASCu/TCu"); see Table 14-11 for details of the search-ellipse orientations.

Table 14-3: Fracture – Intensity Estimation Parameters

Structural Domains, Paleozoic Sediments, Precambrian Rocks						
	Search - Ellipse Ranges (ft)			Composite Constraints		
Estimation Pass	Major	Semi-Major	Minor	Min	Max	Max/hole
1	700	700	233	4	10	4
2	1000	1000	333	1	10	4
Texas Canyon Quartz Monzonite						
	Search-Ellipse Ranges (ft)			Composite Constraints		
Estimation Pass	Major	Semi-Major	Minor	Min	Max	Max/hole
1	700	700	700	4	10	4

Figure 14-4 is an east-west cross section showing the fracture-intensity model in the deposit.

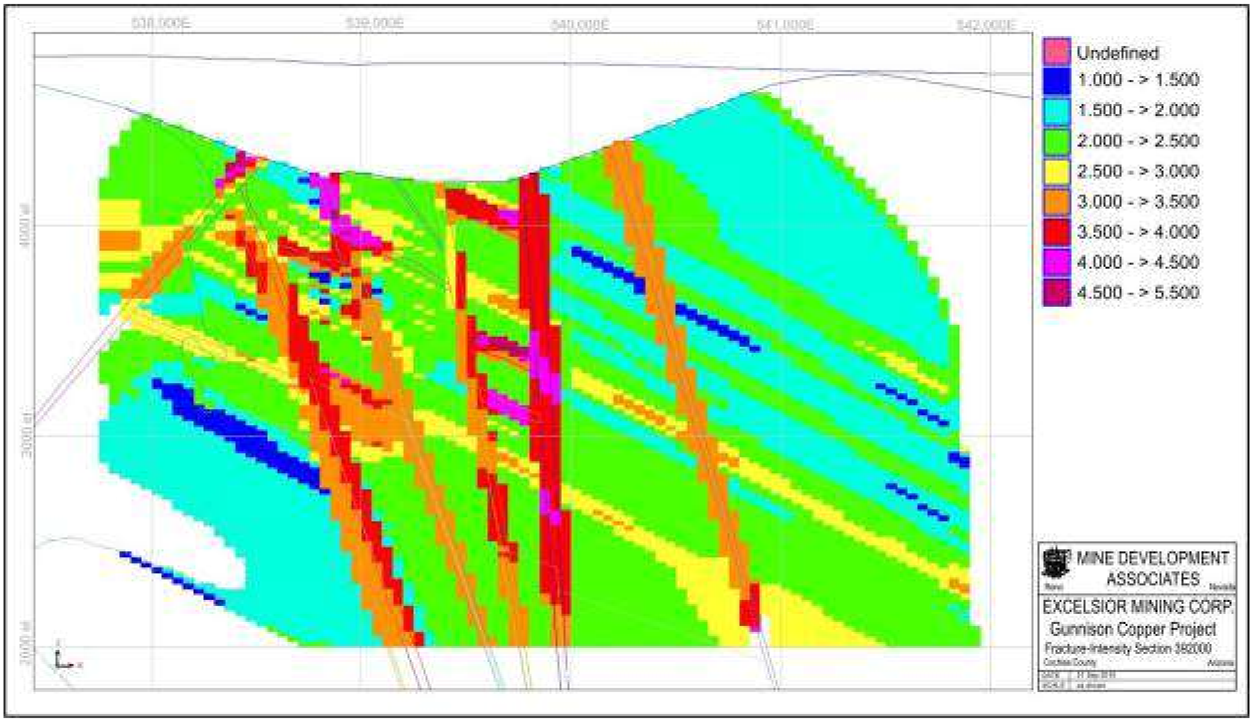


Figure 14-4: Fracture – Intensity Model Cross Section 392000N

14.2.6 Density Modeling

Specific-gravity ("SG") determinations were made by Excelsior for every assay sample in zones of mineralization and an additional 10 feet beyond the limits of each mineralized zone. The logging geologist determined where SG measurements were taken with regards to mineralized and non-mineralized materials; determinations were made on core from the NSD-series holes as well as the NSM-series metallurgical holes. The water-displacement method was used to determine the SG values using whole-core samples, which were not wrapped or waxed for the measurements. RESPEC notes that this methodology does not allow for the determination of actual in-situ bulk specific gravity in zones of highly broken core, because natural void spaces cannot be properly measured, leading to some overstatement of SG in these cases.

Model tonnage factors were assigned based on the combination of lithologic, oxidation, and total-copper mineral-domain coding of each block in the model. The TCu mineral-domain codes (discussed in Section 14.2.7) include domain 100 (low-grade), domain 200 (high-grade), or domain 0 (un-modeled/un-mineralized). Table 14-4 shows descriptive statistics of the underlying SG data by these categories, as well as the tonnage factors assigned to the model blocks (calculated from the SG means).

Table 14-4: Specific Gravity Statistics and Model Coding of Tonnage Factors

Unit	TCu Domain	Oxidation Zone	Specific Gravity					Tonnage Factor
			Mean	Median	Min	Max	Count	(ft^3/ton)
Qal	0	ox	2.5	2.54	2.28	2.74	17	12.81
Tqm	200	ox + trans	2.61	2.59	2.27	3.14	35	12.27
	100	ox + trans	2.57	2.58	2.33	3.06	115	12.47
	0	ox + trans	2.56	2.58	2.14	2.88	177	12.51
	100	unox	2.59	2.6	2.16	3.18	237	12.37
	0	unox	2.56	2.59	2.21	2.7	80	12.51
Ppu	100	ox + trans	2.72	2.67	2.58	3.47	27	11.78
	0	ox + trans	2.71	2.67	2.36	3.46	137	11.82
Me	200	ox + trans	2.96	3.04	2.03	3.58	63	10.82
	100	ox + trans	2.84	2.7	2.42	3.67	101	11.28
	0	ox + trans	2.68	2.66	2.26	3.69	125	11.95
Dm	200	ox + trans	2.79	2.76	2.18	3.81	478	11.48
	100	ox + trans	2.82	2.75	2.12	3.66	125	11.36
	0	ox + trans	2.72	2.71	1.97	4.23	444	11.78
	200	unox	2.9	2.85	2.6	3.25	31	11.05
	100	unox	2.85	2.86	2.46	3.21	26	11.24
	0	unox	2.86	2.85	2.7	3.11	10	11.2
Cau	200	ox + trans	2.82	2.83	2.14	3.75	337	11.36
	100	ox + trans	2.85	2.85	2.27	3.32	277	11.24
	0	ox + trans	2.75	2.77	2.07	3.54	332	11.65
	200	unox	2.98	2.99	2.46	4.11	89	10.75
	100	unox	2.88	2.87	2.44	3.42	59	11.12
	0	unox	2.85	2.81	2.42	3.43	42	11.24
Cam	200	ox + trans	2.85	2.81	2.1	4.55	368	11.24
	100	ox + trans	2.96	2.96	2.1	3.41	201	10.82
	0	ox + trans	2.91	2.88	2.24	3.84	239	11.01
	200	unox	2.9	2.89	2.38	3.65	81	11.05
	100	unox	2.98	2.96	2.47	3.48	79	10.75
	0	unox	3.05	3.03	2.41	3.67	177	10.5
Cal	200	ox + trans	2.71	2.7	1.79	3.72	269	11.82
	100	ox + trans	2.66	2.66	2.32	3.01	97	12.04
	0	ox + trans	2.66	2.66	2.34	3.01	32	12.04
	200	unox	2.75	2.73	2.15	3.59	472	11.65
	100	unox	2.72	2.69	2.3	3.57	293	11.78
	0	unox	2.81	2.76	2.42	3.41	90	11.4
Cb	100	ox + trans	2.75	2.64	2.61	3	3	11.65
	200	unox	2.62	2.61	2.47	2.9	30	12.23
	100	unox	2.64	2.64	2.31	3	173	12.14
	0	unox	2.63	2.62	2.48	2.99	48	12.18
Pcu	0	ox + trans	2.7	2.7	2.26	3.01	85	11.87
	200	unox	2.69	2.69	2.56	2.87	15	11.91
	100	unox	2.74	2.73	2.43	3.11	94	11.69
	0	unox	2.69	2.69	2.25	3.01	155	11.91

14.2.7 Total Copper and Acid – Soluble Copper Modeling

The North Star deposit mineral domains were modeled jointly by RESPEC and Excelsior to respect the detailed lithologic, structural, and oxidation modeling completed by Excelsior. Following a statistical evaluation of the drillhole copper data, TCu mineral domains were interpreted on 100-foot spaced, east-west vertical cross sections that span the 2.1-mile north-south and 1.3-mile east-west extents of the deposit. The TCu domains were then used to explicitly constrain the estimation of copper grades into 50 x 100 x 25-foot (x, y, z) model blocks using 20-foot composites and inverse-distance interpolation. The total copper grade estimation was further controlled by the incorporation of a number of unique search ellipses that reflect the various orientations of the modeled structural domains, as well as the strike and dip of the favorable stratigraphic units in areas outside the structural domains. The estimation of the ASCu/TCu ratios was constrained by modified versions of the TCu mineral domains, as well as by oxidation zone (oxide, transitional, and sulfide).

Mineral Domains. A mineral domain encompasses a volume of ground that is ideally characterized by a single, natural, population of a metal grade that occurs within a specific geologic environment. In order to define the mineral domains at the North Star deposit, the natural TCu grade populations were identified on population-distribution graphs for all drillhole samples in the North Star deposit area. This analysis led to the identification of low-grade and high-grade populations, with a gradational change between the two. Ideally, each of these populations can be correlated with specific geologic characteristics that are captured in the project database, which then can be used in conjunction with the grade populations to interpret the bounds of each of the TCu mineral domains. The approximate grade ranges of the low-(domain 100) and high- (domain 200) grade domains are listed in Table 14-5.

Table 14-5: Approximate Grade Ranges of Total – Copper Mineral Domains

Domain	Total Copper (%)
100	~0.01 to ~0.15
200	> ~0.15

Using these grade populations in conjunction with Excelsior's lithologic and structural interpretations, the North Star TCu mineralization was modeled by interpreting mineral-domain polygons on the set of 100-foot spaced cross sections described in Section 14.2.3. The interpretation of the TCu mineral-domain polygons was guided by the lithologic, structural, and fracture-intensity controls described in Section 14.2.2.

Representative cross sections showing the TCu mineral-domain interpretations are shown in Figure 14-5 and Figure 14-6.

As discussed further below, ASCu was not estimated directly into the block model, but was instead derived from the estimations of TCu grade and ASCu/TCu ratio. In addition to other constraints discussed below, an ASCu/TCu ratio domain was created to envelope an area of anomalously low ratios in the Paleozoic sedimentary rocks and Precambrian rocks within the oxide zone. This low-ratio mineral domain, interpreted on the project cross sections, models a low-ratio rind that more-or-less lies along the contact of the sedimentary units with the Texas Canyon Quartz Monzonite. This low-ratio contact zone appears to be related more to clay mineralogy than to oxidation.

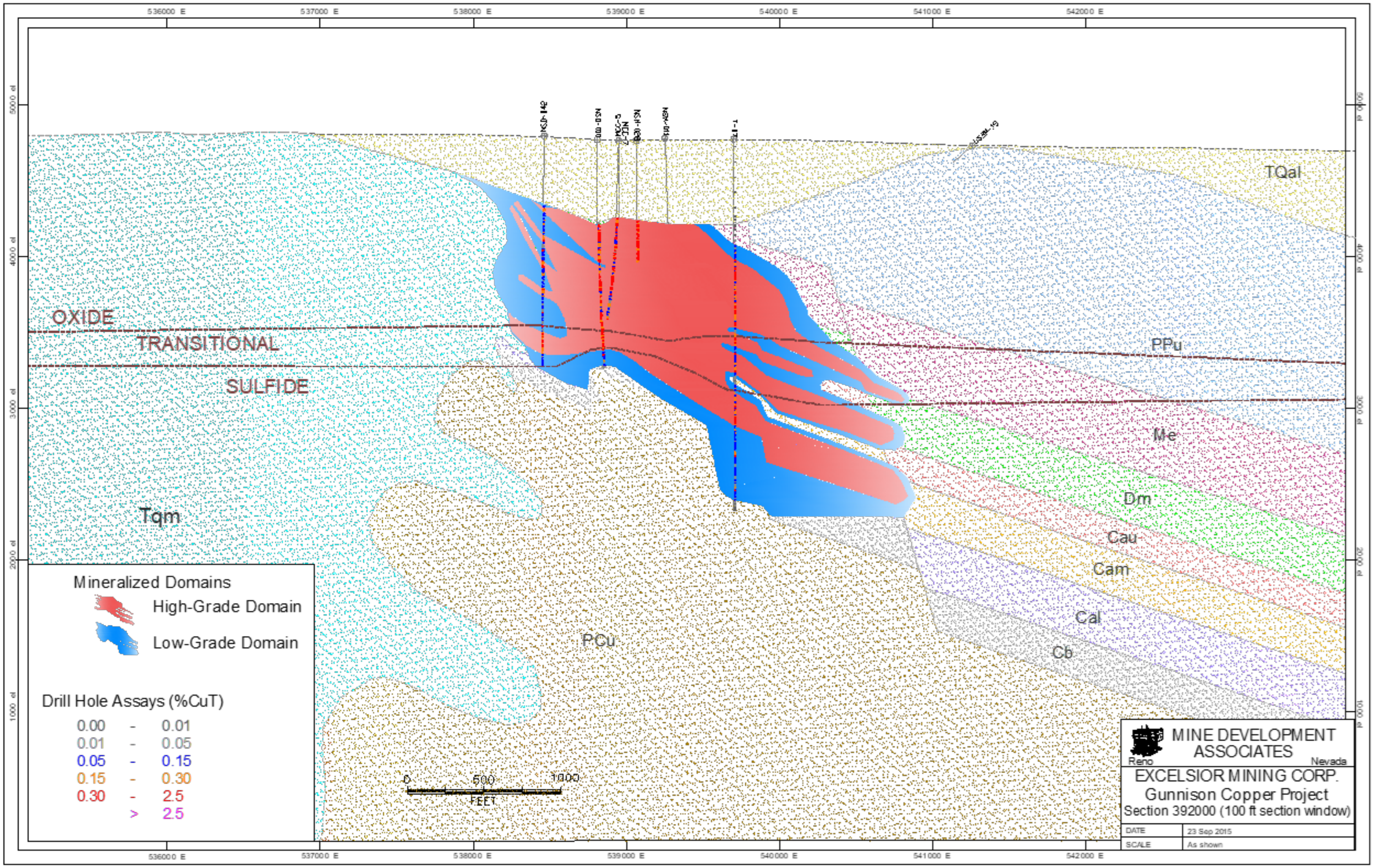


Figure 14-5: Cross Section 392000 N Showing Total – Copper Mineral Domains

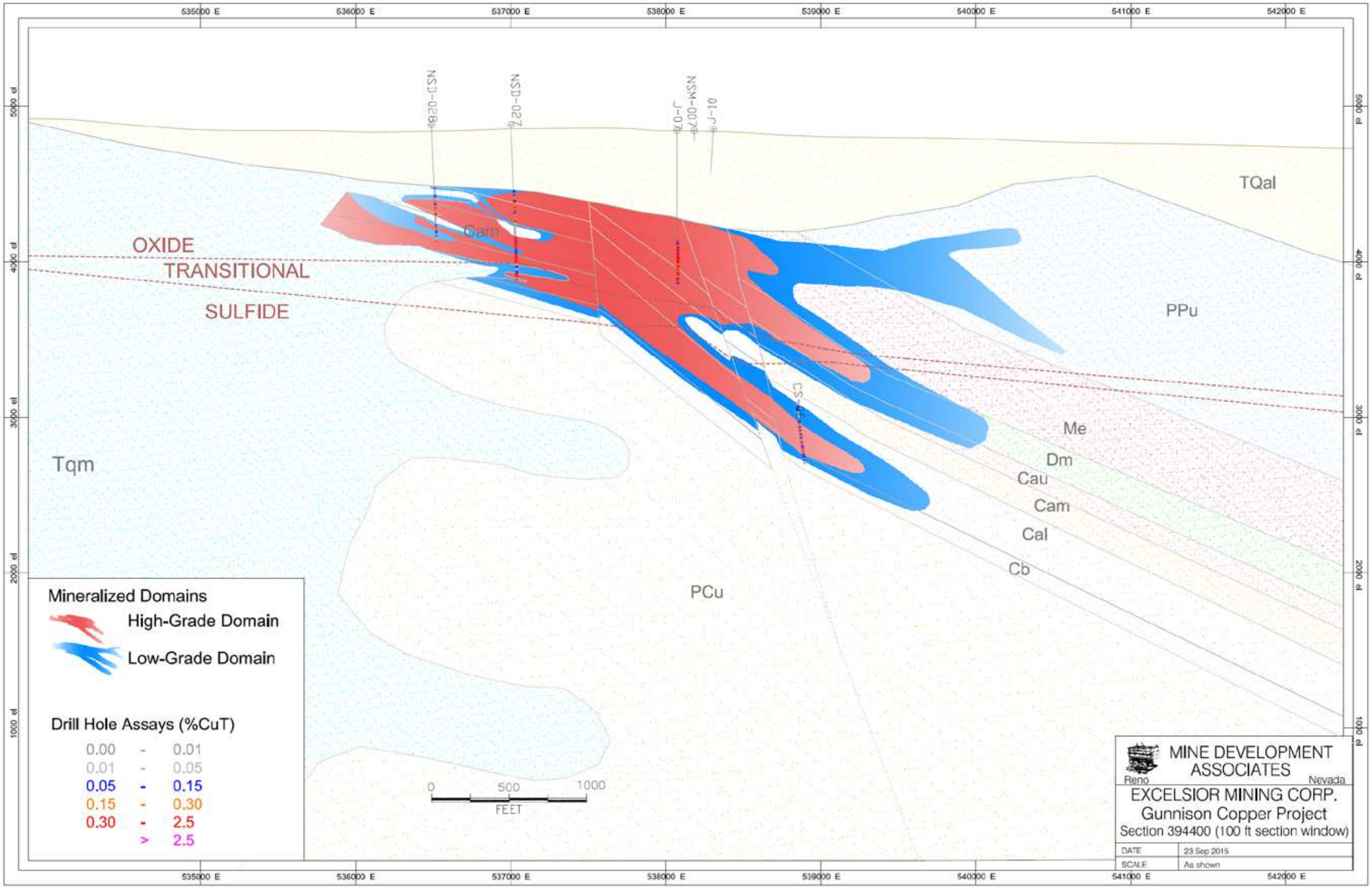


Figure 14-6: Cross Section 394400 Showing Total – Copper Mineral Domains

Assay Coding, Capping, and Compositing. The TCu cross-sectional mineral-domain polygons were used to code drillhole TCu intervals to their respective mineral domains. The ASCu database intervals were coded to the oxide, transitional, sulfide, and low-ratio domains using the oxidation surfaces and low-ratio sectional polygons. Only those intervals that were also coded to one of the two TCu mineral domains were coded to one of the four ASCu domains. As an additional constraint, ASCu intervals were not coded if the TCu value was less than 0.03%, in order to alleviate spurious ASCu/TCu ratios caused by analyses of either species close to, or at, the analytical detection limits.

Descriptive statistics of the coded TCu analyses and ASCu/TCu ratios are provided in Table 14-6 and Table 14-7 respectively.

Table 14-6: Descriptive Statistics of Coded Total – Copper Analyses

Domain	Assays	Count	Mean (Cu%)	Median (Cu%)	Std. Dev	CV	Min. (Cu%)	Max. (Cu%)
100	Cu	3075	0.09	0.07	0.15	1.57	0.00	9.00
	Cu Cap	3075	0.09	0.07	0.09	1.04	0.00	1.50
200	Cu	4498	0.40	0.30	0.37	0.94	0.00	10.95
	Cu Cap	4498	0.40	0.30	0.37	0.94	0.00	10.95
All	Cu	7573	0.27	0.17	0.34	1.23	0.00	10.95
	Cu Cap	7573	0.27	0.17	0.34	1.21	0.00	10.95

Table 14-7: Descriptive Statistics of Acid – Soluble to Total – Copper Ratios

Domain	Count	Mean	Median	Std. Dev.	CV	Min.	Max.
Oxide	4079	0.75	0.78	0.16	0.22	0.01	1.00
Low-Ratio	292	0.45	0.44	0.19	0.43	0.01	1.00
Transition	1540	0.31	0.25	0.24	0.78	0.00	1.00
Sulfide	1040	0.09	0.06	0.10	1.12	0.00	0.97
All	6951	0.54	0.65	0.31	0.58	0.00	1.00

The process of determining TCu capping levels (Table 14-8) included the evaluation of population distribution plots of the coded analyses by domain to identify potential high-grade outliers. Descriptive statistics of the coded assays by domain and visual reviews of the spatial relationships of the possible outliers and their potential impacts during grade interpolation were also considered. ASCu/TCu ratios were capped at 1.00.

Table 14-8: Total – Copper Assay Caps by Mineral Domain

Domain	TCu%	Number Capped (% of Samples)
100	1.5	7 (<1%)
200	-	-

The capped TCu analyses and ASCu/TCu ratios in the database were composited at 20-foot down-hole intervals that respect the mineral domains; composites less than 10 feet in length were eliminated. The 20-foot composite length was chosen because it is a multiple of the dominant 10-foot sample length.

Descriptive statistics of TCu and ASCu/TCu-ratio composites are shown in Table 14-9 and Table 14-10, respectively.

Table 14-9: Descriptive Statistics of Total – Copper Composites

Domain	Count	Mean (Cu%)	Median (Cu%)	Std. Dev.	CV	Min. (Cu%)	Max. (Cu%)
100	1,352	0.09	0.08	0.06	0.71	0	0.81
200	1,915	0.4	0.33	0.29	0.72	0.01	2.9
All	3,267	0.27	0.19	0.27	1	0	2.9

Table 14-10: Descriptive Statistics of Acid – Soluble to Total – Copper Composites

Domain	Count	Mean	Median	Std. Dev.	CV	Min.	Max.
100	139	0.44	0.43	0.17	0.37	0.1	0.81
210	17,766	0.75	0.77	0.14	0.18	0.03	1
220	694	0.31	0.28	0.23	0.73	0.01	1
230	428	0.09	0.07	0.09	1	0.01	0.74
All	3,027	0.54	0.65	0.3	0.56	0.01	1

Block Model Coding. The percentage of each block that lies below the topographic surface was coded into the block model, as well as the lithologic, structural, fracture intensity, oxidation, and density coding discussed in previous subsections of this report. The TCu domains were coded using the 100-foot spaced mineral-domain polygons, and the low-ASCu/TCu ratio domain was similarly coded. All of this coding was done on a block-in-block-out basis (i.e., each block received only one lithologic code, one oxidation code, one TCu domain code, etc.).

The model was also coded by land, including the unpatented claims on BLM lands, State of Arizona lands, and Connie Johnson mineral rights, all controlled by Excelsior, as well as “Other” lands (not controlled by Excelsior).

*Variograph*y. Using all TCu composites, variogram ranges of 1,200 feet along the strike of the sedimentary units (340°) and 700 feet in the dip direction (-35° at 070°) were obtained. Due to the inclusion of composites in this analysis from the structure domains and the Texas Canyon Quartz Monzonite, which have a variety of orientations and whose strikes and especially dips are quite different than the orientation of the sedimentary units, these ranges are considered to be minimums.

Acid-Soluble Copper Modeling. There are two methods for estimating ASCu: directly, using composites of the ASCu analyses in the database; or indirectly, by estimating ASCu/TCu ratios. In the latter case, the ratios are determined for each drill interval that has both ASCu and TCu analyses, and these ratios are then coded, composited, and used to estimate the ratios into the model blocks. The estimated ASCu model values are then derived by multiplying the estimated ASCu/TCu ratio by the estimated TCu value in each block.

There is no evidence of significant leaching and remobilization of the supergene copper at the North Star deposit, which is probably due to remnant carbonate minerals in the host units that would have restricted the movement of acidic solutions during oxidation. In a scenario of limited to no remobilization of oxidized copper species, ASCu/TCu ratios reflect the degree of oxidation of the hypogene copper mineralization. At North Star, the ASCu/TCu ratios are relatively uniform within each of the oxidation zones, with some indication of decreasing ratios (decreasing oxidation) with depth.

The use of ASCu/TCu ratios in the estimation of ASCu values can negate possible biases created by sample intervals that were selectively analyzed for TCu but not ASCu. There are 259 sample intervals coded to the TCu domains that have no ASCu analyses, which represents approximately 3.5% of the coded intervals.

RESPEC decided to use estimated ASCu/TCu ratios to model the North Star ASCu values. The ASCu/TCu ratio estimation was confined to blocks with estimated TCu values. The ratios of blocks coded to the oxide, transitional, and sulfide zones, as well as the low-ratio zone discussed above, were all estimated independently.

Estimation. The search ellipses used for the TCu and ASCu/TCu ratio interpolations are shown in Table 14-11 and other estimation parameters are summarized in Table 14-12.

Table 14-11: Search Ellipse Orientations

Search Ellipse Orientations			
Total Copper and Fracture Intensity	Major Bearing	Plunge	Tilt
Inside Structural Domains: All Rock Types	005°	0°	-85°
	025°	0°	-80°
	045°	0°	-65°
	090°	0°	-90°
	145°	0°	-50°
	165°	0°	-35°
	340°	0°	-25°
	345°	0°	-70°
Outside Structural Domains: Paleozoic + Precambrian Units	340°	0°	-35°
Outside Structural Domains: Texas Canyon Quartz Monzonite	0°	0°	0°
Acid-Soluble to Total-Copper Ratio	Major Bearing	Plunge	Tilt
All Domains	0°	0°	0°

Table 14-12: Estimation Parameters

Total Copper – All Units Except Quartz Monzonite						
Estimation	Search Ranges (ft)			Composite Constraints		
Pass	Major	S-Major	Minor	Min	Max	Max/hole
1	300	300	100	3	12	3
2	700	700	233	3	12	3
3	2000	2000	667	1	12	3
Total Copper – Quartz Monzonite						
1	300	300	100	3	12	3
2	700	700	233	1	12	3
Acid-Soluble to Total-Copper Ratio						
1	700	700	233	3	12	3
2	2000	2000	667	1	12	3

The estimation passes were performed independently for each of the TCu mineral domains, so only composites coded to a particular domain were used to estimate grade into blocks coded by that domain.

Inverse-distance to the third power (ID3) and ordinary kriging estimations were run for both total copper and the ASCu/TCu ratios; nearest-neighbor estimations were also completed for evaluation purposes. Ultimately, the (ID3) results were selected for reporting of the project resources.

14.3 NORTH STAR DEPOSIT MINERAL RESOURCES

The North Star deposit mineral resources are reported at cut-offs that are reasonable given anticipated mining methods, processing costs, and economic conditions, which fulfills regulatory requirements that a resource exists "in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction."

The oxide + transitional mineral resources are tabulated using a cut-off grade of 0.05% TCu, representing resources potentially available for in-situ recovery. The sulfide mineral resources are reported at a 0.30% TCu cut-off to capture mineralization that is potentially available for open-pit extraction. Both of these cut-offs are the same as the cut-offs used for the previously reported resources (M3, 2016).

No resources were estimated within overburden (Tertiary/Quaternary alluvium), and the reported resources are restricted to lands controlled by Excelsior.

The North Star deposit TCu resources are listed in Table 14-13.

Table 14-13: North Star Deposit Total – Copper Resources

Oxide Resources @ 0.05% TCu Cut-off			
Resource Class	Short Tons (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	157.2	0.38	1.201
Indicated	502.1	0.28	2.782
Measured + Indicated	659.3	0.30	3.983
Inferred	108.0	0.16	0.351
Transitional Resources @ 0.05% TCu Cut-off			
Resource Class	Short Ton (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	41.9	0.27	0.227
Indicated	172.0	0.23	0.785
Measured + Indicated	213.9	0.24	1.02
Inferred	79.2	0.18	0.279
Oxide + Transitional Resources @ 0.05% TCu Cut-off			
Resource Class	Short Tons (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	199.1	0.36	1.427
Indicated	674.0	0.27	3.567
Measured + Indicated	873.2	0.29	4.995
Inferred	187.2	0.17	0.630
Sulfide Resources @ 0.30% TCu Cut-off			
Resource Class	Short Tons (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	1.6	0.39	0.012
Indicated	36.8	0.42	0.308
Measured + Indicated	38.4	0.42	0.32
Inferred	53.7	0.41	0.44

Notes:

1. Mineral Resources are inclusive of Mineral Reserves.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Oxidized + Transitional Mineral Resources are reported at a 0.05% total-copper cut-off in consideration of potential mining by in-situ recovery.
4. Sulfide Mineral Resources are reported at a 0.30% total-copper cut-off in consideration of potential mining by open-pit extraction.
5. Rounding may result in apparent discrepancies between tons, grade, and contained metal content.
6. The Effective Date of the mineral resource estimate is October 1, 2016.

The North Star deposit resources are classified on the basis of a combination of: (i) a minimum number of composites used to interpolate TCu grades into a block; (ii) the number of holes from which the composites are derived; and (iii) the distance of the composites to the block (Table 14-14).

Table 14-14: North Star Deposit Classification Parameters

Class	Min. Number of Composites	Additional Constraints
Measured	2	Minimum of 2 holes within an average distance of 200 feet from the block
Indicated	2	Minimum of 2 holes within an average distance of 400 feet from the block
Inferred	all other estimated blocks	

When evaluating the results produced by the classification criteria, it became apparent that a small, isolated zone of blocks classified as Inferred occurred within a mass of Indicated blocks near the southern limit of the well-drilled portion of the deposit. This Inferred material created a classification discontinuity in the deposit, where confidence in the modeling is high, and the classification was therefore changed to Indicated. This change resulted in an increase of one percent of the resource tonnes classified as Indicated.

The average ASCu/TCu ratios estimated for the oxide, transition, and sulfide resources reported in Table 14-13 are 0.74, 0.30, and 0.09, respectively.

Total project resources, obtained by adding the oxide, transitional, and sulfide resources in Table 14-13, are tabulated in Table 14-15.

Table 14-15: Combined Oxide, Transitional, and Sulfide Resources

Total Resources (Oxide + Transitional + Sulfide)			
Resource Class	Short Tons (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	200.7	0.36	1.439
Indicated	710.8	0.27	3.875
Measured + Indicated	911.6	0.29	5.315
Inferred	240.9	0.22	1.070
0.05% TCu Cut-off for Oxide + Transitional; 0.30% TCu Cut-off for Sulfide			

The average ASCu/TCu ratio of the combined resources is 0.57.

The modeled North Star deposit mineralization is tabulated at additional cut-offs in Table 14-16 to provide grade-distribution information, as well as to evaluate the sensitivity of the reported resources to economic conditions and/or mining scenarios other than those envisioned in this study.

Table 14-16: Modeled Mineralization at Various Cut-offs

Oxide + Transitional Mineralization @0.10% TCu Cut-off			
Resource Class	Short Tons (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	165.7	0.42	1.378
Indicated	495.7	0.33	3.302
Measured + Indicated	661.4	0.35	4.680
Inferred	93.2	0.26	0.490
Oxide + Transitional Mineralization @ 0.30% TCu Cut-off			
Resource Class	Short Tons (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	113.9	0.51	1.167
Indicated	264.5	0.45	2.372
Measured + Indicated	378.4	0.47	3.539
Inferred	34.0	0.41	0.277
Oxide + Transitional Mineralization @ 0.50% TCu Cut-off			
Resource Class	Short Tons (millions)	Total Cu (%)	Cu Pounds (billions)
Measured	42.5	0.72	0.615
Indicated	67.3	0.63	0.852
Measured + Indicated	109.8	0.67	1.467
Inferred	5.6	0.59	0.066
Sulfide Mineralization @ 0.50% TCu Cut-off			
Resource Class	Short tons (millions	Total Cu (%)	Cu Pounds (billions)
Measured	0.2	0.55	0.002
Indicated	6.3	0.6	0.076
Measured + indicated	6.5	0.6	0.078
Inferred	5.3	0.58	0.062

Figure 14-7 and Figure 14-8 show cross section of the block model that correspond to the mineral-domain cross sections presented above.

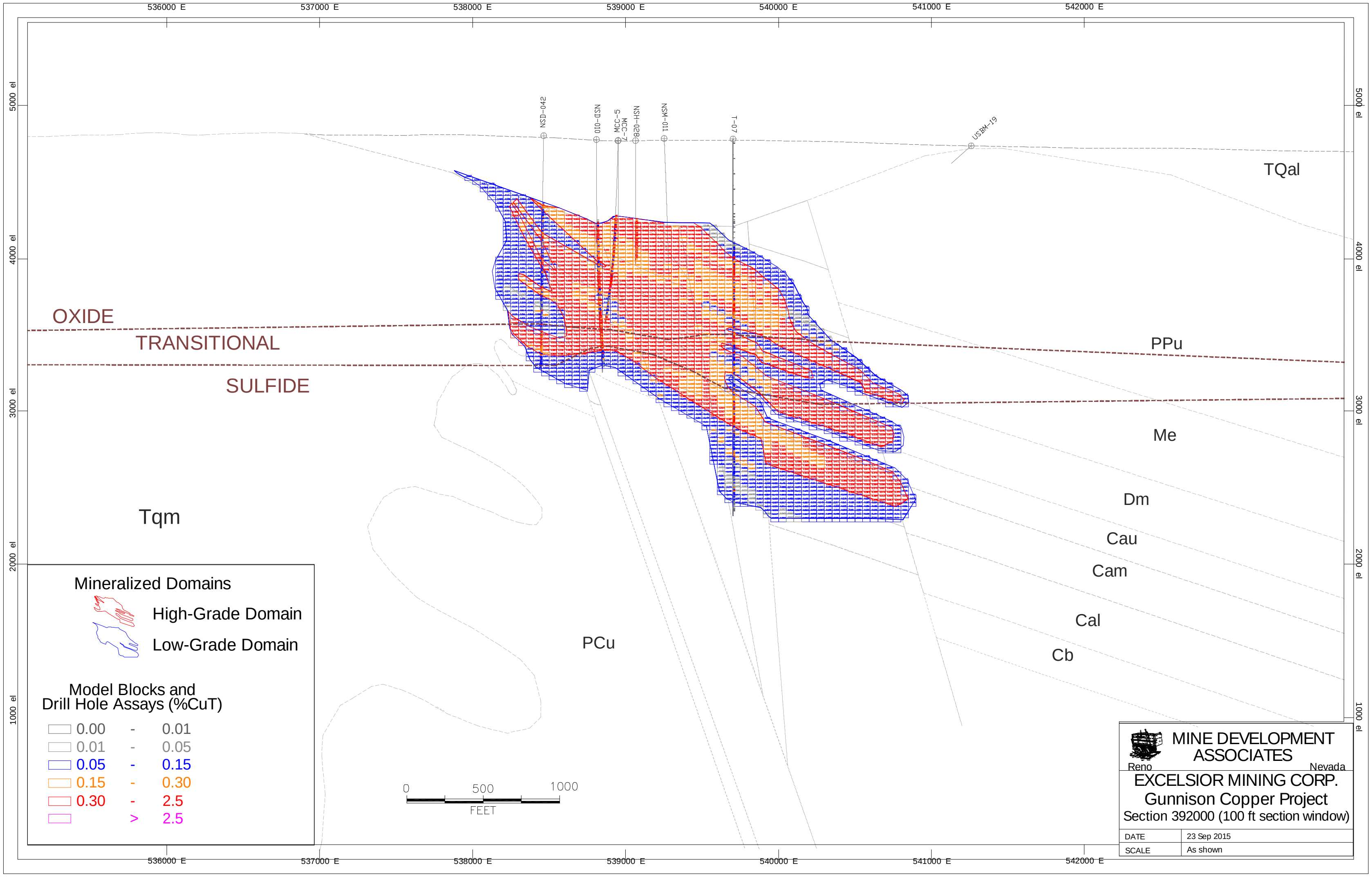


Figure 14-7: North Star Cross Section 392000 Showing Block Model Copper Grades

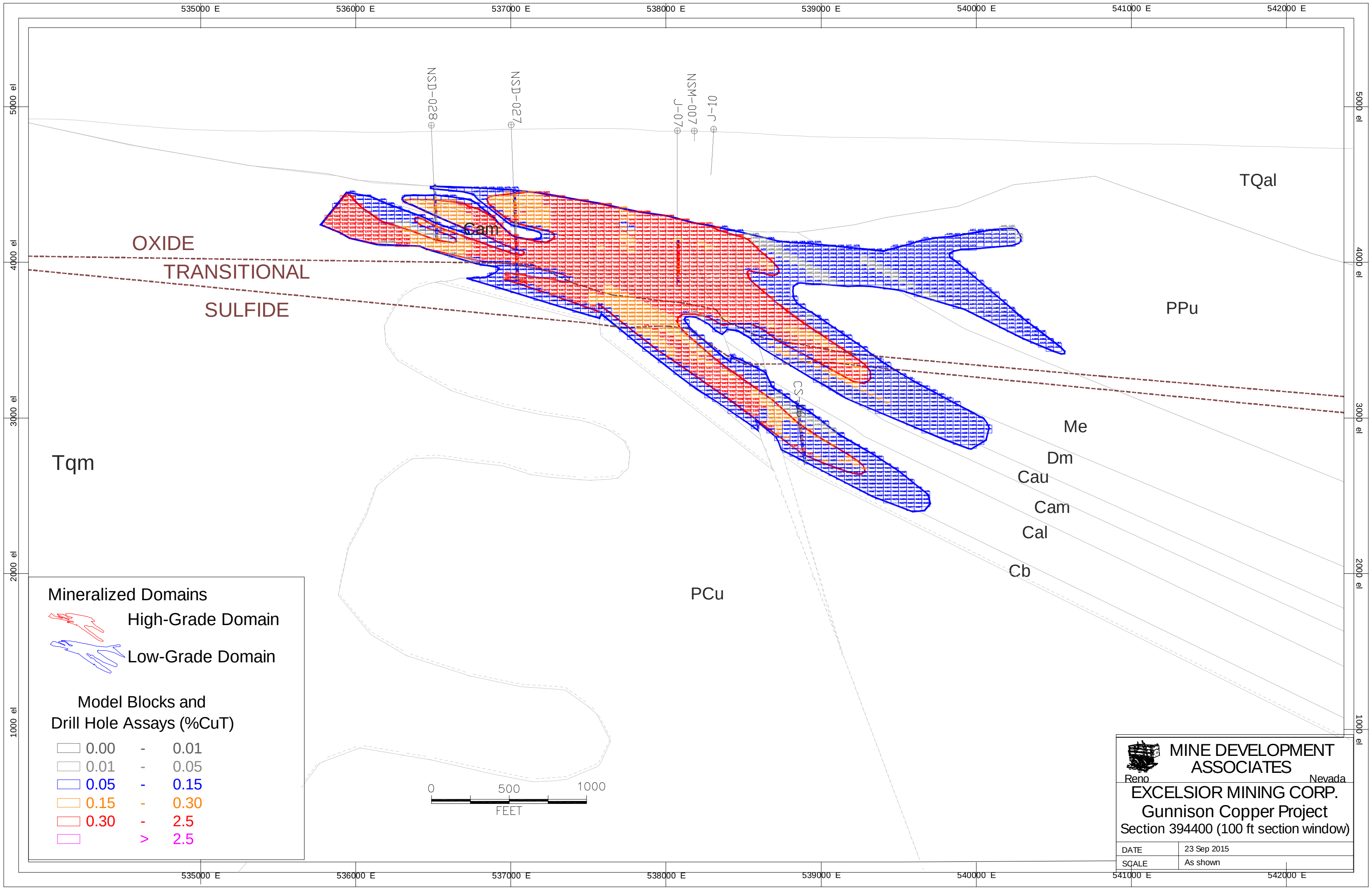


Figure 14-8: North Star Cross Section 394400N Showing Block Model Copper Grades

14.3.1 Copper Block Model Checks

Volumes derived from the sectional mineral-domain modeling were compared to the coded block-model volumes to assure close agreement, and all block-model coding described herein was checked visually. The inverse-distance results, from which the reported project resources are tabulated, were compared to those from: (i) a polygonal estimate based on the cross-sectional interpretations; and (ii) the nearest-neighbor and ordinary kriging estimates of the modeled resources, all at 0 cut-off grade. The ID3, ordinary kriging, and nearest-neighbor grades are identical, and the polygonal tons and grade are as expected. Various grade-distribution plots of assays, composites, and nearest-neighbor, ordinary kriging, and ID3 block grades were evaluated as a check on the both the global and local estimation results, with no anomalous relationships. Finally, the ID3 grades were visually compared to the drillhole assay data to assure that reasonable results were obtained.

14.3.2 Comments on the Resource Block Model Estimates

A subsequent estimate of the project resources could be improved with the incorporation of additional geologic input into the modeling. Specifically, the modeling of the western extremities of the deposit could be improved where the large mass of mineralization that typifies the core, central portion of the deposit breaks up into lenses that follow favorable stratigraphic horizons. The correlations of some of these ‘arms’ of mineralization with specific stratigraphic units might be improved with additional drill data and further review and consideration.

Mr. Bickel has reviewed the data from the 2018-2019 production wellfield drilling and determined that the addition of these data is not material to the estimate of mineral resources reported herein. The lack of materiality to the resource is primarily based on the scale of the wellfield compared to that of the entire resource. Mr. Bickel’s conclusions on this matter are discussed in Section 12.6 of this report.

15 MINERAL RESERVE ESTIMATES

The mineral resources discussed in Section 14 (Table 14-1) were used to estimate the Probable Mineral Reserves for North Star (Gunnison Project). Details of the process to determine the mineral reserve are outlined in this section.

The mineral resources for this study are the resources reported in the January 2017 Feasibility Study. Although there was some copper production in 2021, the net reported copper production of 385,238 pounds during this period is not material and the resource nor the reserve estimates were not revised for this report from the 2017 Feasibility Study.

Table 15-1 shows the diluted Probable diluted mineral reserves as defined for the Gunnison Project's Feasibility Study. The Probable mineral reserves include material classified in the Measured and Indicated mineral resource categories. No Inferred mineral resources were added to the tabulation of mineral reserves. No material from the sulfide zone was included in the mineral reserves either.

Table 15-1: Probable Diluted Mineral Reserve Estimate (October 2016 Estimate)

Item	Value
Short Tons (million)	782
TCu Grade (%)	0.29
TCu Contained Copper (million lbs)	4,505
Average Total Copper Recovery (%)	48.4
Recoverable Copper** (million lbs)	2,154

**Probable reserves were defined from measured and indicated resources. Inferred resources were not converted into reserves.*
*** Total includes losses to water treatment.*

The Probable mineral reserves summary prepared for this report were estimated using data and input from the qualified persons responsible for this section in the 2016 Feasibility Study. Probable mineral reserves were defined using updated cost and metal recovery estimates. They were also constrained to take into account lost mineral resources beneath Interstate 10 and along some of the lease boundaries. The production from blocks under Interstate 10 is factored by 50% to estimate mining losses there. RESPEC's mineral resource estimate detailed in Section 14 and the ISR mine production schedule developed for the Update in Section 16 served as the basis for the mineral reserves. Figure 15-1 shows the resulting outline for the Probable Mineral Reserve as the black outline within the limits of the mineral resource (blue outline).

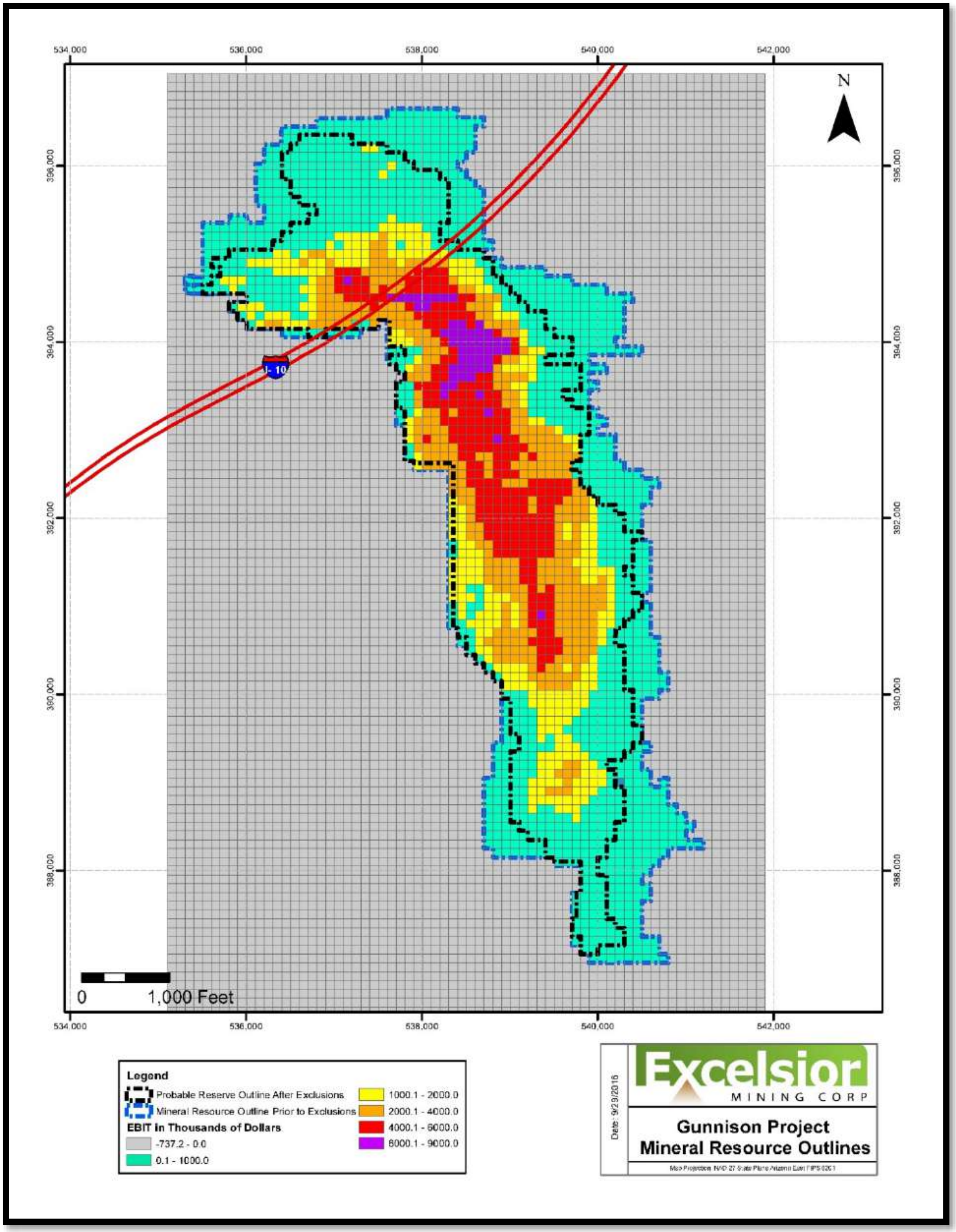


Figure 15-1: Mineral Resource and Mineral Reserve Outlines

15.1 ECONOMIC EVALUATION

The economic evaluation of RESPEC's mineral resource model was the starting point for developing the mine plan. The block resource model contains 100' x 50' x 25' resource blocks which were individually coded with estimated tons, total and acid soluble copper grades, oxidation designation, resource class (Measured, Indicated, or Inferred), specific gravity, formation (rock type), fracture intensity, and mineral lease designation. For the purpose of the in-situ mining plan, two columns of resource blocks (100' x 100' x 25') from the model are combined to represent a production cell (5 spot pattern of one injection well surrounded by four recovery wells). Figure 15-2 shows a typical wellfield layout with the repeated 5 spot patterns. The recovery wells are spaced 100' apart at each corner of the production cell. The injection well is positioned in the middle of the production cell.

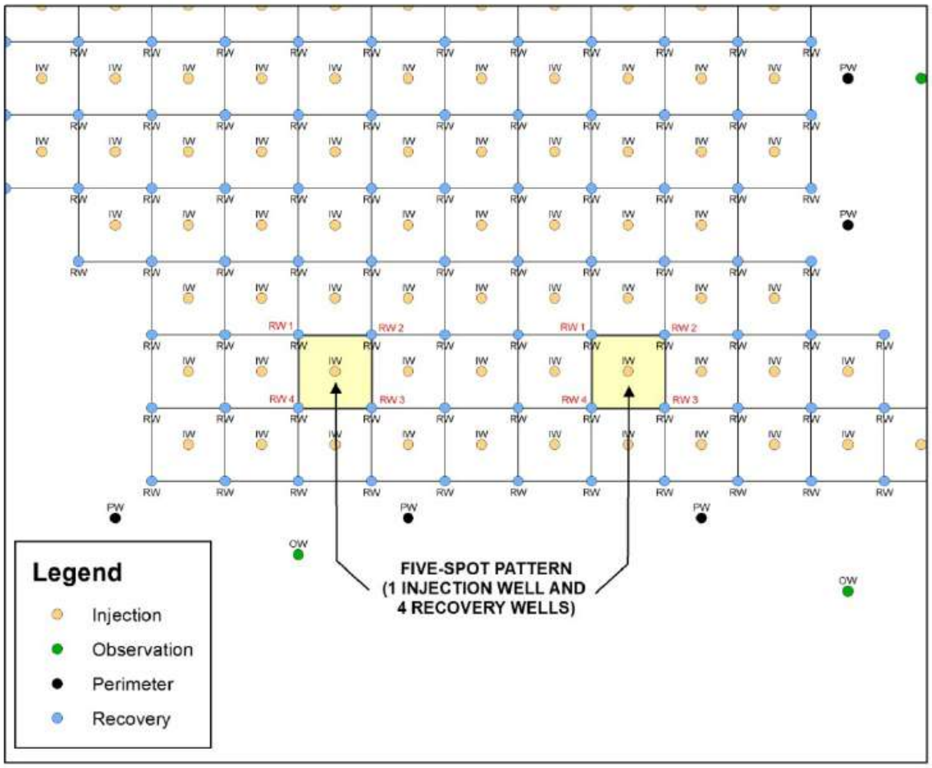


Figure 15-2: Wellfield Design Layout

The updated production schedule for the first 24 years of production was used to test the pre-tax economic of the Project. The Gunnison Project is defined by three stages of increasing growth of production.

The updated estimated capital costs, operating cost and sustaining capital cost was obtained from M3 to test the pretax economics of the Project. The cost (capital cost + operating cost + sustaining capital) of the Project per pound of copper must be less than the price of copper of $3.75 per pound used in the study. Table 15-2 summarizes the economics of each stage of production of the Project.

Table 15-2: Economic Test of Project

Category	Stage 1 Yrs 1 – 3	Stage 2 Yrs 4 – 6	Stage 3 Yrs 7 – 24	Total Life Yrs 1-24
Total Copper Production (klbs)	62,682	206,842	1,884,340	2,153,864
Total Copper Sales ($000)	235,057	775,659	7,066,275	8,076,991
Total Sales ($/lb)	3.75	3.75	3.75	3.75
Total Initial Capital ($000)	47,621			47,621
Total Sustaining Capital ($000)	218,316	299,537	515,335	1,033,188
Total Capital	265,937	299,537	515,335	1,080,809
Total Capital ($/lb)	**4.24**	**1.45**	**0.27**	**0.50**
Operating Costs				
G&A ($000)	22,464	23,873	153,140	199,477
Wellfield Opex ($000)	57,604	154,928	581,980	794,512
SX-EW Opex ($000)	22,089	58,374	431,369	511,832
WTP Opex ($000)	25,793	42,104	462,709	530,606
Other Expenses ($000)	14,543	47,838	539,285	601,666
Total Operating + Expenses ($000)	**142,493**	**327,117**	**2,168,483**	**2,638,093**
G&A ($/lb)	0.36	0.12	0.08	0.09
Wellfield Opex ($/lb)	0.92	0.75	0.31	0.37
SX-EW Opex ($/lb)	0.35	0.28	0.23	0.24
WTP Opex ($lb)	0.41	0.20	0.25	0.25
Other Expenses ($/lb)	0.23	0.23	0.29	0.28
Total operating + expenses ($/lb)	2.27	1.58	1.15	1.22
Total Capital + Operating ($/lb)	**6.52**	**3.03**	**1.42**	**1.73**

Table 15-2 illustrates that the cost per pound of copper of the proposed project (capital + sustaining capital + operating) is $1.73 per pound of copper, which is well below the average realized copper price of $3.75 per pound used in the study.

15.2 TABULATION OF MINERAL RESERVE

A summary of the diluted mineral reserve is shown in Table 15-1 and Table 15-3 for the deposit by rock type. The formations have been combined with like acid consumption. The dilution included in the mineral reserve is from blocks within the well that are below the 0.05% TCu cut-off grade but are within the production column for a particular well. The drilling cost through these dilution zones is carried by the positive value blocks located below them. The diluting tonnage is 30,151 kilotons (ktons) at an average grade of 0.041% TCu.

The effective date of the mineral reserve is October 1, 2016, and it is RESPEC's opinion that it is a fair representation of the mineral reserve. The mineral reserve tabulated by Excelsior is from a database extracted from the mineral resource block model to develop the well extraction columns. RESPEC has checked the tabulation of the undiluted mineral reserve by flagging the blocks within the mineral resource model that constitute the mineral reserve and tabulating them from the block model. This tabulation checks are within 1 percent of the tonnage and grades. In addition, RESPEC spot checked a number of the reserve blocks for model data and calculated data. Calculated data includes copper recovery, pounds of copper recovered, and the time period of copper recovery and found no issues with the calculations of the reserves. RESPEC checked the production schedule with the blocks scheduled and found no errors.

Table 15-3: Diluted Mineral Reserve by Formation Type

Formation (1)	ktons	TCu %	ASCu %	Lbs Cu x 1000	Lbs ASCu x 1000	Recoverable Lbs x 1000	Average Recovery, %	
							TCu	ASCu
OXIDE								
Dm	226,137	0.33	0.26	1,486,729	1,180,788	849,837	0.58	0.73
Cau	142,830	0.32	0.24	900,548	677,449	498,990	0.56	0.75
Cam	129,548	0.29	0.21	755,486	536,662	397,292	0.53	0.75
Cal	75,563	0.27	0.17	405,812	262,740	197,374	0.49	0.76
Cb	14,400	0.10	0.05	28,053	14,742	11,176	0.40	0.77
Total	588,479	0.30	0.23	3,576,627	2,672,381	1,954,668	0.55	0.74
TRANSITION								
Dm	8,489	0.27	0.08	45,819	13,172	9,191	0.20	0.71
Cau	21,962	0.21	0.06	92,979	24,402	17,754	0.19	0.74
Cam	39,740	0.29	0.08	227,948	65,720	48,870	0.22	0.75
Cal	83,896	0.28	0.08	473,961	140,881	99,992	0.21	0.72
Cb	39,587	0.11	0.04	87,934	31,866	23,144	0.27	0.74
Total	193,674	0.24	0.07	928,641	276,041	198,950	0.22	0.73
TOTAL								
Dm	234,626	0.33	0.25	1,532,549	1,193,960	859,028	0.57	0.73
Cau	164,792	0.30	0.21	993,527	701,850	516,744	0.53	0.75
Cam	169,289	0.29	0.18	983,433	602,382	446,162	0.46	0.75
Cal	159,459	0.28	0.13	879,773	403,621	297,366	0.34	0.75
Cb	53,987	0.11	0.04	115,987	46,608	34,320	0.30	0.75
Total	782,153	0.29	0.19	4,505,268	2,948,422	2,153,168	0.48	0.74

Notes:
Formation: Dm = Martin, Escabrosa, and Naco Group Undivided; Cau = Upper Abrigo; Cam = Middle Abrigo; Cal = Lower Abrigo; Cb = Texas Quartz Monzonite, Bolsa Qtz, & Undivided Precambrian Rocks
ktons = short tons x 1000
Lbs Cu = pounds of copper in the ground
Lbs ASCu = pounds of soluble copper in the ground
Recoverable Lbs = pounds of expected recoverable copper using sweep (recovery) factor and adjusted for copper losses in water treatment.

15.3 POTENTIAL FOR RESERVE EXPANSION

An upgrade of the mineral reserve at Gunnison is possible with continued resource drilling by Excelsior. Material categorized as inferred within the resource has the potential to be converted into the measured and indicated resource categories as it spatially borders the existing measured and indicated resources. Table 15-4 lists the inferred mineral resources at Gunnison as defined by 2016 Resource Model described in Section 14. Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that any economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 15-4: Inferred Mineral Resources at Gunnison (October, 2016)

Short Tons (million)	187
TCu Grade (%)	0.17
TCu Contained Copper (million lbs)	630

**Inferred resources as defined by RESPEC's resource model. Calculated for blocks existing inside Excelsior lease boundaries, within the oxide and transition zones, and at a 0.05% TCu cut-off.*

16 MINING METHODS

16.1 IN-SITU RECOVERY

Excelsior proposes to use the In-Situ Recovery (ISR) method to extract copper from oxide mineralization located within the Gunnison deposit (Figure 4-1). ISR was chosen based on the fractured nature of the host rock, the presence of water saturated joints and fractures within the ore body, copper mineralization that preferentially occurs along fracture surfaces, the ability to operate in the vicinity of Interstate 10, and to avoid the challenges of open pit mining in an area with alluvium overburden thickness ranging from approximately 300 ft to 800 ft.

In the ISR process, a low pH raffinate solution ("lixiviant") is injected into the ore body via a series of injection wells. As the lixiviant migrates through the joints and fractures within the mineralized bedrock, copper is dissolved. This pregnant leach solution (PLS) is recovered by a series of recovery wells that surround each respective injection well (Figure 16-1).

The PLS is pumped to the surface where the copper is stripped from the solution using the solvent extraction/electrowinning (SX-EW) process. The SX-EW process begins with the SX plant extracting and concentrating the dissolved copper from the PLS, after which the EW plant reduces the concentrated copper to copper cathode. Once the copper is recovered by SX, the barren solution is re-acidified with sulfuric acid to create new lixiviant which is pumped back to the well field and re-injected. The total volume of lixiviant injected and PLS extracted will remain effectively equal throughout ISR operations.

After ISR in a production block is complete, as determined by degradation of the PLS grade below the economic cut-off, the bedrock within the completed production block will be rinsed in compliance with appropriate permit conditions.

Economic recovery of acid soluble copper using ISR requires certain hydrogeological conditions be present within an ore body, such as: (1) a saturated ore body; (2) sufficient hydraulic conductivity within the fractured bedrock; (3) hydraulic connection between the injection and recovery wells so lixiviant can circulate through the mineralized bedrock; and (4) lixiviant/mineral contact and adequate lixiviant retention time. These conditions allow for lixiviant to be circulated through the ore body, with sufficient contact and retention time with acid soluble copper in the ore body to meet the required PLS grade. Site characterization efforts described in this chapter have focused on gathering data to assess these hydrogeological conditions.

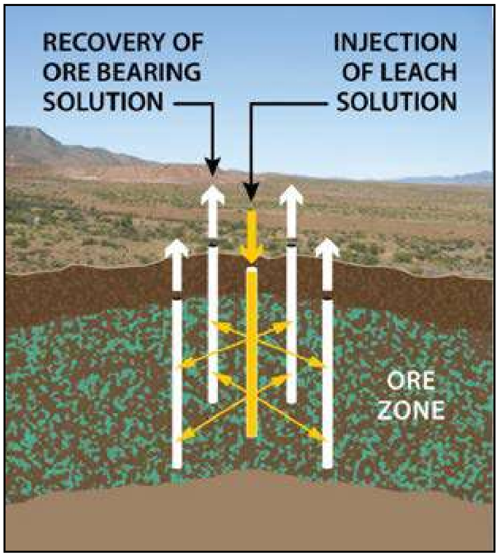


Figure 16-1: Conceptual Schematic of ISR Injection and Recovery

16.2 HYDROGEOLOGICAL CHARACTERIZATION

The rock units in the study area range in age from Precambrian to Quaternary. The basement rock is comprised primarily of the Pinal Schist, Lower Precambrian in age. The mineralized ore is hosted by the Abrigo (Upper Cambrian), Martin (Upper Devonian) and, to a limited degree, Escabrosa (Lower Mississippian) Formations. Bounding the sub-basin to the east are the Horquilla Formation of the Naco Group (Middle Mississippian) with outcrops in the Gunnison Hills, and to the west the Texas Canyon quartz monzonite (a Lower Tertiary intrusive unit), cropping out as the Texas Canyon Summit. The bedrock formations are uncomformably overlain by Basin Fill of upper Tertiary and Quaternary age. The thickness of the Basin Fill over the ISR wellfield varies from 300 to 800 feet and increases in thickness towards the Gunnison Hills.

16.2.1 Water-Bearing Units

The following water-bearing units have been identified within and adjacent to the Project area:

- Basin Fill Aquifer; and,
- Bedrock Aquifer.

16.2.1.1 Basin Fill

Depending on the location, basin fill in the area may be unsaturated or partially saturated. The basin fill aquifer is used for water supply in the Dragoon area, and also historically as a source of water for the Johnson Camp Mine, north of the Site.

At the Project site within the ISR wellfield, the thickness of basin fill ranges from approximately 300 to 800 feet. In general, the basin fill within the boundary of the ISR wellfield is unsaturated, and thus, not an aquifer. Thin, isolated occurrences of saturated basin fill were identified within the ISR wellfield during drilling in 2011 at two locations: NSH-006 and NSD-020. Thirty to 40 feet of saturation were observed at these locations which are within a low spot on the bedrock surface. In October 2021, NSH-006 had 33 feet of saturated alluvium; NSD-020 has been abandoned. The thickness of basin fill to the east of the Project site increases to approximately 1800 feet along the western flank of the Gunnison Hills (Harshbarger, 1973). The saturated thickness of the basin fill aquifer also increases toward the east.

16.2.1.2 Bedrock

Data collected from hydrogeological investigation wells completed in bedrock at the Project indicate that the mineralized zone in bedrock is mostly saturated. In general, groundwater is present at or near the bedrock-basin fill contact. Depending on the location there may be an interval of unsaturated bedrock, generally less than 50 feet in thickness, above the saturated bedrock. In other locations (as discussed above), the potentiometric surface has been observed slightly above the bedrock-basin fill contact.

Figure 16-2 shows the geology of the bedrock surface and locations of cross sections A-A’ (Figure 16-3), and C-C’ (Figure 16-4).

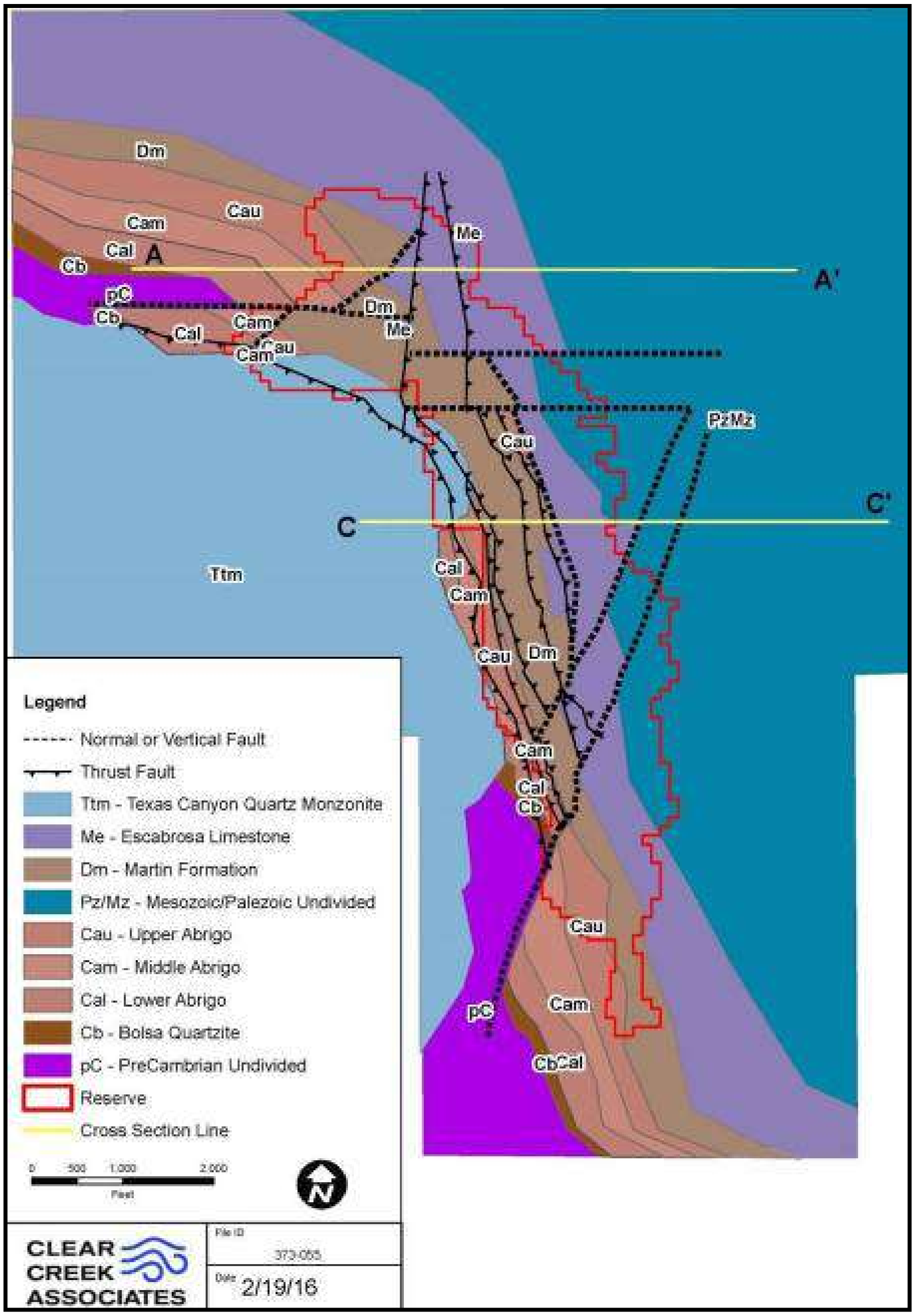


Figure 16-2: Geologic Map of Bedrock Surface

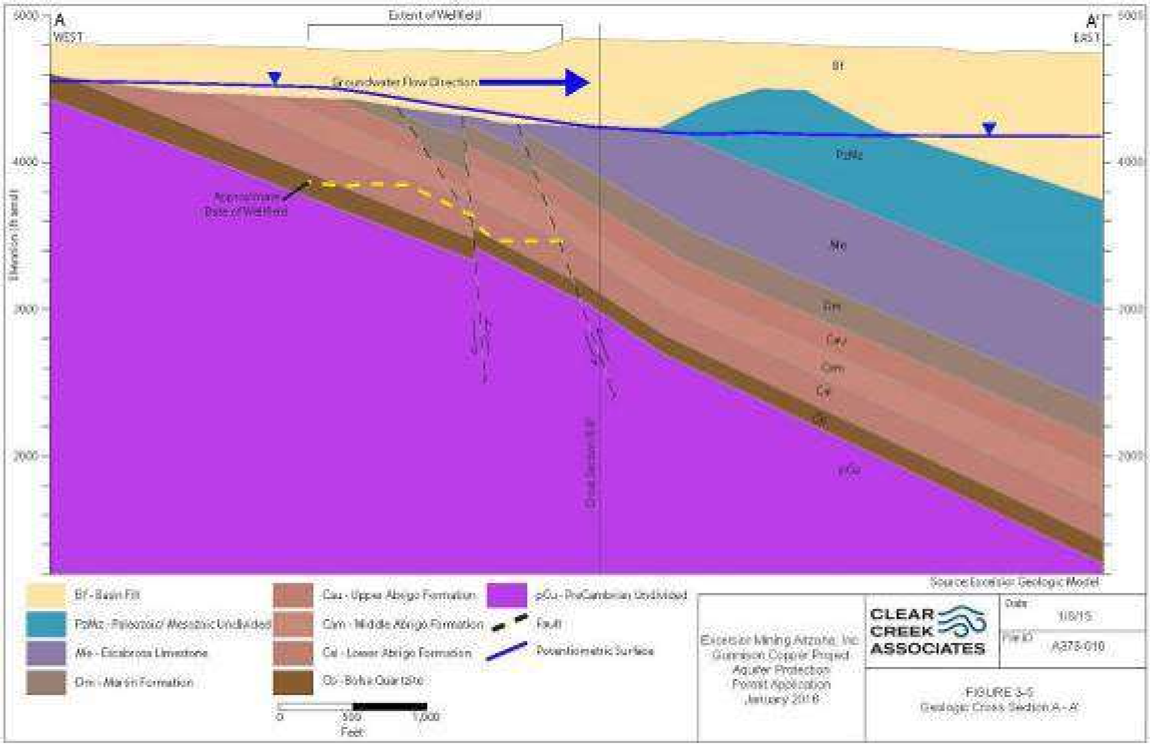


Figure 16-3: Cross Section A-A (EMC, 2016)

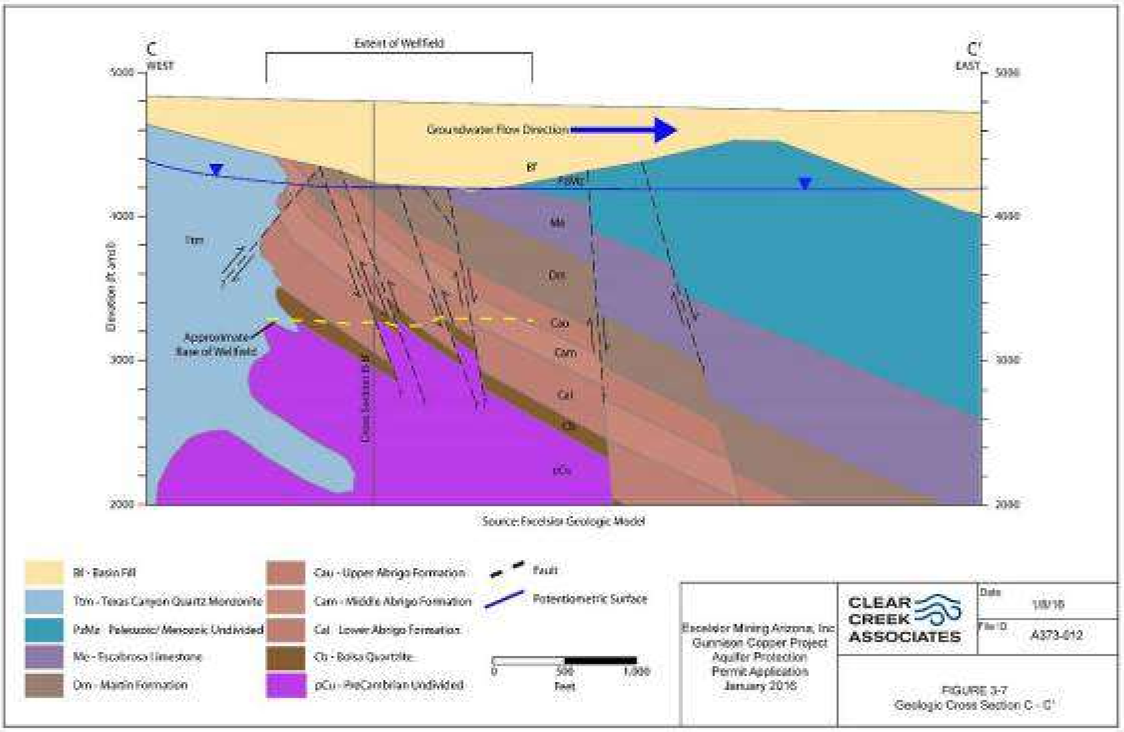


Figure 16-4: Cross Section C-C (EMC, 2016)

16.2.2 Depth to Groundwater

A depth-to-groundwater map, based on a water level sweep conducted in the fourth quarter of 2021, is presented on Figure 16-5. Depths to water ranged from 263 feet below land surface (bls) at exploration drillhole NSD-030 in the northwest part of the Project, to 763 feet bls at MCC-3 south of the Year 1 wellfield.

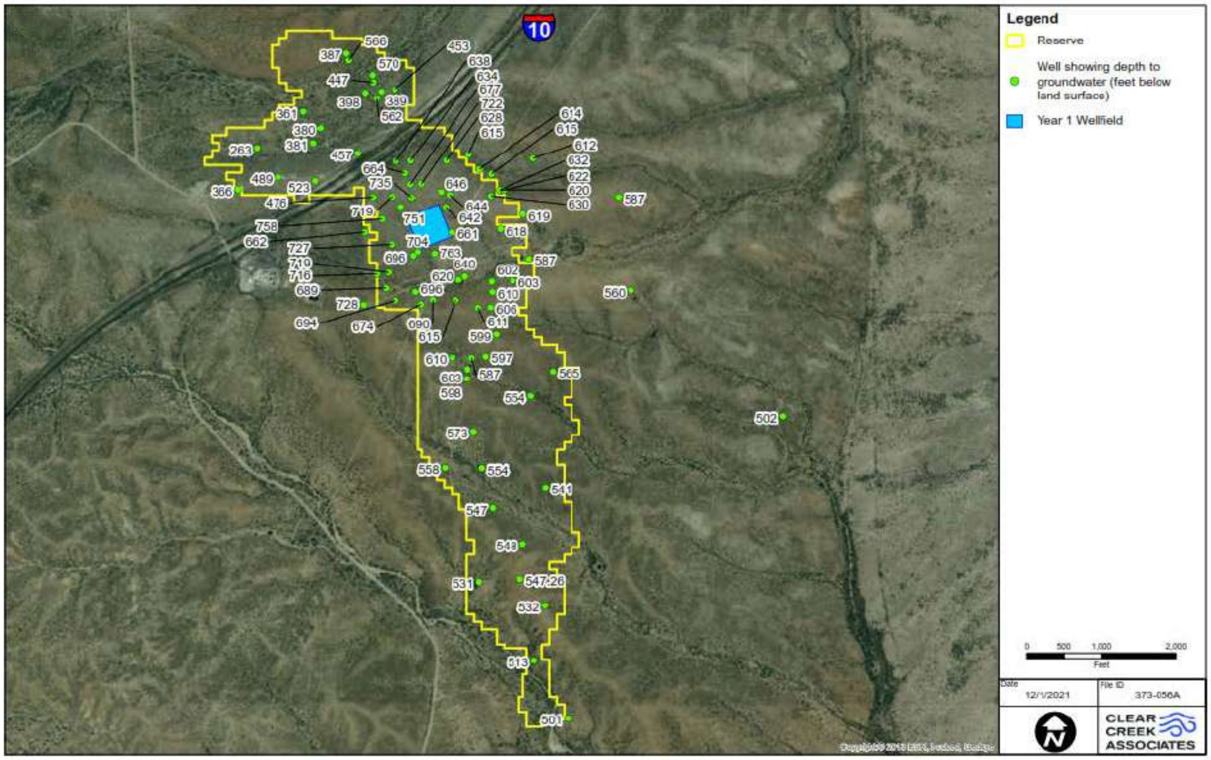


Figure 16-5: Depth to Groundwater, Q4 2021

In the fourth quarter of 2021, bedrock water level elevations ranged from 4180 to 4411 feet above mean sea level, except in the immediate vicinity of the active wellfield where water levels may be lower (Figure 16-5). The overall regional groundwater flow direction to the east, towards the center of the Dragoon sub-basin; this is consistent with a west to east regional groundwater flow direction reported at the Johnson Camp mine (Dickens, 2003). Groundwater levels in wells at the site and in the surrounding area have declined since in-situ mining began due to the 1% net pumping required to maintain hydraulic control.

16.2.3 Fractured Bedrock Characteristics

The Gunnison deposit is the result of the intrusion of the Texas Canyon Quartz Monzonite into the Paleozoic sedimentary rocks. The formation of the skarn deposit created denser minerals and removed carbon dioxide, resulting in a volume reduction of the rocks. This volume reduction resulted in significant fracturing that allows for hydraulic connection within the ISR wellfield. Weitz (1976) calculated a 30% volume reduction in some of the Paleozoic sediments at the Project site. The resulting fractures allowed for the mineralizing fluids to coat the fracture faces with copper-bearing sulfide minerals. The copper sulfide minerals were subsequently oxidized by circulating meteoric groundwater.

Hydrologic characterization activities indicate that groundwater is present within open joints and fractures within the ore body. The permeability of the bedrock varies depending upon the degree of fracturing present. At the Project site, the Paleozoic sedimentary rocks are more permeable than the Pinal Schist or Texas Canyon Quartz Monzonite. Furthermore, the skarn deposit is more fractured than the un-mineralized Paleozoic sedimentary rocks due to the intrusion of the quartz monzonite and the subsequent volume reduction.

16.2.3.1 Porosity

Excelsior estimated the porosity of bedrock at the Project by reviewing published values in the literature, analyzing pumping test results, and analyzing gamma-gamma density logs from seven boreholes. Porosity values consistent with these sources were used in the groundwater flow model and rinsing for closure strategy.

- Literature review data indicate porosities of bedrock range from less than 1% (for fresh igneous plutonic rock) to up to 10% for fractured rock. A study by Kim et al. (2015) for a skarn deposit in Korea found porosities that ranged from less than 1% to over 8% and averaged approximately 4%.
- Aquifer testing data from the Project site were analyzed to evaluate porosity using a method created by Ramsahoye and Lang (1961). The method resulted in estimated porosities ranging from 0.1% to 1.6%. However, the results are considered underestimates because the analysis is based on key assumptions (a homogeneous and isotropic aquifer and a cone of depression at equilibrium) that were not observed during aquifer testing.
- Measurements of fracture porosity were calculated from eight gamma-gamma density logs. Average porosity values for the boreholes ranged from 1.31% to 5.73%; the overall average (weighted to account for different borehole lengths) was 2.77%.

16.2.3.2 Hydraulic Conductivity

Excelsior conducted aquifer testing at the Project to determine the hydraulic conductivity and storativity of the ore body. The tests also provide valuable information and data regarding the hydraulic communication between pumping wells and observation wells and core holes. The tested locations cover the range of hydrogeological conditions observed at the site and include tests in typical fractured zones within the different geologic bedrock units, fault intersects, rock masses with limited faulting and highly mineralized zones as well as unmineralized rock formations. Thirty-one (31) tests (including four in the Year 1 wellfield) were conducted in wells completed in fractured bedrock and water level responses were recorded in observation wells. Tested wells are shown on Figure 16-6.

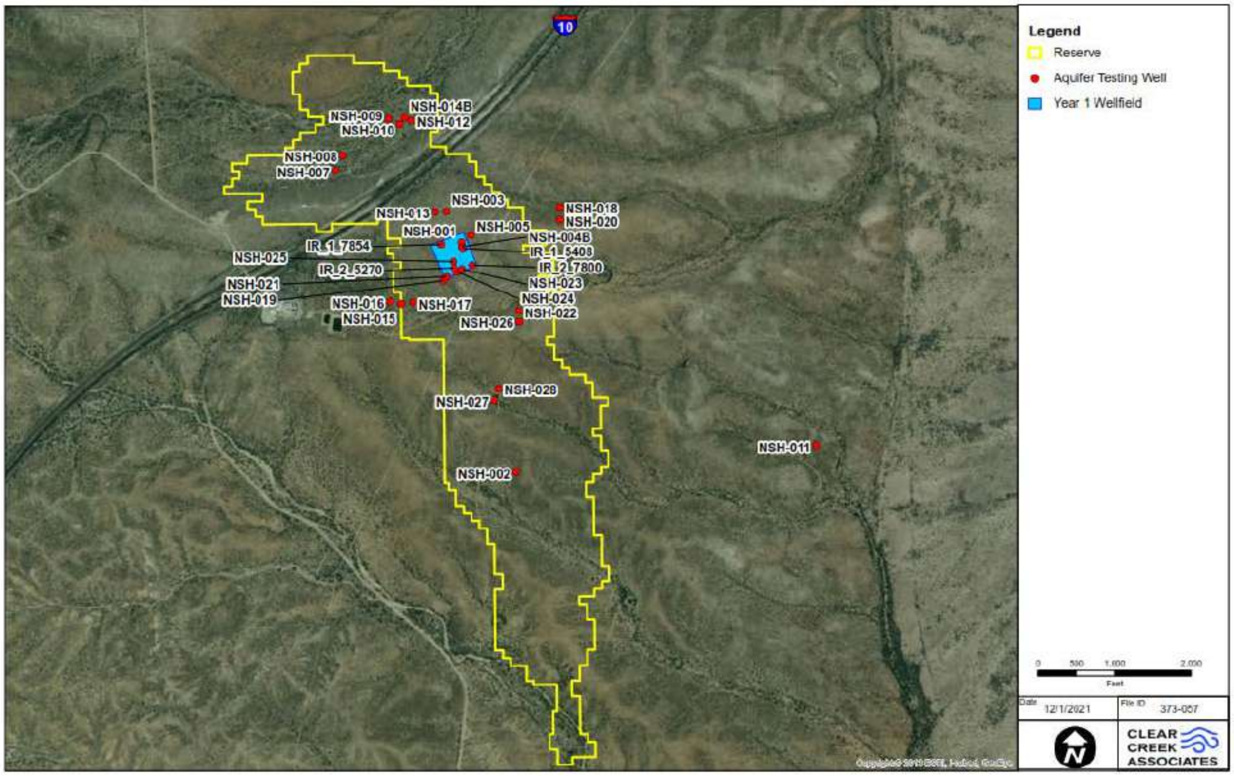


Figure 16-6: Hydraulic Testing Location Map

Testing shows significant variation in hydraulic conductivity values depending upon the fracture density and faulting. The minimum hydraulic conductivity in bedrock is 0.01 ft/day, the maximum is 9 ft/day, and the average is 1.1 ft/day. Testing results are consistent with a confined bedrock system based on the following observations:

- propagation of signal (i.e., connection between wells) over large distances (>1400 feet)
- instantaneous to rapid response to pump start in observation wells
- calculated storativities of less than 10^{-5} (dimensionless)

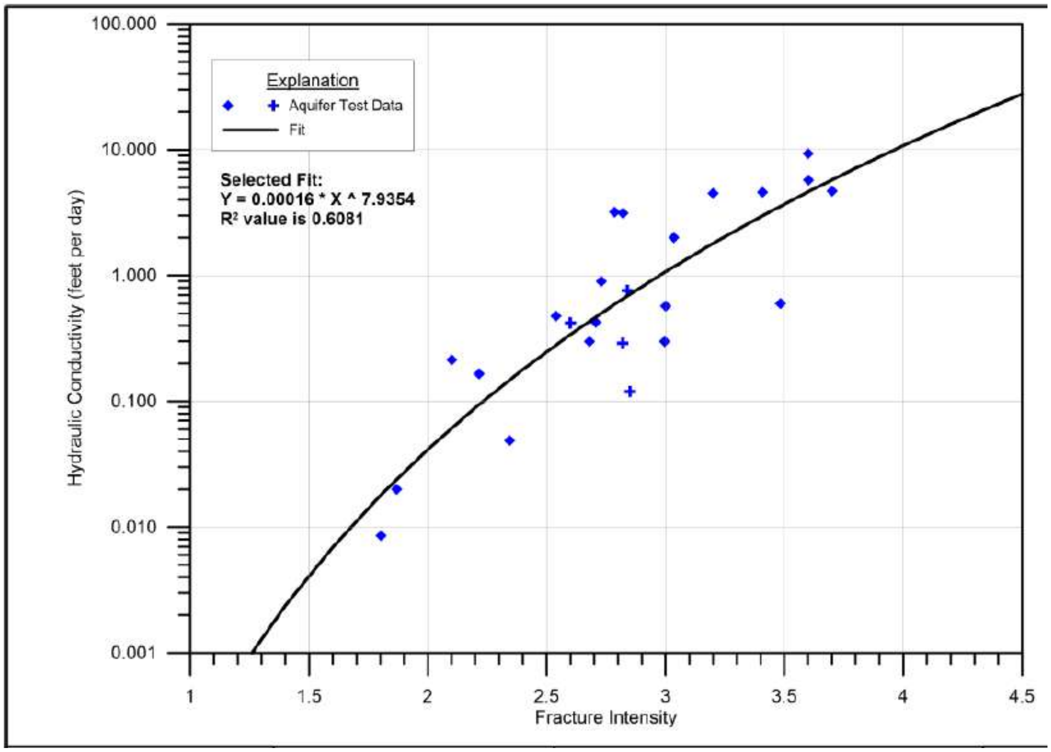
Two bedrock wells completed in the sulfide zone below the oxide zone, NSH-014B and NSH-025, were tested to characterize the permeability of this zone with regard to penetration of lixiviant into an ore where it is not effective for in-situ recovery. The testing showed that the average hydraulic conductivity of the sulfide zone is significantly lower (two orders of magnitude) than the oxide zone, indicating that lixiviant will be maintained within the oxide zone where it can be effective in copper recovery.

Finally, packer testing was conducted in nine wells. Results were generally consistent with the pumping tests. The limited variability in fracture gradients with depth within the ore body supports the concept that hydraulic conductivity is evenly distributed across the vertical dimension in the oxide ore.

Based on the testing results, the significant degree of fracturing, and the responses to pumping in observation wells, modeling bedrock as an equivalent porous medium is a justifiable approach. This is also a reasonable assumption based on the observed hydraulic conductivity and fracture orientations.

16.2.4 Fracture Intensity versus Hydraulic Conductivity

Excelsior estimated fracture intensity from core samples from numerous boreholes, including direct inspection of core and borehole televiewer logs. These data were used for establishing the model hydraulic properties using a correlation relationship between hydraulic conductivity and fracture intensity from the geology block model. Figure 16-7 illustrates a comparison of hydraulic conductivity estimates from aquifer testing with average fracture intensity for each tested zone. The dots represent individual test data, and the fracture intensity interpretation was provided by Excelsior. A power law fit was estimated and is presented on Figure 16-7.



(Data Provided by Excelsior Mining Corp., 2015, updated 2021)

Figure 16-7: Relationship of Fracture Intensity and Hydraulic Conductivity

16.2.5 Sweep Efficiency

The sweep efficiency is defined as the percentage of mineralized fractured bedrock that is in contact with the lixiviant as it circulates between the injection well and surrounding recovery wells. The sweep efficiency is a component of the "recovery factor" applied to the Gunnison resource block model and is based on the fracture intensity of the rocks, which in turn is based on the 3D geologic structural model.

Because the lixiviant must be in contact with the mineralized fractures to recover acid soluble copper, the sweep efficiency influences the amount of acid soluble copper recovered during ISR. "The overall copper recovery from the ore is calculated by multiplying the "metallurgical recovery" by the "sweep efficiency." The metallurgical recovery factor applied to the resource block model was developed by Leach, Inc. The sweep efficiency factor has been selected by Excelsior. A discussion of Excelsior's assumed relationship between the fracture intensity and corresponding sweep efficiency used in the resource block model is provided in Section 13.2. Overall, Excelsior has estimated an average fracture intensity of 2.79, which translates to a sweep efficiency for the deposit of approximately 74%.

16.2.6 Hydraulic Control and Net Groundwater Extraction

The Gunnison Copper Project groundwater flow model was constructed in 2015 by Clear Creek Associates. The model uses the finite difference model code "MODFLOW-NWT" as implemented in the graphical user interface known as "Groundwater Vistas" (v.6.78; Environmental Simulations, Inc., 2011). The finite difference grid consists of 209 rows, 209 columns and 7 layers for a total of 305,767 calculation cells, 173,523 of which are active. The model domain covers an area of 87.8 square miles and encompasses the major hydrologic drainages in the vicinity of the Project (Figure 16-8).

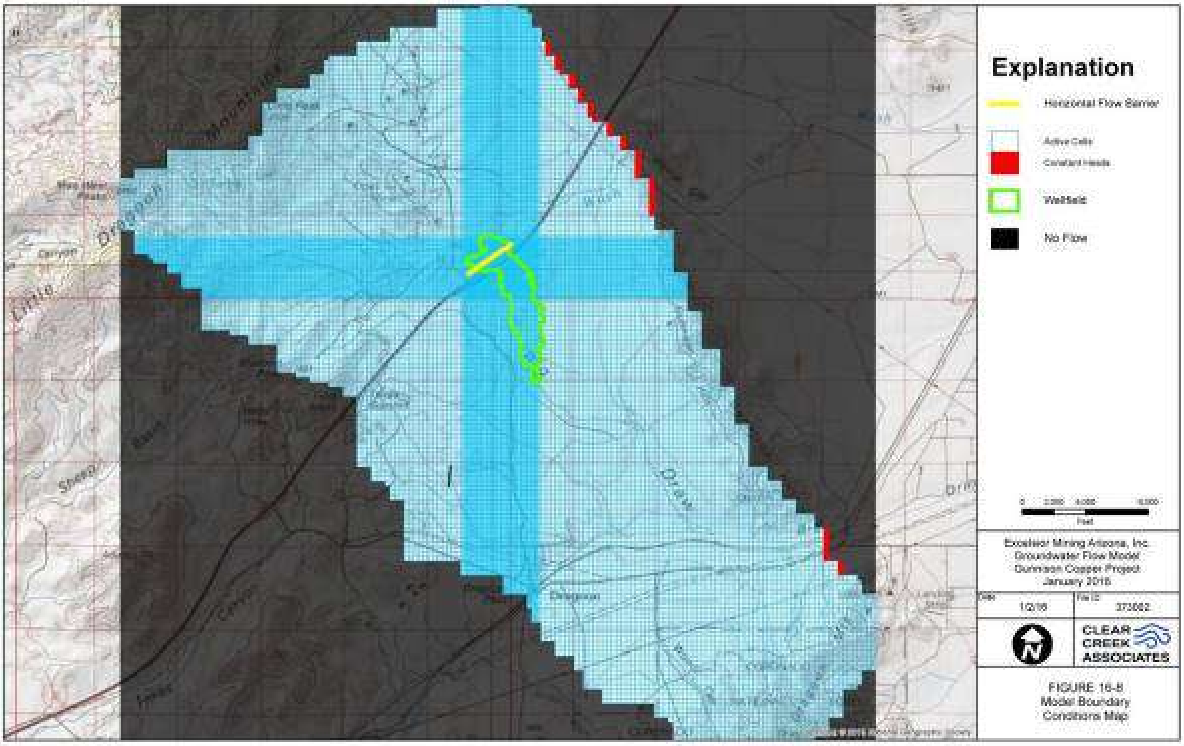


Figure 16-8: Model Grid and Boundary Conditions

The model was constructed using a number of extensive datasets created by Excelsior, including a detailed mapping of fracture intensity, which is key to groundwater flow in the Project area, and the other hydrogeological characterization data discussed above. The model calibrated acceptably, and the statistical match of measured water levels and simulated levels was good with statistical measures better than industry-accepted standards. The model demonstrates that control of mining solutions can be maintained with hydraulic control wells located around the wellfield. Hydraulic control wells (which will supply water to the Project) will generate cones of depression to contain solutions within the ISR wellfield.

16.2.7 Conceptual Hydrogeological Model

Based on the hydrogeological characterization studies discussed above, the conceptual hydrogeological model for the Project consists of the following elements:

- The Project is located within a structurally controlled basin filled with sediments (basin fill). The basin fill thickness in the Project area ranges from about 300 to 800 feet.

- Bedrock beneath the basin fill consists of Paleozoic sediments which have been fractured and altered by the intrusion of the Texas Canyon Quartz Monzonite which is also present in the subsurface at the Project site.
- Bedrock is generally saturated in the Project area. Groundwater flow is within secondary porosity (fractures) that are related to the intrusion of the Texas Canyon Quartz Monzonite and the resulting mineralization. The basin fill is generally unsaturated within the wellfield.
- Aquifer testing results indicate that there is hydraulic communication between wells through bedrock fractures.
- Control of lixiviant in the subsurface can be maintained by pumping of hydraulic control wells around the ISR wellfield.

16.3 WELL DESIGN

Wells at the Project include injection, recovery, hydraulic control, observation, and point-of-compliance (POC) monitoring wells. With the exception of the POC monitoring wells, these wells are and will be constructed to meet Underground Injection Control (UIC) Class III requirements.

Boreholes may be drilling using air rotary, mud rotary, reverse circulation mud rotary, or casing advance methods depending upon which method is most suitable for a given well. Wells may be drilled using two general approaches: either single pass or double-pass approach. The single-pass approach entails drilling to total depth prior to running surface casing whereas the double-pass approach entails running surface casing prior to drilling the completion zone. Standard drilling equipment, casing and bit sizes are utilized in well drilling and completion operations. Exact bit size and casing/liner size may vary depending on hole conditions, drilling results, and material availability.

The injection, recovery, and hydraulic control wells have open-hole completions within the ore body, which ranges from approximately 50 to 1,250 feet in thickness. However, if the borehole is unstable, well screen may be installed. Several possible well designs, including a range of diameters, are used for the injection, recovery, and hydraulic control wells to allow for operational flexibility. Borehole diameters are sufficient to allow for installation of casing that will accommodate the pumps. The cased portions of the boreholes are 12-inch nominal (for small diameter injection/recovery wells and hydraulic control wells), and 17-inch nominal (for large diameter injection/recovery wells). The open borehole sections within bedrock are 7 to 17-inch nominal. (Figure 16-9). Injection/recovery wells are designed to be dual purpose; they can be switched from one function to the other, depending on the need.

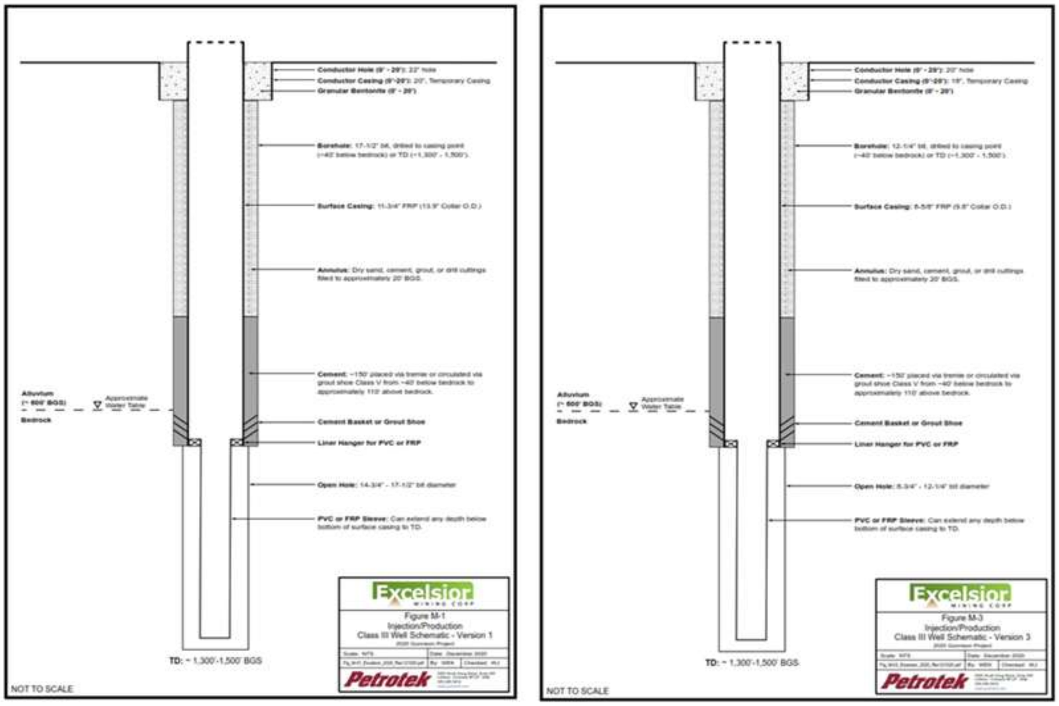


Figure 16-9: Examples of Large and Small Diameter Injection and Recovery Well Designs

Observation wells and POC wells are designed with 9-inch nominal boreholes and have well screen and filter pack (Figure 16-10).

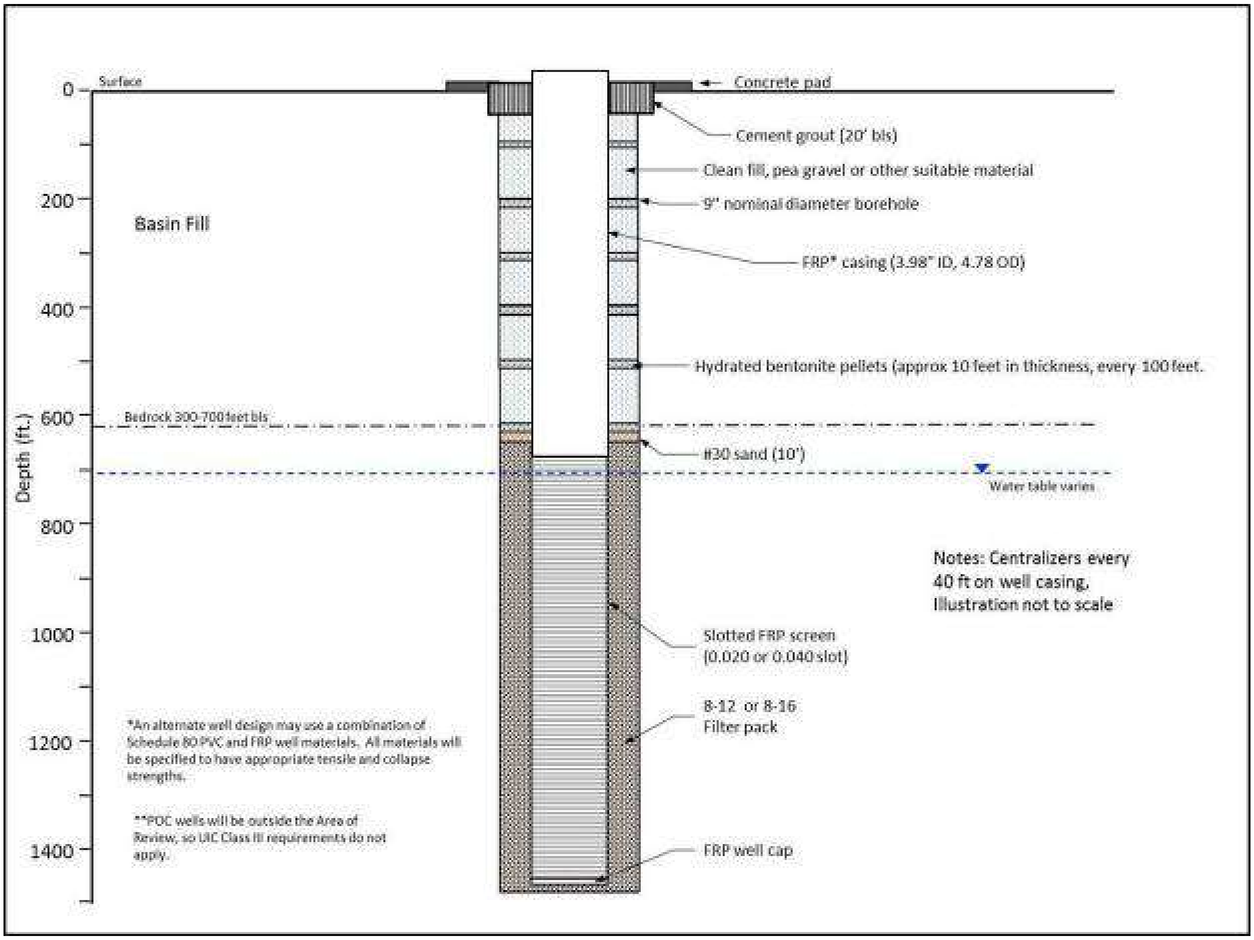


Figure 16-10: Observation and POC Well Design

For all types of wells, casing strings (including the well screen if the well has a screened completion) is of appropriate size and grade to have sufficient collapse, pressure, and tensional strengths to maintain integrity during well construction and for the life of the well. Well materials are compatible with injected fluids and formation fluids with which they are expected to come into contact. Casing centralizers are placed every 40 ft along the casing (and screen, if used) length. The casing string is suspended in the borehole until the annular materials are installed.

The casing annulus of all Class III wells is grouted to at least 100 ft above the basin fill/bedrock contact (or static groundwater level, whichever is shallower) to prevent solutions from moving vertically in the casing annulus, as required by UIC regulations.

16.4 COPPER EXTRACTION FORECAST

The copper production for the Gunnison Copper Project increase in stages. From Years 1-4, production will range from 12 to 55 mppa. In Years 5 and 6, production will be 75 mppa. Starting in Year 7, production will be approximately 125 mppa, and in Year 20 production will drop below 100 mppa. Production will average approximately 98 mppa from Years 1 through 20, with a decline in production beginning in Year 20 (85 million pounds) through the end of the mine life (8.7 million pounds in Year 24). The total amount of copper production forecast over the 24-year LoM is approximately 2,165 million pounds.

Predicted PLS throughput to the SX-EW plant increases from 3,722 gpm in Year 1 to 18,930 gpm in Year 10, with relatively consistent PLS throughput of approximately 18,000-19,000 gpm from Year 7 through Year 18. Predicted average net PLS grade ranges from 0.5 g/L to 1.99 g/L, with the average net PLS grade of 1.51 g/L for the LoM. The copper extraction schedule is provided in Table 16-1.

Table 16-1: Copper Extraction Schedule

Copper Production Schedule (Summary)	Units	Totals	Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Plant Capacity	mppa				25	25	25	75	75	75	125	125	125	125	125	125	125	125	125	125	125	125	125	125	125	125	125	125
Recovered Cu Metal	MM lb	2,153.63			12.08	25.51	26.37	56.48	76.77	73.39	125.13	124.67	126.27	124.42	125.67	122.74	125.68	122.99	126.71	121.12	120.38	103.86	117.94	84.19	94.97	77.14	30.45	8.69
Copper Produced & Sold	MM lb	2,153.63			12.08	25.51	26.37	56.48	76.77	73.39	125.13	124.67	126.27	124.42	125.67	122.74	125.68	122.99	126.71	121.12	120.38	103.86	117.94	84.19	94.97	77.14	30.45	8.69
Copper Produced & Sold from Arizona State Lease	MM lb	210.19			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	44.9	69.2	42.6	39.3	10.0	4.2	-
Net Acid Consumed	MM lb	20,262			113	169	413	407	701	774	1,109	1,006	1,329	1,136	1,272	1,074	1,288	1,031	1,335	922	1,167	905	929	865	961	589	520	248
Average acid consumption	lb/lb	9.36	average		7.90	6.56	16.09	7.01	9.24	10.35	8.81	8.01	10.51	9.05	10.15	8.63	10.32	8.24	10.63	7.45	9.80	8.50	7.90	10.21	10.00	7.63	17.07	28.47
Number of Production CELLS in service	#	229	average		13	18	23	62	54	113	156	172	213	213	197	218	192	302	285	511	367	534	352	346	486	271	259	140
Number of Production WELLS in service	#	303	average		22	28	36	83	78	161	205	237	276	264	234	259	225	353	325	605	589	673	675	419	614	373	349	198
Average Pump Rate per Recovery well	gpm/well	45.5	average		211	149	105	111	120	77	90	77	68	72	79	72	82	53	56	30	26	27	24	31	26	28	25	20
Average Flow rate to Neutralization	gpm	3,524	average	1,250	2,500	0	6,070	460	6,530	6,600	5,810	3,450	5,600	3,120	6,300	1,440	7,865	2,280	8,276	0	8,745	4,050	4,445	3,315	0	0	0	0
Average Flow rate to SX-EW	gpm	13,747	average		3,722	3,746	3,780	9,250	9,350	12,415	18,370	18,250	18,875	18,930	18,465	18,650	18,450	18,610	18,325	18,376	15,236	18,383	16,381	12,950	16,260	10,620	8,584	3,960
Average PLS grade	g/l	1.5	average		1.00	1.57	1.55	1.43	1.85	1.37	1.56	1.57	1.53	1.51	1.55	1.52	1.54	1.53	1.56	1.53	1.78	1.32	1.64	1.49	1.35	1.66	0.81	0.5

The following inputs and assumptions were used to generate the copper extraction forecast:

- Key physical parameters from RESPEC 100-foot x 50-foot resource block model such as rock type, specific gravity of each rock type, percent total copper and percent acid soluble copper, fracture intensity, ore thickness, water table elevation, ore greater than 0.05% total copper, and lease boundaries (see Section 14 for details).
- Incremental acid soluble copper recovery curves over a 4-year recovery period and recovery factor

As discussed in Section 14, the resource block model consists of stacked cells with dimensions of 100 feet x 50 feet x 25 feet thick. Because each vertical column of the resource block model is 100 feet x 100 feet in area, it corresponds to a 100-foot x 100 foot 5-spot well pattern. The resource block model after averaging side by side blocks therefore approximates the well field model.

Based on the estimated incremental acid consumption and copper recovery from metallurgical testing (see Section 13), copper recovery in each resource block is expected to be complete in four years for all rock types. The resource block model therefore estimates the mineral reserve available for extraction (see Section 15). The copper production schedule uses the reserve estimate and assigns recovery rates that will extract the copper resource over a 4-year time period using desired PLS flow rates and PLS grade.

16.4.1 Copper Extraction Sequence

Nine geographical subdivisions were identified to facilitate well field design and production scheduling, resulting in mine “Groups” 1 through 9 (Figure 16-11). The Groups were further subdivided into 29 mine “Blocks” to aid in production scheduling (Figure 16-12).

As shown in Figure 16-12, the production schedule sequence is generally from north to south along the western perimeter of the ISR area (Blocks 1 through 25) followed by south to north production along the eastern perimeter (Blocks 25, 26, and 27), generally following land surface topography from high to low elevation, and the overall west to east groundwater flow direction. The blocks north of Interstate 10 (Blocks 28 and 29)are scheduled to go into production near the end of the LoM, in Years 18 and 20. The production schedule was generated with the goal of producing an average of approximately 125 million pounds of copper per year once the ramp up is complete. Wells are brought on line in sequential blocks over the LoM to maintain this desired production rate.

Note that while the Year 1 wellfield (shown on Figure 16-11 and Figure 16-12) roughly coincides with Mine group 1 and mine block 1, it was rotated to conform with geologic structural elements. This does not change the production scheduling.

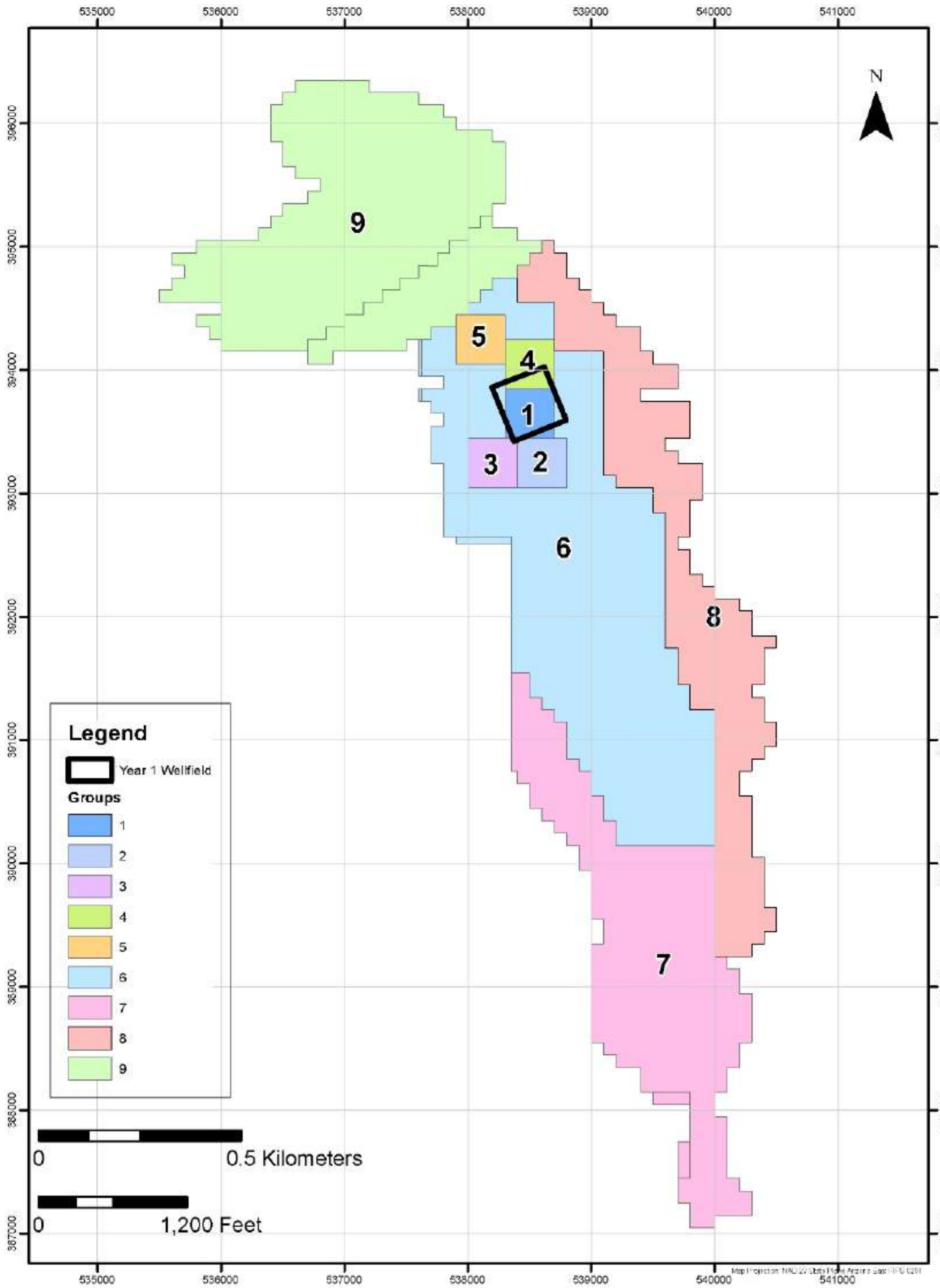


Figure 16-11: Mine Groups

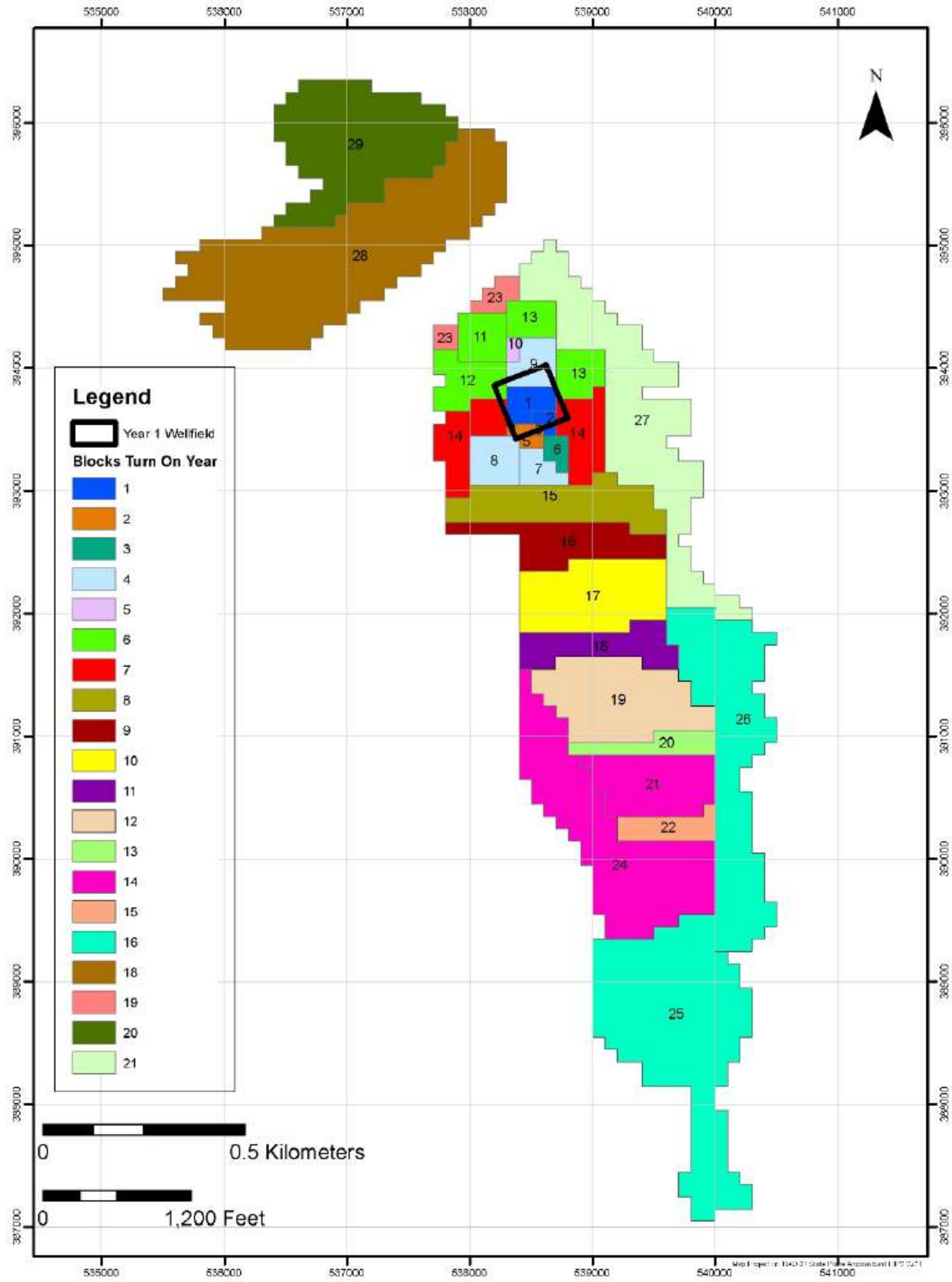
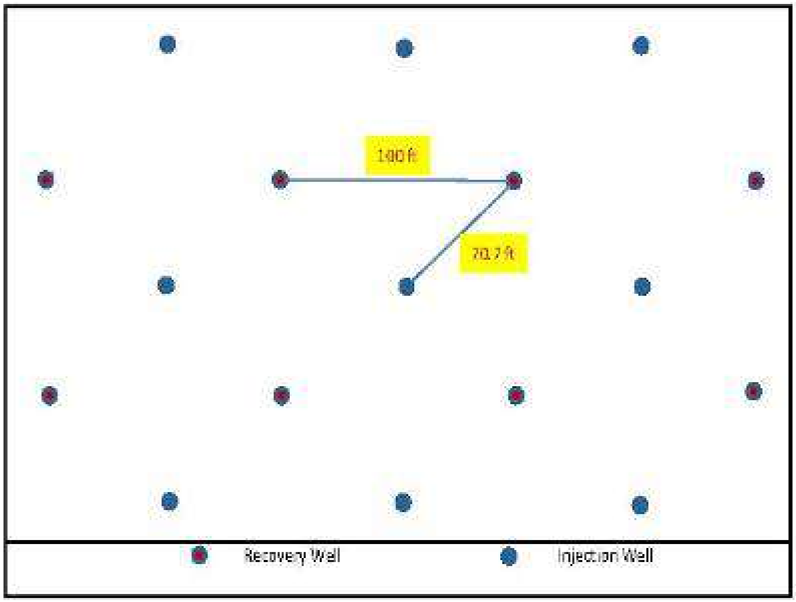


Figure 16-12: Mining Block Sequence Map

16.4.2 Number of Operational Wells

ISR requires injection, recovery, hydraulic control, and observation wells during operation. Injection and recovery wells will be interspaced in an alternating and repeating pattern throughout the wellfield. According to Excelsior’s production schedule, there will be a total of 3000-3100 Class III injection/recovery wells in the wellfield during the life of mine. New wells are anticipated to be installed the year prior for each block brought on line for production. Because injection and recovery wells will be constructed alike, a well can be converted from injection to recovery (and vice versa) by changing out the equipment and wellhead instrumentation.

Figure 16-13 shows a five-spot pattern in which each injection well is surrounded by four recovery wells with a 100-feet spacing between wells in a row and 50 feet between rows. This configuration results in approximately 71 feet between each injection and recovery well. In practice, this arrangement may be revised to optimize recovery, based on geologic and hydrogeological conditions observed during the installation of the wellfield. Aquifer testing will be performed at installation and used to determine the optimal wellfield array configuration.



(Source: 2012 Preliminary Economic Assessment)

Figure 16-13: Conceptual 5-Spot Pattern

Hydraulic control wells will be located around the perimeter of the wellfield, at locations indicated by the groundwater flow model, to prevent the flow of solutions from the ISR area. Observation well pairs will be used to monitor groundwater levels and demonstrate hydraulic gradients toward the ISR wellfield. POC wells for monitoring groundwater quality will be located outside the wellfield to meet monitoring requirements of the Aquifer Protection and Underground Injection Control permits.

16.4.3 PLS Solution and Flow Rates

The annual PLS flow rate for each resource block was estimated using the number of recovery wells per resource block during the operational year and per well recovery rates. Recovery (and injection) rates are expected to vary and will depend on the thickness of the mineralized material under leach (i.e., the recovery well screen length) and the degree of fracturing. Recovery rates will be approximately equal to injection rates to avoid de-watering the ore zone. The average annual PLS grade is predicted to range from 0.5 g/L to 1.99 g/L during the LoM, with an average net PLS grade of 1.51 g/L after adjusting for copper in the raffinate.

Individual recovery well pumping rates are anticipated to range from 20 gpm to 250 gpm (Table 16-2), with an average of approximately 68 gpm during the LoM. Aquifer testing conducted in support of the Aquifer Protection Permit and Underground Injection Permit Applications indicates that these flow rates are achievable. The annual PLS flow to the SX-EW plant is expected to range from a peak of 18,930 gpm in year 10 to 3722 gpm during year 1, with an average PLS flow rate of 13,661 gpm over the LoM.

Table 16-2: Individual Recovery and/or Rinse Well Pumping Rates in GPM

Block	Year 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27
1	208	139	100	40	17	-	10	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	250	146	100	80	80	9	-	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	-	191	150	80	80	30	4	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
4	-	179	150	80	80	30	-	-	12	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
5	-	100	100	80	80	30	-	-	7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
6	-	-	85	85	85	15	54	-	9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
7	-	-	-	109	109	70	70	9	-	6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
8	-	-	-	130	130	70	70	18	-	12	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
9	-	-	-	107	107	50	50	19	-	6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
10	-	-	-	-	150	150	100	80	10	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
11	-	-	-	-	-	100	100	80	20	7	-	4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
12	-	-	-	-	-	80	80	80	55	11	-	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
13	-	-	-	-	-	80	80	80	55	11	-	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
14	-	-	-	-	-	-	120	75	75	50	8	-	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-
15	-	-	-	-	-	-	-	-	80	80	80	50	-	5	-	-	-	-	-	-	-	-	-	-	-	-	-
16	-	-	-	-	-	-	-	-	80	80	80	50	9	-	5	-	-	-	-	-	-	-	-	-	-	-	-
17	-	-	-	-	-	-	-	-	-	80	80	80	80	10	-	6	-	-	-	-	-	-	-	-	-	-	-
18	-	-	-	-	-	-	-	-	-	-	80	80	80	50	10	-	5	-	-	-	-	-	-	-	-	-	-
19	-	-	-	-	-	-	-	-	-	-	-	80	80	60	50	12	-	6	-	-	-	-	-	-	-	-	-
20	-	-	-	-	-	-	-	-	-	-	-	-	80	80	80	45	12	-	4	-	-	-	-	-	-	-	-
21	-	-	-	-	-	-	-	-	-	-	-	-	-	80	80	22	22	10	-	5	-	-	-	-	-	-	-
22	-	-	-	-	-	-	-	-	-	-	-	-	-	-	140	80	80	70	7	-	4	-	-	-	-	-	-
23	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	120	80	80	80	4	-	2	-	-
24	-	-	-	-	-	-	-	-	-	-	-	-	-	35	35	20	16	7	-	4	-	-	-	-	-	-	-
25	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	35	35	24	18	6	-	4	-	-	-	-	-
26	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	30	30	20	13	5	-	3	-	-	-	-	-
27	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	30	30	22	12	4	-	3
28	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	30	30	20	15	5	-	3	-	-	-
29	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	40	20	20	20	5	-	3	-

16.4.4 Hydraulic Control Solution Flow Rates

Hydraulic control wells located around the perimeter of the ISR wellfield will be pumped at rates needed to maintain a hydraulic gradient toward the wellfield. The gradient will be demonstrated by water levels measured in observation well pairs. Hydraulic pumping rates will vary from year to year, based on the extent of ISR operations. Hydraulic pumping will continue until rinsing is complete. The groundwater flow model has demonstrated that individual hydraulic control pumping rates will range from 2 to 17 gpm. These rates will be sufficient, according to model simulations, to contain solutions within the ISR wellfield.

16.4.5 Rinse Solution Control Flow Rates

Each block is scheduled for rinsing once ISR is completed based on the PLS grade economic cut-off. A rinse solution will be injected and recovered within the mining block with the goal of returning formation water quality to Aquifer Water Quality Standards (AWQSs) using a rinse-rest-rinse strategy. Since ISR duration is anticipated to be four years rinsing operations will not start until Year 5 and will continue until year 27.

A geochemical model utilizing industry standard software was prepared using a combination of host rock mineralogical and geochemical data, hydrological data and rinsing data from prior and the most recent metallurgical test work. The geochemical modeling indicates that five pore volumes, based on a 3% porosity[1], will be required to adequately flush the formation during rinsing. This volume was scheduled over a 2-to-4-year time period for each block under rinse to accommodate waste water treatment system flow capacity and to take advantage of the natural attenuation properties of the host rocks. After three pore volumes of rinsing, the system will be allowed to rest. During the rest period, the solution will reach circumneutral pH. Two additional pore volumes of rinsing will follow the rest period. Individual rinsing well production rates range from 2 to 19 gpm, with an average rinsing pumping rate of 7 gpm per rinsing well over the LoM. The annual total flow rate from rinsing ranges from 36 to 2,900 gpm, with an average of 839 gpm over the LoM.

---

[1] Based on weighted gamma-gamma borehole logging results, with an added safety factor.

16.4.6 Limitations/Opportunities

The copper extraction forecast only includes measured and indicated copper oxide mineral resources as defined in the resource block model prepared by RESPEC (see Section 14). The inferred copper oxide mineral resources present within the ISR area are not included in the copper extraction forecast or the economic analysis. Opportunity exists to add inferred mineral resources to future production by appropriately converting these inferred mineral resources to measured or indicated mineral resources. However, recovery of this additional copper would also result in additional acid consumption not accounted for in the current copper production schedule[2].

Prior to production of the ISR wellfield, the pre-development well installation will undergo geological, geophysical, and hydrological testing and modeling. This newly collected hydrological data and modeling will be used to optimize the wellfield design, pumping rates and the production schedule. The recently acquired data will refine aspects of the extraction plan, well design, and hydrologic performance of the ore body, and will help define the effects of individual geologic structures on well field geometry and refinement the well field geometry and hydraulic control.

16.5 CONVENTIONAL MINING FLEET

This Project is an in-situ recovery project, and as such, does not have a conventional mining fleet. The Project includes drilling and well servicing equipment required to develop the ISR wellfield. This equipment, which is included in the sustaining capital cost estimate and financial model, includes:

Table 16-3: Equipment Quantity

Quantity	Equipment
3	Air Compressor/ Booster
1	Water Truck
1	Forklift
1	Boom Truck
3	Reverse Circulation Rigs
3	Service Rigs
1	Wire Line Logging Truck
1	Fuel Truck
1	10yd Dump Truck

---

[2] Acid consumption is currently based on pounds of copper produced, not tons of material in contact with the leaching solution.

17 RECOVERY METHODS

The Gunnison Project uses solvent extraction (SX) and electrowinning (EW) to recover copper from an in-situ recovery (ISR) wellfield. The Gunnison Project is planned for development in three stages. In Stage 1, the existing JCM plant is used to recover 25 million pounds per annum (mppa) of copper cathode from the Gunnison wellfield. In Stage 2, a 50 mppa SX-EW plant will be constructed in the Gunnison Project area south of I-10. This new plant will operate independent of the JCM plant. In Stage 3, the Gunnison plant capacity will be doubled to 100 mppa, resulting in an aggregate capacity of 125 mppa of copper cathode. The SX-EW facilities are designed to recover copper from pregnant leach solution (PLS) to produce cathode-quality copper with 99.99% purity. The process consists of the following elements (schematic representation in Figure 17-1):

- ISR wellfield
- SX mixer-settlers which transfer copper from PLS to the electrolyte solution
- EW cells which recover copper on cathodes that are stripped with an automatic stripping machine
- Tanks which store and handle process liquids at a centralized tank farm
- Evaporation ponds and a water treatment plant which handle excess solutions and provide clean water for conditioning and rinsing leached blocks

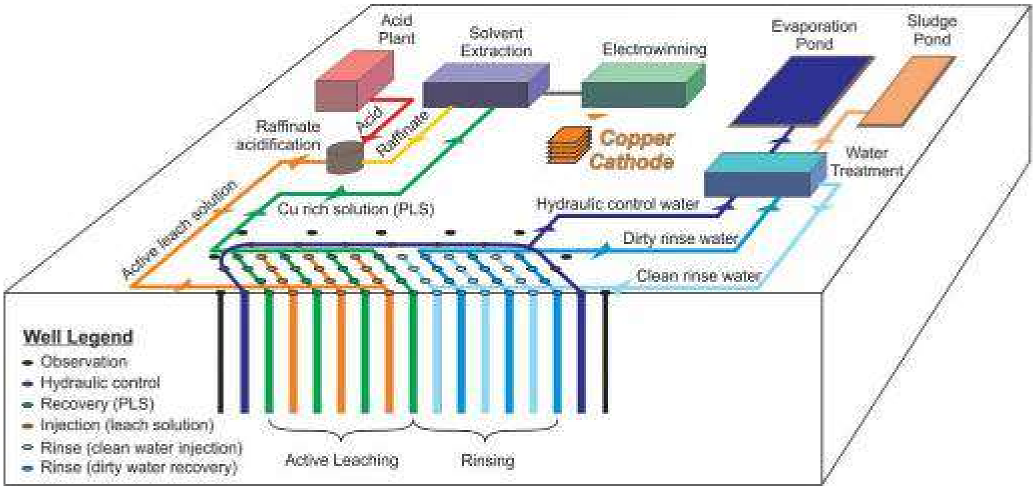


Figure 17-1: Recovery Process

All three stages of the Gunnison production use essentially the same process, as described in Section 17.1. There are minor differences in the design of the Stage 1 JCM plant and the Gunnison Stage 2 and 3 plant. Table 17-1 provides the design criteria for each stage.

Table 17-1: Process Design Criteria by Stage

Parameter	Units	Stage 1 JCM Plant	Stage 2 Gunnison Plant	Stage 3 Gunnison Plant
Nominal production	million pounds per year	25	50	100
Nominal flow rate to SX	gallons per minute	3,890	7,800	15,600
PLS Concentration	copper grams per liter	1.63	1.63	1.63
Extractant	type / concentration	AGORA M5774 or equal	LIX 984N or equal / 11.6%	LIX 984N or equal / 11.6%
Copper/iron transfer ratio	design	1,200:1	1,200:1	1,200:1
Number of SX trains	design	2	1	2
Extraction				
Extraction arrangement	extraction stages / arrangement	2 / series	2 / parallel	2 / parallel
Extraction flow rates	overall / per settler	3,880 / 1,940	7,800 / 3,900	15,600 / 3,900
Organic to Aqueous	ratio	1:1	1:1	1:1
Settler-specific flow rate	gallons per minute per foot	2.0	2.0	2.0
Linear flow velocity maximum	inches per second	2.76	2.76	2.76
Combined SX copper recovery	percent	92	92	92
Stripping				
Stripping arrangement	stripping stages	1	1	1
Stripping flow rates	overall / per settler	3,880 / 1,940	3,900 / 3,900	7,800 / 3,900
Organic to Aqueous	ratio	1:1	1:1	1:1
Settler-specific flow rate	gallons per minute per foot	2.0	2.0	2.0
Linear flow velocity maximum	inches per second	2.76	2.76	2.76
Nominal change in concentratio	copper grams per liter	15	15	15
Electrowinning				
Number of EW cells	number	Block 1 = 56 Block 2 = 32	80	160
Cell construction	material / type	polymer concrete / cross flow	polymer concrete / cross flow	polymer concrete / cross flow
Current density	amperes per square foot operating / design	Block 1 = 28.8 / 28.8 Block 2 = 21.9 / 21.9	22.8 / 28	22.8 / 28
Cathodes	type	316L SS mother blanks	316L SS mother blanks	316L SS mother blanks
Cathodes per cell	number	Block 1 = 21 Block 2 = 36	63	63
Cathode plating dimensions	width x height in inches	36.4 x 46.25	39.375 x 39.375	39.375 x 39.375
Anodes	type	Pb-Ca-Sn rolled	Pb-Ca-Sn rolled	Pb-Ca-Sn rolled
Anodes per cell	number	Block 1 = 22 Block 2 = 37	64	64
Anode dimensions	width x height in inches	33.5 x 46.5	37.0 / 47.75	37.0 / 47.75
Rectifers	number	2	1	2
Rectifer voltage	volts	Block 1 = 120 Block 2 = 70	276	276
Rectifier amps	nominal / maximum	Block 1 = 13,000 / 13,000 Block 2 = 17,000 / 17,000	30,500 / 38,000	30,500 / 38,000
Rich electrolyte concentration	copper grams per liter, nominal	46	46	46
Rich electrolyte concentration	sulfuric acid grams per liter	165	159	159
Rich electrolyte concentration	cobalt parts per million	150	90	90
Lean electrolyte concentration	copper grams per liter, nominal	36	36	36
Lean electrolyte concentration	sulfuric acid grams per liter	180	183.2	183.2
Cell Feed solution concentration	copper grams per liter, nominal	38	36.5	36.5
Cell Feed solution concentration	sulfuric acid grams per liter	176	180	180
Cell Feed solution flowrate	gallons per minute per square foot	0.049	0.049	0.049
Cell Feed solution flowrate	gallons per minute per cell	Block 1 = 24 Block 2 = 41.2	67	67

17.1 PROCESS DESCRIPTION

The copper recovery process for the Gunnison Project uses a conventional SX-EW flowsheet to recover soluble copper from in-situ mineralization using injection and recovery wells. Acidified leach solution from SX raffinate is injected into oxide copper mineralization in the subsurface below the water table through a network of wells. The injection wells are interspersed with recovery wells that pump copper-bearing pregnant leach solution (PLS) from the subsurface at a rate of flow slightly greater than the rate of injection. The PLS from the recovery wells is combined to provide the feed for the SX-EW process. Copper is extracted from PLS and transferred to a high-acid electrolyte in the SX process. The copper-bearing electrolyte is pumped to EW where the copper is plated on stainless steel cathodes. Sheets of plated copper are stripped from the cathodes, bundled, tested, and weighed prior to being shipped to market. The following sections provide details of the copper recovery process. When a block of mineralized material has been depleted of its recoverable copper, the same injection and recovery wells are then used to rinse the formation.

17.1.1 Leaching

Leaching of copper from the subsurface mineralization is accomplished by using injection and recovery wells in the ISR wellfield, as described in Section 16. Raffinate from the SX-EW plant is acidified and pumped to the ISR wellfield through a network of process piping to a series of injection wells that are each surrounded by four recovery wells. The recovery wells create a hydraulic gradient that promotes flow of the acidified raffinate through the mineralized formation. Acid-soluble copper is drawn into solution as it migrates toward the recovery wells. The PLS extracted by the recovery wells is collected in the PLS pond through a network of process piping.

Production wells are arranged in a repeating 5-spot pattern with the central injection well being surrounded by recovery wells. Each of the recovery wells in the interior of the wellfield is surrounded, in turn, by four injection wells. The rate of PLS extraction must be adjusted to slightly exceed the rate of injection for the operating wellfield.

The injection and recovery wells throughout the center of the wellfield are reversible, enabling each well to be used for injection or recovery. This facilitates backflushing of the recovery wells to remove precipitated sulfates on the well screens and well pumps during the initial stages of operation. It also enables the modification of flow paths within the wellfield to remove copper more efficiently.

Blocks of injection and recovery wells are connected by piping (typically 4” diameter) to a valve skid at the edge of the block. The valve skid contains control valves, flow meters, piping connections, and the instrumentation and controls needed to monitor the wellfield operation and track the flows into and out of the wellfield. The headers are located near valve skids to supply leach solution to injection wells and receive PLS from recovery wells. The header pipes (18 to 36 inches in diameter) convey solutions to and from the SX-EW plant.

Barren leach solution (acidified raffinate) is delivered to the injection header with sufficient pressure to distribute through injection piping to each of the injection wells served by an individual valve skid. The injection piping includes an isolation valve, flow meter, control valve, and pressure gauge (Figure 17-3).

17.1.2 Wellfield Conditioning

Flow rates through the wellfield are observed to decline considerably during the early stages of introducing acidified leach solution to a new block. Scientific investigations indicate that the reduction in flow is due primarily to sulfuric acid in the solution dissolves calcite in the fractures, which releases CO_2. The CO_2 gas builds up in the fractures causing “vapor locking,” inhibiting the flow of liquid through the fractures. Dissolving the CO_2 in the fractures is a crucial step to restoring the flow rates.

Excelsior plans to address this situation in new ore blocks by alternating acidified leach solution, which dissolves calcite and creates the CO_2, with neutralized solution, which will dissolve the CO_2 bubbles and restore flow. Alternating acidified with neutralized flows will continue until the calcite is sufficiently removed from the ore block that acidified leach solution flows can be maintained at pre-acid flow rates. Alternating pulses of acidified and neutralized raffinate on approximately one-month cycles are expected to be effective in removing the calcite and CO_2 blockage. The duration of wellfield conditioning for each block is determined by the amount of calcite along fractures in the block, the relative size of the fractures, and the distribution of the calcite within the block. Conditioning is anticipated to require up to 15 months or less for each added block of wells. When the pre-acid flow rates have been restored, conventional copper in-situ leaching can proceed.

A water treatment plant (WTP) is required in the plant design to neutralize the solutions to enable dissolution of the CO_2 during conditioning. The WTP uses milk of lime to bring up the pH while dropping out solids as gypsum, metal hydroxides, and other sulfates. The solids are settled in a flocculating clarifier and partially recycled to the reactor tank to increase the particle size to facilitate settling. The remaining underflow from the clarifier will be discharged to lined ponds or impoundments where the solids can settle, and the supernatant water can be decanted and returned to the process.

17.1.3 Solvent Extraction

PLS is pumped from each recovery well by an electric submersible pump. The pump discharge is conveyed to the valve skid where the control valves direct the flow through the flow meter and into the PLS header. Aggregate flow from the recovery wells is collected in the header and conveyed to the PLS pond and then pumped to the SX circuit(s) for extraction of copper. The SX circuits for the Gunnison Project consist of trains of mixer-settlers that strip copper from the PLS and transfer it to the lean electrolyte solution. Each train has two extraction settlers and one strip settler (Figure 17-4). The extraction settlers use an extractant contained in a petroleum-based liquid (“organic”) to extract the copper from the aqueous phase. The strip settlers (one in each train) use a high-acid aqueous phase (electrolyte) to strip the copper from the organic phase. The electrolyte is then pumped to EW for recovery by electrowinning.

The SX trains for the Gunnison Project are designed to operate in parallel, which means that half the PLS goes to each extraction settler in the train. The SX trains for the JCM plant are operated in series such that the entire PLS flow through each train passes through both of extraction settlers in the train. The organic passes through both extraction settlers, extracting copper from the PLS and becoming “loaded organic.” The copper-bearing loaded organic is mixed with lean electrolyte in the stripper pumper mixers to transfer the copper from the extractant in the organic phase to electrolyte solution. The stripper settler allows the immiscible liquids to separate in laminar flow. The rich electrolyte then flows to the Electrolyte Filter Feed Tank.

Stripped organic is sent to the extraction pumper mixers where agitated contact between the organic and PLS solutions promotes adsorption of the copper by the extractant in the organic phase. The extraction settlers allow the immiscible liquids to separate in laminar flow so that the aqueous solution (raffinate) and organic solution can be collected in separate launders at the end of the settler. Raffinate is re-acidified in the aqueous launder of the second extraction settler and flows by gravity to the Raffinate Pond. The partially loaded organic from the second extraction settler flows to the pumper mixers of the first extraction settler and adsorbs copper from the other half of the PLS stream. Fully loaded organic from the first extraction settler flows to the Loaded Organic Tank. The SX process is designed to extract 92% of the copper contained within the PLS at an incoming copper grade of 1.63 grams per liter (g/L).

Gunnison wellfield: Wellhead (foreground), valve skid (middle ground), and VFD skid (background)

Valve skid detail showing piping manifold and connection to main PLS and raffinate pipelines.

VFD skid detail showing switchboard, distribution panel, VFDs, and exit cabling to wells

Reversible wellhead showing raffinate and PLS pipeline connections instrumentation, and valving

Figure 17-2: Wellfield Development Photos

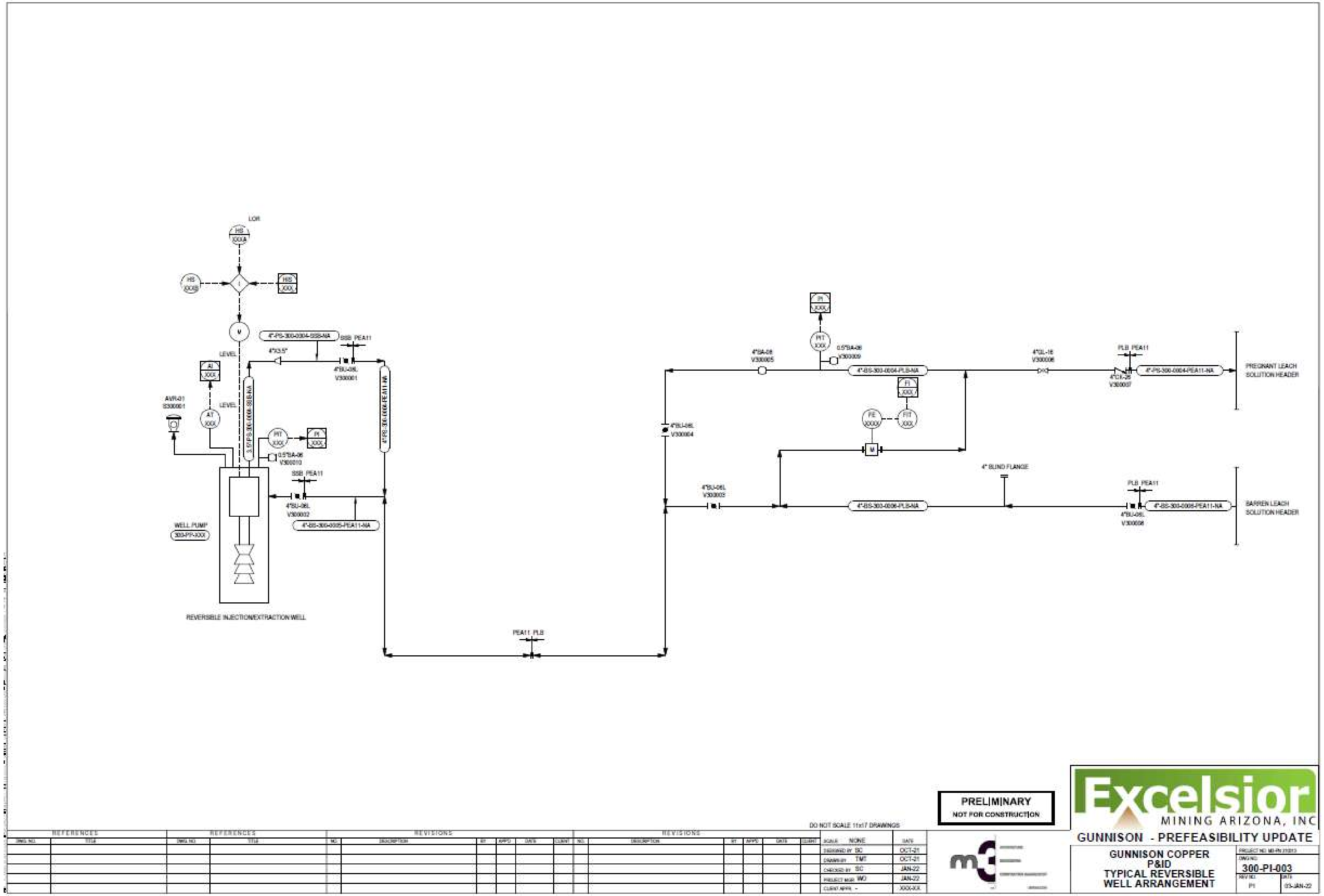


Figure 17-3: Reversible Injection/Recovery Well Controls

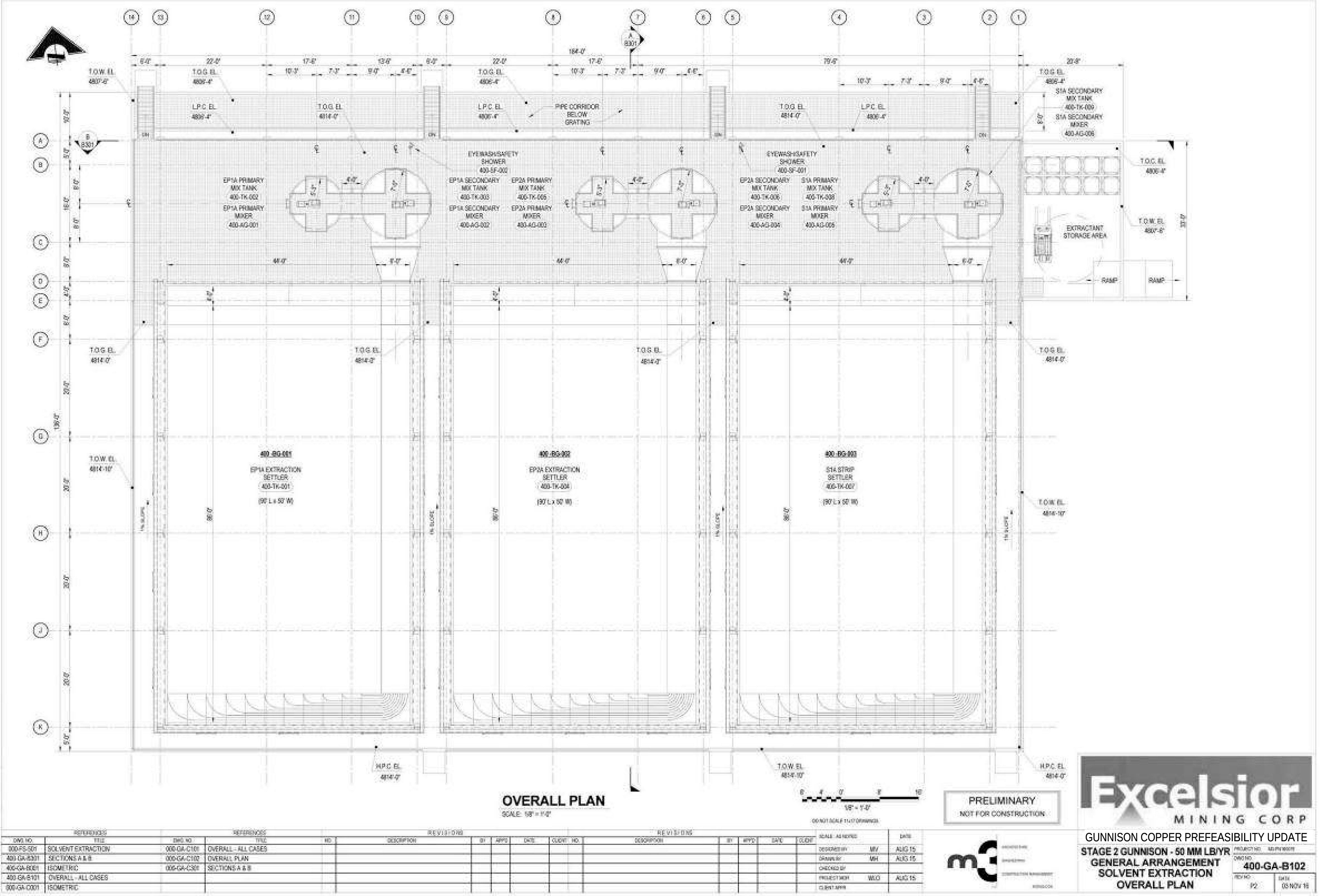


Figure 17-4: Solvent Extraction General Arrangement

17.1.4 Electrowinning

Removing copper from the rich electrolyte solution is accomplished by electrowinning and takes place in the Electrowinning Building or "Tankhouse" (Figure 17-5). Rich electrolyte solution advanced from the solvent extraction area flows by gravity to the Electrolyte Filter Feed Tank. Electrolyte is pumped from this tank through two electrolyte filters to remove entrained organic emulsion and particulates from electrolyte prior to electrowinning. The filters are backwashed periodically with water (or lean electrolyte solution) and air from an air scour blower. In Stage 1, filter backwash solution flows by gravity to the JCM Raffinate Pond. In the Stage 2 and 3 plant, the filters are backwashed with lean electrolyte and the backwash solution is pumped to the PLS Pond.

Filtered electrolyte solution is pumped to an electrolyte recirculation tank through the electrolyte heat exchangers. The filtered rich electrolyte flows through one heat exchanger and is warmed by lean electrolyte returning to solvent extraction from electrowinning. Rich electrolyte is heated to the final temperature for electrowinning in the trim heater, when required, with supplemental heat from a hot water heating system. When supplemental heat is not required, lean electrolyte flows through the trim heater, countercurrent to the flow of rich electrolyte being heated.

In Stage 1, heated electrolyte solution enters an electrolyte recirculation tank and is mixed with electrolyte solution flowing in from the Lean Electrolyte Tank. In Stages 2 and 3, comes from the lean electrolyte portion of the Electrolyte Recirculation Tank. The electrolyte solution exits the EW cells and flows by gravity to the Lean Electrolyte Tank (Stage 1) or the lean side of the Electrolyte Recirculation Tank (Stages 2 and 3), which are equipped with pumps for sending electrolyte to the SX stripping circuits. Excess lean electrolyte is mixed with rich electrolyte for feeding the electrowinning cells.

Copper is plated onto stainless steel cathode blanks in the EW cells. The copper cathodes are harvested on a weekly basis. The tankhouse has an overhead bridge crane for transporting cathodes (and anodes) to and from the cells using a cathode (anode) lifting strongback. Harvested cathodes are washed in the Cathode Wash Tanks using circulation pumps. Washed cathodes are removed from the stainless-steel blanks, sampled, weighed, and banded using a semi-automatic stripping machine. Copper produced by this process is LME Grade A for sale on the world market in 2- to 3-ton packages.

17.1.5 Tank Farm

The tank farm (Figure 17-6) for each plant contains tanks, pumps, and filters for handling solutions needed for the SX-EW process. The primary process function of the tank farm is storage and transfer of solutions. There are two process functions that take place in the tank farm: electrolyte filtration and crud treatment.

Electrolyte filters in the tank farm remove impurities from the rich electrolyte returning from SX to prevent contamination of the tankhouse and electrolyte system. Rich electrolyte flows by gravity to the Electrolyte Filter Feed Tank and is pumped through one or more anthracite-garnet filters to remove entrained organic and particulates that could interfere with the electrowinning process. Filtered rich electrolyte flows to the Electrolyte Recirculation Tank. The filters are periodically backwashed to remove impurities and maintain design flow rates through the filter media.

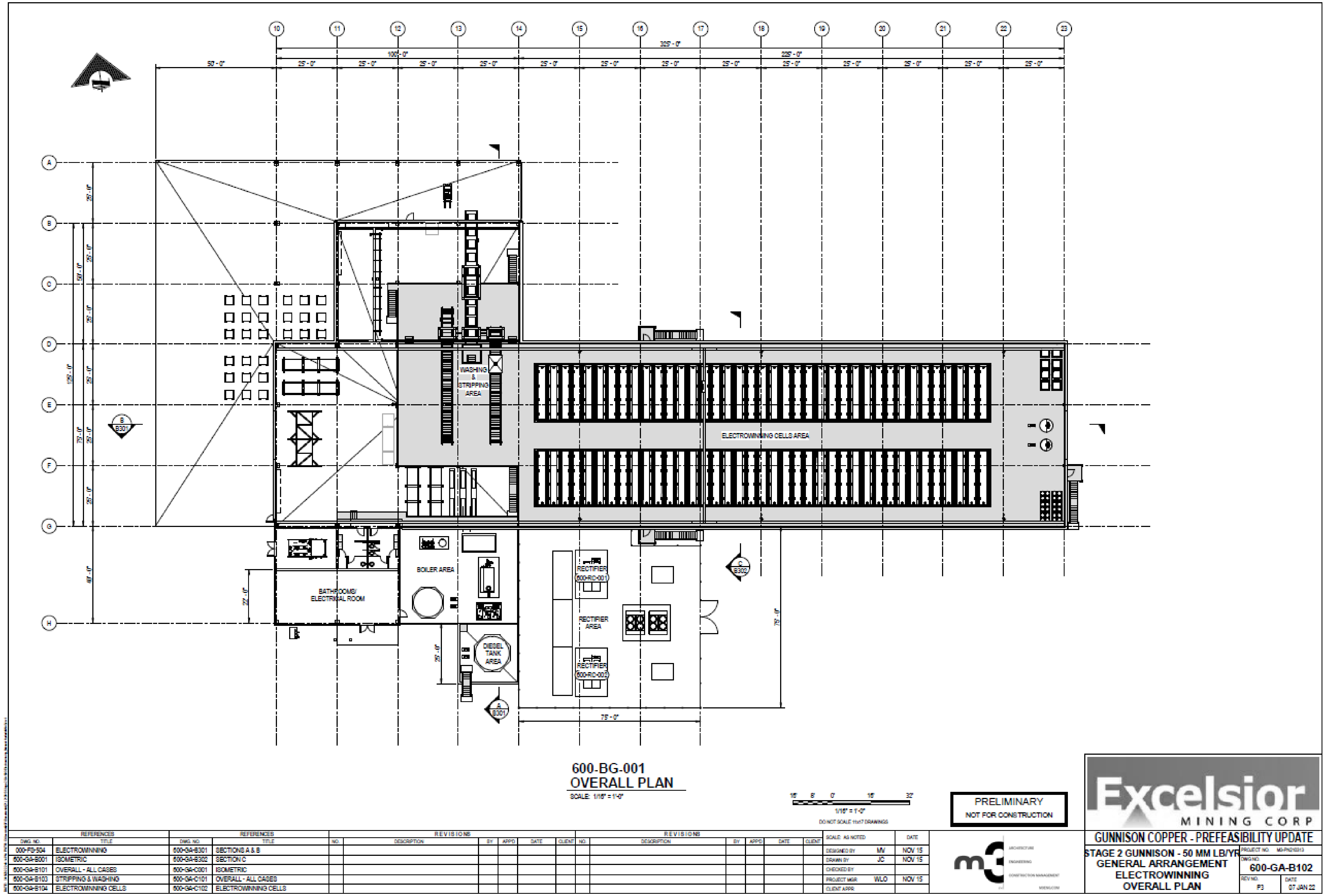


Figure 17-5: Electrowinning Overall Plan

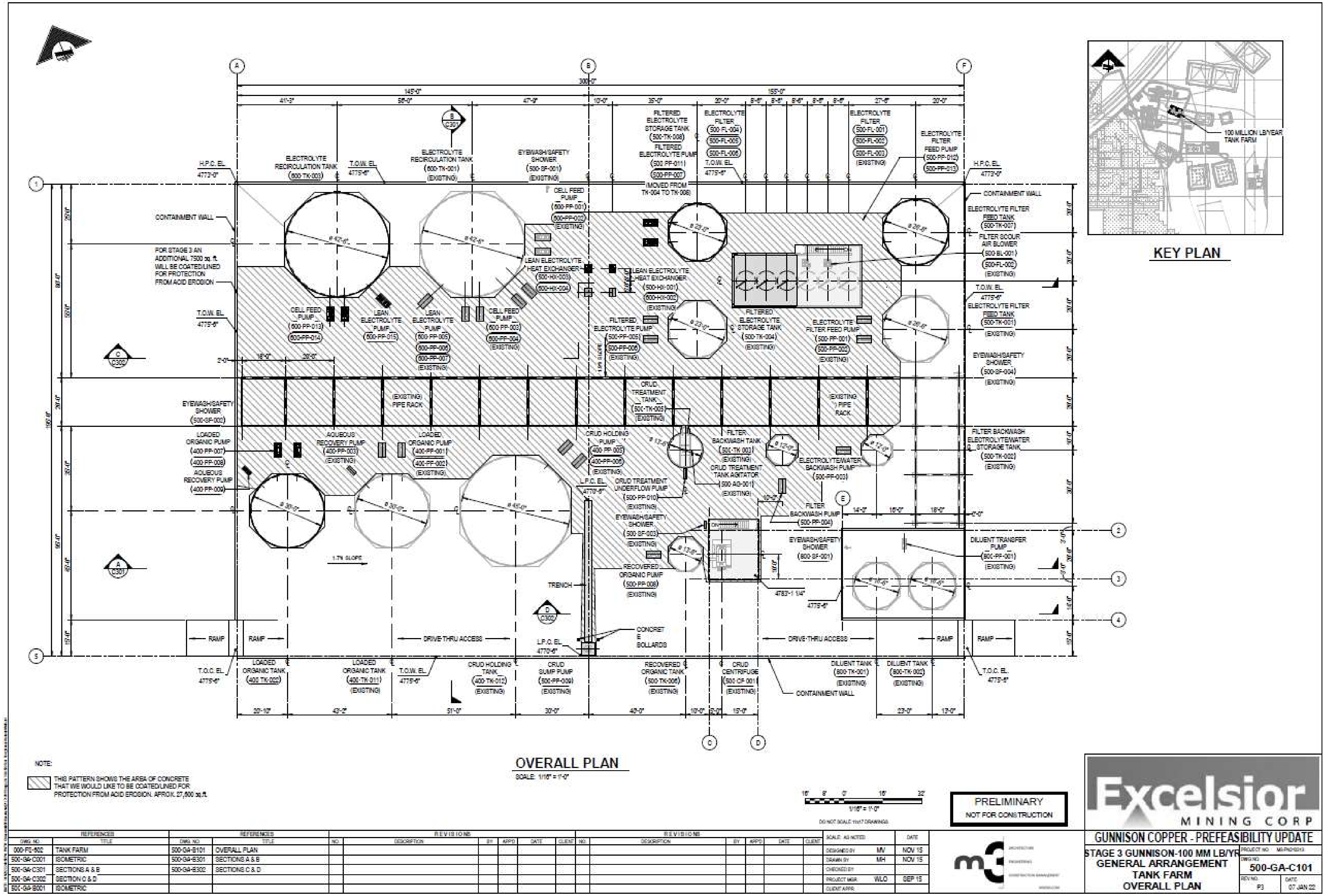


Figure 17-6: Tank Farm Overall Plan

Crud is a mixture of solids, organic liquid, and aqueous solution that accumulates at the organic/aqueous interface in the settlers or any mixture of aqueous and organic liquids that requires separation. Crud is removed by suction from the settlers and needs to be treated to separate the three phases for reuse in the process or, in the case of the solids, for disposal. Crud also comes from the mixture of aqueous, organic, and solids that accumulates in the electrolyte filters. The crud treatment system consists of the following major equipment.

- Crud Holding Tank
- Crud Treatment Tank
- Crud Centrifuge (Tricanter)
- Recovered Organic Tank

Crud from the Crud Holding Tank will be pumped to the Crud Treatment Tank, an agitated, cone-bottom tank. Amendments including clay and diatomaceous earth can be added to the Crud Treatment Tank to assist in separation of the phases. The Crud Centrifuge is a horizontal-axis centrifuge that separates the crud into its three component phases, allowing aqueous and organic liquids to be returned to the process. Solids are collected in a container for offsite disposal.

17.1.6 Rinsing

The mineralized formation becomes depleted of its leachable copper in approximately four years. The formation is then rinsed using the same injection and recovery wells that were used during leaching. Clean water from water supply wells or permeate from the WTP is injected to flush out the remaining leach solution and reduce the concentrations of dissolved solids in the formation. Rinse water (rinsate) from the recovery wells is directed to the PLS pond if it contains recoverable copper or the WTP. The WTP is expanded in Year 7 to add membrane filtration that is necessary to make low sulfate permeate that is necessary to reduce constituent level in the formation sufficiently to comply with reclamation and closure standards.

17.1.7 Evaporation

The Gunnison Project is designed as a “zero-discharge” facility. All excess process solutions and mine-impacted waters will be sent to Evaporation Pond #1. A double-lined evaporation pond with leak collection and removal system (LCRS) is used to contain and evaporate excess water. The pond is equipped with shore-mounted, ducted-fan type mechanical evaporator units, installed around the edge of the pond in positions commonly upwind from the pond. Water is pumped at high pressure through the spray head and blown with a fan out across the pond. The sprayers will automatically shut off when adverse wind directions and/or wind velocity exceeds a level that may result in overspray. Mechanical evaporators will be supplied solution from floating submersible pumps to allow sediments to settle on the pond bottom and minimize interference with and clogging of the sprayers. After operations have ceased, the liquids will be evaporated and the remaining solids will be covered, graded to shed surface water, and revegetated so that evapotranspiration exceeds annual precipitation infiltration.

This pond was constructed as part of the Stage 1 construction execution and is operating as designed. Excelsior added a floating evaporator header within the pond after commissioning and operating of the pond. It is currently operating at 70% of capacity.

17.1.8 Solids Dewatering

Solids produced by the WTP from neutralization, coagulation, and settling are pumped to solids impoundments for containment, dewatering, and solidification. Solids slurry is discharged into the impoundment at a slurry density of 10 to 20 percent by weight. The slurry settles in the impoundment and the excess water is recirculated to the WTP.

17.1.9 Reagents

There are several reagents required for the SX-EW process.

- Sulfuric Acid
- Diluent
- Extractant
- Cobalt sulfate
- Guar
- FC-1100
- Mist suppressor

Sulfuric acid storage tanks are provided to store approximately 14 days of the acid supply required for leaching and making the electrolyte for the EW process. Concentrated sulfuric acid is delivered by tanker trucks for Stages 1 and 2 of the Project. A molten sulfur burning sulfuric acid plant is planned for construction in Stage 3 to provide the acid necessary for leaching and SX-EW process make-up for Stage 3 production.

Diluent provides a petroleum liquid base for the extractant used as the organic phase of SX. The Diluent Tank stores makeup liquid to compensate for evaporative and process loss of organic. Other reagents include extractant, the active ingredient in the organic phase that transfers copper from PLS to electrolyte; cobalt sulfate, an additive to the electrolyte to improve plating; guar, a cathode smoothing agent; and mist suppressor, a chemical added to the electrolyte to inhibit the formation of acid mist in the tankhouse.

17.2 SUPPORTING SYSTEMS

There are several systems that are necessary to support the SX-EW operation. These include systems to contain solutions, convey solutions, provide water, control the process, suppress fires, and ensure that mine-influenced solutions in the subsurface do not migrate offsite.

17.2.1 Central Piping and Power Corridor

The ISR wellfield is managed using valve skids that each serve a block of injection and recovery wells. Valve skids are connected to the processing plant through a central piping corridor (Figure 17-7). The corridor contains the large diameter piping necessary to convey solutions to and from the piping manifolds on the valve skids in operation at any given time. Connections at the valve skids to header pipes enable the operators to direct solutions to various process ponds and tanks over the operating life of the Project.

Power and communications will be delivered to the VFD skids via pole-mounted power distribution system and fiber-optic cables along the main piping corridors. A pad-mounted transformer near a single VFD skid for each mining block will drop the voltage to 480-volt, 3-phase current to provide power to the recovery well pumps and to operate controls in the well heads and valve skids. Since each mining block has four VFD skids, power from the skid near the transformer will connect to the distal VFD skids on the perimeter of the mining block.

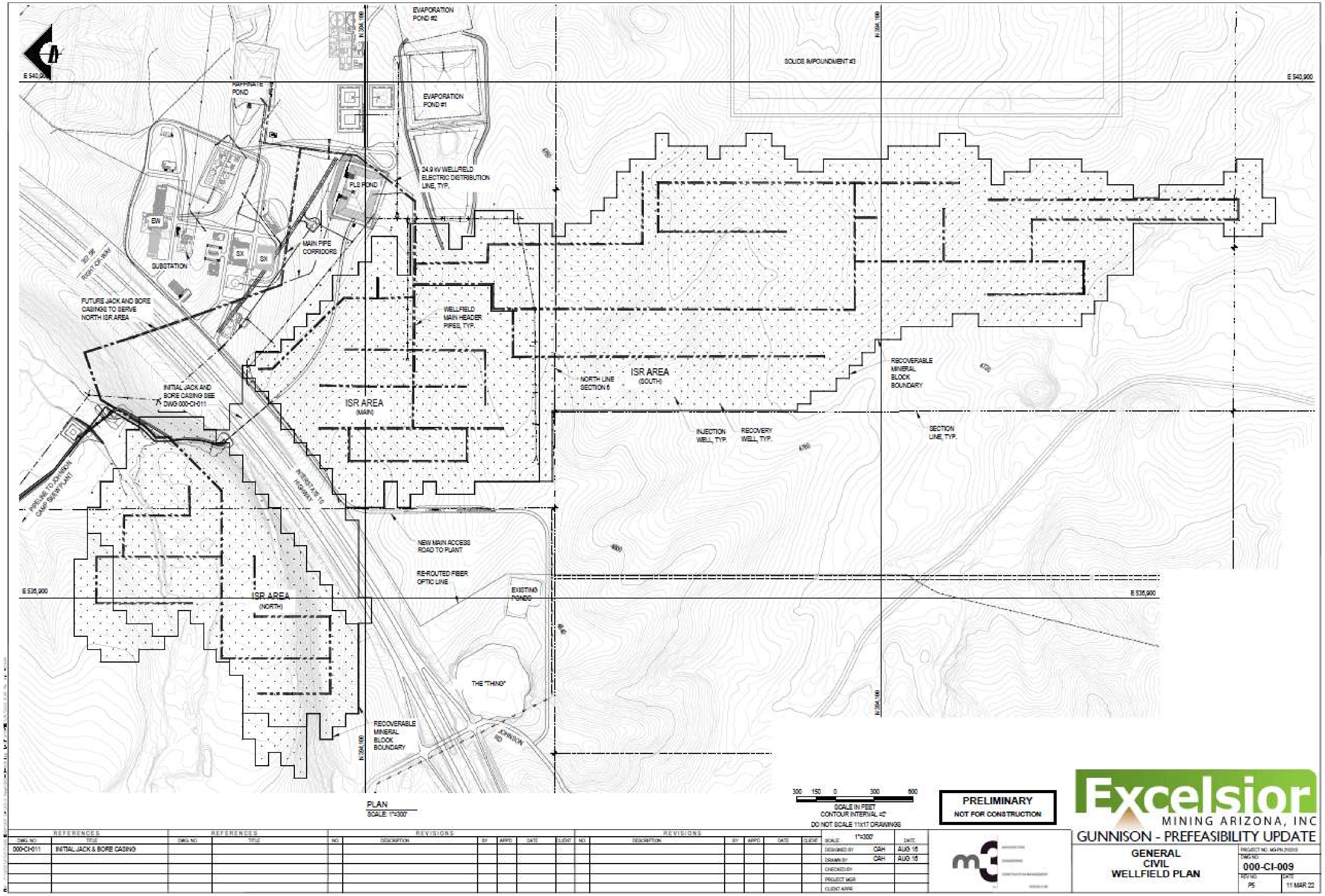


Figure 17-7: Civil Wellfield Plan

17.2.2 Process Control and Monitoring

The operational data from instrumentation in the wellfield is transmitted via fiber-optic cables to the control room in the EW building where it is monitored by a computerized plant control system (PCS). Communication between the PCS and the main control enclosures is by fiber-optic cable. The operator in the control room uses the PCS to monitor conditions at each well and communicates any abnormal conditions to the wellfield operators. The control room operator can turn off pumps, adjust flow conditions, and monitoring line pressures from the control room, but restarting pumps, is reserved for the wellfield operators.

The PCS is also equipped with data loggers to record information from the instruments at each well to enable the operator to examine trends, calculate local and cumulative flows, set alarm conditions, and maintain production records. The PCS provides trending, historical and alarm data for level sensors, flow meters, and any other instrumentation required in this system. Alarms are triggered when monitored parameters are out of limits set by the operator. Alarms will also be generated when there is a communications fault, equipment or instrument failure, or a process that is out of control limits.

17.2.3 Process Ponds

Process ponds are used to store and handle the various liquids and liquid-solid mixtures that are involved in the SX-EW process. PLS ponds collect copper-bearing solutions from the ISR wellfield, allow particulates to settle, and provide a source for feeding the SX plant. The Stage 2 PLS pond has already been constructed as part of Stage 1. Raffinate ponds collect the solution from which copper has been removed (raffinate) and provide a source of acidified solution for leaching to the ISR wellfield. These ponds are managed so that they have a reserve of solutions to maintain SX-EW and ISR operations if one or the other is interrupted and surge capacity to contain the solutions if the other part of the operation is not operating. Both sets of ponds are equipped with pumps and piping to remove the stored solutions and deliver them to the necessary destination at the variable flows and adequate pressures.

Other ponds for the Gunnison Project include the Pipeline Drain Pond (already constructed), Clean Water Pond, Recycled Water Pond, Evaporation Pond #1 (constructed in Stage 1), Water Treatment Feed Pond, and solids impoundments. The Pipeline Drain Pond is situated at a low point between the Gunnison site and the JCM site that allows the contents of the pipelines between the facilities to be drained, if necessary, to perform maintenance or repair work. The Clean Water Pond is a reservoir of well water from the water supply system and permeate from the WTP for rinsing of depleted well blocks. The Recycled Water Pond receives neutralized solutions from the WTP that are pumped to the wellfield for wellfield conditioning prior to copper production from new blocks of wells.

17.2.4 Hydraulic Control Wells

Hydraulic control wells are used at the margins of the ISR wellfield to ensure that groundwater impacted by the leaching process is contained within defined boundaries. Hydraulic control and observation wells are used to control the hydraulic gradient and ensure that the flow of groundwater is toward the ISR wellfield throughout its perimeter. Hydraulic control wells are positioned on the “downgradient” perimeter of the wellfield to cause a depression in the phreatic (water table) surface to “capture” any impacted groundwater. The hydraulic control wells ensure an inward hydraulic gradient i.e., groundwater movement is toward the operating wellfield. The hydraulic control wells are designed by location and extraction rate to capture PLS before it flows out of the permitted wellfield area.

Observation wells are located outside of the hydraulic control wells to demonstrate that the groundwater gradient (i.e., flow direction) is inward (i.e., toward the wellfield). Water levels are measured in pairs of observation wells, one near the hydraulic control wells and the other farther away in the direction of natural groundwater movement, to verify that the phreatic surface of the aquifer near the wells is at a lower elevation than that of the one farther away, indicating that the flow direction is toward the wellfield. If not, extraction (pumping) rates in the hydraulic control wells are increased until the flow direction is once again toward the wellfield.

Hydraulic control water pumped from these wells is directed into one of two collecting pipelines. One of the pipelines conveys hydraulic control water that is unimpacted by ISR operations to Evaporation Pond #1 or the Clean Water Pond after it is constructed, where it can be used for wellfield rinsing. Hydraulic control water that is from wells that have been impacted by the ISR operations are conveyed to Evaporation Pond #1.

Additional hydraulic control and observation wells will be installed as necessary as the wellfield develops. Pumping rates will be increased at hydraulic control wells in response to observation wells in their vicinity that suggest the inward gradient is not being maintained. Groundwater sampling and analytical testing at observation wells will also be conducted on a regular basis to evaluate for any evidence that impacted groundwater is migrating past the hydraulic control perimeter.

18 PROJECT INFRASTRUCTURE

18.1 SITE LOCATION

The Gunnison Project is in Cochise County, Arizona, on the southeastern flank of the Little Dragoon Mountains in the Johnson Camp Mining District. The property is about 65 miles east of Tucson, Arizona, along Interstate Highway 10 (I-10), between Benson, Arizona and Willcox, Arizona (Figure 18-1). Initial (Stage 1) production from the Gunnison wellfield is being processed at the Johnson Camp Mine (JCM) SX-EW plant north of the interstate. Stage 1 modifications and connections to the JCM SX-EW plant were completed in 2020. Processing and support facilities for Stage 2, including a new SX-EW plant, will be constructed south of the interstate and expanded in Stage 3. The Stage 2 and 3 process facilities are located east of the ISR well field and south of I-10 in Section 31, which is referred to as the Connie Johnson property.

18.2 ACCESS ROADS

The primary access to the site will be from I-10 via the North Johnson Road exit between Benson and Willcox, Arizona. The exit is at the location of “The Thing” attraction on the south side of I-10. Stage 1 processing facilities are located at the JCM site north of the interstate, and the Gunnison Stage 2 and 3 processing facilities and initial wellfield are located south of the interstate (Figure 18-2). The roads to and within the JCM site are existing and there are no plans for additions or improvements.

The Gunnison site is accessed by a new gravel road which connects to Johnson Road south of “The Thing” attraction and runs along the south line of Section 36, (T15S, R22E) to the guard house. That road will be expanded and improved during the construction of the Stage 2 plant, which will include access to the new facility areas (Figure 18-2).

18.3 PROCESS BUILDINGS

The Stage 1 process facilities are present at the JCM plant site. Existing mixer-settlers, tank farm, and electrowinning building will be used to process copper-bearing solutions pumped up to JCM from the Gunnison wellfield (Figure 18-3).

Stages 2 and 3 of the Project include the addition of process facilities on the Gunnison side consisting of solvent extraction mixer-settlers, a tank farm, and an electrowinning building (Figure 18-4). For Stage 2, the solvent extraction settlers consist of three, covered mixer-settler tanks (Figure 17-4), and the electrowinning building (tankhouse) consists of a steel building with metal roofing and siding. The Stage 2 electrowinning cell area is on one end of the building and the automatic stripping machine, and the cathode handling equipment are on the other, with a paved cathode storage area outdoors (Figure 17-5). An electrical equipment room and a control room above are located near the cathode stripping area so that personnel in the control room can observe the entire operation. Cathode handling, weighing, and banding is performed at the cathode handling section. A paved storage yard is provided outside the cathode handling area to allow cathode storage and loading of cathodes onto flatbed trailers for shipment to market. The building is provided with ventilation fans to circulate air in the cell area. The transformer-rectifiers which provide direct electrical current for electrowinning are located outside and upwind of the building to minimize impacts from mist and vapors evolved during electrowinning.

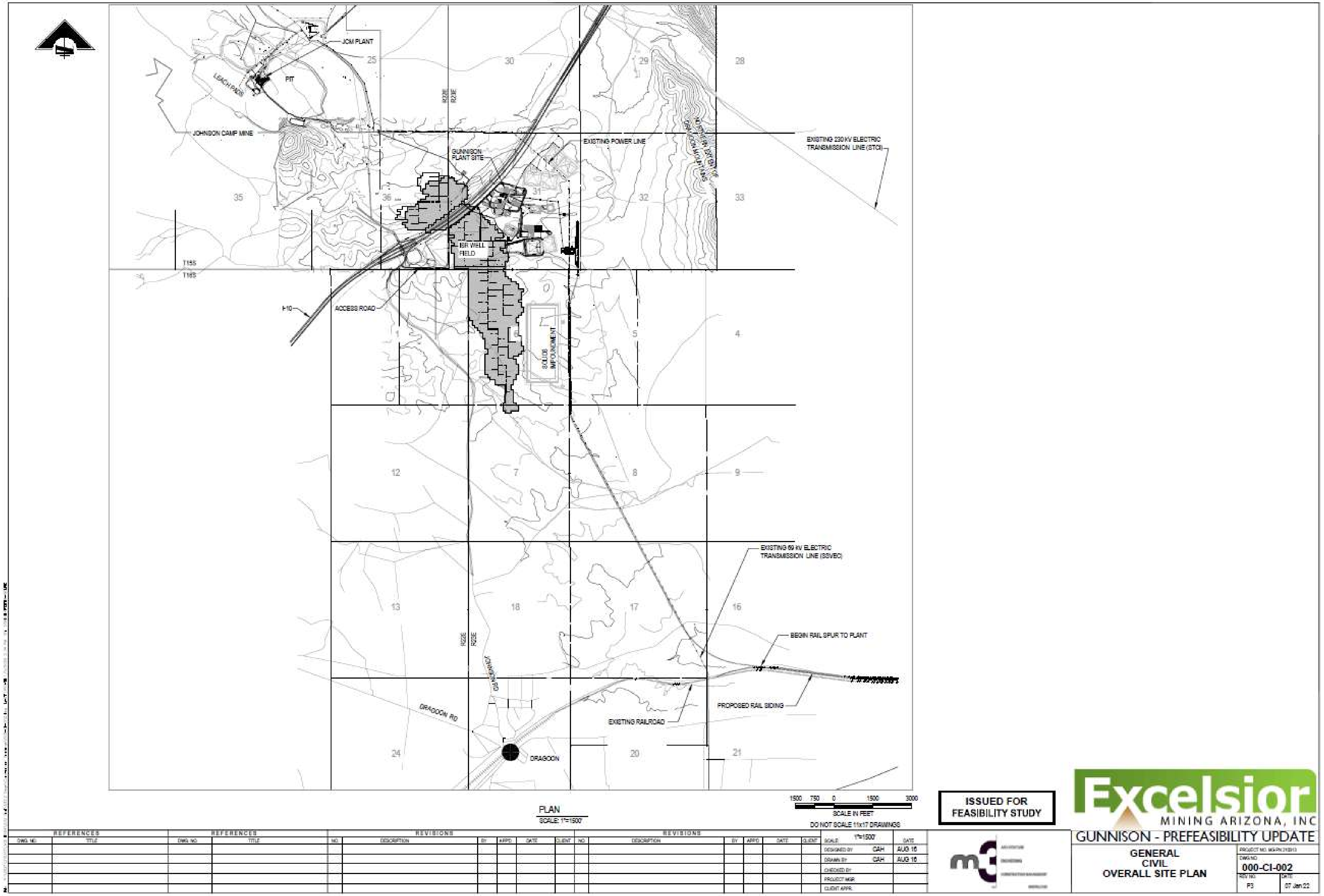


Figure 18-1: Overall Site Plan

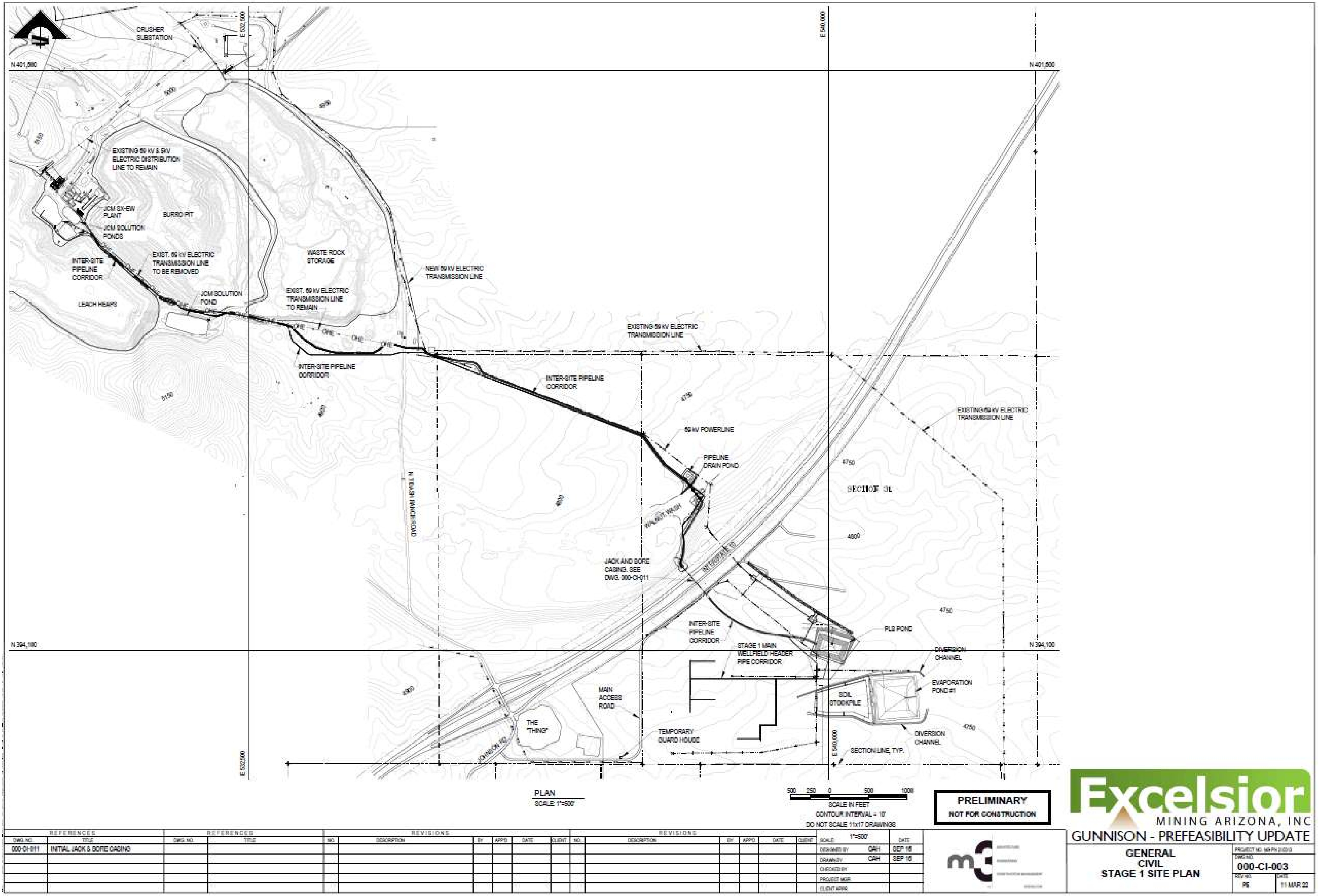


Figure 18-2: Stage 1 Facilities and Infrastructure

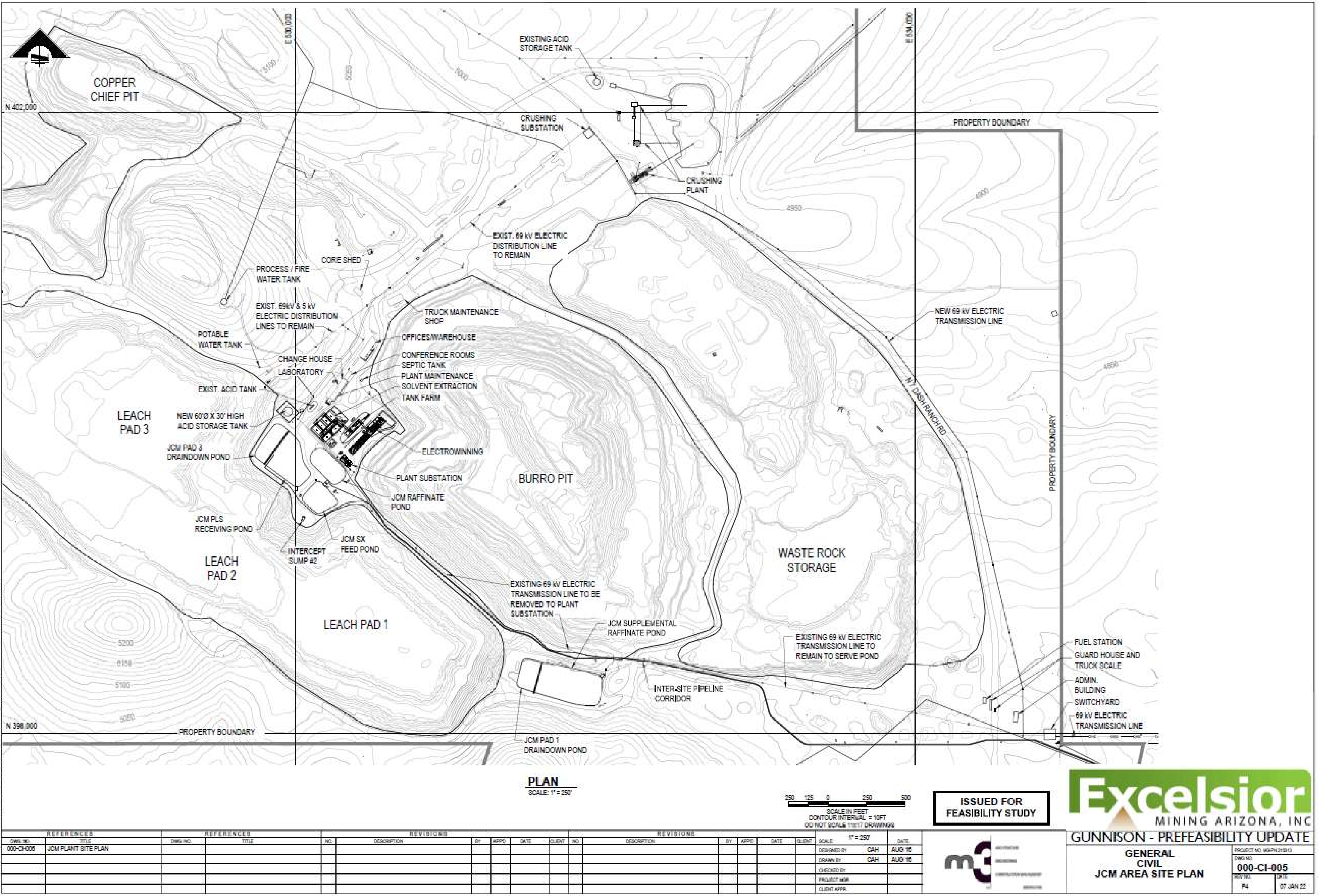


Figure 18-3: Johnson Camp Mine Facilities Arrangement

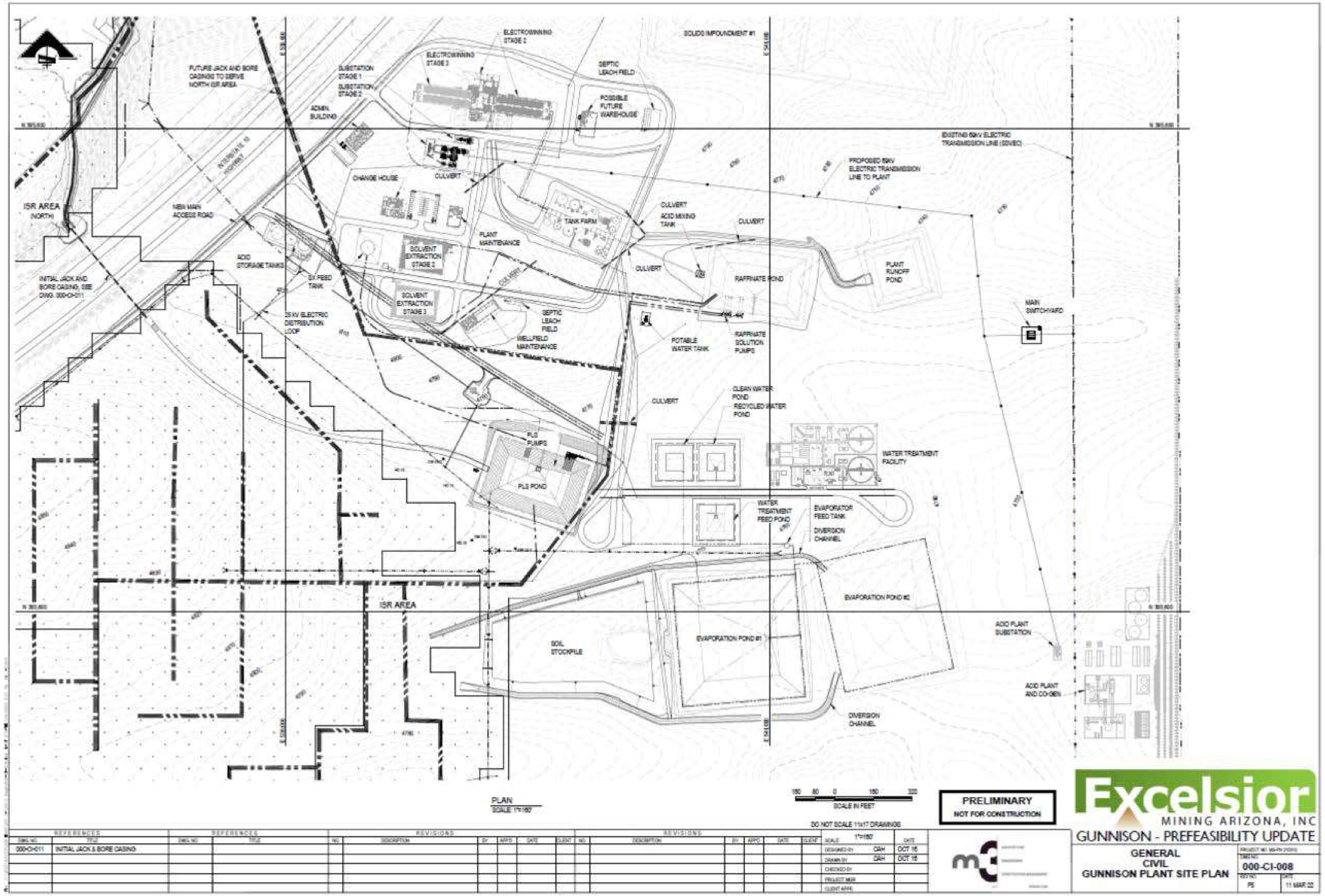


Figure 18-4: Site Plan of Stage 2 and 3 Facilities and Infrastructure

The Stage 2 tank farm is uncovered and located downhill from the mixer-settlers and the electrowinning building to facilitate gravity drainage of fluids to the tank farm. The tank farm contains tanks, pumps, filters, and heat exchangers involved in the handling of aqueous and organic solutions used in the process (Figure 17-6). The tank farm has a containment area that drains to a sump with an oil-water separator to return spilled liquid to the proper location for recycling. A drain line is also provided to drain the tank farm sump to the Raffinate Pond in case of a process upset during power outage.

For Stage 3 operation, a second train with three additional mixer-settlers is added to SX, additional tanks and electrolyte filters are added to the tank farm, and additional cells are added to the electrowinning building on the opposite side of the cathode handling area (Figure 18-4). The additional electrowinning cells will be served by a second electrowinning bridge crane but will share the stripping machine and most of the other cathode handling infrastructure.

18.4 ANCILLARY FACILITIES

Ancillary buildings are needed to support the Gunnison Copper Project at both the JCM site and at the Gunnison site in Stages 2 and 3.

18.4.1 JCM Ancillaries

The ancillary buildings at the JCM site are existing buildings that may need minor modification to serve as intended for the Gunnison project. The administration building, guard house, weigh scale, and fuel station are located at the main gate (Figure 18-3). The former truck shop will be modified to act as a wellfield warehouse. The existing offices/warehouse building will be used as a warehouse for plant operations. The conference rooms will be used for operations and safety meetings. The change house, sample preparation area, laboratory, and plant maintenance areas will be modified as necessary to perform the same functions for operations at the JCM site.

18.4.2 Gunnison Ancillaries

Additional ancillary buildings will be constructed at the Gunnison site for Stage 2 and 3 operations (Figure 18-4). Ancillary buildings include a guard house, an administration building, change house, plant maintenance building, and wellfield maintenance building.

18.4.2.1 Guard House

The guard house is located near the main gate along the access road on the west side of the property. The guard house is a modular building which includes security office, training room, restroom, check-in area, and storage. The area also includes a scale to weight trucks entering and leaving the property. This facility was installed with the construction of Stage 1.

18.4.2.2 Administration Complex

The administration complex includes the administration building, change house, and plant maintenance building, all located north of the SX facilities (Figure 18-4). The administration building is a single-story pre-engineered steel building that includes offices for the administrative and supervisory personnel for the operation. The change house is a single-story, pre-engineered steel building for workers coming and going at shift change. The change house includes showers and locker rooms for men and women; meeting room; offices for safety and training personnel; exam, first aid, and nurse’s room; supply rooms; and records room. The plant maintenance building is a two-story, pre-engineered steel building for maintenance of equipment used in the SX-EW process. The first floor of the maintenance building includes working areas, tool cribs, instrument room, overhead crane, offices, and restrooms. The second story present at one end of the building includes offices and meeting rooms for planning and supervisory personnel.

18.4.2.3 Wellfield Maintenance Facilities

A wellfield maintenance building is located near the well field, northwest of the PLS Pond (Figure 18-4). The wellfield maintenance building is a single-story, pre-engineered steel building for maintenance of well field pumps, valves, and instrumentation, and analysis of samples and data collected during the installation of the wells associated with the ISR well field. This building will contain offices for wellfield maintenance supervisors, geologists, and hydrogeologists, storage space for wellfield pumps, motors, valves, controls, and instrumentation and the maintenance bays to work on them.

18.5 WATER TREATMENT PLANT

Water treatment is required for two primary purposes, neutralizing raffinate for dissolution of carbon dioxide during wellfield conditioning and removing acid, metals, and sulfate from solutions to rinse the formation after it is depleted of copper. The neutralization process requires raising the pH to near neutral (~7). The removal of metals and sulfate requires nanofiltration in addition to the neutralization. Rinsing of the formation is not scheduled to begin until Year 8 of the mine plan, so the water treatment plant (WTP) is planned for construction in phases.

Since wellfield conditioning is required to prepare the ore blocks for copper production, Phase 1 of the WTP (Train A) will be constructed in Year -1 and is located north of the Evaporation Pond (Figure 18-4). Phase 2 is required to increase the capacity of the Train A neutralization system in advance of the Stage 2 ISR production expansion to 75 mppa. The Phase 3 WTP expansion adds a second train (Train B) that includes two stages of pH adjustment, clarification, filtration, nanofiltration, and desaturation to produce low-sulfate water for rinsing. Phase 4 adds additional capacity to Train B to produce a higher flow rate of low-sulfate water for rinsing. A block flow diagram illustrating the Phase 4 design flows is presented in Figure 18-5. Train A is illustrated at the top of the figure and Train B with nanofiltration is illustrated at the bottom of the figure.

The WTP is designed to provide treatment for mine-influenced water (MIW) comprising wellfield conditioning and rinse water return from the ISR wellfield, raffinate bleed, and impacted hydraulic control water (Figure 18-6). The Phase 1 Train A system takes up to 1,260 gpm of excess solutions primarily comprising raffinate from the Evaporation Pond to provide neutralized solutions for CO_2 removal in the wellfield.

The Train A high-density solids (HDS) treatment process includes lime neutralization, clarification, and multimedia filtration. The process includes a conditioning tank, reaction tank, flocculating clarifier, and multimedia filtration including a feed tank, backwash tank, and filtered water tank. Reagents include milk of lime, polymer, and filter aid. The influent flow reacts with lime (calcium hydroxide), forming metal hydroxides and gypsum (calcium sulfate) as solids. HDS incorporates significant recycling of solids from the clarifier underflow to the reactor tank, which increases the size and density of precipitated solids. The filtered water tank discharge is pumped to the wellfield for injection to dissolve CO_2 in the formation. Discharge slurry from the underflow of the clarifier that is not recycled to the reaction tank is pumped to the Evaporation Pond.

In Phase 2, a reaction tank and four multimedia filters are added to increase the capacity to at least 3,500 gpm. A WTP Feed Pond to provide surge capacity for the feed and a Recycled Water Pond for surge capacity of the treated water discharge are added in Phase 2. A Solids Impoundment (No. 1) is added in Phase 2 to contain and drain the increased clarifier underflow for settling and recovery of the supernatant liquid.

The Phase 3 expansion will include the construction of Train B capable of treating acidic rinse solutions from the wellfield to produce low sulfate permeate that can be returned to the wellfield for rinsing. Train B includes the following major equipment:

- Conditioning tank

- Reactor tank
- Flocculating clarifier
- Second-stage reactor tank
- High-rate clarifier
- Multimedia filters (4) including
  - Feed tank
  - Backwash tank
  - Filtered water tank
- Nanofiltration skids (2)
- NF Permeate Tank
- Desaturation tank
- Desaturation clarifier

Phase 3 construction also includes the addition of a Clean Water Pond to receive the permeate and pump it back to the wellfield for rinse injection.

The Train B process starts with the same HDS process as Train A but adds a second stage reaction tank at a higher pH to remove additional metals. The solids are removed in a high-rate ballasted clarifier prior to multimedia filtration. The filtered water is conditioned and subjected to nanofiltration to produce a low sulfate permeate (75%) and a reject brine (25%) that is subjected to desaturation by the addition of lime and additional solids separation. Most of the overflow from this process is recycled back to the lime neutralization reaction tank, but a portion of the overflow (‘blowdown”) is sent to the evaporation pond to prevent buildup of sodium, chloride, and other dissolved solids in the process. The permeate is pumped to the Clean Water Pond for use in rinsing.

All of the solids from the various clarifiers are discharged to a solids impoundment for dewatering and final solids disposal. Water drained from the solids impoundment or pumped from the supernatant pool in the impoundment is returned to the WTP as influent to Train A.

18.6 SULFURIC ACID PLANT

The sulfuric acid plant is scheduled to be constructed for use in Stage 3. The sulfuric acid plant is located east of Evaporation Pond #1 along the railroad spur that runs along the eastern margin of the Project area (Figure 18-4).

A PFS-level design and cost estimate were produced for this PFS update by NORAM Engineering (2022). The plant is designed to produce 1,650 tonnes of concentrated sulfuric acid per day. Sulfuric acid generation uses molten sulfur to make sulfuric acid through the process of oxidation, which produces heat. Waste heat from the acid making process produces steam as a by-product to generate 9 MW of electrical power, which reduces operating costs from $150/short ton to $52/short ton of acid. The facility includes molten sulfur day tanks, sulfur burner and waste-heat boiler, drying and adsorption tower area, cogeneration building, water treatment building, power distribution building and substation, cooling towers, office building, sulfuric acid storage area, and a rail yard for unloading molten sulfur and sulfuric acid (Figure 18-7).

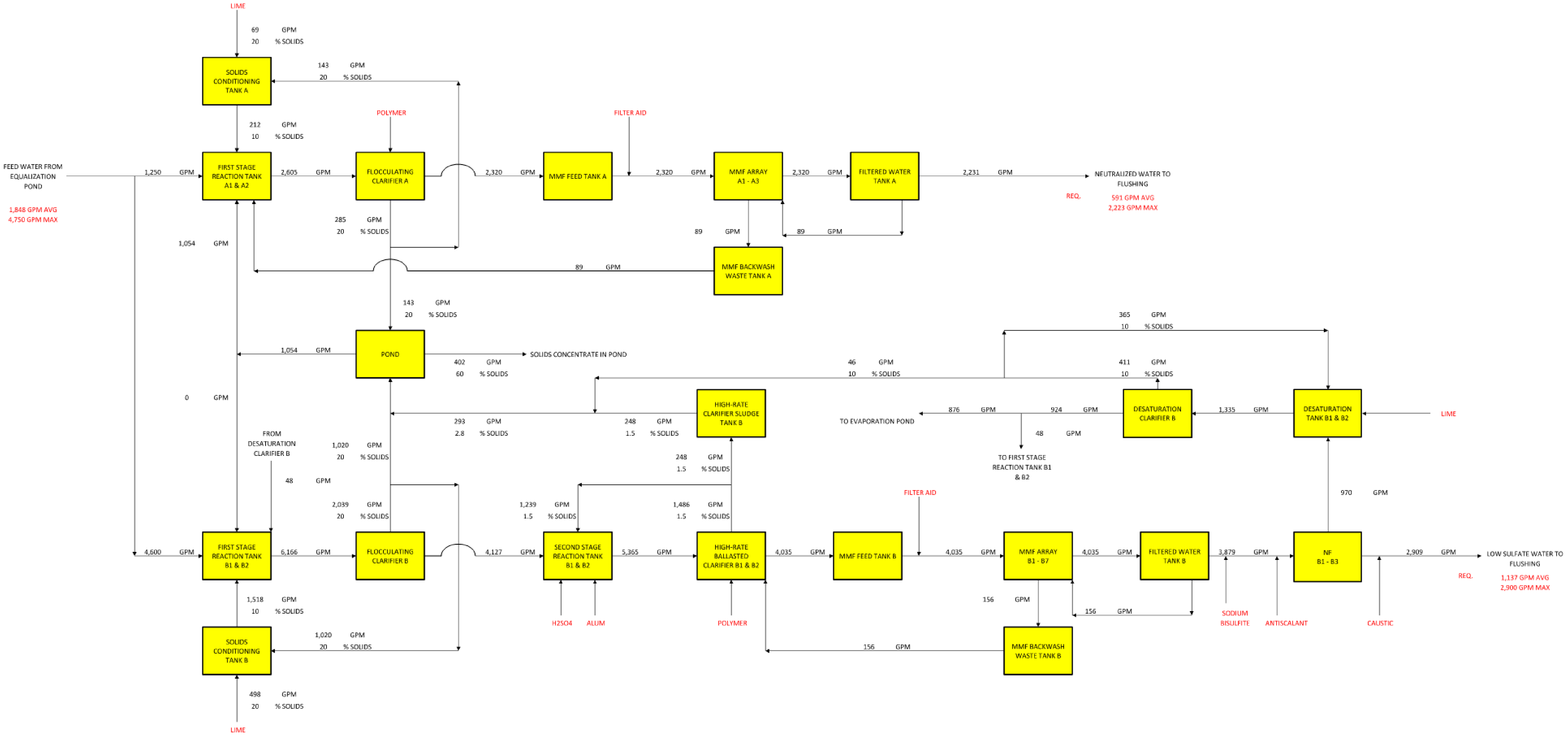


Figure 18-5: Phase 4 Water Treatment Plant Block Flow Diagram

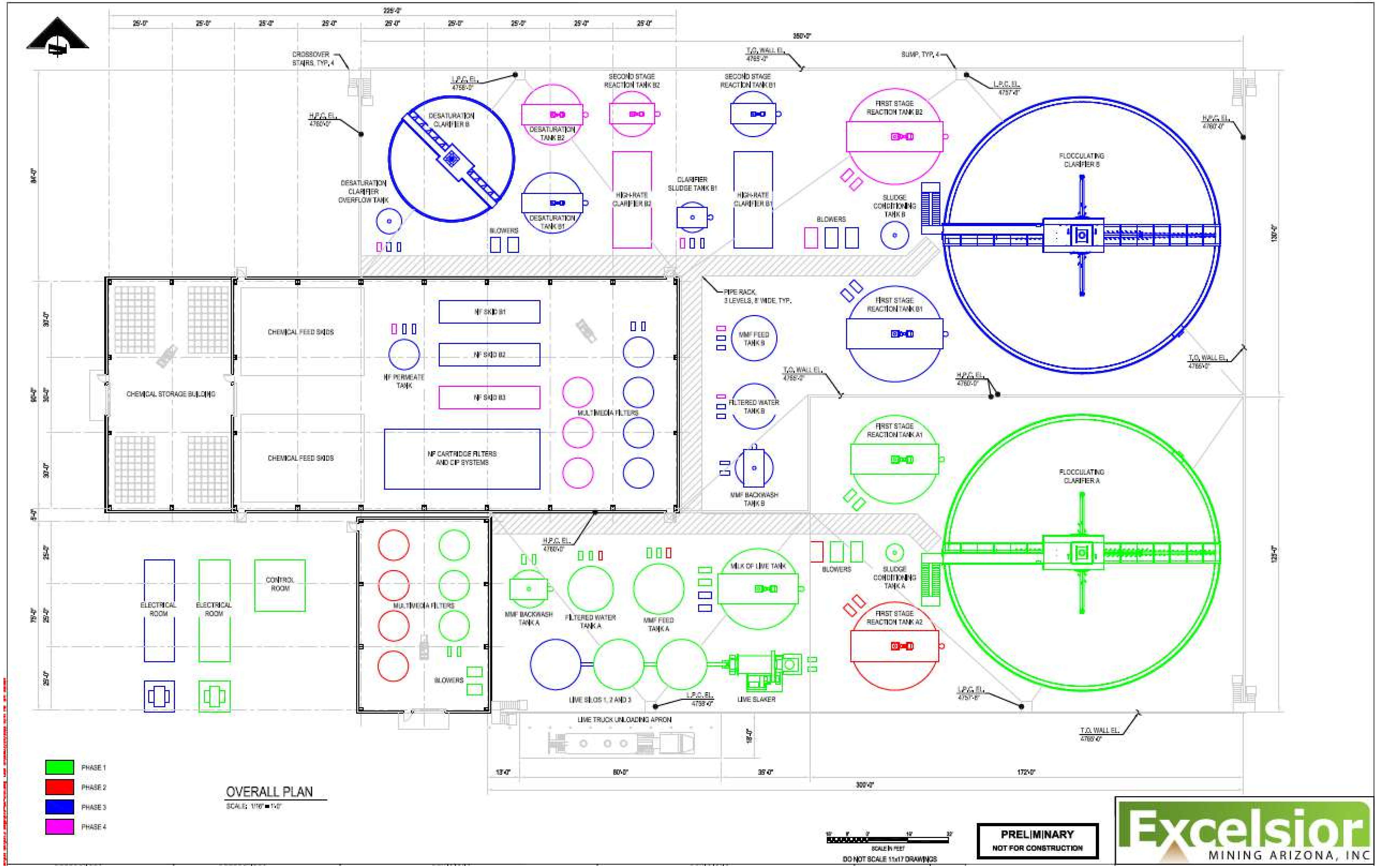


Figure 18-6: General Arrangement of Gunnison Water Treatment Plant

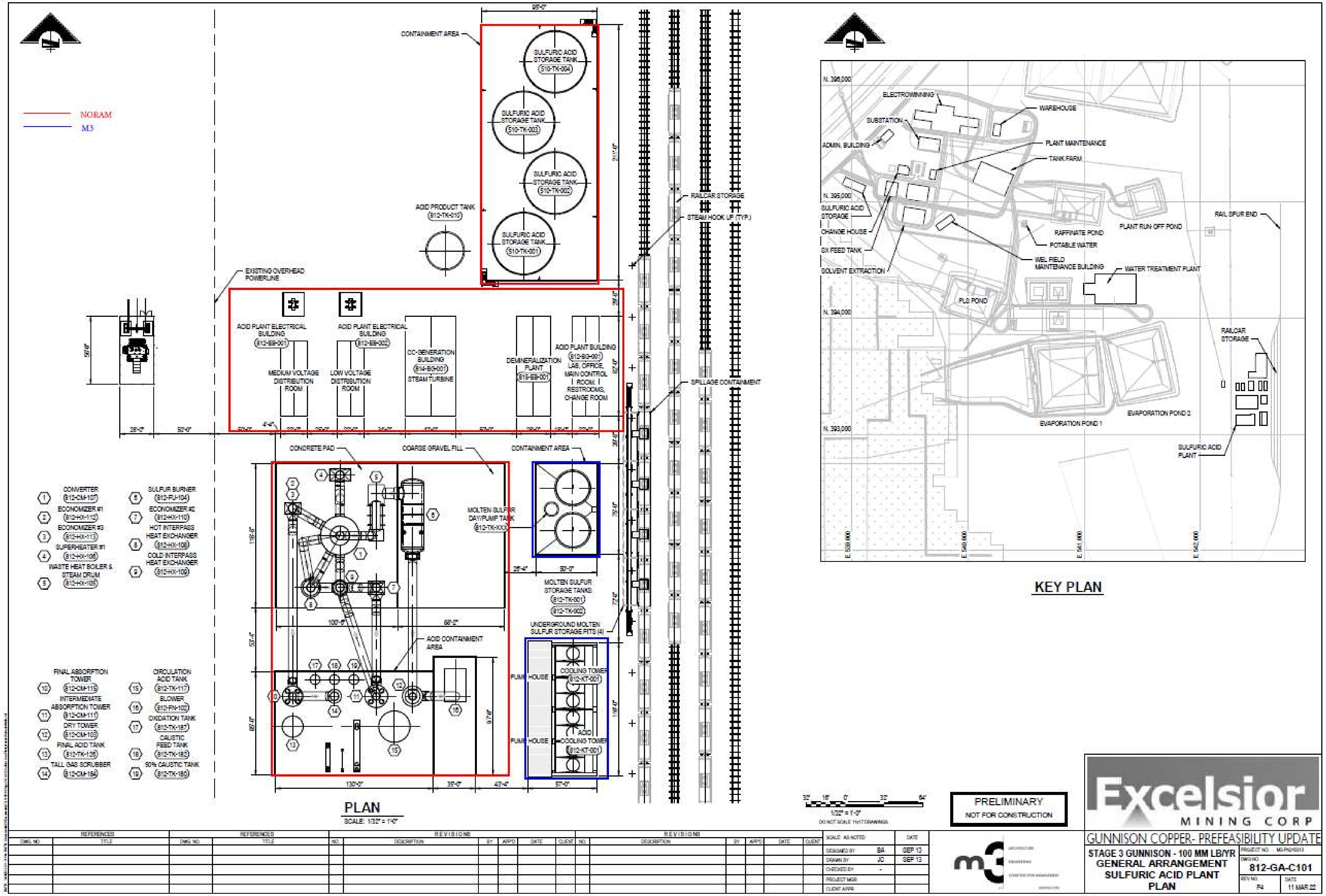


Figure 18-7: Sulfuric Acid Plant

The proposed acid plant is a double-contact double-absorption acid plant which provides the highest conversion rate and lowest emission of sulfur dioxide (SO_2), less than 500 ppm by volume. The sulfur-burning sulfuric acid plant is sized for 1,650 tonnes per day (100% H_2SO_4), with the product acid strength of 98.5% H_2SO_4. Allowing for 10 days down time each year for maintenance, the acid plant operates at an average of 80% capacity for the first 20 years. In seven of those years, the demand is projected to be greater than 100% of capacity.

The process to make concentrated sulfuric acid from molten sulfur is a multi-step process that generates a great deal of heat that can be used to generate electrical power. Molten sulfur is burned to produce SO_2 that is catalytically converted to sulfur trioxide (SO_3). SO_3 is hydrated in absorption towers to produce concentrated H_2SO_4. Burning sulfur, catalytic conversion to SO_3, and hydration to H_2SO_4 all produce considerable heat. Some of that heat is captured to make high-pressure steam for electrical power generation. Low-pressure steam is used for sulfur heating among other uses. Cooling towers are necessary to dissipate waste heat from the processes.

Molten sulfur is received at the plant in rail tank cars with a payload capacity of approximately 100 tons. The rail cars must be heated by steam to liquefy the sulfur since heat loss in the car during transit solidifies some of the sulfur. When re-heated, the molten sulfur is discharged to a receiving pit and pumped into heated storage tanks. A heated pump tank is provided at the rail unloading siding and heated storage tanks located near the acid plant.

Molten sulfur is pumped from the storage tanks to the sulfur furnace where it is mixed with dehumidified high-pressure air to atomize and oxidize the sulfur to SO_2. A bleed stream of sulfur is recirculated back to the sulfur storage tanks to ensure a consistent feed of sulfur to the sulfur burners. Excess air is provided at the burners to ensure complete combustion and sufficient excess oxygen in the off gas for the conversion of SO_2 to SO_3 in the acid plant. SO_2 in the off-gas is catalytically converted to SO_3 in a four-bed converter with vanadium pentoxide as the catalyst.

Mass transfer from the gas phase to the acid phase takes place in the Absorption Towers. All acid towers have an acid distributor designed to spread acid uniformly over the tower cross-section packing to promote gas to liquid mass transfer, mist-eliminators on top outlet to capture entrained acid mist and spray, and a screen in the acid outlet to capture packing chips. The acid circulates in a closed loop in all towers, starting from a pump tank with an acid cooler provided in the loop. The acid circulation rate through the towers is typically between 10 to 20 times that of the acid production rate. Sufficient acid is circulated to wet the packing and to limit the temperature rise of the acid due to the heat of dilution and reaction.

- Moisture in the air transfers to the acid thereby diluting the acid in the Drying Tower.
- SO_3 gas transfers to the acid phase in the Interpass and Final Towers.

In cold climates, typically two grades of acid are produced, 93% H_2SO_4 in winter and 98.5% H_2SO_4 in summer. The stronger acid freezes at +5 ℃ while the weaker acid freezes at -34 ℃. To make 98.5% acid, a split stream of acid from the Interpass Tower circulation is sent to the Final Tower, where it produces 98.5% acid, which is sent to storage. A second split stream of acid from the Interpass Tower circulation is sent to the Drying Tower to absorb moisture from the air. A corresponding stream of diluted acid at 93% is returned from the Drying Tower. To make 93% acid, a split stream of the 93% acid circulation returning from the Drying Tower is taken to storage.

Steam produced in the Waste Heat Boiler from cooling the sulfur burner is superheated and used to create electrical power in the steam turbine generator (STG). Steam production is proportional to the acid production: approximately 1.25 tons of steam per ton of acid. The Start-up/Emergency Boiler creates low-pressure steam needed to start up the sulfur burner and provide low-pressure steam when the process is down. Some low-pressure steam is extracted from the STG and used in the deaerator and molten sulfur heating system during the acid-making process. Condensate from the STG system is collected and polished (treated) to be reused as waste heat boiler feed water.

Boiler feed water is received in a deaerator, where oxygen is stripped by low pressure steam. Two boiler feed water pumps are provided, one motor driven and one steam turbine driven. The boiler feed water is heated in Economizers and Interpass Heat Exchangers. The heated water then goes to the Waste Heat Boiler to maintain inventory.

The Acid Plant Cooling Tower provides cooling water for heat regulation in the acid section of the plant. The Drying Tower, Interpass Tower, Final Tower, and Product Acid Coolers are heat exchangers that operate in parallel to control process temperatures. A cooler in the main blower lubrication system also uses cooling water supplied from the Acid Plant Cooling Tower.

18.7 PONDS AND IMPOUNDMENTS

Several lined ponds and impoundments are needed to contain liquids and solids that are not directly related to the SX-EW process. These ponds include a Water Treatment Feed Pond; Plant Runoff Pond to intercept potentially impacted surface drainage; Clean Water Pond and Recycled Water Pond that are associated with the wellfield rinsing and water treatment systems; Evaporation Pond #1 located southeast of the SX-EW area for evaporating excess solutions and reject brines from the water treatment plant; and solids impoundments to contain metal hydroxide and sulfate precipitates from the pH neutralization of wellfield rinse water (Figure 18-8).

Each impoundment is designed to include an LCRS, underdrain, and decant systems to manage clarifier underflow from the WTP, allow dewatering of the solids, and recirculate water back into the WTP. The impoundments are designed with an HDPE-lined berm in the middle separating it into two compartments for management of solids and liquids. One compartment can be “rested” and permitted to settle and densify while the clarifier underflow is directed to the other compartment. Moisture drains from the slurry as it densifies and is collected in the underdrain pipes or forms a supernatant pool on top of the solids that is pumped to associated seepage ponds. Water is then pumped from the seepage pond and is recycled back into the WTP. Solids are expected to drain to an ultimate density of 60% by weight. All solids impoundments are expected to be closed in place by covering the dried solids with clean topsoil, grading the area to drain surface water, and revegetating the surface so that evapotranspiration losses exceed annual precipitation infiltration.

18.8 RAILROAD FACILITIES

The Union Pacific main line railroad passes through the town of Dragoon, Arizona. A new rail siding will connect to the main line about 1 mile northeast of the town of Dragoon. A new rail spur will generally follow an existing power line alignment northwest to the plant site. The rail spur is about 4 miles long and terminates at the east side of the site near Evaporation Pond #1 (Figure 18-1). Sulfuric acid will be received during initial operations, replaced by molten sulfur shipments when the acid plant is constructed. The rail loading facility near the plant consists of three sidings in addition to the spur line to accommodate up to 25 cars each: one for unloading, one for empties, and one for switching. The new siding (drop-pull track) will consist of three tracks and will be of sufficient length to accommodate an 80-car unit train. It is assumed that the Union Pacific will service the property from Dragoon.

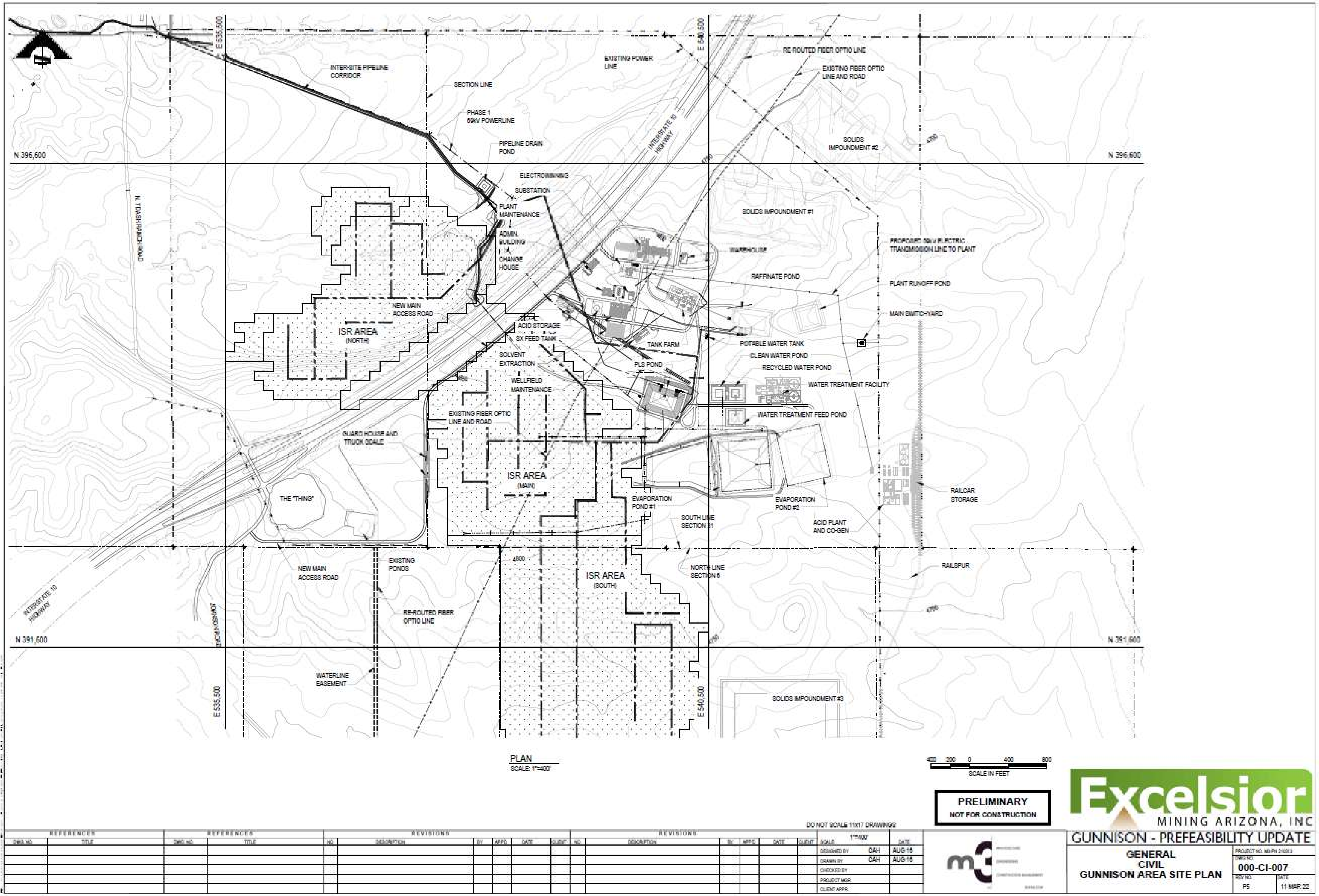


Figure 18-8: Gunnison Area Site Plan

18.9 POWER SUPPLY & DISTRIBUTION

Power for the facility will be taken from an existing 69 kV electric transmission line feeding the existing Johnson Camp mine located on the north side of I-10. The existing power line is owned by the Sulfur Springs Valley Electric Cooperative Inc. located in Willcox, Arizona. The power line approaches the plant site along the eastern boundary of Section 31. For Stage 1, a 69kV line was installed along the piping corridor connecting the wellfield to the JCM plant. The overhead powerline crosses the Interstate and feeds a temporary substation near the PLS Pond (Figure 18-2). In Stages 2 and 3, the temporary substation will be replaced by a main substation (Figure 18-4). At the main substation, power will be transformed to 24.9 kV for distribution throughout the plant and wellfield. Additional transformers will be provided in the various process areas to provide medium voltage (4,160 V) and low voltage (480 V) to feed the end users. A second main substation will be located near the sulfuric acid plant (Section 18.6 and Figure 18-4) to supply and transmit the power generated by the steam turbine from waste heat produced in the acid plant.

18.10 WATER SUPPLY & DISTRIBUTION

Fresh water is supplied from existing wells on the JCM property and pumped to an existing process/fire water storage tank (Figure 18-9). The lower portion of the storage tank is reserved for fire water. Process water for plant use is taken from the storage tank above the fire water reserve level. Potable water for the JCM site is provided by the existing Section 19 well, chlorinator building, and potable water tank.

In Stage 2, process water and fire water pipelines will be constructed from the JCM process/fire water tank to the Gunnison site. The elevation difference provides sufficient hydraulic head for process and fire water pressure demands without pumping.

Also, for Stage 2, a water well will be constructed northeast of the Gunnison site for potable water supply to the Gunnison plant. A potable water tank and chlorination system will be provided for the potable water system. Potable water will be used for offices, labs, restrooms, and eye wash stations.

18.11 SANITARY WASTE DISPOSAL

Sanitary wastes from sinks, lavatories, toilets, and showers will be handled by septic systems. The septic systems will be typically dedicated to an individual building, but it is possible that adjacent buildings might share a septic tank or leach field. The septic systems will be designed and permitted in accordance with Cochise County regulations.

Sinks and drains where chemical handling operations are taking place will either drain to the tank farm sump and ultimately report to the Raffinate Pond or be contained in dedicated piping to a chemical containment tank. Any containment tanks will be serviced by licensed hazardous materials handling contractors in accordance with federal, state, and local regulations.

18.12 WASTE MANAGEMENT

Solid wastes will be collected in approved containers, removed from site by a solid waste contractor, and disposed in accordance with federal, state, and local regulations. Excess construction materials and construction debris will be removed from site by the generating contractor.

Recyclable materials that are non-hazardous, such as scrap metal, paper, used oil, batteries, wood products, etc., will be collected in suitable containers and recycled with appropriate vendors.

Hazardous materials, such as contaminated greases, chemicals, paint, and reagents, will be collected and recycled, whenever possible, or shipped off-site for destruction, treatment, or disposal.

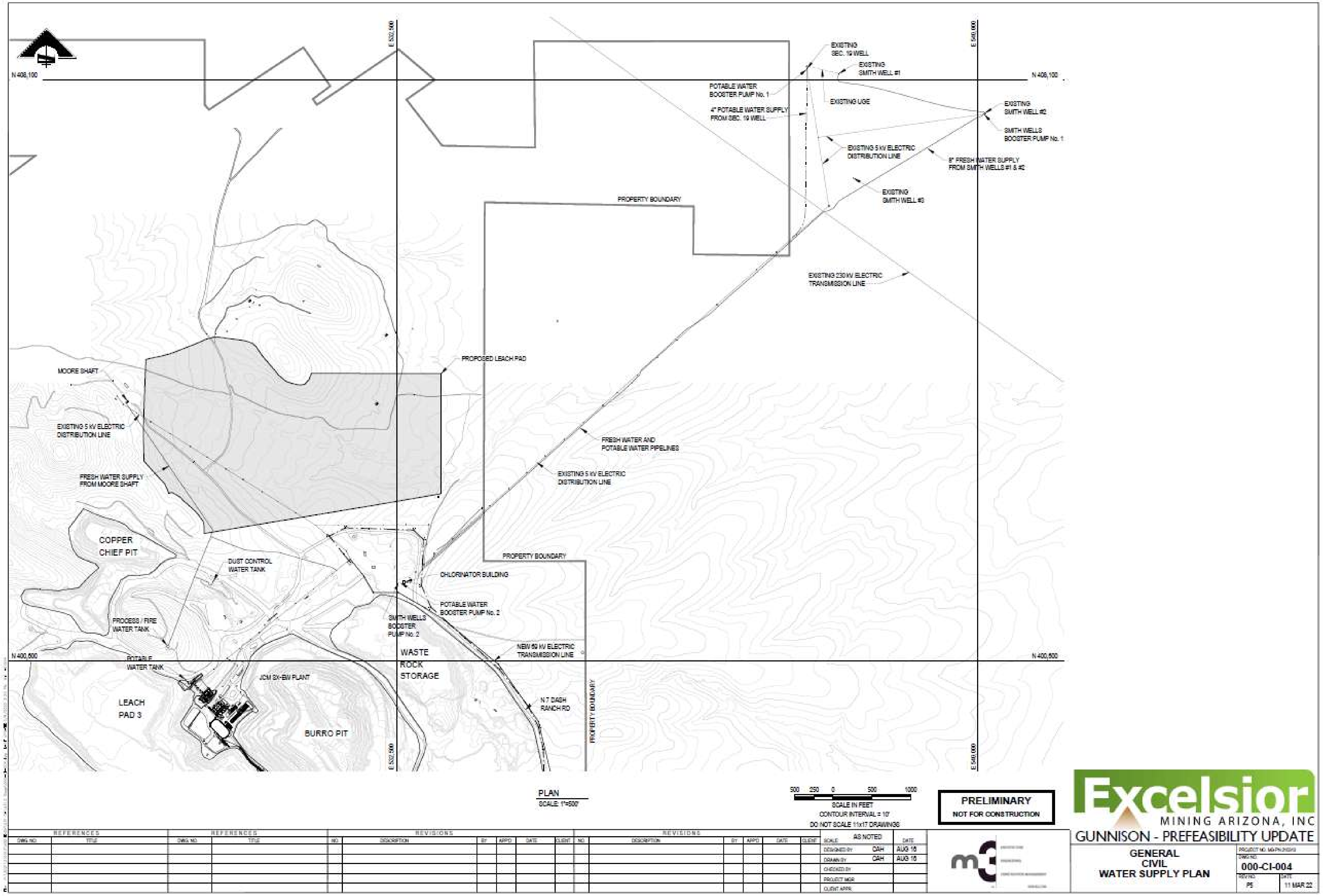


Figure 18-9: Water Supply Plan

18.13 SURFACE WATER CONTROL

Storm water run-off will be diverted around the plant facilities as much as possible. The natural gradient of the land generally slopes from the northwest to the southeast.

Stormwater, process water, or fresh water falling on or running on to potentially impacted areas of the site is considered potentially contaminated “contact” water and will be directed to containment ponds and sumps to prevent contamination of the natural drainage ways. Collected contact water will be pumped to the evaporation pond or to the recycled water pond (after it is constructed with Phase 2 of the WTP).

18.14 TRANSPORTATION & SHIPPING

All materials coming into the JCM and Gunnison plants will be brought in by truck. Sulfuric acid is being received at the JCM site in Stage 1 and will continue to be received by truck at both the JCM and Gunnison sites by truck for Stage 2 operations. Beginning in Stage 3, sulfuric acid and molten sulfur will arrive by rail and be unloaded into their respective storage tanks at the rail yard. The Gunnison site may continue to receive sulfuric acid by truck during Stage 3, if needed. Incoming materials include reagents, pebble lime, extractant, diluent, diesel, warehouse stock, well construction materials, and spare parts.

The primary product leaving the plant is cathode copper, which will be by flatbed tractor trailers. Recycled materials leaving the plant will also be by truck. Scales to weigh full and empty trucks coming into and leaving the site are provided at the main gate for highway trucks and at the rail spur for the rail cars of sulfuric acid and sulfur.

18.15 COMMUNICATIONS

The connection to telephone and internet service has not been confirmed at this time; however, telephone service is available at the Johnson Camp property one mile north and at the town of Dragoon, four miles to the south, which is located on a major intercontinental fiber optic communications line. The telecommunication system will be integrated with the onsite data network system utilizing a voice over internet protocol (VoIP) phone system. A dedicated server will be provided for setup and maintenance of the VoIP system. Handsets will plug into any network connection in the system for telecommunications. The office network will support accounting, payroll, maintenance, and other servers as well as individual user computers. High bandwidth routers and switches will be used to logically segment the system and provide the ability to monitor and control traffic over the network.

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

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

19 MARKET STUDIES AND CONTRACTS

19.1 MARKET STUDIES

19.1.1 Copper price

The anticipated long-term demand for copper cathode is not easily determined but for the purpose of this report, it has been assumed that markets for this product will remain steady. To date, no market study has been conducted for this Project. The Company has engaged with an Offtaker to purchase the copper cathode produced at the average monthly HG Copper COMEX settlement price. The contract is renegotiated annually. The copper market historically has been robust as to consumption requirements.

The use of consensus prices obtained by collating the prices used by peers or as provided by industry observers, such as analysts for example, can be used for reports of this nature. This methodology is recognized by the Canadian Institute of Mining and Metallurgy (CIM) and has the advantage of providing prices that are acceptable to a wide body of industry professionals (peers). These prices are generally acceptable for most common commodities, major industrial minerals, and some minor minerals.

Table 19-1 shows the consensus pricing as of the end of December 2021.

Table 19-1: Copper Forward Curve as of December 31, 2021

Tenor (Years)	Copper Forward Curve ($/lb Cu)
Spot	$4.46
1	$4.41
2	$4.33
3	$4.32
4	$4.41
5	$4.43
6	$4.39
7	$4.35
8	$4.30
9	$4.26
10	$4.21
11	$4.16
12	$4.12
13	$4.07
14	$4.03
15	$3.98
16	$3.94
17	$3.89
18	$3.85
19	$3.80
20	$3.76
21	$3.71
22	$3.67
23	$3.62
24	$3.58
25	$3.53
26	$3.49

For the financial analysis for this technical report, the copper prices by year are held static throughout the life of the mine (LoM) at $3.75/lb.

19.1.2 Sulfuric Acid Price

Sulfuric acid is one of two main consumables for the Project, the other being lime for neutralization of raffinate solution and for rinsing the wellfield blocks post-mining. The price of sulfuric acid can be volatile based on local supply aberrations. The current pricing, $230/ton of acid delivered is considered an outlier price to the historical and forecast price for sulfuric acid, partly due to the pandemic and partly due to a shortage of supply from smelters due to a fire at one of the main US producers in 2021.

Sulfuric acid pricing from Excelsior's supplier has been at the rates below:

2018	$96 per short ton
2019	$125 per short ton
2020	$111 per short ton
2021	$115 per short ton
2022	$231 per short ton

The current forecast for sulfuric acid for this technical report is:

2024 thru end of mine life: $150 per short ton with deduction of $20/st in Years 7 to 24 for rail delivery

When the sulfuric acid plant is online in Stage 3, there can be price reduction based on the credit returned from co-generating 16 MW of power, of which 9.6 MW can be sold back to the utility or consumed internally at the Gunnison SX-EW plant. The credit for the reduced consumption cost of power is approximately $11 per ton of acid purchased.

19.1.3 Sulfur Pricing

Sulfur prices for projects in Arizona have a historical price range of $40 to $80 per short ton over the last ten to fifteen years, not including 2021 and 2022. Currently, there is a spike associated with sulfur prices related to supply chain effects of the Covid-19 pandemic that is not expected to be sustained.

Table 19-2 below reports the historical quarterly price for molten sulfur covering the Years 2015 to 2021 (Source – Trafigura and other sources). It includes the ex-Tampa charge (delivery charges) that ranges between $30 and $40 per ton.

Looking forward, none of us can predict how much sulfur will be extracted at the refineries, nor what the demand from the fertilizer industry might be. Today's crop prices, which are leading to additional fertilizer usage and sulfur consumption, have not held up historically which lends some additional credibility to the recommended price. The sulfuric acid plant will be constructed and ready to produce acid approximately eight years from a funding decision for the Gunnison Project.

Table 19-2: Historical Delivered Sulfur Prices 2015 to 2022 (in $USD per ton)

2015	2016	2017	2018	2019	2020	2021	2022	Average
132	75	82	123	80	54	167	316	129

Average for the period is $129/ton so rounding and using $130/ton would appear to be reasonable for delivery to southeastern Arizona for a project located on the main UPSP rail line.

19.1.4 Quick Lime Pricing

The annual consumption of quick lime for the Gunnison Project ranges for 50 per day to 600 tons per day, based on the most recent model for operation of the Gunnison water treatment plant. Detailed planning will smooth out the lime consumptions from year to year, which currently vary by an order of magnitude depending on the number of wells drilled.

Quick lime is produced by Lhoist in Arizona at its Peach Springs plant and at its North Apex plant near Las Vegas, Nevada. A fresh quote from for the purchase of pebble and delivery by rail is $170 per short. Before Stage 3, there will not be a rail siding and rail yard at the Gunnison site where lime can be offloaded at the plant site. Railyards in Benson, Willcox, and Port of Tucson could accept shipments of pebble lime. There will be an incremental cost of $21.38/ton added to the years before Stage 3 to the cost of quick lime for rehandling and trucking the consumable to site. Storage facilities will be included in the capital cost.

19.2 CONTRACTS

Principal activities for Excelsior are project financing, community relations and permitting, and related engineering activities that support the development of the Gunnison Copper Project. During this period, contracting activities will continue to be driven by the need to acquire specialists and professional services firms to assist Excelsior with these various activities.

A number of contracts will need to be put into place in order to complete the proposed studies. Some are already in place and others are still proposed. These include:

- Project financing,
- Community relations,
- Land use,
- Environmental studies and permitting,
- Hydrology and hydrogeology,
- Metallurgical and process engineering support,
- Detail engineering and procurement,
- Site safety and health services,
- Professional Services,
- Drilling services contractors, and
- Sulfuric acid contract.

Contractors will be pre-qualified by Excelsior on the basis of their:

- Safety record,
- Previous experience on similar projects,
- Quality of workmanship on previous projects,
- Quality/experience of on-site management,
- Local availability in region,
- Previous schedule performance,
- Financial stability, and
- Cost competitiveness.

Areas with clearly defined scopes of work will be required as unit price or lump sum contracts.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The Gunnison Copper Project is fully permitted and has been in operation since December 30, 2019. The Project is in compliance with all existing permits. There have been no new environmental related studies since the issuance of the various permits, therefore, the discussions on plans have been removed from this updated report.

20.2 ENVIRONMENTAL STUDIES AND PERMITTING

This section identifies applicable key environmental permits. Federal, state, and local government existing environmental permits are listed in Table 20-1, as are permits that may be required when the Project expands onto BLM lands, or if additional processing facilities are built at the wellfield.

Table 20-1: Environmental Permits

Agency	Permit	Description	Citation	When Required/ Permit No.
Federal				
Bureau of Land Management	Mining	1. Notice Level Operations may not exceed 5 acres. 2. All operations on public lands that disturb the surface require a Plan of Operations will require an environmental assessment or environmental impact statement and posting of a reclamation bond.	43 CFR §3809	Applicable only when mining on BLM lands
US Environmental Protection Agency (EPA)	Underground Injection Control	Establishes an Area of Review (AOR), beyond which mining related solutions shall not pass. Covers all subsurface well activities, i.e., monitor wells and injection/recovery wells located within the AOR. Will require amendment for life-of-mine production.	40 CFR §§124, 144, 146, 147 and 148	R9UIC-AZ3-FY16-1
US Fish & Wildlife Service	Incidental Take Permit	Mining activities that may affect species listed as endangered or threatened need to conduct studies to identify any targeted species and to apply for a permit to conduct their activities. Any identified threatened or endangered species identified in pre-mining surveys would need to be mitigated before mining could proceed.	50 CFR Sections 7 and 10	None previously identified. New studies required prior to disturbing new ground
Nation Historic Preservation Act	Consultation and Mitigation	Requires Federal agencies to take into account the effects of their undertakings, such as construction projects, on properties covered by the NHPA.	42 CFR §137.88	None previously identified. New studies required prior disturbing new ground.
Section 404 of the Clean Water Act	Jurisdictional Waters of the US	Regulates the discharge of dredged or fill material into waters of the United States	33 CFR §323	No jurisdictional waters identified
State of Arizona				
Arizona Department of Environmental Quality (ADEQ)				
Air Quality Division	Air Quality Control Permit	Ensures air pollutants from any source does not exceed the National Ambient Air Quality	ARS §49-402	AQP-71633

Agency	Permit	Description	Citation	When Required/ Permit No.
		Standards. Will require amendment to incorporate for the Acid Plant option.		
Groundwater Section	Aquifer Protection Permit	Covers surface impoundments, solid waste disposal facilities, mine tailings piles and ponds, heap leaching operations. This permit requires designs for the proper management of process facilities, ponds, tailings impoundments, and includes monitoring requirements to ensure compliance with the permit. Will require amendment for life-of-mine production.	AAC R18-9 Articles 1 - 4	P-511633
	Reclamation & Closure Plan for Facilities covered by APP	Reclamation plan; estimated cost of executing reclamation plan and surety bond. The reclamation plan includes reclamation activities and post-closure monitoring, and bonding estimate must be approved by the agencies and the bond must be posted prior to commencement of construction. Will require amendment for life-of-mine production.	AAC R18-9 Articles 1 - 4	P-511633
Waste Management Division	EPA ID Number	Generators of hazardous waste must have an EPA ID prior to offering the waste for shipment.	ARS §49-922	Currently covered under Johnson Camp
	Pollution Prevention Plan	Plan identifying opportunities to reduce waste.	ARS §49-961 thru 973	Annually
	Toxic Release Inventory	Submit Form R for quantity of copper in waste rock.	40 CFR 372	Annually
Arizona Dept of Water Resources	Dam Safety Regulations	Obtain permit for qualifying dams and ponds	ARS §45-1201	Not Required
Arizona State Mine Inspector	Mined Land Reclamation Plan and Bond	Exploration and mining activities on private land with greater than 5 acres disturbance. Does not include facilities covered in Aquifer Protection Permit.	AAC R11-2-101 thru 822	Approved Oct 9, 2018
Arizona Department of Agriculture	Notice of Intent to Clear Land	Ensures enforcement of Arizona Native Plant Laws	ARS §3-904	60 days prior to disturbance
Arizona Game and Fish Department		Ascertain whether or not the mining operation would endanger fish and game habitat, etc.	AAC Title 12	No T&E Species identified
Arizona Department of Transportation	Encroachment Permit	Obtained to allow jack and bore installation of process solution pipelines under I-10.	AAC R17-3-502	Completed
State Historic Preservation Office		Submit a legal description with map of the area to be disturbed SHPO can inform applicants whether work will occur in a state designated historic district	ARS §43-861	Only applies to public lands

20.2.1 Underground Injection Control

Region 9 of the Environmental Protection Agency (EPA) issued an Underground Injection Control (UIC) permit, R9UIC-AZ3-FY16-1, in June 2018. The permit establishes the Area of Review (AOR), which encompasses the north portion of the ore body. The UIC permit focuses on the design, construction, operation, and closure of the wellfield within the AOR boundary. Concurrently, Excelsior obtained an aquifer exemption in June 2018 for the oxide ore zone, the active mining area. Excelsior is compliant with permit conditions. The UIC permit will require a major modification prior to expansion to the southern portion of the Project.

The following groundwater studies were conducted in support of the UIC. These studies may be required to be updated or expanded in the project area is expanded.

- Aquifer Properties – included hydraulic conductivity, transmissivity and storativity, existing hydraulic connections between wells, estimates of porosity and permeability.
- Fracture Gradient Testing – established the minimum injection pressure at which rock would fracture in each well; these values were used to establish maximum allowable injection pressures in the wellfield.
- Groundwater Quality – monitored sufficiently to establish background water quality prior to the commencement of mining.
- Groundwater Modeling – used existing data and industry standard software to construct a numerical groundwater model to simulate groundwater conditions prior to mining, during mining and post mining. The model was updated at the end of the first year of mining and will be updated as required by the permit.
- Geochemical Modeling – used to create the post-mining rinsing and closure protocol. The model will require updating with expansion of the project area and/or if there are significant changes in the injectate.

20.2.2 Aquifer Protection Permit

The Gunnison Solution Mine began operating with the wellfield permitted by both a US Environmental Protection Agency (EPA) Underground Injection Control (UIC) Permit and an Arizona Aquifer Protection Permit (APP), issued by Arizona Department of Environmental Quality (ADEQ). On September 30, 2021, House Bill 2042 became law. This bill removed any provisions within an existing APP which are covered in the EPA UIC permit. Therefore, the existing Gunnison APP P-511633 now covers above ground process related facilities, while the UIC continues to cover subsurface activities within the Area of Review (AOR) boundary.

APP-regulated surface activities related to solution mining include Solids Ponds 1a and 1b, 2a and 2b, and Solids Pond 3, Evaporation Ponds 1 and 2 (if needed), the Recycled Water Pond, PLS Pond, Raffinate Pond, the Plant Runoff Pond, and the Pipeline Drain Pond. They are sized to have sufficient capacities and freeboard. Process solution impoundments have a double liner with a leak collection and removal system (LCRS) between the liners. Excelsior must submit an amendment application to the existing APP prior to constructing additional processing facilities. Should the Project require additional pond storage capacity, an amendment to the APP will be submitted. Because ponds are designed according to prescriptive BADCT, permitting timeframes are expected to be minimized.

20.3 WATER AND WASTE MANAGEMENT

Solution mining does not generate traditional mine wastes such as waste rock and tailings. “Mine wastes” produced from solution mining are primarily in the form of excess solutions and chemical precipitates from the treatment of those excess solutions. Excess water is evaporated, and the solid residues generated by treatment of excess solutions will be contained in lined impoundments.

A water management plan has been implemented to make the most efficient use of water resources and eliminate discharges. A number of impoundments are used to manage process solutions. Excelsior is currently utilizing both the processing plant and raffinate ponds at JCM and PLS and evaporation ponds at Gunnison. In addition to the process ponds, Drain Pond is located at the topographic low point between JCM and the Gunnison site to contain pipeline drainage in the event that one of the pipelines needs to be repaired.

Additional ponds will be constructed on the east side of the wellfield as production increases and SX-EW facilities are constructed south of Interstate 10. These include: the Gunnison Raffinate Pond, Plant Runoff Pond, Clean Water pond, Recycled Water pond, and Solids Impoundments, which will contain the precipitate from the Water Treatment Plant planned for construction in Year 7. The Plant Runoff Pond, Pipeline Drain Pond, and the Clean Water Pond will be constructed with a single HDPE liner and no leak detection. The PLS, Raffinate, Recycle, Evaporation, and Solids ponds will be constructed with a double liner and a LCRS between the liners in accordance with prescriptive BADCT designs for the APP.

The ISR process involves: a) delivering acidified leach solution (raffinate) to the mineralized blocks through injection wells, b) recovering pregnant leach solution (PLS) from adjacent recovery wells, c) stripping dissolved copper from PLS, d) re-acidifying the mostly barren raffinate, and e) delivering it back into the wellfield. Most of the solution is recycled in the “closed loop” of injection, recovery, solvent extraction of copper, and re-acidification. A small amount of solution is added from the process (acid addition, electrolyte filter backwashing, cathode washing, etc.). Groundwater pumped from the formation to maintain hydraulic control is added to create an excess flow of impacted water that must be managed. Excess solutions will initially be routed to evaporation ponds where mechanical evaporators will be used to assist natural solar evaporation in reducing the water volume. Formation rinsing after mineralized blocks have been depleted by injecting clean water and recovering rinse solutions (rinsate) from the wellfield will add to the excess solutions that must be managed. The rinsate will also be sent to the Evaporation Pond and evaporated.

A Water Treatment Plant (WTP) will be constructed to allow for the rinsate and other excess solutions to be treated. Approximately 80% will be returned for reuse in the process and as ‘clean water’ for rinsing. The WTP is a high-density solids (HDS) lime neutralization system coupled with membrane filtration to produce permeate that will be pumped to the Clean Water Pond.

The solids from the WTP process will be pumped to the Solids Impoundments as a slurry containing precipitated solids. The concentrate brine and filter backwash from the WTP will be pumped to the Evaporation Pond(s). Groundwater produced from hydraulic control pumping will be conveyed to the Clean Water Pond or, if impacted by PLS, to the Evaporation Pond.

The Solids Impoundments will be double-lined ponds with a leak collection and removal system constructed in accordance with Arizona prescriptive BADCT guidance (ADEQ, 2004). The impoundments will include an overliner drainage system piped to a drainage collection pond. The Solids Impoundments will be used to dewater the WTP solids and recover water, which will be returned to the equalization pond for the WTP. Solids are expected to settle to approximately 50% by volume. The impoundments will be closed in place at the end of their functional life in accordance with Arizona BADCT (ADEQ, 2004).

Pond sizes are based on the chemistry of the process solutions, process flow streams, and the volume of solids that will result after evaporation or water treatment. The ponds will be constructed with excess capacity to meet the freeboard requirements of prescriptive BADCT.

20.4 CLOSURE AND RECLAMATION COSTS

All APP-regulated facilities must be closed in accordance with the stipulations of the permit at the end of operations. Closure activities described in the APP refer only to APP facilities. Non-APP facilities, such as buildings and infrastructure, will be reclaimed in accordance with the approved Mined Land Reclamation Program overseen by the Arizona State Mine Inspector’s Office. This reclamation plan ensures safe and stable post-mining land use. Re-grading and resurfacing needs, if any, will be completed with good engineering practices minimizing unwanted surface disturbances. The closure and reclamation plans includes cost estimates and financial assurance for implementing the plans. Excelsior maintains a surety bond, posted with Arizona State Mine Inspector (ASMI). Both the reclamation plan and surety bond must be updated to reflect changes in the surface processing facilities.

Closure of the ISR wellfield requires rinsing and neutralization of the portions of the formation that have been exposed to leaching. Clean water for rinsing will be provided by water supply wells and recycled water from the Water Treatment Plant. Extracted rinse water will be treated with greater than 80 percent returned for additional rinsing and the remainder being entrained in the Solids Impoundments or disposed of in the Evaporation Ponds. Rinsing is considered complete when the concentrations of regulated constituents are at or below acceptance criteria.

Wells that are accepted as being sufficiently rinsed will be abandoned in accordance with the UIC permit. The wells will be grouted from the bottom upward using a tremie pipe to eliminate the well's ability to act as a conduit for solution migration.

APP-regulated impoundments, including the PLS, Raffinate, Recycled Water, and Evaporation Ponds will be closed in accordance with the approved closure plan. The solution ponds containing liquids (PLS, raffinate, pipeline draindown, etc.) will be emptied and cleaned. Liners will be inspected for signs of leakage. The soils beneath prospective defects will be investigated and remediated as necessary. After clearance, the liner materials will be folded into the bottom of the pond for burial in place. Perimeter berms above the natural land surface will be pushed into the pond to cover the liner, contoured, and revegetated to shed surface runoff and minimize infiltration. The impoundments containing solids (Evaporation and Solids Impoundments) will be closed in place and covered to minimize infiltration. The edges of the liner will be folded inward and covered with a low permeability cap. The cap will be contoured and revegetated to shed surface runoff and minimize infiltration.

20.5 COMMUNITY RELATIONS

Excelsior consistently seeks to build sustainable partnerships and bring value to the communities where it operates. Excelsior's approach to community relations reinforces its core values and provides guidelines for making decisions on a variety of issues, ranging from charitable giving to resource development. To that end, Excelsior maintains a broad-based community relations and stakeholder outreach program in support of the Gunnison Project. Various levels of activity and outreach occur as a function of the development of the Project from prefeasibility and feasibility studies, through Project construction and operations, to closure and rehabilitation. Elements of this program include:

- Targeted stakeholder outreach to government, community, business, non-profit and special interest groups, and leaders at the local, county and state level.
- Development of community relation and communication tools and resources (e.g., Project website, Project e-newsletter, and presentation materials);
- Public open houses and technical briefings when appropriate.

Crucial elements of Excelsior's community relations efforts will involve ensuring consistent and ongoing communication with all stakeholders and providing opportunities for meaningful two-way dialogue and active public involvement. Excelsior will focus on ensuring the public benefits related to the Gunnison Project, such as employment opportunities, supplier services, infrastructure development and community investment are optimized for the local community.

Excelsior's social and environmental license to operate is further confirmed by a settlement agreement that was reached with a number of environmental activist groups that resulted in the withdrawal of a permit appeal.

20.6 ECONOMIC BENEFITS

Prior to commencement of permitting and mining, Excelsior commissioned an Economic Impact Study through Arizona State University's W. P. Carey School of Business which forecasted the increase in economic activity within Arizona during the construction phase and life of the mine. The study utilized an Arizona-specific version of the Regional

Economic Models Inc. (REMI) regional forecasting model[3] to make projections about the direct benefit and multiplier effect of the Gunnison Project. The economic impact of mine development to surrounding communities and the State in General are outlined below.

- On average, during the lifetime of the Project, annualized Arizona jobs added is 819[4]. This employment creation includes 108 direct jobs created and 711 indirect "secondary" jobs, with employment increasing by 283 individuals within Cochise County.
- Employment benefits are distributed through many sectors. The largest impacts are in mining, construction, professional/technical services, and government sectors. Additional significant impacts are in real estate, retail trade, health care, and accommodation/food services among others.
- The annual average value added to Arizona's Gross State Product (GSP) during the entire Project life – pre-production, production, and closure – is approximately $109 million with approximately $28 million added within Cochise County. The total addition to the GSP is $2.9 billion, with $757 million locally within Cochise County.

Economic modeling predicts the Project will have an average annual impact on state revenues of $10.9 million for a total impact of $295 million. Activity for Cochise County has been forecasted to average $3.6 million with a total impact for county revenues of $98 million.

---

[3] This study was based on a projected 20-year production phase. The current measured and indicated copper resource is planned for a 24-year production phase with similar pre- and post-production time periods.

[4] The values reported here reflect the "acid plant scenario" in which Excelsior constructs and manages an internal sulfuric acid production facility.

21 CAPITAL AND OPERATING COSTS

Capital and operating costs for the Gunnison Project were estimated on the basis of a prefeasibility level of design for Pre-Stage 2, Stages 2 and 3. A small amount of Pre-Stage 2 costs are considered initial capital costs, but the bulk of costs presented in this section are considered sustaining capital costs. These estimates included construction materials; construction and operating labor based on those designs; consumption of power, reagents, and consumables; budgetary quotes for major process and ancillary equipment; and estimates from other consultants for the Water Treatment Plant (WTP) and sulfuric acid plant.

21.1 CAPITAL COST

Capital cost (CAPEX) is divided into initial and sustaining capital costs. Stage 1 of the original Gunnison Project was constructed in 2020 with acid injection commencing in December of 2020.

For this study, Pre-Stage 2 initial capital is defined as improvements to Stage 1 in Years -2 and -1 of the Gunnison wellfield, mainly the addition of the Phase 1 water treatment plant. This plant is needed to neutralize raffinate to flush CO_2 from the subsurface as described in Section 18. The new water treatment plant includes a substation (transformer and E-building) to power it and there will be some new wellfield drilling included in Pre-Stage 2 initial capital. Wellfield infrastructure development (piping & electrical) is also required so that new mining blocks can be pre-conditioned right away.

Sustaining capital costs include the ongoing year-by-year additions to wellfield drilling and development, construction of the Stage 2 SX-EW and Stage 3 SX-EW plants on the Gunnison side of the property, each adding 50 million pounds per annum (mppa) of copper cathode capacity, the addition of a new 69 kV to 24.9 kV Gunnison substation, three expansions of the water treatment plant, the addition of water ponds and solids ponds to support plant operation and water treatment, the construction of a sulfur burning sulfuric acid and cogeneration plant, and the addition of a railroad siding and railcar unloading facility.

Table 21-1 summarizes the various categories of capital cost described in this section.

Table 21-1: Summary of Capital Cost over Life of Project

Stage	Copper Production	Description	Total ($000)
Initial Capital	25 mppa	Pre-production wellfield drilling, development & operations; Installation & operation of Phase 1 Water Treatment Plant	$47,621
Phase 2 WTP (Year 2)		First expansion of water treatment; Installation of Feed Water Pond, Recycled Water Pond; & Solids Ponds 1A & 1B	$7,629
Stage 2 (Years 2& 3)	75 mppa	Gunnison 50 mppa SX-EW; 80 EW cells; New Raffinate pond; new Gunnison substation, Gunnison ancillary bldgs. to support drilling and ISR mining, and the Railyard	$178,043
Stage 3 (Year 5 & 6)	125 mppa	Wellfield Expansion; Gunnison 50 mppa SX-EW; 80 EW cells; Water Treatment Plant (WTP); Wellfield expansion; Railroad Siding & Railcar Unloading	$104,263
Acid Plant (Years 5 & 6)		Sulfuric Acid Plant, Molten Sulfur Handling, Cogen Plant; Boiler Water Treatment (Optional)	$159,860
Phase 3 WTP (Year 7)		Second expansion of water treatment plant for membrane filtration	$47,435
Wellfield Drilling Sustaining Capital (All years)		All wellfield drilling costs and wellfield capital equipment and wellfield infrastructure development, Solids Ponds	$526,990
Phase 4 WTP (Year 17)		Third expansion of water treatment plant for additional membrane filtration capacity	$8,968
Total		**Initial & Sustaining Capital Cost**	**$1,080,808**

A Basis of Capital Cost Estimate specification (210313-1027) was prepared by M3 to provide information regarding the sources of capital cost information, assumptions that were used in the estimation, exclusions, and project-specific conditions.

The following costs and quantity estimates used by M3 were provided by others:

- Hatch (February 2022) provided phased design, capital cost for equipment and reagent consumption and of the Water Treatment Plant. The new design of the water treatment plant treats two streams of water: raffinate to neutralize for use in flushing the wellfield and water returned from the wellfield for rinsing operations in areas that have been depleted (in-situ leached) of economically recoverable copper.
- Kinley Exploration LLC (Kinley) (December 2021) provided update cost estimates for installation and development of extraction, injection, and hydraulic control wells, as well as well abandonment costs for existing wells and core holes and production wells that have been rinsed and are out of service.
- NORAM Engineering & Constructors of Vancouver, B.C. (January, 2022) prepared a new PFS study for the sulfuric acid plant. They provided capital and operating cost for the sulfuric acid plant which will be constructed in Years 5 & 6 for operation in Year 7. The new study designs and costs equipment for plant producing 1,650 tons per day of concentrated sulfuric acid. The previous NORAM study (2013) was based on a plant that produced 1,350 tons per day.
- MHF Services (2016), a railroad consulting company, estimated the capital costs to install a railroad siding off of the Union Pacific Southern Pacific railroad and rail transfer and unloading yard for deliveries of acid and/or sulfur. This study was escalated for the current study and the MTO provided in the MHF study was re-estimated.

21.1.1 Basis of Capital Cost

The capital cost estimates on which this prefeasibility study is based were prepared from a level of engineering commensurate with a +/- 20% level of accuracy except where noted. Among the components used for the capital cost estimates were the following items:

- Subsoil investigation
- Facilities plot plan and site location maps
- Electronic topography with a contour interval of 2 feet
- Plant, road railroad, and other site preparation drawings
- Process flowsheets and process design criteria
- Equipment selection and sizing
- General arrangement drawings
- Architectural design criteria
- Ancillary building general arrangements and elevations
- Development of final utility requirements and heat balances
- P&IDs and Design Criteria
- Substation design and specifications
- Single line drawings
- Final motor list
- Material take-offs for civil, concrete, structural, piping, electrical, & instrumentation
- Updated pricing for plant equipment and materials constituting >90% of equipment value
- Development of estimate construction hours and labor rates by craft using sources like RS Means, Davis Bacon, and M3 internal historical data for installation and construction.

- Indirect costs based on M3 historical rates from domestic mining projects (described in Section 21.2).

Some parts of the capital cost estimates were based on third party consulting reports. These reports were prepared by:

- Kinley Exploration – Updated drilling & completion costs; well abandonment costs, well maintenance costs and capital equipment needed to support the wellfield drilling. Estimate accuracy +30%/-15%
- Hatch Water - Water Treatment Plant, estimate accuracy +30%/-15%
- NORAM Engineering & Constructors – Sulfuric acid plant, estimate accuracy +35%/-25%
- MHF Consulting – Railroad siding and rail unloading yard, estimate accuracy +30%/-15%

The estimates by Kinley, Hatch, NORAM, and MHF Consulting were prepared at a prefeasibility level of detail. In the case of the water treatment plant and the sulfuric acid plant, piping, electrical, and some instrumentation were factored leading to a lower estimate accuracy. However, other costs were estimated by MTO so both capital cost estimates are better than fully factored or scoping level estimates.

21.1.2 Initial Capital

Already mentioned in Section 21.1, Excelsior constructed the initial wellfield of 41 wells (drilling and development) for the first mining block of the Gunnison wellfield in 2019 & 2020. Also installed were a set of 24” pipelines for PLS and raffinate between the Gunnison wellfield and the Johnson Camp Mine (JCM), and minor improvements to the JCM plant and ponds.

The current initial capital, which is linked to the development of the Stage 1 wellfield, is the addition of the water treatment plant to mitigate the build-up of CO_2 in the formations that are impeding flow between injection and recovery wells (see Section 13.2). The first “phase” of the Water Treatment Plant is a raffinate neutralization circuit using lime (CaO) as a consumable to lower the pH of solutions fed back to the wellfield during pre-conditioning of the formation to remove calcite. The capital cost for the wellfield provided by Hatch (2022) includes flowsheets, a sized equipment list for each phase of development but aggregate pricing of capital equipment, and an operating cost estimate (power, chemicals, maintenance) by phase. M3 prepared a functional PFS level general arrangement for the complete water treatment plant and prepared material take-offs for civil, concrete, steelwork, piping, electrical and instrumentation costs by factoring.

To provide power to the Phase 1 Water Treatment Plant, M3 costed a substation including a 69 kV to 24 kV transformer and an E-building including MCCs, VFDs, and switchgear). The incoming power from the current Gunnison substation is 5 kV, therefore an additional 5 kV to 480V transformer will be needed prior to the first expansion of the water treatment plant in Year 2. Electrical equipment and the new power line costs were built up from material take-offs.

In Year -1, the next set of injection and recovery wells for the next mining block consisting of 13 holes will be drilled, piped, and powered.

The civil, concrete, and steelwork quantities were estimated from GAs. Materials for construction were based on updated pricing from recent M3 projects in southern Arizona and nearby areas. New piping material pricing was solicited for this study. Electrical and instrumentation for this Project were derived from pricing from recent M3 projects in the US. Excelsior provided some pricing via invoices for valve skids, VFD skids, flowmeters, and pressure transducers that they had purchased for the initial Stage 1 development. Construction labor was adjusted for current Davis-Bacon prevailing shop wages in Arizona for Q1 2022. M3 provided the design and cost estimate for modifications to the process ponds and evaporation pond at Johnson Camp. The accuracy range of the estimate is +30% to -20%

suitable to support a Prefeasibility Study. Table 21-2 summarizes direct and indirect costs that make up the initial capital costs by Area.

Table 21-2: Initial Capital Costs

Category	Items	Cost ($000)
Initial Capex		
Wellfield Drilling	Drilling 13 new holes	$5,441
Wellfield Infrastructure	New valve skid & VFD skid, Piping new wells, power to wells and VFD skids,	$636
SX-EW	None	-
Water Treatment Plant	Phase 1 WTP, reactor, clarifier, filter, and substation	$21,179
Site Infrastructure	none	-
Owner's Cost	none	-
Indirects	Mobilization, Temp construction facilities/power	$4,678
Contingency (15%)	Excludes well drilling	$6,464
Subtotal Initial Capital Costs		$38,398
Capitalized Pre-production		
Wellfield Pre-production	Wellfield staffing, power, consumables	$8,434
Water Treatment Plant Pre-production	WTP staffing, power, consumables	$789
Subtotal Capitalized Pre-production		$9,223
Total Initial Capital Costs		$47,621

21.1.3 Wellfield Infrastructure

Initial wellfield infrastructure includes site roads, valve skids and piping manifolds, wellfield piping infrastructure, VFD-motor starter skids, electrical and communications infrastructure. Valve skids and pipe manifolds replaced the header houses that were designed and costed in the 2017 Gunnison Feasibility Study. Cabling for power to well pumps goes to a VFD skid that accommodates 9 wells per skid. Overhead powerlines carrying 24.9 kV are stepped down to 480 kV at each mine unit block which includes in general 41 wells total of which 25 are equipped with well pumps. Piping manifolds on valve skids are connected to trunk piping by three sets of pipes that bring raffinate to the wellfield, send PLS to the PLS pond for processing, and flush water for wellfield conditioning at the commencement of a new mining block. After the mining block is depleted, clean water rinsate uses the same piping network to remove sulfuric acid from the formation.

21.1.4 Gunnison Evaporation Ponds

The first Evaporation Pond was constructed in 2019 as part of the Stage 1 development of the Gunnison wellfield. That pond is currently in use. It receives and evaporates excess solutions generated by the ISR operation. The Evaporation Pond was be constructed with a double HDPE liner system with a LCRS system in accordance with prescriptive Arizona BADCT design requirements. Excess solutions including water pumped from hydraulic control wells, immature PLS solutions, raffinate, and reject water from the WTP already are or will be discharged to the Evaporation Pond. Multiple mechanical evaporators are used to enhance natural evaporation to dispose of the excess water. A second Evaporation Pond with a capacity of approximately 31 million gallons of storage is scheduled in Year 13 at an estimated cost of $1.62 million in direct costs.

21.2 INDIRECT COSTS

Indirect capital costs were generally factored from the direct field cost. Below is a list of indirect items in the capital cost estimate:

- Indirect field mobilization is 1.5% of the direct field cost without mobile equipment.
- Temporary construction facilities is 0.5% of direct cost less mobile equipment.
- Construction power is 0.1% of direct cost less mobile equipment.
- Engineering Procurement and Construction management is 16.8% for Stages 2 and 3 of the direct cost plus the indirect cost listed above. Stage 1 EPCM was estimated to be 13.5% since Excelsior will self-perform much of the construction management.
- EPCM temporary facilities and utility setup were estimated as 0.5% of total constructed cost.
- Commissioning was estimated to cost 1% of plant equipment less mobile equipment.
- Vendor supervision is estimated as 1.5% of plant equipment costs during construction and 0.5% of plant equipment costs, each, for pre-commissioning and commissioning.
- Capital spare parts are estimated as 2.0% of plant equipment and commissioning spares are 0.5% of plant equipment.

Contingency for both wellfield development and plant improvements have been included at 20% of the total direct and indirect costs.

21.3 OWNER'S COSTS

There is no Owner's cost since the initial Gunnison wellfield has already been constructed. All costs going forward are either initial capital, sustaining capital, operating costs, or G&A costs.

21.4 SUSTAINING CAPITAL COST

Sustaining capital costs commence in Year 1 of the mine schedule and include all capital expenditures that occur after pre-conditioning of the existing well block is completed by the end of Year -1. Starting in Year 1, Excelsior expects that Stage 1 production of PLS production will ramp up to a rate of 4,000 gallons per minute (gpm). Stage 1 production will proceed for a period of three years during which various wellfield installations will be made.

In Year 4, Excelsior anticipates commencing Stage 2 production from the Gunnison wellfield using both the JCM SX-EW tankhouse and the new Gunnison SX-EW facility. In Year 7, Stage 3 production is scheduled to commence with the doubling of the Gunnison SX-EW facility. During all of this time and through the end of the mine life, there will be expansions of the wellfield, the installation of three expansions of the Water Treatment Plant, the installation of a sulfur burning sulfuric acid plant, the installation of a railyard to bring in acid, lime, and sulfur and to ship copper cathode to market. During the Life-of-Mine, various ponds are added to the Gunnison side of the Project. A second tunnel is bored beneath I-10 to access the Gunnison wellfield on the north side of the highway. All of these activities are included under sustaining capital.

Concurrent reclamation is not a sustaining capital cost. The sections below describe the various categories of sustaining capital cost.

21.4.1 Wellfield Drilling

Estimates for well drilling and completion for the Project use both contractor drilling rates and drilling costs based on owning and operating the drilling equipment in house. The Stage 1 wellfield drilling and completion costs were estimated using contractor costs to reduce initial capital expenditures. The cost for purchasing Owner operated drilling equipment is deferred to Year 2. Well drilling includes four main categories: drilling, well pump replacement/sand fill, well abandonment, and capital equipment purchases. Only direct well drilling costs and capital equipment costs are included in the CAPEX. The wellfield initial capital cost in Year -1 for the next 13 wells is estimated to be $2.5 million. This total covers 18,969 feet of drilling and completion of injection wells and recovery wells. The estimated contractor cost, $100 per foot, is based on Kinley's estimate for drilling cost for wells using a contractor's drill rig plus a nominal amount for well supplies. These supplies include well casing and installation materials, well pumps, drop pipes, cabling, and downhole instrumentation, sand, and cement. The cost for well supplies equates to 4.05/foot during contract drilling although each year, the cost of well supplies varies slightly depending on what is scheduled to be drilled. Over the life-of-mine, the cost of well supplies averages $23.05/foot.

For years after Year 2, Excelsior plans to self-perform drilling using its own fleet of drill rigs and support equipment. The capital cost in Year 3 for purchasing the drilling fleet and support equipment is $9.7 million. Kinley has prepared a detailed cost estimate for well drilling using labor build-ups per drill and service rig, hours, and days of operation per rig, hours and costs per rig activity including drilling labor, drilling consumables (mud, drill bits, etc.), casing installation, cementing, integrity testing, hole surveying and geophysical testing, pump installation, and well abandonment costs. The cost of Owner supplied drilling averages $72.30/foot. In Years 4 and 16, relatively little footage is drilled (a quirk of the current mine plan), so the fixed drilling crew labor costs skew the per foot costs to $177/foot and $305/foot, respectively. In practice, production drilling will be smoothed out to prevent these anomalies in drilling cost.

21.4.2 Year-by-Year Sustaining Capital

Annual sustaining capital costs include well construction, wellfield equipment replacement, and wellfield infrastructure development for new mining blocks. Wellfield sustaining capital costs accrue from continued well drilling and completion through Year 19 of the mine schedule. Beyond Year 19, there are wellfield equipment replacement costs that continue through Year 27 and well abandonment costs.

Mining block piping and electrical includes the addition of valve skids, VFD skids, piping, electrical, and instrumentation for approximately 41 wells.

Each mining block includes a pipe manifold, the valve skid, the four VFD skids, and the piping and cabling to the wells, the installation of reversible well heads, instruments, and communications cabling. M3 prepared a build-up for the development cost of a mining block exclusive of wellfield drilling and resolved that cost to a cost per well. This way, mining block cost estimation can be achieved by using the number of wells drilled per year. The cost per well for wellfield mining block development is $57,802 per well drilled.

In addition to the mining block development, there are annual incremental increases to the trunk and lateral piping that feed the wellfield valve skids. Trunk and lateral piping have been laid out and an MTO has been developed by year. These annual piping MTOs for trunk and lateral piping have been priced and are included in the year-by-year sustaining cost. Labor to install the wellfield piping has been costed with outside contractors at local rates; however, it is likely that the Gunnison mine operations staff will actually install the wellfield piping infrastructure and well block additions, which would result in capital savings. Overhead powerlines will run to the corner of each mining block where the power will be stepped down from 24.9 kV to 480V. The costs for wellfield electrical distribution have also been accounted for in wellfield sustaining capital costs. The costs for overall wellfield infrastructure are captured in the category, Wellfield Trunk Piping and Power Loop.

Table 21-3 summarizes the year-by-year capital cost growths in the wellfield including well drilling, wellfield development and ponds.

Table 21-3: Wellfield Sustaining Capital Schedule ($000)

Year	New Wells	Well Drilling (Kinley)	Piping & Power Loop	Mining Block Piping & Electrical	Main Events
1	13		$565	$751	
2	86	$18,684	$273	$4,971	
3	4	$2,982	$4,187	$231	
4	153	$15,650	$141	$8,844	Stage 2 Commences
5	97	$11,670	$5,035	$5,607	Solids Pond 2
6	118	$12,730	$888	$6,821	
7	89	$12,255	$2,148	$5,144	Stage 3 Commences
8	135	$14,209	$21,240	$7,803	Solids Pond 3
9	73	$8,198	$1,747	$4,220	
10	160	$21,139	$1,207	$9,248	
11	35	$5,289	$2,028	$2,023	
12	373	$34,761	$414	$21,560	
13	26	$5,671	$2,592	$1,503	
14	644	$45,291	$361	$37,224	
15	0	$3,610	$4,867	$0	
16	480	$27,749		$27,745	
17	28	$6,754	$8,490	$1,618	Jack & Bore Tunnel 2
18	246	$20,429	$346	$14,219	
19	290	$22,290	$1,761	$16,762	
20		$282	$2,563	$0	
21		$35		$0	well abandonments
22		$57			well abandonments
23		$84			well abandonments
24		$20			well abandonments
Totals	3,050	$289,841	$60,854	$176,295	

A major plant expansion is scheduled in Year 2 & 3 for the Stage 2 SX-EW plant (see Section 22.3) so that Stage 2 production can commence in Year 4 at a PLS flow rate of 9,000 gpm to produce an additional 50 million pounds per annum (mppa). Construction of the Gunnison site ancillary facilities, enlargement of the Gunnison electrical substation, construction of the Gunnison process ponds, and much of the plant infrastructure occur in Stage 2. The rail siding and car unloading yard (the railyard), was previously scheduled for Stage 3 and is now included in Stage 2.

The second expansion of the WTP occurs in Year 7 to include membrane filtration. This is a major expansion with a total cost of $47.4 million inclusive of $15.2 million in capital equipment as well as a new slab, interconnect piping, a new E-house, power transformer, construction costs, indirects, and contingency.

A final major plant expansion (Stage 3) occurs in Year 5 & 6 that includes the doubling of the Stage 2 SX-EW facilities for an additional 50 mppa of copper, bringing the total production for the Gunnison Project to 125 mppa.

The capital cost estimates for Stage 2 and Stage 3 have been handled as discrete cost build-ups based on flowsheets, general arrangement drawings, material take-off for construction material quantities and capital equipment budgetary pricing.

In addition to the Stage 2 and Stage 3 plant constructions, there are the year-by-year capital costs associated mostly with drilling and development of the wellfield, the addition of ponds for water management and solids storage, and for expansion of the Water Treatment Plant.

21.4.3 Stage 2 SX-EW Plant Capital Costs

The construction of the Stage 2 SX-EW plant is scheduled to go into operation in Year 4. The plant includes three SX settlers, and a complete Tank Farm scaled for Stage 2 production, and an EW facility with 80 electrowinning cells and an Automatic Stripping Machine. All site grading, infrastructure development and the new Gunnison 69 kV to 24.9 kV substation are included in Stage 2. The plant includes the Gunnison raffinate pond, the Plant Run-off Pond, the Recycled Water Pond and the Clean Water Pond. The Stage 2 Plant also includes a sulfuric acid storage area for acid that is trucked in, prior to making acid on site in Stage 3. Ancillary buildings include a Security Building with a truck scale, an Administration Building, a Change House, a Plant Warehouse, a Plant Maintenance Building, and a Wellfield Maintenance Building.

Stage 2 capital costs are summarized by Area in Table 21-4 below. The total cost of Stage 2 plant development is $178.0 million not including wellfield development in Years 2 & 3 that increase the PLS flow to 9000 gpm from 4000 gpm. The 50 mppa copper production addition brings the total Gunnison Project capacity to 75 mppa.

Table 21-4: Stage 2 – 50 mppa SX-EW Plant Capital Cost

Area	Area Name	Cost ($000)	Main Items
000	Plant General	$24,099	Plant site grading; Plant Runoff Pond; Sitewide Communications
200	Wellfield Drilling		In Financial Model
300	Wellfield Infrastructure	-	In Sustaining Capex
350	Solution ponds	$9,007	Raffinate Ponds, VT Pumps, SS Piping, Electrical Distribution
400	Solvent Extraction	$22,304	SS Piping; power distribution, instrumentation
500	Tank Farm	$10,502	Tankage; crud tricanter; electrolyte filters; heat exchangers
600	Electrowinning	$28,419	80 EW Cells; Production bridge crane, automated stripping
605	Electrowinning (JC)	$2,083	Replacement of MacroAmp transformer-rectifier system
650	Fresh Water Systems	$2,177	Clean Water Pond installation, potable water well and piping
655	Fire Water Systems	$1,005	Installation of foam fire system; yard piping
660	Reclaim Water Systems	$6,170	Installation of Recycled Water Pond
700	Main Substation	$4,704	Expansion of Gunnison Substation to power the SX-EW plant
710	Transmission Power Line	$310	Overhead line to Gunnison Plant
800	Reagents	$183	Diluent storage, pumping, and piping
810	Sulfuric Acid Storage	$4,601	Installation of 2 sulfuric acid storage tanks, piping, and containment
900	Ancillary Facilities – General	$1,461	Installation of compressor, boiler, diesel feed pumps, piping and electrical
901	Security Building	$113	New prefabricated security building
902	Truck Scale	$157	Installation of truck scale
903	Administration Building	$940	Pre-engineered metal building plus architectural, plumbing, electrical
905	Change House	$1,157	
910	Plant Warehouse	$194	
921	Plant Maintenance Building	$484	
922	Wellfield Maintenance Building	$1,398	
	Freight	$6,528	
	Total Direct Cost	$127,997	
	Indirect Costs	$26,243	
	Contingency (15%)	$23,136	
	Owners Costs	$667	First Fills of SX area
	Total Stage 2 Capital Costs	$178,043	

21.4.4 Stage 3 SX-EW Plant Capital Costs

The Stage 3 plant expansion adds another 50 mppa in copper cathode capacity bringing the total plant capacity to 125 mppa. This expansion includes adding another 80 EW cells to the Gunnison tankhouse, three additional settlers to the SX circuit, additional tankage and electrolyte filters to the Tank Farm, and Table 21-5 summarizes the costs by Area for Stage 3 construction which comes to $104.3 million.

Table 21-5: Stage 3 – 100 mppa Capital Costs

Area	Area Name	Cost ($000)	Main Items
000	Plant General	-	
200	Wellfield Drilling (Kinley)	-	Drilling Capex in Financial Model
300	Wellfield Infrastructure	-	In Year-by-Year Sustaining Capex
350	Solution ponds	$2,648	Pumps, electrical improvements
400	Solvent Extraction	$21,869	3 more settlers; 6 mix tanks, pumper-mixers, SS piping, electrical
500	Tank Farm	$6,357	Tankage; electrolyte filters; heat exchangers
600	Electrowinning	$23,356	80 EW Cells; Production bridge crane
655	Fire Water Systems	$393	Installation of foam fire system; yard piping
660	Reclaim Water Systems	$636	Solids Ponds
710	Transmission Power Line	$8,500	Water Treatment Plant Capex
800	Reagents	$212	Additional Diluent Tank
810	Sulfuric Acid Storage	$1,912	Additional Sulfuric Acid Storage Tank
811	Sulfuric Acid Rail Terminal	$587	Railroad siding and sulfur unloading yard; transloading platform
812	Sulfuric Acid Infrastructure	$3,474	Railroad siding and sulfur unloading yard; transloading platform
900	Ancillary Facilities	$193	Additional compressor
	Freight	$4,253	
	Total Direct Cost	$74,390	
	Indirect Costs	$16,273	
	Total Contracted Cost	$90,663	
	Contingency (15%)	$13,600	
	Owners Costs (First Fill)	-	
	Total Stage 3 Capital Costs	$104,263	

21.4.5 Water Treatment Plant Addition

A water treatment plant (WTP) for the Gunnison Project is staged in over four Phases. The first phase included the installation of a facility to neutralize raffinate for the purpose to dissolve CO_2 that forms from injecting sulfuric acid into the formation. Phase 1 of the WTP is included in the initial CAPEX for this Project. Phase 2 is an incremental expansion of the raffinate neutralization facility by adding new reactors and tanks but in the same areas as Phase 2.

Excelsior will begin to treat the water from rinsing depleted blocks of the ISR wellfield beginning in Year 8. The rinsing solutions must be treated by membrane filtration. The Phase 3 expansion of the water treatment facility adds a new slab and equipment to produce the rinsing solutions in Years 6 & 7 to commence rinsing in Year 8.

In Year 18, as rinsing accelerates, the membrane filtration plant expands in Phase 4 to meet the demand for rinsate.

The conceptual water treatment plant design was prepared by Hatch based on water quality and flow information provided to them from this project (Hatch, 2022). The capital cost estimate was developed assuming the Project will be executed using a design-build (DB) approach. Major equipment costs to support the capital cost estimate are based on budgetary vendor quotes and Hatch's data from similar projects. Typical costs for engineering, procurement, and construction management are covered by a standard percentage of the construction cost. The non-binding estimate of probable cost, with an assumed accuracy of -15% to +30% (AACE Class 4 Estimate), is presented below for project planning and evaluation purposes only. Table 21-6 is an overall estimation of the four phases of the WTP.

Table 21-6: Water Treatment Plant Capex by Phase

Phase	Phase Name	Year	Direct Cost ($000)	Freight ($000)	Indirects ($000)	Contingency (25%)	Total Costs	Main Items
1	Raffinate Neuralization Circuit	-1	$19.676	$1,504	$4,678	$6,464	$32,322	Reactor tank, Flocculating clarifier, Lime slaker & silo, MOL circuit, multimedia filters, reagent skids, Control Rm, E-Building
2	Raffinate Neutralization Circuit Expansion	1	$3,661	$280	$870	$1,203	$6,013	Add reactor, MMFs
3	NanoFiltration Circuit	7	$28,876	$2,207	$6,865	$9,487	$47,435	Add Flocculating and high-rate clarifiers, Reactor tanks, Nanofiltration skids, MMFs
4	NanoFiltration Circuit Expansion	17	$5,459	$417	$1,298	$1,794	$8,968	Add reactor tanks and high-rate clarifier
	Total Water Treatment Plant Capex		$57,672	$4,408	$13,711	$18,948	$94,738	

21.4.6 Sulfuric Acid Plant

For the Base Case, a sulfuric acid plant is planned for construction in Years 5 & 6 to reduce the cost of sulfuric acid necessary to the ISR process. The cost savings comes from the cogeneration of electricity from waste heat. In the Alternate Case, the sulfuric acid plant is omitted.

The 1,650 stpd sulfur burning sulfuric acid plant was estimated at a scoping level based (+35%/-25%) on the revised study by NORAM (2022), who provided a layout design, capital equipment specifications and rolled up costs for the main capital equipment, and costs for piping, ductwork, and instrumentation within the battery limits of their study. The main equipment within NORAM's scope of work includes:

- Main blower and steam generator
- Drying Tower
- Sulfur Furnace
- Waste Heat Boiler
- Converter
- Superheater
- Hot and Cold Pass Heat Exchangers
- Economizers
- Intermediate and Final Absorption Towers
- Circulation Acid Tank w/ Acid Cooler and pump
- Product and Final Acid Coolers and pumps
- Sulfuric Acid Storage Tanks (3)
- Deaerator
- Continuous Blowdown Tank
- Dump and Surface Condensers

- Caustic Feed Tank, pump, and cooler

Civil, concrete and steel quantities were developed by MTO.

Equipment outside NORAM's scope of supply includes:

- Sulfur unloading pumps
- Railcar steaming equipment
- Sulfur storage pit
- Sulfur day tanks and steam jacketing system
- Demineralized water plant.
- Cooling tower.
- Turbo generator.
- Emergency boiler

Interconnect piping, electrical equipment and distribution, and plant instrumentation were built up by using standard M3 factors based on capital equipment costs.M3 also included a dedicated substation to provide power to the sulfuric acid plant.

These prices were estimated by M3.

Table 21-7: Sulfuric Acid Capital Cost Summary

Area	Area Name	Cost ($000)	Main Items
812	Sulfuric Acid Plant	$75,565	NORAM sulfuric acid plant
813	Molten Sulfur Handling	$2,938	Molten sulfur unloading, pit, day tanks, and pumps
814	Power Cogeneration Plant	$11,911	Turbo generator, cooling tower
815	Boiler Water Treatment	$1,065	Water Treatment Package
	Freight	$7,505	
	Total Direct Cost	$98,984	
	Indirect Costs	$23,985	
	Contingency (30%)	$36,891	
	Total Sulfuric Acid Plant Capex	$159,860	

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

21.5 OPERATING COST

21.5.1 ISR Wellfield Operating Cost

Wellfield operating costs include labor, sulfuric acid, and consumables which are shown in Table 21-8 below. Initial wellfield operations involve injection of acidified raffinate from the SX-EW plant into injection wells, recovery of PLS from production wells, pumping the recovered PLS to a collection tank or pond for treatment in the Johnson Camp or Gunnison SX-EW plants, maintenance of the wells and wellfield, reconfiguring well equipment, and expanding piping, cabling, and instrumentation within the wellfield as required. Stage 1 wellfield production supports 3,800 gpm of PLS flow to Johnson Camp; Stage 2 production supports a combined PLS flow of 9,500 gpm to both plant; and Stage 3 supports up to 19,500 gpm of PLS flow.

Wellfield labor consists of operators who service the wells and valve skids, take samples of solutions, read instrumentation, and monitor performance. Wellfield maintenance personnel repair the wellfield piping, switch pipelines from leaching service to rinsing service, and replace faulty instrumentation. The wellfield labor grows from 11 personnel in Stage 1 to 23 personnel in Stage 2 to 27 personnel in Stage 3. Accordingly, the labor cost per pound drops from $0.035/lb Cu in Stage 1 to $0.015/lb Cu in Stage 2 to $0.011/lb Cu in Stage 3.

Electric power costs are proportional to the number of wells operating in the wellfield. While the total power cost attributed to the wellfield pumping increases with increasing production, the unit power cost for life-of-mine is $0.029/lb Cu.

Costs shown are for example years of producing wellfield during each Stage.

When a mining block is completely leached, the wells will be reconfigured for groundwater rinsing, typically after four years of leaching operation. Rinsing will involve circulating fresh water through the mining block with periods of rest between circulations. Once rinsing has met permit conditions, the wells will be abandoned as described elsewhere.

Table 21-8: ISR Wellfield Operating Cost Breakdown

Cost Element	Stage 1 (Year 3)		Stage 2 (Year 6)		Stage 3 (Year 9)	
	Annual Cost ($000)	$/lb Copper	Annual Cost ($000)	$/lb Copper	Annual Cost ($000)	$/lb Copper
Wellfield Labor	$868	$0.035	$1,116	$0.015	$1,363	$0.011
Electrical Power	$832	$0.033	$2,150	$0.029	$2,885	$0.023
Sulfuric Acid (Wellfield make-up)	$30,283	$1.219	$58,874	$0.797	$38,999	$0.270
Maintenance Parts & Services	$319	$0.013	$1,397	$0.019	$2,356	$0.019
Supplies & Services	$66	$0.003	$196	$0.003	$332	$0.003
Total Wellfield Operating Costs	$32,367	$1.30	$63,733	$0.86	$45,225	$0.33

21.5.2 SX-EW Operating Cost

The operating cost for the combined SX-EW facilities averages $21.4 million per year or $0.238 per pound of copper produced for life-of-mine, not including G&A, water treatment costs, or evaporation ponds. Stage 1 (Years 1 through 3) plant operating costs average $8.1 million or $0.33 per pound. Stage 2 (Years 4 through 6) plant operating costs average $21.4 million per year or $0.29 per pound. Stage 3 (Years 7 through 24) plant operating costs average $27.4 million or $0.22 per pound. Table 21-9 gives example years within Stages 1, 2, and 3 showing the breakdown of SX-EW operating cost by operating labor, reagents, power, maintenance labor and spare parts, and operating supplies.

Table 21-9: Summary SX-EW Operating Cost

Cost Element	Stage 1 (Year 3)		Stage 2 (Year 6)		Stage 3 (Year 9)	
	Annual Cost ($000)	$/lb Copper	Annual Cost ($000)	$/lb Copper	Annual Cost ($000)	$/lb Copper
SX-EW Labor	$1,657	$0.07	$3,020	$0.04	$3,111	$0.02
Electrical Power	$3,659	$0.15	$8,824	$0.12	$13,817	$0.11
Reagents	$833	$0.03	$2,479	$0.03	$2,948	$0.02
Maintenance Parts & Services	$1,752	$0.07	$6,642	$0.09	$6,780	$0.05
Supplies & Services	$197	$0.01	$508	$0.01	$799	$0.01
Total SX-EW Operating Costs	$8,098	$0.33	$21,473	$0.29	$27,454	$0.22

The labor cost for the SX-EW area is based on an average wage rate of $67,390 and 35% fringe benefits. Stage 1 starts with a staffing plan of 39 personnel for the Johnson Camp plant at an annual labor cost of $1.7 million. In Stage 2, the Gunnison SX-EW plant is placed into operation with an additional 27 plant personnel increasing the total annual labor cost to $3.0 million. For the Stage 3 addition to the Gunnison SX-EW plant, 4 additional cathode harvesters will be added, bringing the annual plant labor staff to 70 personnel and increasing the annual labor cost to $3.1 million. At full staffing, the SX-EW plant will consist of 13 Operations Administration personnel, 34 operations personnel, and 23 maintenance personnel. Table 21-10 lists the SX-EW plant operator and maintenance positions, rates, and quantity of personnel at Year 9. The cost of labor is totaled for full production.

Table 21-10: SX-EW Operating Labor Cost Summary (Year 9)

		Annual Salary / Position		Extended Annual Labor Costs		
Department and Position	Number of Personnel	Annual Salary ($)	Benefits (35%)	Annual Labor ($)	Annual Benefit ($)	Total Annual Labor Cost ($)
Administration - Operations						
SX-EW Manager	1	$140,000	$49,000	$140,000	$49,000	$189,000
Process Manager	1	$120,000	$42,000	$120,000	$42,000	$162,000
SX-EW Superintendent	1	$99,996	$34,999	$99,996	$34,999	$134,995
SX-EW Shift Supervisor	8	$65,000	$22,750	$520,000	$182,000	$702,000
Administrative Assistant	2	$32,000	$11,200	$64,000	$22,400	$86,400
SX-EW Operations						
SX Operator	8	$45,800	$16,030	$366,400	$128,240	$494,640
SX Helper	4	$37,500	$13,125	$150,000	$52,500	$202,500
EW Operator	6	$46,000	$16,100	$276,000	$96,600	$372,600
EW Helper	8	$37,500	$13,125	$300,000	$105,000	$405,000
Laborer	8	$33,500	$11,725	$268,000	$93,800	$361,800
SX-EW Maintenance						
Maintenance Supervisors (Mech+Elec)	1	$85,008	$29,753	$85,008	$29,753	$114,761
Maintenance Planner	1	$65,040	$22,764	$65,040	$22,764	$87,804
Shift Supervisor	4	$65,000	$22,750	$260,000	$91,000	$351,000
SX-EW Maintenance						
Mechanic/Welder	8	$50,000	$17,500	$400,000	$140,000	$540,000
Mechanics helper	2	$52,000	$18,200	$104,000	$36,400	$140,400
Electrical Supervisor	1	$75,000	$26,250	$75,000	$26,250	$101,250
Electrician	3	$65,000	$22,750	$195,000	$68,250	$263,250
Instrument Tech	3	$69,000	$24,150	$207,000	$72,450	$279,450
Total SX-EW Labor	70			$3,695,444	$1,293,405	$4,988,849
SX-EW Labor: Average/employee				$52,792.06		$71,269.28

The annual cost of reagents for the SX-EW area is $3.0 million at the peak of production (Year 9) and includes extractant, diluent, cobalt sulfate, guar, mist suppressor (FC-1100), and sulfuric acid for electrolyte makeup. Sulfuric acid for the wellfield leaching is included in the wellfield operating cost estimate. The annual consumption of SX-EW reagents and cost for a typical year at full production is shown in Table 21-11 below.

Table 21-11: SX-EW Reagent Consumption and Costs

Copper Cathode Produced (Year 9)	125,697,500	lbs / year	
Reagent	**Quantity/Year (lbs)**	**Annual Cost**	**$/lb Copper**
Extractant	125,698	$541,756	$0.0043
Diluent	2,011,160	$1,005,580	$0.0080
Sulfuric Acid	27,653,450	$715,635	$0.0057
Cobalt Sulfate	12,570	$60,586	$0.0005
Guar	75,419	$471,366	$0.0038
Mist Suppressor (FC 1100)	3,771	$62,371	$0.0005
Total Reagents: SX-EW Plant		$2,857,294	$0.0227

21.5.3 General and Administrative Cost

The breakdown of G&A cost and labor detail is shown in Table 21-12 and Table 21-13. Allowances were made for non-labor components of general and administrative expenses, which include office supplies, fuels, communications, small vehicle maintenance, claims assessments, legal and auditing, insurances, travel, meals and expenses, community relations, recruiting and relocation expenses, and janitorial services.

Table 21-12: General and Administrative Cost Breakdown

	Year 3		Year 6		Year 9	
Copper Cathode Produced	24,851,262		73,906,508		125,697,500	
Cost Element	Annual Cost	$ / lb Copper	Annual Cost	$/lb Copper	Annual Cost	$/lb Copper
Labor & Fringes	$3,667,054	$0.148	$4,124,423	$0.056	$4,124,423	$0.033
Accounting (excluding labor)	$25,000	$0.001	$25,000	$0.000	$25,000	$0.000
Safety & Environmental (excluding labor)	$25,000	$0.001	$25,000	$0.000	$25,000	$0.000
Human Resources (excluding labor)	$25,000	$0.001	$25,000	$0.000	$25,000	$0.000
Security (excluding labor)	$25,000	$0.001	$25,000	$0.000	$25,000	$0.000
Assay Lab (excluding labor)	$300,000	$0.012	$300,000	$0.004	$300,000	$0.002
Office Operating Supplies and Postage	$40,000	$0.002	$40,000	$0.001	$40,000	$0.000
Maintenance Supplies	$306,516	$0.012	$306,516	$0.004	$306,516	$0.002
Propane Power	$36,183	$0.001	$47,337	$0.001	$47,501	$36,183
Communications	$70,000	$0.003	$70,000	$0.001	$70,000	$0.001
Small Vehicles	$125,000	$0.005	$125,000	$0.002	$125,000	$0.001
Claims Assessment	$10,000	$0.000	$10,000	$0.000	$10,000	$0.000
Legal & Audit	$300,000	$0.012	$300,000	$0.004	$300,000	$0.002
Consultants	$150,000	$0.006	$150,000	$0.002	$150,000	$0.001
Janitorial Services	$50,000	$0.002	$50,000	$0.001	$50,000	$0.000
Insurances	$2,000,000	$0.080	$2,000,000	$0.027	$2,000,000	$0.016
Subs, Dues, PR, and Donations	$60,000	$0.002	$60,000	$0.001	$60,000	$0.000
Travel, Lodging, and Meals	$150,000	$0.006	$150,000	$0.002	$150,000	$0.001
Recruiting/Relocation	$125,000	$0.005	$125,000	$0.002	$125,000	$0.001
Total General & Administrative Cost	$7,489,753	$0.301	$7,958,276	$0.108	$7,958,440	$0.063

Table 21-13: General and Administrative Labor Cost Summary (Year 9)

		Annual Salary / Position		Extended Annual Labor Costs		
Department and Position	Number of Personnel	Annual Salary ($)	Benefits (35%)	Annual Labor ($)	Annual Benefit ($)	Total Annual Labor Cost
General & Administrative						
General Manager	1	$192,000	$67,200	$192,000	$67,200	$259,200
Administrative Assistant 1	1	$50,400	$17,640	$50,400	$17,640	$68,040
Security Guard	12	$32,000	$11,200	$384,000	$134,400	$518,400
EHS Manager	1	$95,040	$33,264	$95,040	$33,264	$128,304
Safety Specialist	1	$65,000	$22,750	$65,000	$22,750	$87,750
HR Manager	1	$120,000	$42,000	$120,000	$42,000	$162,000
HR Administrative Assistant	1	$34,000	$11,900	$34,000	$11,900	$45,900
IT Technician	2	$64,800	$22,680	$129,600	$45,360	$174,960
Controller	1	$120,000	$42,000	$120,000	$42,000	$162,000
Staff Accountant	1	$72,000	$25,200	$72,000	$25,200	$97,200
Accounts Payable	1	$35,000	$12,250	$35,000	$12,250	$47,250
Cost Accounting	1	$65,000	$22,750	$65,000	$22,750	$87,750
Purchasing/Warehouse Manager	1	$120,000	$42,000	$120,000	$42,000	$162,000
Purchasing Agent	1	$80,004	$28,001	$80,004	$28,001	$108,005
Purchasing Assistant	1	$34,000	$11,900	$34,000	$11,900	$45,900
Warehouseman	4	$48,900	$17,115	$195,600	$68,460	$264,060
Administrative Assistant 2	1	$42,000	$14,700	$42,000	$14,700	$56,700
Process Engineer	1	$75,000	$26,250	$75,000	$26,250	$101,250
Surveyor/Technician	1	$54,000	$18,900	$54,000	$18,900	$72,900
Environmental Manager	1	$129,996	$45,499	$129,996	$45,499	$175,495
Environmental Engineer	1	$70,000	$24,500	$70,000	$24,500	$94,500
Environmental Technician	1	$52,000	$18,200	$52,000	$18,200	$70,200
Senior Geologist	1	$105,000	$36,750	$105,000	$36,750	$141,750
Geologist	1	$99,996	$34,999	$99,996	$34,999	$134,995
Metallurgist	2	$75,000	$26,250	$150,000	$52,500	$202,500
Chief Chemist	1	$94,500	$33,075	$94,500	$33,075	$127,575
Lab Technician	4	$54,000	$18,900	$216,000	$75,600	$291,600
Sr. Hydrologist	1	$65,000	$22,750	$65,000	$22,750	$87,750
Core/Hydro Techs	2	$54,996	$19,249	$109,992	$38,497	$148,489
Total General & Administration Labor	49			$3,055,128	$1,069,295	$4,124,423

21.5.4 Water Treatment Plant

An estimate of annual OPEX has also been developed based on vendor data, previous estimates for similar treatment systems and plant operating experience (Hatch, 2022). Major OPEX categories include labor, utility power, chemical reagents, process consumables, waste disposal and compliance sampling, analysis, and reporting. Annual wages for operators and electrical power cost are site specific and were provided by M3.

Line-item costs for the general categories include the following:

Labor

- Maintenance technicians
- Environmental health and safety support
- Administrative support
- Engineering
- Environmental compliance support
- Utilities
- Electrical power consumption based on estimate of online factor and connected loads

Process consumables and chemical reagents:

- Lime
- Alum
- Flocculant
- Caustic (NaOH)
- Sulfuric Acid
- Hydrochloric Acid
- Filter Media
- Sodium Bisulfite
- Sodium EDTA
- Antiscalant
- Biocide

A summary of operating costs for the Water Treatment Plant is provided in Table 21-14.

Table 21-14: Water Treatment Plant Operating Cost Summary

Cost Element	Minimum (Year 4) ($000)	Maximum (Year 20) ($000)	LoM Costs ($000)
Labor	$906	$906	$25,371
Power	$94	$4,289	$38,484
Reagents	$1,853	$47,223	$469,686
Maintenance	$8	$319	$5,499
Total WTP Operating Costs	$2,861	$52,738	$539,040

21.5.5 Sulfuric Acid Plant

The annual operating costs for the sulfuric acid plant, power plant, and associated facilities at full production are summarized in Table 21-15 below.

Table 21-15: Sulfuric Acid Plant Operating Costs

Annual Sulfuric Acid Production	589,475	short tons / year	
Annual Average Copper Production	124,672,205	lbs / year	
Cost Element	**Annual Cost ($000)**	**$ / Short ton Acid**	**$ / lb Copper**
Labor	$5,114	$8.68	$0.04
Reagents	$25,053	$42.50	$0.20
Fuel (propane)	$631	$1.07	$0.01
Power (Credit)	($6,385)	($10.83)	($0.05)
Maintenance	$3,232	$5.48	$0.03
Supplies	$2,865	$4.86	$0.02
Total Acid Plant Operating Costs	$30,510	$51.76	$0.24

21.5.5.1 Labor

Labor cost is based on a staffing plan of 28 operators and 28 maintenance personnel. The operating crew consists of a general foreman and technician on day shift, five days per week, and a control room operator and field operator each shift seven days per week.

21.5.5.2 Reagents

Reagents needed for the sulfuric acid plant includes the sulfur required for acid production and water treatment chemicals for the cooling tower and boiler feed water systems. One ton of sulfur will produce a little over 3 tons of sulfuric acid. Based on a requirement of 589,475 tons of sulfuric acid annually, approximately 192,482 tons of sulfur will be required. The cost of sulfur used is $130.00 per ton delivered to site and is based on the average published cost for U.S. west coast sulfur and Houston sulfur over the last eleven years with freight allowed to the project site. An allowance of $30,000 per year was used for the water treatment chemicals.

21.5.5.3 Fuels (Propane)

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

21.5.5.4 Power Credit

The power requirements to produce sulfuric acid was estimated to be 6,000 kW or $4.0 million annually at the cost of power of $0.08/ kWh. The turbine generator is expected to produce approximately 15.6 MW of power at a value of $10.4 million annually at the same cost of power. The excess power can displace purchased power for the SX-EW and leach facilities or sold back to the power company. The net power credit is $6.4 million annually. The power consumption and power produced were factored from existing in-house data on similar sulfur burning acid plants.

21.5.5.5 Maintenance

Annual maintenance cost for the sulfuric acid plant was estimated to be 4% of the installed cost of the acid plant or $2.4 million. The annual maintenance for the power plant was estimated to be $0.01 / kWh or $0.85 million. Total maintenance cost is $3.2 million annually. The maintenance cost includes an accrual for major repairs that will occur at intervals of 1.5 to 2 years.

21.5.5.6 Operating Supplies and Services

Operating supplies and services were estimated at 2.5% of the total installed cost of the acid plant and power plant or $1.5 million annually which equals $2.53/ton of acid.

Railroad services for unloading sulfur and operating the railcar unloading and storage facility is estimated to be $1.37 million annually which equals $2.33/ton of acid.

General and administration costs are included in the SX-EW costs. No additional G&A costs are required for the acid plant.

The total operating supplies and services estimate comes to $4.86/ton of acid produced.

21.6 RECLAMATION AND CLOSURE COST

In the 2014 Gunnison Copper PFS, reclamation and closure costs were accounted for in the financial model as sustaining capital costs. In the current FS, reclamation costs have been refined and are now accounted for as expenses (operating costs) with some concurrent well abandonment and JCM leach pad closure. The reclamation and closure costs used in the financial model are estimated to be $60.0 million. These costs are summarized in Table 21-16 below.

Four main components comprise the reclamation costs:

- Johnson Camp Mine, Ponds, Leach Pad & Waste Dumps
- Well Abandonment
- Gunnison Plant, Ponds, & Infrastructure
- Bond Fees

Table 21-16: Summary Reclamation and Closure Cost

Area	Reclamation & Closure Cost ($000)
JCM Buildings, Ponds, Waste Dump & Heap	$5,084
Well Abandonment	$17,708
Gunnison Plant, Ponds	$24,647
Bond Fees	$12,444
Total Reclamation & Closure	$59,884

In the case of the Johnson Camp Mine, M3 used the reclamation quantities developed by Bikerman (2008) in its NI 43-101 Feasibility Study for the reopening of the Johnson Camp Mine. M3 used its own labor hours and costs to develop the reclamation and closure cost for the Gunnison site.

Well abandonment is a concurrent closure activity that is captured by Kinley in its 2021 updated wellfield development cost estimate. The cost of well abandonment, $17.7 million excludes the abandonment of exploration holes that are captured in Year -1, which were capitalized as pre-production. The cost for rinsing is captured in the water treatment and comingled with wellfield operating costs although these costs could be considered reclamation and closure activities.

The Gunnison area reclamation and closure cost estimate includes closure estimates for each process and water treatment pond as well as an aggregate cost for demolition and reclamation of the Gunnison plant site. The reclamation cost includes dismantling all buildings and equipment and removing from the site. Above ground concrete will be demolished and removed from site or buried on site. Below ground concrete will remain and be covered.

Solution ponds will be drained, and the top lining removed to inspect the bottom lining for leaks. If there is evidence of leaks, the bottom lining must be removed, the soil at the leak tested for contamination, and any required remediation performed before the pond can be covered. If no evidence of leaks is found, the top lining can be folded over in place and the pond covered. The ponds must be filled to form a mound to prevent storm water from collecting over the pond and migrating into the pond.

The plant site will be graded to existing contours. Roads will be left in place; however, asphalt will be removed. The plant site and solution and evaporation ponds will be hydro-seeded for plant growth.

The bond fees are non-refundable expenses for covering the cost of project bonding. The estimated amount is based on 3% annually of the amount bonded.

In the current study, the reclamation costs for the acid plant were not included in the financial model because the intention is for the acid plant to be operated as a going concern as a separate business entity. The timing of reclamation of the acid plant is not likely to occur simultaneously with the reclamation and closure of the Gunnison Project.

22 ECONOMIC ANALYSIS

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

22.1 WELLFIELD STATISTICS

Wellfield production is reported as soluble copper removed from the ISR operation. The annual production figures were obtained from the extraction plan as presented earlier in this report.

22.2 PLANT PRODUCTION STATISTICS

The design basis for the process plant for each stage is:

- Stage 1 – 4,860 gpm nameplate SX flow; 3,890 gpm nominal SX flow
- Stage 2 – 14,610 gpm nameplate SX flow; 11,690 gpm nominal SX flow
- Stage 3 – 19,500 gpm nameplate SX flow; 15,600 gpm nominal SX flow
- 1.5 g/L recovered at the SX Plant

Average annual full-rate production is projected to be approximately:

- Stage 1 – 25 million pounds
- Stage 2 – 75 million pounds
- Stage 3 - 125 million pounds

Total production for the life of operation is projected at approximately 2,154 million pounds of copper.

22.2.1 Copper Sales

The copper cathodes are assumed to be shipped to buyers in the US market, with sales terms negotiated with each buyer. The financial model assumptions are based on experience with copper sales from similar operations in the US.

22.3 CAPITAL EXPENDITURE

Capital expenditures for this project include the construction of the in-situ recovery (ISR) wellfield, solvent extraction - electrowinning (SX-EW) plant and water treatment plant. Initial capital items include expenditures that are necessary to bring the plant into production. The estimated initial capital for Stage 1 production is $38.4 million. An additional $9.2 million of pre-production mining costs are capitalized as initial capital. Sustaining capital items include wellfield expansion, construction of the Gunnison SX-EW plant in two stages (Stage 2 and Stage 3), several ponds for process solutions, and for storing solids from the water treatment plant, the expansion of the water treatment plant, a sulfuric acid plant and a railroad siding and railcar unloading facility, and the replacement of capital equipment is estimated to be $1.03 billion.

The Alternate Case was developed which does not include an acid plant to be constructed and reduces the sustaining capital cost by $160million in Years 5, 6 & 7 of operations. In the Alternate Case, the railroad siding and railcar unloading is still included.

22.3.1 Initial Capital

The financial indicators have been determined with 100% equity financing of the initial capital. Any acquisition cost or expenditures, such as property acquisition, permitting, and study costs, prior to project authorization have been treated as “sunk” cost and have not been included in the analysis.

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

Table 22-1: Initial Capital Requirement (millions)

Category	Initial Capital ($M)
Wellfield Development	$5.4
Wellfield Infrastructure	$0.6
SX-EW Plant Improvements	$0.0
Water Treatment Plant	$32.3
Capitalized Pre-Production Cost	$9.2
Owner's Cost	$0.0
Initial Capital Cost	$47.6

22.3.2 Sustaining Capital

A schedule of capital cost expenditures during the production period was estimated and included in the financial analysis under the category of sustaining capital. The total life of operation sustaining capital is estimated to be $1.03 billion. This capital will be expended during a 30-year period and consists of $289.8 million for wellfield development, equipment, and abandonment, wellfield infrastructure, $237.1 million for wellfield infrastructure development, $282.3 million to construct the Gunnison SX-EW plant in two stages and water/solids management ponds, $160.0 million for an acid plant, $64.0 million for water treatment plant.

The Alternate Case leaves out the sulfuric acid plant, thereby reducing the sustaining capital by $160.0 million.

22.3.3 Working Capital

A 15-day delay of receipt of revenue from sales is used for accounts receivables. A delay of payment for accounts payable of 30 days is also incorporated into the financial model. In addition, working capital allowance of approximately $2.5 million for plant consumable inventory is estimated over Year -1 and Year 1. All the working capital is recaptured at the end of the mine life and the final value of these accounts is zero.

22.4 REVENUE

Annual revenue is determined by applying estimated metal prices to the annual payable metal estimated for each operating year. Sales prices have been applied to all life of operation production without escalation or hedging. The revenue is the gross value of payable metals sold before treatment charges and transportation charges. The average copper price used in the evaluation is $3.75/lb. for the life of the mine.

22.5 TOTAL OPERATING COST

The average Cash Operating Cost over the life of the operation is estimated to be $0.95 per pound of copper produced, excluding the cost of the capitalized pre-production leaching. Cash Operating Cost includes wellfield operations, process plant operations, water treatment, and general administrative cost. Table 22-2 below shows the estimated operating cost by area per pound of copper produced with the acid plant and Table 22-3 shows without the acid plant.

Table 22-2: Life of Operation Operating Cost – with Acid Plant

Operating Cost	$/lb Copper
Wellfield	$0.37
SX-EW Plant	$0.24
Water Treatment Plant	$0.25
General and Administrative	$0.09
Sub-Total: Operating Cash Cost	$0.95
Royalties, Taxes (excludes Income Tax), Reclamation & Salvage	$0.28
Total Cash Cost	$1.22

Table 22-3: Life of Operation Operating Cost – without Acid Plant

Operating Cost	$/lb Copper
Wellfield	$0.77
SX-EW Plant	$0.25
Water Treatment Plant	$0.25
General and Administrative	$0.09
Sub-Total: Operating Cash Cost	$1.35
Royalties, Taxes (excludes Income Tax), Reclamation & Salvage	$0.27
Total Cash Cost	$1.63

22.6 TOTAL CASH COST

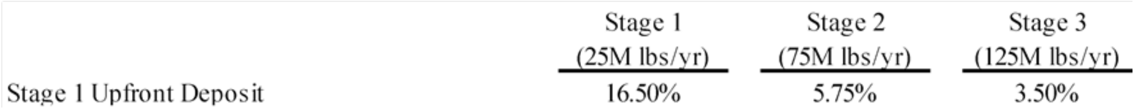
Total Cash Cost is the Total Operating Cost plus royalties, property, severance taxes, and reclamation and closure costs. The average Total Cash Cost over the life of the operation is estimated to be $1.22 per pound of copper produced for the Base Case and $1.63 per pound of copper produced for the Alternate Case, which is shown above.

22.6.1 Royalty

There are three entities that are entitled to royalties: the State of Arizona, Greenstone and Altius. The State has a sliding scale royalty between 2-8%. The royalty calculation takes into account the upper price limit based on the 60-month trailing average plus one standard deviation of the price and a lower price limit of $1.69 per pound. The sliding scale factor is estimated by dividing 6% by the difference of the annual upper price and the lower price. Then, the royalty is calculated by multiplying the sliding scale factor by the annual estimated price spread plus the minimum State royalty rate (2%).

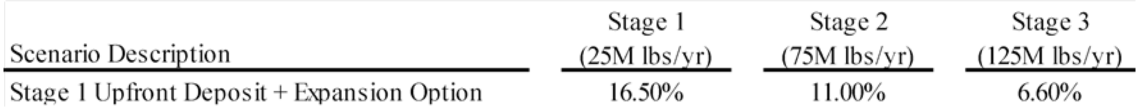
The Greenstone royalty is paid at the rate of 3.0% of the value of copper produced, up 1% from 2015, while the Altius royalty is paid at a rate of 1.5% of the copper value produced.

There is a streaming agreement with Triple Flag Mining Finance Bermuda Ltd. ("Triple Flag") that is based on levels of copper production. The percentages applicable at certain production levels are detailed in the table below.



	Stage 1 (25M lbs/yr)	Stage 2 (75M lbs/yr)	Stage 3 (125M lbs/yr)
Stage 1 Upfront Deposit	16.50%	5.75%	3.50%

Following a decision by Excelsior to expand the production capacity, Triple Flag will have the option to invest a further $65,000,000 in exchange for an increase in its entitlement to copper under the Stream ("Expansion Option"). The table below shows the range of percentage of production to be purchased by Triple Flag based on specified production levels and that includes Triple Flag's Expansion Option. Actual amounts will be calculated within the range, based on the proven production history.



Scenario Description	Stage 1 (25M lbs/yr)	Stage 2 (75M lbs/yr)	Stage 3 (125M lbs/yr)
Stage 1 Upfront Deposit + Expansion Option	16.50%	11.00%	6.60%

Modelling in this PFS is based on Triple Flag not exercising their Expansion Option.

Royalties for the life of the operation are estimated at $417.0 million and average $0.19 per pound of copper recovered.

22.6.2 Property and Severance Taxes

Property and severance taxes are estimated to be $112.2 million and average $0.05 per pound of copper recovered. Property taxes were estimated to be approximately $1.7 million per year during production, totaling $43.2 million for the life of the operation. Severance taxes are calculated as 2.5% of net proceeds before taxes from mining. Severance taxes are estimated to be approximately $69.0 million for the life of the operation.

22.6.3 Reclamation and Closure

Reclamation and closure costs include well abandonment costs for core holes and production wells, closure of process water impoundments, demolition of processing facilities and ancillary structures, and restoration of the land surface to pre-development conditions. The total cost for reclamation and closure is estimated to be $60.0 million and averages $0.03 per pound of copper recovered.

22.6.4 Income Taxes

Taxable income for income tax purposes is defined as metal revenues minus operating expenses, royalty, property and severance taxes, reclamation and closure expense, depreciation, and depletion. The combined federal and state corporate income tax rate in Arizona is 25.9 percent and is applied to 'taxable income' derived from the Gunnison Project.

Income taxes are estimated by applying state and federal tax rates to taxable income. The primary adjustments to taxable income are tax depreciation and the depletion deduction. Income taxes estimated in this manner total $697.6 million for the life of the Project.

22.6.5 Net Cash Flow

Net cash flow after all operating costs, capital costs and income taxes is estimated to be $ $3,185 million for the Base Case. Table 22-4 shows the project cash flow tabulation. The Alternate Case is estimated to be $2,514 million.

Table 22-4: Financial Analysis – Base Case

With Acid Plant		Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11
SXEW Operations															
SX Flow Rate (gpm)		9,427	-	-	3,722	3,746	3,780	9,250	9,350	12,415	18,370	18,250	18,875	18,930	18,465
SXEW PLS (g/l)		1.50	-	-	1.004	1.573	1.551	1.428	1.851	1.373	1.561	1.567	1.527	1.513	1.548
Contained Copper (klbs)		2,165,031	-	-	12,084	25,836	25,648	57,953	75,901	74,773	125,762	125,439	126,433	125,637	125,402
Copper Process Loss (klbs)		(11,166)	-	-	-	(89)	(797)	(60)	(857)	(867)	(763)	(453)	(735)	(410)	(827)
Recovered Copper (klbs)		2,153,864	-	-	12,084	25,747	24,851	57,892	75,044	73,907	124,999	124,986	125,698	125,227	124,574
Revenues ($000)															
Payable Metals															
Payable Copper (klbs)		2,153,864	-	-	12,084	25,747	24,851	57,892	75,044	73,907	124,999	124,986	125,698	125,227	124,574
Sulfuric Acid (t)		1,288	-	-	-	-	-	-	-	-	-	26	-	-	-
Income Statement ($000)															
UNIT Prices - Revenue															
Copper ($/lb.)		$ 3.75	$0.00	$0.00	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75
Sulfuric Acid ($/t)		$ 120.00	$0.00	$0.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00
Copper Revenue		$ 8,076,990	$ -	$ -	$ 45,314	$ 96,550	$ 93,192	$ 217,097	$ 281,413	$ 277,149	$ 468,747	$ 468,697	$ 471,366	$ 469,602	$ 467,154
Copper Revenue to Triple Flag (75% of Price)		$ (630,385)	$ -	$ -	$ (5,608)	$ (6,670)	$ (6,168)	$ (10,245)	$ (35,223)	$ (12,129)	$ (12,305)	$ (12,305)	$ (12,305)	$ (12,305)	$ (12,305)
Net Copper Revenue		$ 7,446,605	$ -	$ -	$ 39,706	$ 89,880	$ 87,024	$ 206,852	$ 246,190	$ 265,020	$ 456,442	$ 456,393	$ 459,061	$ 457,298	$ 454,849
Sulfuric Acid		$ 154,533			$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,159	$ -	$ -	$ -
Total Revenues		$ 7,601,139	$ -	$ -	$ 39,706	$ 89,880	$ 87,024	$ 206,852	$ 246,190	$ 265,020	$ 456,442	$ 459,551	$ 459,061	$ 457,298	$ 454,849
Operating Cost ($000)															
Well Field	$0.37	$ 794,512	$ -	$ -	$ 9,894	$ 14,741	$ 32,969	$ 33,710	$ 56,925	$ 64,294	$ 34,601	$ 32,086	$ 42,203	$ 36,108	$ 39,873
SXEW Plant	$0.24	$ 511,831	$ -	$ -	$ 5,726	$ 8,264	$ 8,098	$ 17,283	$ 19,617	$ 21,473	$ 27,283	$ 27,281	$ 27,364	$ 27,309	$ 27,234
Water Treatment Plant	$0.25	$ 530,606	$ -	$ -	$ 4,015	$ 3,222	$ 18,556	$ 2,861	$ 20,452	$ 18,791	$ 17,144	$ 14,146	$ 13,323	$ 22,132	$ 22,445
General Administration	$0.09	$ 199,476	$ -	$ -	$ 7,484	$ 7,490	$ 7,490	$ 7,956	$ 7,958	$ 7,958	$ 7,958	$ 7,958	$ 7,958	$ 7,958	$ 7,958
Total Operating Cash Cost	$0.95	$ 2,036,426	-	-	27,119	33,718	67,113	61,810	104,953	112,516	86,986	81,472	90,847	93,509	97,511
Other Expenses ($000)															
Miscellaneous	$0.01	$ 21,539	$ -	$ -	$ 121	$ 257	$ 249	$ 579	$ 750	$ 739	$ 1,250	$ 1,250	$ 1,257	$ 1,252	$ 1,246
Royalties	$0.19	$ 416,998	$ -	$ -	$ 2,096	$ 4,465	$ 4,310	$ 10,041	$ 12,664	$ 12,472	$ 21,094	$ 21,091	$ 21,211	$ 21,132	$ 21,022
Property Tax	$0.02	$ 43,237	$ -	$ -	$ 242	$ 515	$ 497	$ 1,158	$ 1,501	$ 1,478	$ 2,500	$ 2,500	$ 2,514	$ 2,505	$ 2,491
Severance Tax	$0.03	$ 68,986	$ -	$ -	$ 152	$ 689	$ 236	$ 1,788	$ 1,732	$ 1,874	$ 4,566	$ 4,673	$ 4,550	$ 4,490	$ 4,410
Salvage Value	$0.00	$ (8,978)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation & Closure	$0.03	$ 59,884	$ -	$ 97	$ 97	$ 260	$ 260	$ 260	$ 402	$ 402	$ 452	$ 452	$ 452	$ 822	$ 827
Total Production Cost	$1.22	$ 2,638,092	$ -	$ 97	$ 29,826	$ 39,904	$ 72,665	$ 75,635	$ 122,002	$ 129,480	$ 116,848	$ 111,439	$ 120,832	$ 123,709	$ 127,506
Operating Income ($000)		$ 4,963,047	$ -	$ (97)	$ 9,880	$ 49,976	$ 14,360	$ 131,217	$ 124,189	$ 135,540	$ 339,595	$ 348,113	$ 338,229	$ 333,588	$ 327,343
Depreciation															
Initial Capital		$ 47,621			14,710	9,404	6,716	4,796	3,429	3,425	3,429	1,713	-	-	-
Sustaining Capital		$ 1,033,188			$ 1,278	$ 10,699	$ 37,556	$ 51,745	$ 52,701	$ 77,148	$ 98,041	$ 95,497	$ 79,446	$ 61,330	$ 51,417
		$ 1,080,809			$ 15,988	$ 20,102	$ 44,272	$ 56,541	$ 56,130	$ 80,574	$ 101,470	$ 97,210	$ 79,446	$ 61,330	$ 51,417
Net Income before Taxes ($000)		$ 3,882,238	$ -	$ (97)	$ (6,108)	$ 29,873	$ (29,913)	$ 74,676	$ 68,059	$ 54,967	$ 238,125	$ 250,903	$ 258,783	$ 272,258	$ 275,926
Income Taxes		$ 697,606	-	-	-	821	-	2,171	1,692	3,146	42,195	45,257	47,236	50,653	51,656
Operating Income After Taxes ($000)		$ 3,184,631	$ -	$ (97)	$ (6,108)	$ 29,052	$ (29,913)	$ 72,505	$ 66,366	$ 51,821	$ 195,929	$ 205,645	$ 211,548	$ 221,605	$ 224,270

With Acid Plant		Total	12	13	14	15	16	17	18	19	20	21	22	23	24
SXEW Operations															
SX Flow Rate (gpm)		9,427	18,650	18,450	18,610	18,325	18,376	15,236	18,383	16,381	12,950	16,260	10,620	8,584	3,960
SXEW PLS (g/l)		1.50	1.522	1.542	1.533	1.564	1.534	1.784	1.320	1.636	1.493	1.347	1.656	0.809	0.500
Contained Copper (klbs)		2,165,031	124,478	124,824	125,178	125,737	123,629	119,198	106,406	117,549	84,787	96,102	77,143	30,447	8,686
Copper Process Loss (klbs)		(11,166)	(189)	(1,033)	(299)	(1,087)	-	(1,148)	(532)	(584)	(435)	-	-	-	-
Recovered Copper (klbs)		2,153,864	124,289	123,791	124,879	124,650	123,629	118,050	105,874	116,966	84,351	96,102	77,143	30,447	8,686
Revenues ($000)															
Payable Metals															
Payable Copper (klbs)		2,153,864	124,289	123,791	124,879	124,650	123,629	118,050	105,874	116,966	84,351	96,102	77,143	30,447	8,686
Sulfuric Acid (t)		1,288	-	-	7	-	60	-	69	70	97	44	236	271	407
Income Statement ($000)															
UNIT Prices - Revenue															
Copper ($/lb.)		$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75	$ 3.75
Sulfuric Acid ($/t)		$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00	$ 120.00
Copper Revenue		$ 8,076,990	$ 466,084	$ 464,216	$ 468,295	$ 467,438	$ 463,607	$ 442,688	$ 397,027	$ 438,622	$ 316,317	$ 360,382	$ 289,285	$ 114,176	$ 32,571
Copper Revenue to Triple Flag (75% of Price)		$ (630,385)	$ (57,766)	$ (57,926)	$ (12,305)	$ (12,305)	$ (57,371)	$ (55,316)	$ (49,379)	$ (54,550)	$ (39,346)	$ (44,597)	$ (35,799)	$ (12,129)	$ (4,031)
Net Copper Revenue		$ 7,446,605	$ 408,319	$ 406,290	$ 455,991	$ 455,134	$ 406,236	$ 387,372	$ 347,648	$ 384,071	$ 276,971	$ 315,785	$ 253,486	$ 102,047	$ 28,540
Sulfuric Acid		$ 154,533	$ -	$ -	$ 862	$ -	$ 7,212	$ -	$ 8,331	$ 8,408	$ 11,655	$ 5,297	$ 28,334	$ 32,471	$ 48,805
Total Revenues		$ 7,601,139	$ 408,319	$ 406,290	$ 456,853	$ 455,134	$ 413,448	$ 387,372	$ 355,979	$ 392,480	$ 288,626	$ 321,081	$ 281,820	$ 134,518	$ 77,345
Operating Cost ($000)															
Well Field	$0.37	$ 794,512	$ 33,818	$ 41,486	$ 32,461	$ 45,138	$ 29,994	$ 41,590	$ 30,527	$ 33,384	$ 29,577	$ 31,963	$ 19,904	$ 17,160	$ 9,557
SXEW Plant	$0.24	$ 511,831	$ 27,200	$ 27,143	$ 27,269	$ 27,276	$ 27,117	$ 26,249	$ 24,355	$ 25,931	$ 21,006	$ 22,587	$ 19,415	$ 11,597	$ 7,751
Water Treatment Plant	$0.25	$ 530,606	$ 20,014	$ 32,027	$ 21,531	$ 31,234	$ 21,676	$ 29,096	$ 40,041	$ 14,839	$ 52,738	$ 3,560	$ 42,835	$ 2,854	$ 25,900
General Administration	$0.09	$ 199,476	$ 7,958	$ 7,958	$ 7,958	$ 7,958	$ 7,958	$ 7,958	$ 7,958	$ 7,958	$ 7,958	$ 7,936	$ 7,962	$ 7,936	$ 7,959
Total Operating Cash Cost	$0.95	$ 2,036,426	88,990	108,614	89,219	111,607	86,746	104,894	102,881	82,113	111,280	66,046	90,115	39,546	51,167
Other Expenses ($000)															
Miscellaneous	$0.01	$ 21,539	$ 1,243	$ 1,238	$ 1,249	$ 1,247	$ 1,236	$ 1,181	$ 1,059	$ 1,170	$ 844	$ 961	$ 771	$ 304	$ 87
Royalties	$0.19	$ 416,998	$ 20,974	$ 20,890	$ 21,073	$ 21,035	$ 20,862	$ 19,921	$ 29,177	$ 37,190	$ 24,968	$ 26,125	$ 15,541	$ 6,179	$ 1,466
Property Tax	$0.02	$ 43,237	$ 2,486	$ 2,476	$ 2,498	$ 2,493	$ 2,473	$ 2,361	$ 2,117	$ 2,339	$ 1,687	$ 1,922	$ 1,543	$ 609	$ 174
Severance Tax	$0.03	$ 68,986	$ 3,887	$ 3,660	$ 4,535	$ 4,218	$ 4,025	$ 3,471	$ 3,112	$ 3,823	$ 2,165	$ 3,132	$ 2,344	$ 1,152	$ 302
Salvage Value	$0.00	$ (8,978)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (8,978)
Reclamation & Closure	$0.03	$ 59,884	$ 4,647	$ 1,186	$ 1,075	$ 2,369	$ 1,017	$ 1,241	$ 969	$ 1,017	$ 1,594	$ 1,602	$ 1,837	$ 1,873	$ 1,764
Total Production Cost	$1.22	$ 2,638,092	$ 122,227	$ 138,063	$ 119,649	$ 142,968	$ 116,358	$ 133,068	$ 139,316	$ 127,653	$ 142,538	$ 99,787	$ 112,151	$ 49,664	$ 45,981
Operating Income ($000)		$ 4,963,047	$ 286,092	$ 268,227	$ 337,204	$ 312,166	$ 297,089	$ 254,304	$ 216,663	$ 264,827	$ 146,088	$ 221,294	$ 169,669	$ 84,855	$ 31,364
Depreciation															
Initial Capital		$ 47,621	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital		$ 1,033,188	$ 51,103	$ 43,274	$ 37,685	$ 37,149	$ 35,074	$ 37,292	$ 35,847	$ 37,404	$ 33,297	$ 23,921	$ 16,375	$ 11,935	$ 15,974
		$ 1,080,809	$ 51,103	$ 43,274	$ 37,685	$ 37,149	$ 35,074	$ 37,292	$ 35,847	$ 37,404	$ 33,297	$ 23,921	$ 16,375	$ 11,935	$ 15,974
Net Income before Taxes ($000)		$ 3,882,238	$ 234,989	$ 224,953	$ 299,519	$ 275,017	$ 262,015	$ 217,012	$ 180,816	$ 227,424	$ 112,791	$ 197,373	$ 153,294	$ 72,919	$ 15,389
Income Taxes		$ 697,606	43,211	40,791	57,449	51,420	49,741	39,521	31,690	41,920	17,284	37,110	27,612	13,117	1,913
Operating Income After Taxes ($000)		$ 3,184,631	$ 191,778	$ 184,163	$ 242,069	$ 223,598	$ 212,274	$ 177,491	$ 149,126	$ 185,503	$ 95,507	$ 160,263	$ 125,682	$ 59,802	$ 13,476

With Acid Plant		Total	25	26	27	28	29	30	31	32	33
SXEW Operations											
SX Flow Rate (gpm)		9,427	-	-	-	-	-	-	-	-	-
SXEW PLS (g/l)		1.50	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)		2,165,031	-	-	-	-	-	-	-	-	-
Copper Process Loss (klbs)		(11,166)	-	-	-	-	-	-	-	-	-
Recovered Copper (klbs)		2,153,864	-	-	-	-	-	-	-	-	-
Revenues ($000)											
Payable Metals											
Payable Copper (klbs)		2,153,864	-	-	-	-	-	-	-	-	-
Sulfuric Acid (t)		1,288	-	-	-	-	-	-	-	-	-
Income Statement ($000)											
UNIT Prices - Revenue											
Copper ($/lb.)		$ 3.75	$ 3.75	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sulfuric Acid ($/t)		$ 120.00	$ 120.00	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Copper Revenue		$ 8,076,990	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Copper Revenue to Triple Flag (75% of Price)		$ (630,385)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Copper Revenue		$ 7,446,605	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sulfuric Acid		$ 154,533	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Revenues		$ 7,601,139	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Operating Cost ($000)											
Well Field	$0.37	$ 794,512	$ 243	$ 303	$ 5	$ -	$ -	$ -	$ -	$ -	$ -
SXEW Plant	$0.24	$ 511,831	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Water Treatment Plant	$0.25	$ 530,606	$ 16,128	$ 8,486	$ 10,560	$ -	$ -	$ -	$ -	$ -	$ -
General Administration	$0.09	$ 199,476	$ 3,305	$ 3,319	$ 3,305	$ -	$ -	$ -	$ -	$ -	$ -
Total Operating Cash Cost	$0.95	$ 2,036,426	19,677	12,108	13,870	-	-	-	-	-	-
Other Expenses ($000)											
Miscellaneous	$0.01	$ 21,539	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Royalties	$0.19	$ 416,998	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Property Tax	$0.02	$ 43,237	$ 40	$ 40	$ 40	$ 40	$ -	$ -	$ -	$ -	$ -
Severance Tax	$0.03	$ 68,986	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Salvage Value	$0.00	$ (8,978)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation & Closure	$0.03	$ 59,884	$ 2,563	$ 1,648	$ 2,333	$ 2,128	$ 1,198	$ 7,804	$ 7,025	$ 1,666	$ 6,547
Total Production Cost	$1.22	$ 2,638,092	$ 22,280	$ 13,796	$ 16,244	$ 2,168	$ 1,198	$ 7,804	$ 7,025	$ 1,666	$ 6,547
Operating Income ($000)		$ 4,963,047	$ (22,280)	$ (13,796)	$ (16,244)	$ (2,168)	$ (1,198)	$ (7,804)	$ (7,025)	$ (1,666)	$ (6,547)
Depreciation											
Initial Capital		$ 47,621	-	-	-	-	-	-	-	-	-
Sustaining Capital		$ 1,033,188	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
		$ 1,080,809	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Income before Taxes ($000)		$ 3,882,238	$ (22,280)	$ (13,796)	$ (16,244)	$ (2,168)	$ (1,198)	$ (7,804)	$ (7,025)	$ (1,666)	$ (6,547)
Income Taxes		$ 697,606	-	-	-	-	-	-	-	-	-
Operating Income After Taxes ($000)		$ 3,184,631	$ (22,280)	$ (13,796)	$ (16,244)	$ (2,168)	$ (1,198)	$ (7,804)	$ (7,025)	$ (1,666)	$ (6,547)

With Acid Plant	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Cash Flow ($000)														
Operating Income	$ 4,963,047	$ -	$ (97)	$ 9,880	$ 49,976	$ 14,360	$ 131,217	$ 124,189	$ 135,540	$ 339,595	$ 348,113	$ 338,229	$ 333,588	$ 327,343
Working Capital ($000)														
AR/AP/Inventory	$ 0	$ -	$ (1,000)	$ (903)	$ (1,520)	$ 2,862	$ (5,360)	$ 1,929	$ (152)	$ (9,965)	$ (581)	$ 791	$ 291	$ 430
Reclamation Bond	$ (0)	$ -	$ (3,237)	$ -	$ (5,421)	$ -	$ -	$ (4,733)	$ -	$ (1,676)	$ -	$ -	$ -	$ (941)
Total Working Capital	$ (0)	$ -	$ (4,237)	$ (903)	$ (6,941)	$ 2,862	$ (5,360)	$ (2,804)	$ (152)	$ (11,641)	$ (581)	$ 791	$ 291	$ (511)
Capital Expenditures ($000)														
Initial Capital														
Wellfield Drilling	$ 5,441	$ 1,904	$ 3,265	$ 272	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Wellfield	$ 636	$ 223	$ 381	$ 32	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Water Treatment Plant	$ 32,322	$ 11,313	$ 19,393	$ 1,616	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Infrastructure	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Owner's Cost	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Pre-Production Opex (Expensed in Year 1)														
Water Treatment Plant Opex	$ 8,434	$ -	$ 8,434	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Wellfield Opex	$ 789	$ -	$ 789	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital														
Wellfield Drilling	$ 289,841	$ -	$ -	$ -	$ 18,684	$ 2,982	$ 15,650	$ 11,670	$ 12,730	$ 12,255	$ 14,209	$ 8,198	$ 21,139	$ 5,289
Wellfield Infrastructure	$ 60,854	$ -	$ -	$ 565	$ 273	$ 4,187	$ 141	$ 5,035	$ 888	$ 2,148	$ 21,240	$ 1,747	$ 1,207	$ 2,028
Mining Block Development	$ 176,295	$ -	$ -	$ 751	$ 4,971	$ 231	$ 8,844	$ 5,607	$ 6,821	$ 5,144	$ 7,803	$ 4,220	$ 9,248	$ 2,023
Process Plant	$ 282,306	$ -	$ -	$ -	$ **35,609**	$ **142,435**	$ -	$ **31,279**	$ **72,984**	$ -	$ -		$ -	$ -
Acid Plant	$ 159,860	$ -	$ -	$ -	$ -	$ -	$ -	$ **31,972**	$ **95,916**	$ **31,972**	$ -	$ -	$ -	$ -
Water Treatment Plant	$ 64,032			$ **7,629**	$ -	$ -	$ -	$ -	$ -	$ **47,435**	$ -	$ -	$ -	$ -
Total Capital Expenditures ($000)	$ 1,080,809	$ 13,440	$ 32,262	$ 10,866	$ 59,536	$ 149,834	$ 24,635	$ 85,563	$ 189,338	$ 98,953	$ 43,253	$ 14,164	$ 31,594	$ 9,340
Cash Flow before Taxes ($000)	$ 3,882,238	$ (13,440)	$ (36,596)	$ (1,888)	$ (16,502)	$ (132,612)	$ 101,221	$ 35,822	$ (53,950)	$ 229,000	$ 304,279	$ 324,856	$ 302,285	$ 317,492
Cummulative Cash Flow before Taxes		$ (13,440)	$ (50,035)	$ (51,924)	$ (68,425)	$ (201,038)	$ (99,816)	$ (63,994)	$ (117,944)	$ 111,056	$ 415,335	$ 740,190	$ 1,042,476	$ 1,359,968
				1.0	1.0	1.0	1.0	1.0	1.0	0.5	-	-	-	-
Taxes														
Income Taxes ($000)	$ 697,606	$ -	$ -	$ -	$ 821	$ -	$ 2,171	$ 1,692	$ 3,146	$ 42,195	$ 45,257	$ 47,236	$ 50,653	$ 51,656
Cash Flow after Taxes ($000)	$ 3,184,631	$ (13,440)	$ (36,596)	$ (1,888)	$ (17,322)	$ (132,612)	$ 99,051	$ 34,130	$ (57,096)	$ 186,805	$ 259,022	$ 277,620	$ 251,633	$ 265,835
Cummulative Cash Flow after Taxes		$ (13,440)	$ (50,035)	$ (51,924)	$ (69,246)	$ (201,859)	$ (102,808)	$ (68,678)	$ (125,774)	$ 61,030	$ 320,052	$ 597,672	$ 849,305	$ 1,115,140
				1.0	1.0	1.0	1.0	1.0	1.0	0.7	-	-	-	-

Economic Indicators before Taxes ($000)		
NPV @ 7.5%	7.5%	$1,434,789
NPV @ 10.0%	10.0%	$1,057,643
IRR		40.6%
Payback	Years	6.5
Economic Indicators after Taxes ($000)		
NPV @ 7.5%	7.5%	$1,166,505
NPV @ 10.0%	10.0%	$855,369
IRR		37.5%
Payback	Years	6.7

With Acid Plant	Total	12	13	14	15	16	17	18	19	20	21	22	23	24
Cash Flow ($000)														
Operating Income	$ 4,963,047	$ 286,092	$ 268,227	$ 337,204	$ 312,166	$ 297,089	$ 254,304	$ 216,663	$ 264,827	$ 146,088	$ 221,294	$ 169,669	$ 84,855	$ 31,364
Working Capital ($000)														
AR/AP/Inventory	$ 0	$ 1,212	$ 1,696	$ (3,672)	$ 1,911	$ (330)	$ 2,563	$ 1,125	$ (3,207)	$ 6,665	$ (5,052)	$ 3,592	$ 1,897	$ 3,305
Reclamation Bond	$ (0)	$ -	$ 1,389	$ -	$ -	$ 448	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Working Capital	$ (0)	$ 1,212	$ 3,086	$ (3,672)	$ 1,911	$ 118	$ 2,563	$ 1,125	$ (3,207)	$ 6,665	$ (5,052)	$ 3,592	$ 1,897	$ 3,305
Capital Expenditures ($000)														
Initial Capital														
Wellfield Drilling	$ 5,441	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Wellfield	$ 636	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Water Treatment Plant	$ 32,322	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Infrastructure	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Owner's Cost	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Pre-Production Opex (Expensed in Year 1)														
Water Treatment Plant Opex	$ 8,434	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Wellfield Opex	$ 789	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital														
Wellfield Drilling	$ 289,841	$ 34,761	$ 5,671	$ 45,291	$ 3,610	$ 27,749	$ 6,754	$ 20,429	$ 22,290	$ 282	$ 35	$ 57	$ 84	$ 20
Wellfield Infrastructure	$ 60,854	$ 414	$ 2,592	$ 361	$ 4,867	$ -	$ 8,490	$ 346	$ 1,761	$ 2,563	$ -	$ -	$ -	$ -
Mining Block Development	$ 176,295	$ 21,560	$ 1,503	$ 37,224	$ -	$ 27,745	$ 1,618	$ 14,219	$ 16,762	$ -	$ -	$ -	$ -	$ -
Process Plant	$ 282,306		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Acid Plant	$ 159,860	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Water Treatment Plant	$ 64,032	$ -	$ -	$ -	$ -	$ -	$ 8,968	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Capital Expenditures ($000)	$ 1,080,809	$ 56,736	$ 9,766	$ 82,877	$ 8,477	$ 55,494	$ 25,831	$ 34,994	$ 40,813	$ 2,845	$ 35	$ 57	$ 84	$ 20
Cash Flow before Taxes ($000)	$ 3,882,238	$ 230,568	$ 261,547	$ 250,655	$ 305,599	$ 241,714	$ 231,036	$ 182,794	$ 220,807	$ 149,908	$ 216,207	$ 173,203	$ 86,668	$ 34,648
Cummulative Cash Flow before Taxes		$ 1,590,536	$ 1,852,083	$ 2,102,737	$ 2,408,337	$ 2,650,051	$ 2,881,087	$ 3,063,880	$ 3,284,687	$ 3,434,595	$ 3,650,803	$ 3,824,006	$ 3,910,674	$ 3,945,322
		-	-	-										
Taxes														
Income Taxes ($000)	$ 697,606	$ 43,211	$ 40,791	$ 57,449	$ 51,420	$ 49,741	$ 39,521	$ 31,690	$ 41,920	$ 17,284	$ 37,110	$ 27,612	$ 13,117	$ 1,913
Cash Flow after Taxes ($000)	$ 3,184,631	$ 187,358	$ 220,756	$ 193,205	$ 254,180	$ 191,972	$ 191,515	$ 151,103	$ 178,887	$ 132,624	$ 179,097	$ 145,592	$ 73,551	$ 32,735
Cummulative Cash Flow after Taxes		$ 1,302,498	$ 1,523,254	$ 1,716,459	$ 1,970,639	$ 2,162,612	$ 2,354,127	$ 2,505,230	$ 2,684,117	$ 2,816,741	$ 2,995,838	$ 3,141,430	$ 3,214,981	$ 3,247,715
		-	-	-	-	-	-	-	-	-	-	-	-	-

Economic Indicators before Taxes ($000)		
NPV @ 7.5%	7.5%	$1,434,789
NPV @ 10.0%	10.0%	$1,057,643
IRR		40.6%
Payback	Years	6.5
Economic Indicators after Taxes ($000)		
NPV @ 7.5%	7.5%	$1,166,505
NPV @ 10.0%	10.0%	$855,369
IRR		37.5%
Payback	Years	6.7

With Acid Plant	Total	25	26	27	28	29	30	31	32	33
Cash Flow ($000)										
Operating Income	$ 4,963,047	$ (22,280)	$ (13,796)	$ (16,244)	$ (2,168)	$ (1,198)	$ (7,804)	$ (7,025)	$ (1,666)	$ (6,547)
Working Capital ($000)										
AR/AP/Inventory	$ 0	$ 3,090	$ (1,617)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation Bond	$ (0)	$ -	$ 960	$ -	$ 960	$ 159	$ 960	$ 2,988	$ 3,338	$ 4,805
Total Working Capital	$ (0)	$ 3,090	$ (657)	$ -	$ 960	$ 159	$ 960	$ 2,988	$ 3,338	$ 4,805
Capital Expenditures ($000)										
Initial Capital										
Wellfield Drilling	$ 5,441	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Wellfield	$ 636	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Water Treatment Plant	$ 32,322	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Infrastructure	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Owner's Cost	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Pre-Production Opex (Expensed in Year 1)										
Water Treatment Plant Opex	$ 8,434	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Wellfield Opex	$ 789	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital										
Wellfield Drilling	$ 289,841	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Wellfield Infrastructure	$ 60,854	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Mining Block Development	$ 176,295	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ 282,306	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Acid Plant	$ 159,860	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Water Treatment Plant	$ 64,032	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Capital Expenditures ($000)	$ 1,080,809	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Cash Flow before Taxes ($000)	$ 3,882,238	$ (19,189)	$ (14,454)	$ (16,244)	$ (1,208)	$ (1,039)	$ (6,844)	$ (4,037)	$ 1,672	$ (1,742)
Cummulative Cash Flow before Taxes		$ 3,926,133	$ 3,911,679	$ 3,895,435	$ 3,894,227	$ 3,893,188	$ 3,886,345	$ 3,882,308	$ 3,883,980	$ 3,882,238
Taxes										
Income Taxes ($000)	$ 697,606	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Cash Flow after Taxes ($000)	$ 3,184,631	$ (19,189)	$ (14,454)	$ (16,244)	$ (1,208)	$ (1,039)	$ (6,844)	$ (4,037)	$ 1,672	$ (1,742)
Cummulative Cash Flow after Taxes		$ 3,228,526	$ 3,214,073	$ 3,197,829	$ 3,196,621	$ 3,195,582	$ 3,188,738	$ 3,184,702	$ 3,186,374	$ 3,184,631
		-	-	-	-	-	-	-	-	-

Economic Indicators before Taxes ($000)		
NPV @ 7.5%	7.5%	$1,434,789
NPV @ 10.0%	10.0%	$1,057,643
IRR		40.6%
Payback	Years	6.5
Economic Indicators after Taxes ($000)		
NPV @ 7.5%	7.5%	$1,166,505
NPV @ 10.0%	10.0%	$855,369
IRR		37.5%
Payback	Years	6.7

22.7 NPV and IRR

The economic analysis for the Base Case before taxes indicates an Internal Rate of Return (IRR) of 40.4% and a payback period of 6.5 years. The Net Present Value ("NPV") before taxes is $1,435 million at a 7.5% discount rate. The economic analysis for the Base Case after taxes indicates that the Project has an IRR of 37.3% with a payback period of 6.7 years. The NPV after taxes is $1,167 million at a 7.5% discount rate. Note that the payback period covers two major plant expansions that can be partially covered by operating profits. Only the initial capital cost will have to be completely financed through debt or equity. Table 22-5 compares the financial indicators for the Base Case and the Alternate Case.

Table 22-5: Financial Indicators

	Base Case	Alternate Case
Years of Commercial Production	24	24
Total Copper Produced (million lbs)	2,154	2,154
LoM Copper Price (avg $/lb)	$3.75	$3.75
Initial Capital Cost ($M)	$47.6	$47.6
Sustaining Capital Cost ($M)	$1,033	$880
Payback of Capital (pre-tax / after-tax)	6.5 / 6.7	5.9 / 6.0
Internal Rate of Return (pre-tax / after-tax)	40.6% / 37.5%	41.0% / 38.1%
LoM Direct Operating Cost ($/lb Copper recovered)	$0.95	$1.35
LoM Total Production Cost ($/lb Copper recovered)	$1.22	$1.63
Pre-Tax NPV at 7.5% discount rate ($M)	$1,435	$1,178
After-Tax NPV at 7.5% discount rate ($M)	$1,167	$976

Table 22-6 compares the Base Case project financial indicators with the financial indicators when other different variables are applied. Fluctuation in the copper price has the most dramatic impact on project economics. Fluctuation in the initial capital cost has the least impact on project economic indicators.

Table 22-6: After Tax Sensitivities – Base Case (with Acid Plant)

	Copper Price		
	NPV @ 7.5% ($M)	IRR%	Payback
Base Case	$1,167	37.5%	6.7
20%	$1,697	50.4%	4.3
10%	$1,433	44.0%	6.2
-10%	$898	30.8%	7.3
-20%	$627	24.2%	8.0
	Operating Cost		
	NPV @ 7.5% ($M)	IRR%	Payback
Base Case	$1,167	37.5%	6.7
20%	$1,031	33.2%	7.1
10%	$1,099	35.3%	6.9
-10%	$1,233	39.7%	6.5
-20%	$1,299	41.9%	6.3
	Initial Capital		
	NPV @ 7.5% ($M)	IRR%	Payback
Base Case	$1,167	37.5%	6.7
20%	$1,160	36.1%	6.7
10%	$1,163	36.8%	6.7
-10%	$1,170	38.2%	6.7
-20%	$1,173	39.0%	6.6

The Alternate Case economic after-tax sensitivities are shown below in Table 22-7.

Table 22-7: After Tax Sensitivities – Alternate Case (no Acid Plant)

	NPV @ 7.5% ($M)	IRR%	Payback
Copper Price			
Base Case	$976	38.1%	6.0
20%	$1,505	51.7%	4.3
10%	$1,241	45.0%	4.8
-10%	$706	30.8%	6.7
-20%	$432	23.0%	7.5
Operating Cost			
	NPV @ 7.5% ($M)	IRR%	Payback
Base Case	$976	38.1%	6.0
20%	$790	32.7%	6.5
10%	$883	35.4%	6.2
-10%	$1,066	40.8%	5.4
-20%	$1,157	43.4%	5.0
Initial Capital			
	NPV @ 7.5% ($M)	IRR%	Payback
Base Case	$976	38.1%	6.0
20%	$969	36.6%	6.1
10%	$972	37.4%	6.1
-10%	$979	38.9%	6.0
-20%	$982	39.8%	6.0

23 ADJACENT PROPERTIES

The Gunnison Project lies within the porphyry copper metallogenic province of the southwestern United States. It is located in the Cochise Mining District, which is dominated by Cu-Zn skarns. With the acquisition of the Johnson Camp Mine, Excelsior now controls a majority of historical producing properties in the district. Tungsten and minor lead-silver-gold have been produced in adjacent properties in the district (Cooper and Silver, 1964). In particular, tungsten has been historically produced in the area west of the Gunnison Project in the northern half of the Texas Canyon quartz monzonite stock before and during World War I. Lead-silver was also historically produced from Paleozoic limestones in the Gunnison Hills east of the Gunnison Project in the early 1900s (Cooper and Silver, 1964). Mineralization on adjacent properties is not necessarily indicative of the mineralization on the Gunnison Project. The author has relied on reports by others (as referenced) for the information presented in this section and has been unable to verify the information.

24 OTHER RELEVANT DATA AND INFORMATION – JOHNSON CAMP MINE HEAP LEACH - PRELIMINARY ECONOMIC ASSESSMENT

24.1 EXECUTIVE SUMMARY

M3 Engineering & Technology Corporation (M3) was commissioned by Excelsior Mining Corp. (Excelsior) to prepare a preliminary economic assessment (PEA) in accordance with the Canadian National Instrument 43-101 (NI 43-101) standards for reporting mineral properties, for the Johnson Camp Mine Heap Leach Project (the “JCM Project” or the “Project”) in Cochise County, Arizona, USA. The Project’s goal is to supplement mining and heap leaching at Excelsior’s Johnson Camp Mine using conventional heap leaching and processing at the JCM solvent extraction-electrowinning (SX-EW) plant that is fully operational. The plant was upgraded in 2019 and 2020 to treat PLS solutions from the Gunnison ISR Project located nearby to effect copper recovery by SX-EW, producing salable copper cathodes.

The Johnson Camp Mine is located about 65 miles east of Tucson, Arizona, on the southeastern flank of the Little Dragoon Mountains in the Cochise Mining District. The property is within the copper porphyry belt of Arizona. The Johnson Camp Mine contains two open pit mines, the Burro pit and the Copper Chief pit, that contain copper oxide, transition, and sulfide mineralization with associated molybdenum (not recovered by heap leaching), in potentially economic concentrations. Mining by a former owner, Nord Resources Corporation (Nord), ceased in 2012.

Heap leaching of sulfide copper with accelerated pyrite oxidation is proposed in this PEA. The Project plans include mining oxide, sulfide, and transition material from the Burro and Copper Chief pits for 20 years and heap leaching for an additional year to produce copper cathode at a capacity up to 25 million pounds per annum (mppa) by Year 3.

To restart the Johnson Camp Mine for heap leaching, two developments need to take place simultaneously: pre-stripping and mine development, and the construction of a new heap leach pad, Pad 5. Both are considered to require between six and nine months to complete before irrigation of the new leach pad can commence. Piping of PLS and raffinate lines from Pad 5 to the JCM ponds also fits within this time frame.

Excelsior plans to use a contract miner for all mine activities and its own staff for heap leach management, process plant operation, and general site management.

Excelsior selected M3 and other third-party consultants to prepare mine plans, a mineral resource estimate, a conceptual mine plan for economic assessment, a high-level capital cost for mine redevelopment and Pad 5 construction, to complete environmental studies, and prepare a discounted cash flow model to assess the viability of the JCM combined oxide, transition, and sulfide heap leach project, presented in this report. All consultants have the experience and capability to support the Project, as required and within the confines of their expertise.

The costs are based on fourth quarter 2022 U.S. dollars.

24.1.1 Key Data

The key results of this study are as follows.

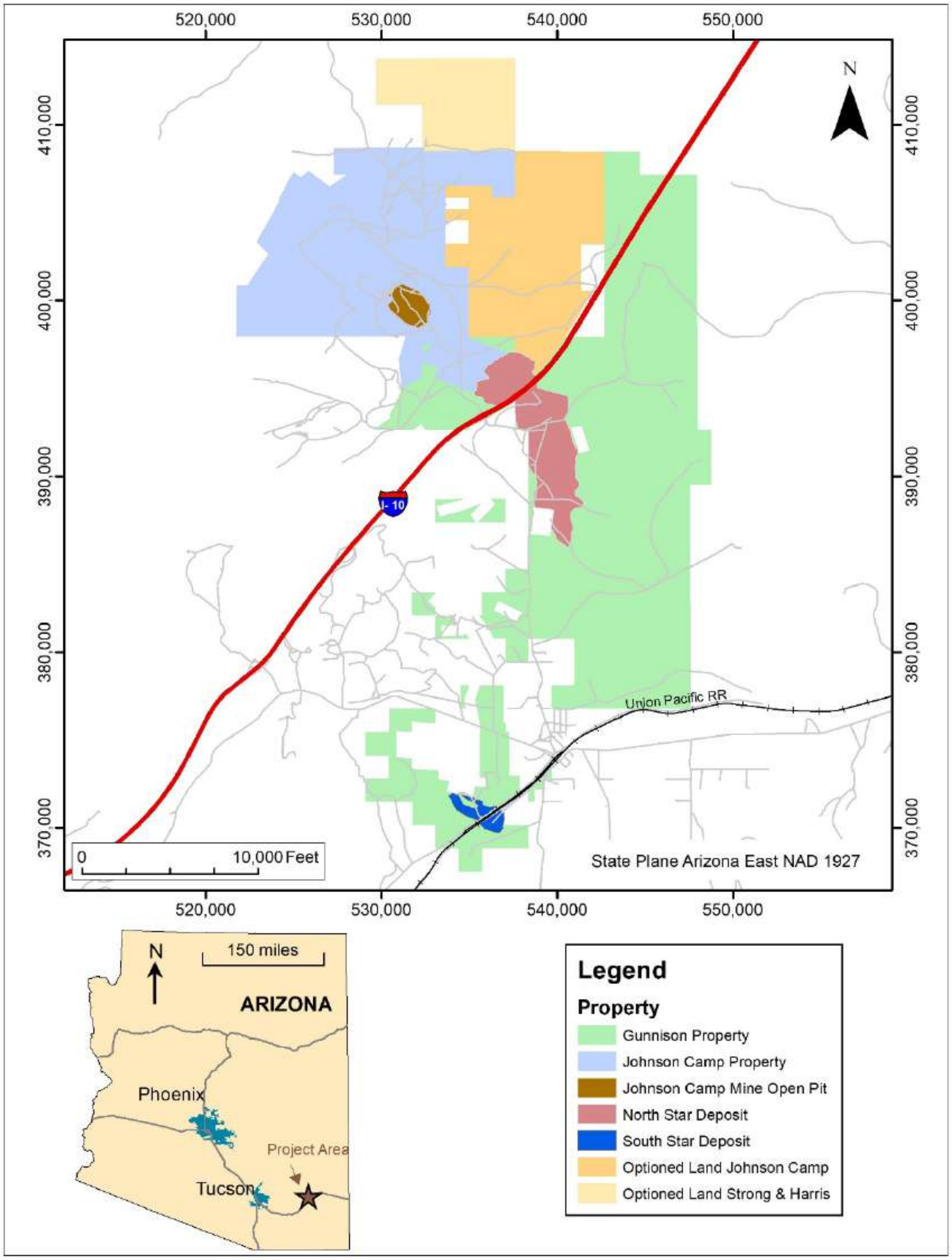
- The Project currently has a pit constrained mineral resource of 20.8 million short tons of measured, 87.1 million short tons of indicated, and 51.0 million short tons of inferred mineral resources with respective total copper grades of 0.31% measured, 0.32% indicated, and 0.32% inferred.

- The conceptual mine plan includes 85.3 million short tons of mineralized material mined over 20 years. The waste tons mined are estimated to be 110.8 million short tons, yielding a stripping ratio of 1.3 to 1 (waste to mineralization).

- The estimated copper production is approximately 492 million lbs of copper over 21 years.
- Life of mine total copper recovery is estimated to be 77%, made up of 95% acid soluble and cyanide soluble copper recovery and 70% primary sulfide copper recovery. Recovery of copper is estimated to be 80% during the first year after placement on the leach pad and 20% during the second year.
- Much of the primary sulfide copper mineralization is chalcopyrite, which typically responds very poorly to conventional heap leaching conditions; however, the unusually high pyrite-to-chalcopyrite ratio of about 3.5-to-1 makes the resource a good candidate for accelerated conditions that are promoted through rapid oxidation of the pyrite by microbial attack with ensuing increased rock and solution temperatures and supplemented by forced aeration.
- Accelerated leaching of all sulfide minerals will be enhanced by crushing and agglomeration with acidified raffinate that has been inoculated with native microbial cultures.
- The estimated initial capital cost is $58.9 million split between mine pre-production costs, fuel and explosives, refurbishment of the crushing-conveying system, and construction of the new Pad 5 leach pad.
- The total cost for reclamation and closure, including demolition of surface piping is estimated to be $15.8 million and averages $0.03 per pound of copper recovered.
- The economic analysis for the Base Case before taxes indicates an Internal Rate of Return (IRR) of 32.2% and a payback period of 4.01 years. Based on a copper price of $3.75 per pound, the Net Present Value (“NPV”) before taxes is $212.5 million at a 7.5% discount rate.
- The economic analysis for the Base Case after taxes indicates that the Project has an IRR of 30.4% with a payback period of 4.04 years. The NPV after taxes is $180.0 million at a 7.5% discount rate.
- Sensitivities for NPV@7.5%, IRR, and payback period for copper price, operating cost, and capital cost were determined for the JCM open pit, heap leach project. At a copper price $4.50/lb, 20% higher than the base study price of $3.75/lb, the after-tax IRR is 49.2% and the NPV is $321 million. A reduction in copper price ($3.00/lb) of 20% yields an after-tax IRR of 11.5% and an NPV@7.5% of $32 million.
- Bacterial oxidation of sulfide minerals will reduce acid consumption for the heap leaching operation so that after Year 2, acid may not be required for the heap leach pad, only for the agglomerator.

24.1.2 Property Description and Location

The Project is located in Cochise County, Arizona, approximately 65 miles east of Tucson in the historic Johnson Camp mining district. Figure 24-1 is a general location map and location of the Johnson Camp Mine on the north side of US Interstate 10 (I-10). The light blue color represents the Johnson Camp property boundary and the brown color shows the location of the Burro pit.



Source: Excelsior, 2023

Figure 24-1: Project Location Map

The Project is held by Excelsior through its wholly owned subsidiary Excelsior Mining Arizona, Inc. (Excelsior Arizona). Acquisition of the Nord Resources Corporation assets took place through a court-appointed receiver in December 2015.

24.1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Project is located in a sparsely populated, flat to slightly undulating ranching and mining area about 65 road miles east of Tucson, Arizona. The Tucson metropolitan area is a major population center (approximately 1,000,000 persons) with a major airport and transportation hub and well-developed infrastructure and services that support the surrounding copper mining and processing industry. The towns of Benson and Willcox are nearby and combined with Tucson can supply sufficient skilled labor for the Project.

Access to the Project is via the I-10 freeway from Tucson and Benson to the west or Willcox to the east. The Johnson Camp Mine can be accessed via good quality dirt roads heading approximately 1 mile north from the Johnson Road exit from I-10.

The elevation on the property ranges from 4,800 to 5,300 feet above mean sea level in the eastern Basin and Range physiographic province of southeastern Arizona. The climate varies with elevation, but in general the summers are hot and dry, and winters are mild.

Vegetation on the property is typical of the upper Sonoran Desert and includes bunchgrasses, yucca, mesquite, and cacti.

24.1.4 History

Modern mining and leaching operations at the Johnson Camp Mine began in the 1970s by Cyprus Minerals. Successor owners and operators include Arimetco, who mined JCM in the 1980s-early 90s, North Star, Summo Minerals, and Nord Resources Corporation who commenced mining in 2009 until 2012. Nord mined fresh material until mid-2010 and maintained leaching operations until late 2015, when the property was purchased by Excelsior.

24.1.5 Geological Setting and Mineralization

The Johnson Camp Mine is located within the Mexican Highland region of the Basin and Range province, which is characterized by fault-bounded mountain ranges, with large intrusions forming the cores of the ranges. The Project lies on the eastern edge of the Little Dragoon Mountains within the Cochise mining district. The Little Dragoon Mountains are an isolated, fault bounded horst block comprised of rocks spanning from 1.4 billion years ago (Ga) Pinal Group schists to Holocene sediments. The southern portion of the Little Dragoon Mountains consists predominately of the Texas Canyon Quartz Monzonite of Tertiary age, whereas the Pinal Group schists and a sequence of Paleozoic sedimentary units dominate the northern half of the range. At Johnson Camp, the important Paleozoic host is the Cambrian Abrigo Formation. The Texas Canyon Quartz Monzonite is porphyritic intrusion that crops out to the southwest of the Burro Pit at the Johnson Camp Mine.

Several deformations have occurred in the area with the most recent being the latest Cretaceous-Paleocene Laramide Orogeny compression, followed by Miocene and younger Basin and Range extension that has modified the topography to its current appearance.

The stratigraphy of the Burro pit and Copper Chief pit includes, from lowest to highest, Pioneer shale, diabase sill, Bolsa quartzite, three members of the Abrigo formation, and the Martin dolomite. Most mineralization is hosted in the lower and middle members of the Abrigo formation.

Moderate to intense calc-silicate alteration including garnet, epidote, and diopside are common in various assemblages, most intense calc-silicate alteration in the Lower and Middle Abrigo formations. Pervasive quartz veining occurs in both the Abrigo Formation and underlying Bolsa Quartzite throughout the Johnson Camp Mine area. Quartz vein orientations are typically sub-parallel to the stratigraphic units.

Primary copper mineralization at the Johnson Camp Mine is dominantly found along bedding planes or in veins and replacements as chalcopyrite along with quartz and pyrite, closely associated with skarn and calc-silicate alteration in the rock. The host formations are generally within the Bolsa Quartzite, Diabase Units, Lower and Middle Abrigo Formations. Oxidized mineralization consists of chrysocolla, malachite, copper limonite, and manganiferous wad; decreases with depth; but penetrates faults and stratigraphic contacts. Supergene chalcocite and occasional native copper occur generally below the oxidized zone. Below the supergene zone, the mineralization transitions to primary sulfides with local zones of supergene mineralization.

24.1.6 Deposit Types

The Johnson Camp Mine copper deposit is a type of copper skarn. The copper skarn at Johnson Camp and collectively in the Cochise mining district is presumably related to the Texas Canyon Quartz Monzonite. Copper skarns generally form in calcareous shales, dolomites, and limestones peripheral or adjacent to the margins of diorite to granite intrusions that range from dikes and sills to large stocks or phases of batholithic intrusions, and frequently are associated with mineralized intrusions. Copper mineralization forms along structurally complex and fractured rocks and convert the calcareous shales and limestones to andradite-rich garnet assemblages near the intrusive body, and to pyroxene and wollastonite rich assemblages at areas more distal to the intrusive that are subject to retrograde alteration with mineral hydrated silicate assemblages that overprint earlier garnet and pyroxene.

Mineralization at Johnson Camp occurs approximately 500 ft northeast of known occurrences of the Texas Canyon Quartz Monzonite intrusion as proximal skarn related to a porphyry copper system. This assumption is supported by the high abundance of garnet-epidote alteration in the mineralized zones, and the characterization of the deposits in numerous historical publications.

24.1.7 Exploration

Open pit mining commenced in 1975 by Cyprus and replaced the underground mining operations following the completion of an exploratory drilling program that defined the reserve of the Burro deposit. Cyprus and Arimetco collectively drilled 254 holes within both the Burro and Copper Chief pits. In 1999, Nord focused drilling exploration efforts on prospective targets outside of the pits that added no copper mineralization could be classified as reserves. Excelsior completed an exploration drilling program in 2022.

24.1.8 Drilling

The Johnson Camp Mine database contains 357 drill holes total 121,536 feet of drilling. Several drilling campaigns and operators span the contents of the database. Based on RESPEC's current knowledge, historical operators of the campaigns include Cyprus Mining (187 drill holes), Arimetco (83 drill holes), Nord (31 drill holes), Sumitomo (12 drill holes), and 16 drill holes were completed by an operator unknown to RESPEC. Excelsior drilled 44 holes. Drilling is concentrated in and immediately around the historically producing open pits.

Figure 24-9 shows the collar locations for the drill holes in the database and Table 24-1 is a breakdown of the drilling and operators in the Johnson Camp Mine area.

Table 24-1: Summary of Johnson Camp Drilling

Operator	Year	Holes	Feet
Cyprus Mining	1960 – 1986	187	61,417
Arimetco	1989 - 1997	83	24,638
Summo USA Corp.	1998	12	5,800
Nord Resources Corp.	2008-2010	31	14,368
Excelsior	2022	44	15,313
	Totals	357	121,536

The drilling sampling procedures provided samples that are representative and of sufficient quality for use in the resource estimations discussed in Section 24.14. The QP is unaware of any sampling or recovery factors that materially impact the mineral resources discussed in Section 24.14.

There is a general lack of down-hole deviation survey data for the historical holes in the Johnson Camp Mine area. The paucity of such data is not unusual for drilling done prior to the 1990s, the lack of deviation data contributes a level of uncertainty as to the exact locations of drill samples at depth. However, these uncertainties are mitigated to a significant extent by the vertical orientation of nearly all drill holes, and the open-pit nature of any potential future mining operation that is based in part on data derived from the historical holes.

24.1.9 Sample Preparation, Analysis and Security

All of the historical drilling, sample preparation and analysis of the samples presented in this report was under the control of the previous property owners. Excelsior drilled forty-four holes in 2022 and conducted core-duplicate sampling in 2016 and 2017.

The laboratory sample preparation and analysis procedures used by the previous owners of the deposits are unknown; however, major commercial laboratories using best practices at the time completed the majority of analyses. Additionally, most of the historical data were generated by well-known mining companies.

The data, information, samples, and core from the deposits have been under the control and security of AzTech Minerals since November 2006 and then Excelsior since October 2010. The original Information and samples are stored at Excelsior’s core storage facility in Casa Grande, with numerous copies held by Excelsior at its Phoenix, Arizona office.

The certification status of some of the historical analytical laboratories is not known. Southwestern Assayers and Chemists is the predecessor to Skyline. Mr. Bickel believes the historical labs were independent commercial laboratories that were widely recognized and used by the mining industry at that time.

Documentation of the methods and procedures used for historical sample preparation, analyses, and sample security, as well as for quality assurance/quality control procedures and results, is incomplete and in many cases not available. Despite this, some of the historical assay certificates have been preserved and Excelsior was able to reasonably duplicate the original results (described in 24.12.2.4). The QP is satisfied that the historical analytical data are adequate to support the current resources, interpretations, conclusions, and recommendations summarized in this report.

Excelsior’s sample preparation and analyses were performed at a well-known certified laboratory, and the sample security and QA/QC procedures are adequate to support the current resources, interpretations, conclusions, and recommendations summarized in this report.

24.1.10 Data Verification

Data verification, the process of confirming that data has been generated with proper procedures, has been accurately transcribed from the original source and is suitable to be used, has been performed by Mr. Bickel through reviews of original data and certificates, drill core, a site visit, and audits and analyses of Excelsior's drill-hole database. As a part of the verification of historical assays, RESPEC also analyzed core-duplicate data generated by Excelsior in 2016 and 2017 and compared the results to historical assays. The results are discussed in Section 24.12. There were no limitations on, or failure to conduct, the data verification for this report other than those discussed in this report. Mr. Bickel has verified that the project data are adequate as used in this report, most significantly to support the estimation and classification of the mineral resources reported herein.

24.1.11 Mineral Processing and Metallurgical Testing

Metallurgical testwork has been conducted in numerous campaigns by previous operators and owners including Superior Oil, Quintana Minerals, Phelps Dodge, Magma Copper, Arimetco, and Nord Resources. Testwork included a number of rounds of bottle roll and column testing. Early test programs indicated that total sulfuric acid consumption (before the electrowinning credit) will be approximately 9 lb H_2SO_4/lb of copper dissolved, that average PLS grade will be as high as 1.5 gpl Cu, and that about 65% of the total copper will dissolve, while about 95% of the ASCu should dissolve after sufficient contact time. This prior test work did not include augmented sulfide and transitional mineral leaching.

Nord Resources conducted eight column tests in 2011 on crushed and agglomerated material and 35 column tests in 2012 on crushed material minus 1" and minus 6". Of these columns, 23 provided useful results to determine copper recovery and acid consumption. The column testing programs are described in Section 24.13.2.1. The results of some of the column tests produced ambiguous results regarding acid consumption (higher in 6" crush than 1" crush).

There were only a few comparisons between fine and coarse column feeds, but they do not always make a strong case for converting JCM from ROM to crushing and agglomeration. A minus 6-inch fragment population probably does not represent ROM very faithfully, so it is possible that ROM underperforms a finer heap feed sufficiently to consider reactivating the crushing and screening plant. Crushing may be especially important as the pits deepen into transition mineralization.

Lacking recent laboratory testing and comparison of results with current heap performance, a meaningful prediction of near-term operating results requires further test work. However, for the purpose of this study it is not unreasonable to expect 95% average ASCu and CNCu extraction and net acid consumptions in pounds per ton of mineralized material as follows: Upper Abrigo, 45; Middle Abrigo, 55; Lower Abrigo, 40; Bolsa Quartzite, 25; and Martin/Escabrosa, 70.

Excelsior management, in collaboration with an industry leading sulfide leaching organization, have launched a sampling and metallurgical column testing program for material from the Burro pit, focusing on sulfide and mixed sulfide/transition/oxide mineralization. As the JCM pits deepen and non-ASCu copper minerals begin to overtake predominantly non-sulfide species, total copper extraction will decline, and the rate of extraction will diminish. Augmented bio-leaching is designed to counteract this effect by leaching the sulfide and transitional mineralization.

Crushing has been done at Johnson Camp, and the original crushing plant could be reactivated after repairing and upgrading primary and secondary crushers and screens. Excelsior should consider conducting additional parallel large-diameter column (or equivalent) tests on a bulk sample. These tests should mimic future operating conditions as faithfully as possible and should record standard parameters, including ORP/EMF.

24.1.12 Mineral Resource Estimate

The mineral resource estimation for the Johnson Camp Mine project was completed for disclosure in accordance with NI 43-101 with an effective date of July 13, 2022. The Johnson Camp Mine mineral resources are classified in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories in accordance with the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" (2014). All mineral resources in this estimate are classified as Inferred. A full description of the Johnson Camp mineral resource estimation methodology is presented in Section 24.14.

The Johnson Camp Mine copper resources were modeled and estimated using information provided by Excelsior. The information is derived from historical core holes drilled by Cyprus Mining, Arimetco, Summo USA Corp., and Nord Resources Corp. The drill hole database also includes analyses performed by Excelsior on the historical core.

Mineral domains were modeled by RESPEC to respect the lithologic and structural interpretations of the deposit. Following statistical evaluation of the drillhole data, mineral domains were modeled on cross sections for total copper ("CuT"). Low-, mid-, and high-grade domains were modeled for total copper and were numbered 100, 200, and 300, respectively. Grade domains were interpreted based on copper grade domains that ideally correspond to the underlying geology. The grade domain ranges are shown in Table 24-2 below:

Table 24-2: Grade Domain Ranges

Domain	Total Copper (%)
100	~0.025 to ~0.15
200	~0.15 to 0.7
300	> ~0.7

Soluble copper ratios were estimated within the total copper domains and lithologic units and used to calculate a soluble copper grade. A full description of the soluble copper estimate is in Section 24.14.6.2.

Mineral resources were estimated for total copper ("CuT"), acid-soluble copper ("CuAs"), cyanide-soluble copper ("CuCN"), and sulfide copper ("CuSu"). Once the final estimate was complete, a pit optimization using the inputs described in Section 24.14.10 were applied to the resource to evaluate if it has reasonable prospects for economic extraction. The contained resources within the cutoff grade defined by the pit optimization are given in Table 24-3.

Table 24-3: Johnson Camp Mineral Resources
(0.1% CuT cut-off)

Classification	Tons	% Cu	% CuAs	% CuCN	%CuSu	lbs CuT	lbs CuAs	lbs CuCN	lbs CuSu
Measured	20,771,000	0.31	0.13	0.05	0.09	127,545,000	54,762,000	22,564,000	37,551,000
Indicated	87,166,000	0.32	0.13	0.05	0.11	550,118,000	218,657,000	82,380,000	184,432,000
Inferred	50,998,000	0.32	0.12	0.04	0.12	322,656,000	119,614,000	45,377,000	122,781,000

1. The Effective Date of the mineral resources is July 13, 2022.
2. The project mineral resources are shown in bold and are comprised of all model blocks at a 0.1 % CuT cut-off that lie within optimized resource pits.
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
4. The estimate of mineral resources may be materially affected by geology, environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
5. Rounding as required by reporting guidelines may result in apparent discrepancies between tons, grade, and contained metal content.

Table 24-4 provides a breakdown of tons and grade of the JCM mineral resources by oxidation groups defined in modeling at a cut-off grade of 0.1% CuT that fit within the simulated economic pit shell.

Table 24-4: Johnson Camp Mineral Resources by Oxidation Group
(0.1% CuT cut-off)

Classification	Oxidation Group	tons	% CuT	% CuAs	% CuCN	%CuSu	lbs CuT	lbs CuAs	lbs CuCN	lbs CuSu
Measured	sulfide	1,257,000	0.29	0.02	0.03	0.24	7,245,000	600,000	727,000	5,918,000
Indicated	sulfide	6,784,000	0.42	0.04	0.04	0.34	56,881,000	5,392,000	5,652,000	45,836,000
Inferred	sulfide	5,876,000	0.35	0.04	0.05	0.26	41,455,000	5,038,000	5,514,000	30,902,000
Measured	transition	6,049,000	0.32	0.09	0.10	0.12	38,593,000	11,092,000	12,648,000	14,853,000
Indicated	transition	8,440,000	0.31	0.10	0.09	0.12	52,354,000	16,043,000	15,657,000	20,654,000
Inferred	transition	2,130,000	0.28	0.09	0.09	0.10	11,902,000	3,665,000	3,896,000	4,342,000
Measured	mixed	7,595,000	0.30	0.11	0.04	0.08	45,486,000	16,302,000	6,825,000	12,524,000
Indicated	mixed	55,824,000	0.30	0.11	0.05	0.09	338,947,000	123,230,000	54,370,000	103,121,000
Inferred	mixed	38,438,000	0.30	0.11	0.04	0.11	229,387,000	82,314,000	33,219,000	81,145,000
Measured	oxide	5,870,000	0.31	0.23	0.02	0.04	36,220,000	26,768,000	2,364,000	4,255,000
Indicated	oxide	16,118,000	0.32	0.23	0.02	0.05	101,935,000	73,991,000	6,700,000	14,821,000
Inferred	oxide	4,555,000	0.44	0.31	0.03	0.07	39,912,000	28,598,000	2,748,000	6,392,000

Future drilling, exploration, and resource definition at Johnson Camp Mine should focus on increasing the understanding of the distribution of cyanide soluble copper mineralization. Infill drilling in key areas to increase drill density, and drill-testing of the unconstrained limits of the deposit, particularly down-dip from known mineralization, should be prioritized.

24.1.13 Mineral Reserve Estimate

No mineral reserves are reported for this PEA.

24.1.14 Mining Method

The Johnson Camp Mine plan has been developed based on a new mineral resource estimate for the Burro and Copper Chief deposits. The mine plan targets the full resource at Johnson Camp over a 20-year period. A contract miner will execute the mining of the pits and deliver material to the primary crusher.

Mining of the deposit is expected to be accomplished with 100-ton haul trucks and front-end loaders. Mining is planned on 20-ft bench heights. The pit configuration is double benched with catch benches every vertical 40 ft.

Mined material is planned to be crushed and agglomerated before being placed on the leach pad using a conveyor stacker system. The mine plan is designed to provide 25 million pounds of recoverable copper per year to the existing SX-EW plant. The mine plan includes 69.7 million tons of M&I and 15.6 million tons of Inferred for placement on the leach pad over 20 years of mining, which includes a year of pre-production stripping and leach pad placement. The mine plan also includes mining and stockpiling of 111 million tons of waste for a LoM stripping ratio of 1.3:1.

24.1.15 Project Infrastructure

The Johnson Camp Mine is an existing and operating copper hydrometallurgical plant. The site includes the open pits, waste dumps, SX-EW plant facilities and mine infrastructure that will be used when mine operations in the Burro and Copper Chief pits resumes.

Water is supplied by two wells on site that produce 200 gpm of process make-up water.

An existing 69 kV power line runs to the JCM substation where power is stepped down to 5 kV for distribution around the JCM mine site.

24.1.16 Market Studies and Contracts

Excelsior has entered into a copper cathode purchase and sale agreement with Trafigura Trading LLC ("Trafigura") for 100% of copper cathode production from Excelsior's mineral projects. The agreement has a one year term and has been renewed on an annual basis each year, most recently to December 31, 2023. Pricing for product is based on Comex settlement prices, including a premium or discount depending on copper grade.

Please refer to Section 19 of this Report for other relevant Market Studies and Contracts.

24.1.17 Environmental and Permitting

The Johnson Camp Mine (JCM) is an inactive open pit mine. A processing (SX-EW) plant and associated ponds located at JCM are used to process pregnant leach solutions (PLS) from the Gunnison Project. A pipeline under I-10 connects Gunnison with JCM. JCM plans to resume mining of the open pit and process the mineralized material in a new heap leach pad. Existing permits will be modified to address resumption of mining at JCM.

Section 24.20 of this report describes the permit modifications that Excelsior will need to address to construct Pad 5 and reopen the two open pits for mining. The Aquifer Protection Permit, APP closure plan and bonding, will need to be amended for Pad 5. Five other state permits may have to be addressed with minor amendments.

24.1.18 Capital and Operating Costs

Mine operating costs reflect the operating costs to mine the Johnson Camp Mine from Year 1 through the end of mining in Year 19. The total mine operating costs are estimated to be $508 million or $2.59/t mined (mineralization and waste), including mining G&A.

The mine capital costs are estimated to be $9.8 million. These costs include contractor mobilization, construction of initial haul roads and the cost of mining for the first three quarters to achieve consistent release of leachable material (pre-production).

Capital and operating costs for the JCM Heap Leach Copper Project were estimated at a PEA level based on previous designs and operations, which included construction of Leach Pad 5. The current plan develops half of Pad 5 including design, excavation/grading, overliner material crushing and placement, and collection, aeration and leach piping and all of the emergency pond, the pump station and pumps and containment trenches. The estimated capital cost to develop Pad 5 and supporting infrastructure to the leach pad is $27.7 million.

An existing crushing and agglomeration plant will be used, requiring refurbishment of existing equipment and procurement and installation of additional equipment including conveyors and a stacker to place the leach material on the pad. Capital costs for refurbishment of the crush-agglomeration circuit and conveying-stacker system is estimated to be $21.4 million.

The plant operating cost includes the management and irrigation of Pad 5, and the JCM SX-EW plant. Components of the operating cost are labor, power, reagents & consumables, spare & maintenance supplies, and services. The heap leaching costs for Pad 5 are summarized in Table 24-5. The largest heap leach operating cost is sulfuric acid for heap leaching. The assumption is that for JCM as a standalone project, acid will have to be purchased at the nominal rate of $150/st. However, if the mining and heap leaching of JCM is done after the sulfuric acid plant for the Gunnison ISR option, the acid cost would be approximately $52/st after credit for power cogeneration.

Table 24-5: JCM Heap Leaching Operating Cost (Heap Leach only)

Cost Element	LoM Operating Cost ($000)	$/st leached material	$/lb Copper
Labor	$23,898	$0.28	$0.05
Power	$32,080	$0.38	$0.07
Reagents	$113,644	$1.33	$0.23
Maintenance	$46,051	$0.54	$0.09
Supplies & Services	$10,553	$0.12	$0.02
Total Leach Pad Costs	$226,226	$2.65	$0.46

Operating costs for the JCM plant are well known from recent operations of the plant. Staffing for plant maintenance labor were provided by Excelsior with updated salaries and benefit rates. Reagent pricing and consumptions (sulfuric acid, extractant, diluent, etc.) are known from ongoing operations. The JCM plant operating costs are summarized in Table 24-6.

Table 24-6: JCM Plant Operating Costs (SX-EW only)

Cost Element	LoM Operating Cost ($000)	$/st leached material	$/lb Copper
Labor	$29,918	$0.35	$0.06
Power	$53,090	$0.62	$0.11
Reagents	$16,598	$0.19	$0.03
Maintenance	$26,345	$0.31	$0.05
Supplies & Services	$6,261	$0.07	$0.01
Total Plant Operating Costs	$132,211	$1.55	$0.27

General and Administrative (G&A) costs include labor and fringe benefits for administration and support personnel and other support expenses detailed in Section 24.21.4.2. G&A expenses are based on the 2023 JCM budget provided by Excelsior and estimates for various services and expenses from recent studies of JCM for the Gunnison Project. The G&A cost for JCM averages $4.1 million annually of which labor is 38% and insurance is 22%. The cost per lb is approximately $0.31/lb Cu.

The reclamation and closure costs for the Project include reclamation and closure activities at both the JCM plant site and reclamation of leach heaps and stockpiles and are estimated to be $15.8 million, which also includes estimated bonding costs.

24.1.19 Economic Analysis

The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the Project. Annual cash flow projections were estimated over the life of the operation based on the estimates of capital expenditures and production cost and sales revenue. The sales revenue is based on the production of copper cathode.

Table 24-7 compares the financial indicators for JCM Heap Leach Project. The preliminary economic assessment is preliminary in nature, that includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Table 24-7: Financial Indicators

Item	LoM
Years of Commercial Production	20
Total Copper Produced (klbs)	491,754
LoM Copper Price (avg $/lb)	$3.75
Initial Capital Cost ($M)	$58.9
Sustaining Capital Cost ($M)	$36.1
Payback of Capital (pre-tax / after-tax)	4.01 / 4.04
Internal Rate of Return (pre-tax / after-tax)	32.2% / 30.4%
LoM Direct Operating Cost ($/lb Copper recovered)	$1.95
LoM Total Production Cost ($/lb Copper recovered)	$2.24
Pre-Tax NPV at 7.5% discount rate ($M)	$212.5
After-Tax NPV at 7.5% discount rate ($M)	$180.0

Table 24-8 provides a sensitivity analysis for the Base Case project financial indicators with the financial indicators when other different variables are applied. The results indicate that Project economics are impacted the most by fluctuation in the copper price. Fluctuation in the initial capital cost has the least impact on Project economic indicators.

Table 24-8: JCM Base Case After – Tax Sensitivities ($millions)

	Copper Price		
	NPV @ 7.5% ($M)	IRR%	Payback (yrs)
	180.0	30.4	4.0
20%	321	49.2	2.1
10%	251	39.9	2.6
-10%	107	20.9	4.9
-20%	32	11.5	10.6
	Operating Cost		
	NPV @ 7.5% ($M)	IRR%	Payback (yrs)
	180.0	30.4	4.0
20%	141	24.7	4.5
10%	161	27.5	4.3
-10%	199	33.3	3.4
-20%	218	36.4	2.9
	Initial Capital		
	NPV @ 7.5% ($M)	IRR%	Payback (yrs)
	180.0	30.4	4.0
20%	171	27.0	4.3
10%	176	28.6	4.2
-10%	184	32.4	3.7
-20%	189	34.9	3.3

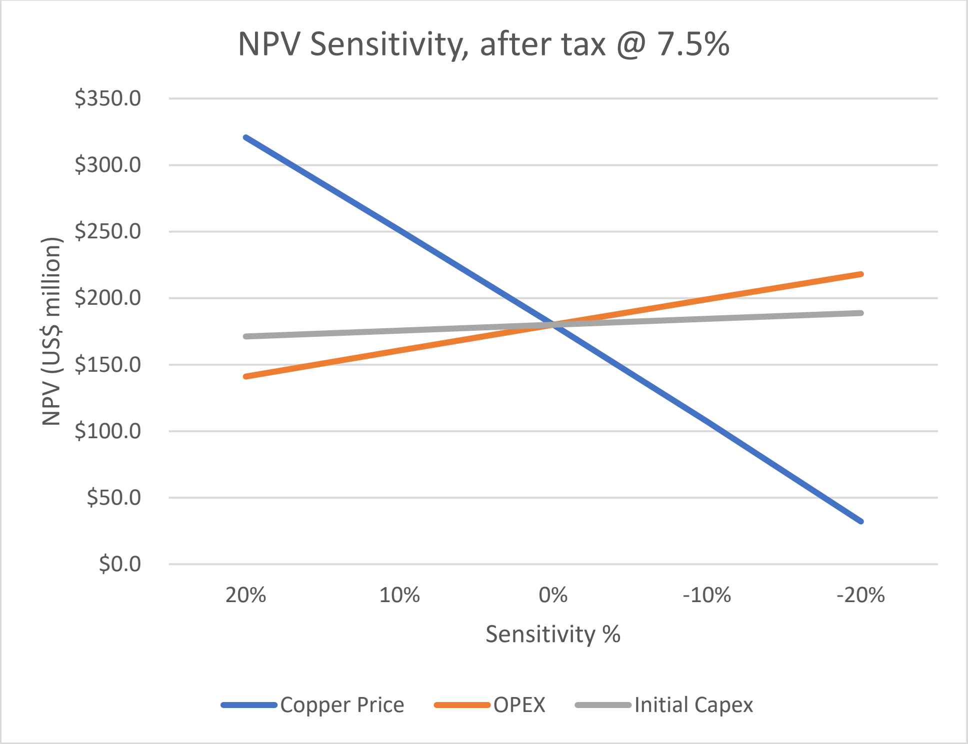


Figure 24-2: JCM NPV Sensitivity- After-Tax

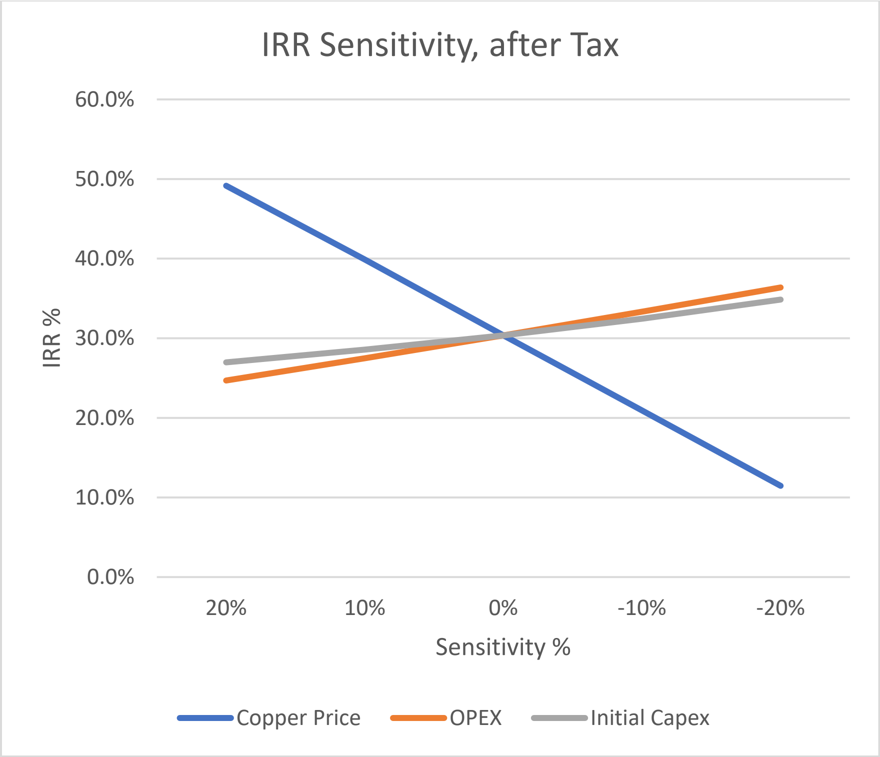


Figure 24-3: JCM IRR Sensitivity – After-Tax

24.1.20 Adjacent Properties

There are no relevant adjacent properties that are not controlled by Excelsior Mining.

24.1.21 Interpretation and Conclusions

The JCM plant has already been upgraded and JCM ponds are fully operational. The crushing plant will be utilized and this capital upgrade has been included along with the construction of the new leach pad, Pad 5.

Based on the current pit shell, mineral resources for the two pits is approximately 108 million tons of M&I and 51 million tons of Inferred at a cut-off grade of 0.1% CuT. The amount that is included in the conceptual mine plan over 19 years of mining is 69.7 million tons of M&I and 15.6 million tons of Inferred. It is possible that the mine life for the JCM open pit operation could be extended for several more years if copper prices continue to be favorable.

The full capital cost for restarting the JCM heap leaching operation between mining pre-production, first fills/Owners costs, leach pad construction, crusher and agglomerator refurbishment, new leach pad stackers and haul road construction is approximately $58.8 million. This project is a low-cost opportunity to exploit existing mineral resources with considerable upside if long-term copper prices and sulfuric acid prices remain favorable.

24.1.22 Recommendations

Excelsior management has launched a sampling and metallurgical testing program to evaluate the leaching strategy proposed in this study. The sampling and testwork program will assess the metallurgical zonation within the pits to estimate copper recoveries more accurately from each zone including testing the solubility of sulfide species. This program will help determine the long-term outlook for open pit mining and heap leaching at JCM.

The current plan includes crushing and agglomeration with conveying and stacking the agglomerated material on the leach pad. Excelsior should refine the cost to reactivate the crushing-agglomerating plant, design the conveyor system, and the stacking plan for the life of the mine.

Excelsior should consider conducting parallel large-diameter column (or equivalent) tests on a bulk sample. Metallurgical testing using bacterial enhancement and aeration should be conducted to more accurately evaluate its application to JCM sulfide mineralization and further evaluate the sulfide recoveries and leaching kinetics.

Excelsior should commission the re-design and estimating of Pad 5 using a footprint that can accommodate all of the leaching material in the mine plan to improve the accuracy of the initial and sustaining capital cost estimates for the leach pad.

Table 24-9: Budget for Recommended JCM Heap Leach Investigations

Detail	Cost US$
Metallurgical Testwork	$250,000
Feasibility Study	$500,000
Detailed Engineering for Leach Pad and Crusher refurbishment	$500,000
Total	**$1,250,000**

24.2 INTRODUCTION

The Johnson Camp Mine has historically been an open pit, heap leach operation since Cyprus Minerals opened the property in the 1970s. The operation includes two open pits, a two-stage crushing-agglomerating circuit, a fully functioning SX-EW plant capable of producing 25 million pounds of cathode copper per year, a complete set of PLS and raffinate ponds, and full infrastructure (ancillary facilities, access, power, water, and communications).

The JCM was operated on and off by three companies: Cyprus Minerals, Arimetco, and Nord Resources (Nord) before Excelsior purchased the property in 2015 to use for the first Stage of its Gunnison in-situ recovery (ISR) operation. As part of its development of the Stage 1 Gunnison Project, Excelsior made numerous upgrades to the JCM SX-EW plant and ponds.

The open pits have not operated since 2012 when Nord stopped mining and actively leaching. Three adjacent heap leach pads, known as Pad 1, 2, and 3 continued with residual leaching through 2017 and drain down from the heaps continues today. Pad 1, 2, and 3 are in the process of closure.

In 2006, Nord commissioned a series of studies to build a new heap leach pad called Pad 5. A feasibility study was prepared by Bikerman Engineering and Technology Associates (2007) that was followed by a detailed engineering design package for Pad 5 by Glasgow Engineering of Littleton, Colorado (2010) and an updated feasibility study by Curtis Associates (2011). The plan for Pad 5 was to build the new leach pad so that one side of it was to be used for higher grade crushed agglomerated material and the other side for lower grade run-of-mine (ROM) heap leach material. Pad 5 was never constructed due to Nord's sinking financial condition.

Excelsior is exploring re-opening the Burro and Copper Chief pits to open pit mining to produce crushed and agglomerated material to be leached on Pad 5 as a means of extracting copper from the remaining mineral resources left within the two pits. The leaching process is planned to include aeration and the use of enhanced bacteria to oxidize sulfides in leach pad to enhance the recovery of sulfide copper. This report is a Preliminary Economic Assessment (PEA) at an estimating accuracy of +/-30%.

24.3 RELIANCE ON OTHER EXPERTS

The authors, as qualified persons, have examined the historical data for the Gunnison Copper Project provided by Excelsior Mining Corp., and have relied upon the basic data to support the statements and opinions presented in this Technical Report. In the opinion of the authors, the Gunnison historical data, in conjunction with borehole assays conducted by Excelsior, are present in sufficient detail to prepare this report and are generally correlative, credible, and verifiable. The Project data are a reasonable representation of the Gunnison Copper Project. Any statements in this report related to deficiency of information are directed at information that, in opinion of the authors, is recommended by the authors to be acquired.

Excelsior relied on reports by John C. Lacy of the law firm, DeConcini, McDonald, Yetwin, & Lacy, for legal determination of lands on the Johnson Camp side of the property. Excelsior also obtained an ALTA Title Insurance Policy from First American Title Insurance Company for the patented mining claims and fee lands of the Johnson Camp property.

Clear Creek Associates (CCA) reviewed and updated the environmental report for the Johnson Camp Mine from the Phase I Site Assessment by Golder (2015) that documented the environmental condition of the Johnson Camp Mine. A new environmental section is provided in Section 24.20 for the Johnson Camp Mine heap leach operation. CCA has relied on information provided by Excelsior operations personnel and reports filed with agencies since the commencement of Stage 1 mining activities at the Gunnison Project in 2020.

24.4 PROPERTY DESCRIPTION AND LOCATION

Please refer to Section 4 of this Report for the relevant Property Description and Location.

24.5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Please refer to Section 5 of this Report for the relevant Accessibility, Climate, Local Resources, Infrastructure, and Physiography.

24.6 HISTORY

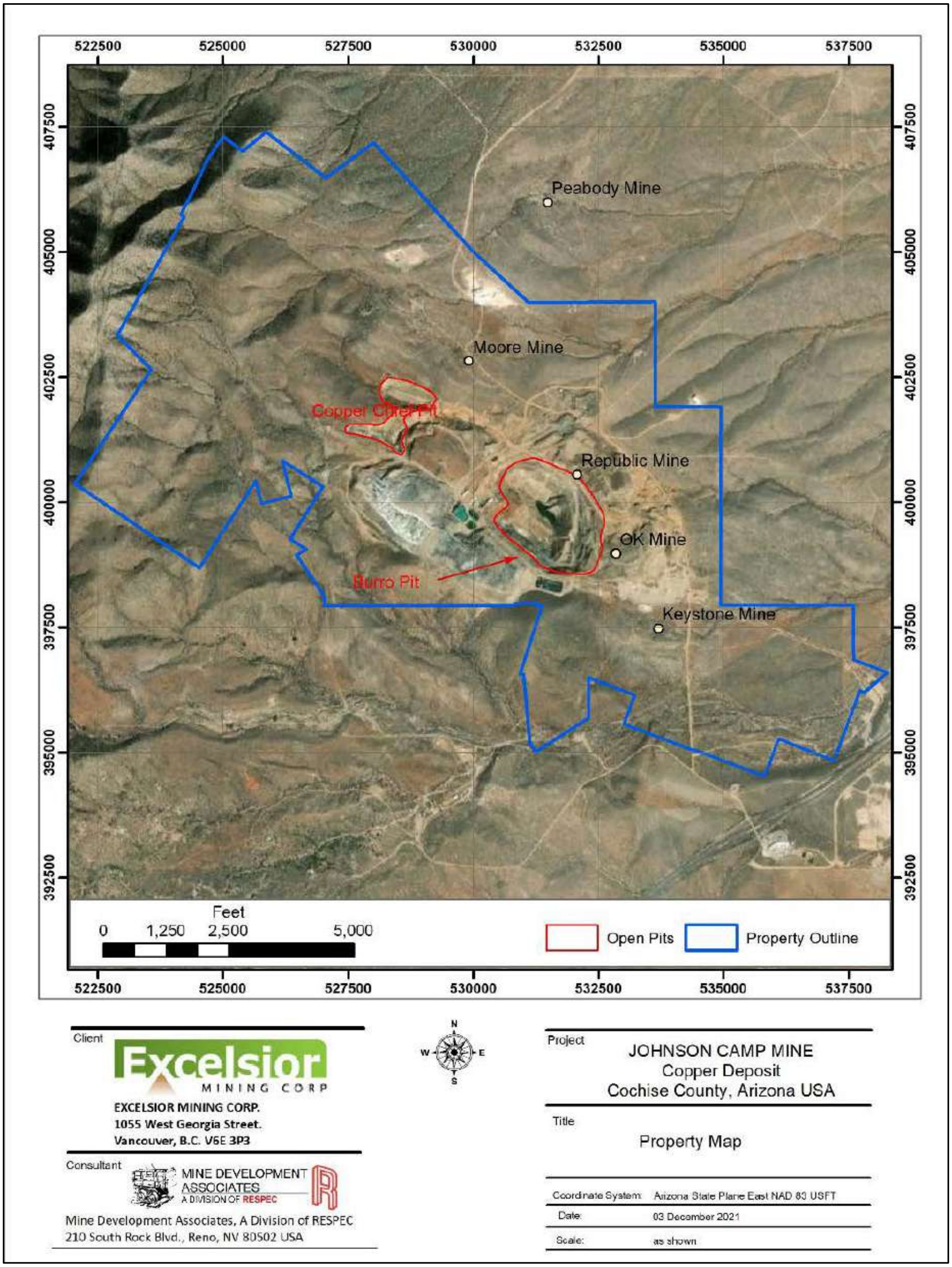
The information summarized in this section has been extracted and modified to a significant extent from (Zimmerman et al., 2016), sources therein, unpublished company files, as well as other sources as cited. The authors have reviewed this information and believe this summary is materially accurate.

24.6.1 District Exploration History

The Cochise district has seen considerable copper, zinc, silver, and tungsten mining beginning in the 1880s and extending to the present day. Prior to the 1880s, miners are said to have worked copper deposits cropping south of the JCM area. Between 1882 and 1981, the district produced 12 million tons of material containing 146 million pounds of copper, 94 million pounds of zinc, 1.3 million pounds of lead, 720 thousand ounces of silver, and minor quantities of gold (Keith et al., 1983). Much of the historical production came from small-scale underground copper-zinc mines located on what is now the Johnson Camp property controlled by Excelsior. The most significant of these producers were the Republic and Moore mines illustrated in Figure 24-4. From 1904-1940, material from these mines reportedly contained 4 to 4.5 percent copper and 0.5-0.75 ounces of silver per ton (Cooper et al., 1964). The zinc content for this period was not reported. After 1940, the material contained 1.5 to 3 percent copper, 5 to 10 percent zinc, and about 0.3 ounces of silver per ton. The Republic mine was the site of the historical concentrating plant in the district. Smaller underground mines in the area, such as the Peabody, reportedly yielded very high-grade mineralized material which averaged 7.5 percent copper, 4 ounces of silver per ton, and contained as much as 44 percent zinc (Cooper et al., 1964).

Copper-oxide mineralization has been mined at the Johnson Camp open-pit operation since 1975, most recently by Nord Resources Corporation from 2008 until 2010 Mining consisted of two open pits (Burro Pit and Copper Chief Pit), which are separated by roughly 2,000 feet along strike. The operation mined copper and processed the material via heap-leach and SX-EW. Previous operators include Cyprus Mines, Arimetco, and Nord Resources. This property is now controlled by Excelsior. Overall, approximately 39 million tons of material and 187 million pounds of copper have been produced out of the Johnson Camp open pits.

A major portion of the district's historical exploration work has taken place about 1.5 to 2 miles southeast of the Johnson Camp Mine. In the 1960s, it was recognized that potentially economic copper-skarn mineralization could be identified remotely by magnetic highs related to the magnetite content of these mineralized bodies. As a result, a magnetic high located southeast of the now nonexistent town of Johnson was drilled in the 1960s and the North Star deposit was discovered in the valley east of the mountain flank, concealed under alluvial cover. Since then, several companies have explored the area with extensive drilling and assaying, magnetic and induce polarization/resistivity ("IP/Res") surveys, metallurgical testing, hydrological studies, and In-situ Recovery ("ISR") tests. Eventually, the North Star copper deposit became known as the Gunnison deposit.



(from Excelsior, December 2021)

Figure 24-4: Historical Mines Near Johnson Camp

24.6.2 Johnson Camp Property History

The information summarized in this section has been extracted and modified from various historical reports and records. The authors have reviewed this information and believe this summary is materially accurate.

In the 1880s, a rail line was constructed approximately four miles south of what is now the JCM area and was the catalyst to the growth of the underground mining operations and processing of copper sulfide, copper oxide, silver, and zinc. By the 1900s, production surged, however, the surge was met with the decrease in copper prices, which eventually led to the forced closure of the Moore, Peabody, and Republic mine shown in Figure 24-4. The following summarizes the historical mining and exploratory operations by company that have taken place at the JCM area in ascending order.

Cyprus Mines Corporation (Cyprus) 1942 – 1986 With the abandonment and forced closure of the Republic mine in the early 1900's, the 20-year vacancy resulted in flooding of the mine. In 1942, Cyprus reopened the mine and began mitigating the flooding and the construction of a mill. As the copper prices rose and fell, Cyprus converted the underground mining operations to an open pit mine with the discovery of the Burro deposit in 1975, that hosted an estimated 22-million-ton reserve. In 1986 operations ceased with the declining economic value of copper. Within that period, Cyprus mined over 100 million pounds of copper bearing material with 51 percent total copper (% CuT) (Curtis Associates, 2011). Note: this section uses "CuT" to describe total copper. This description is equivalent to "TCu" which is used elsewhere in this report.

Arimetco 1989-1998 With the purchase of the Johnson Camp property, Arimetco continued mining the Burro deposit (now known as the Burro Pit). Advancements during Arimetco's ownership included expanding leach pads, construction of a crushing plant for better recovery, and open pit mining at the Copper Chief pit. Within the 10 years, Arimetco mined over 50 million pounds with 43% CuT. The property was eventually sold to Nord Resources Corporation (Curtis Associates, 2011).

Nord Resources Corporation (Nord) 1999 – 2015 Nord continued open pit mining at the Johnson Camp property until the fall of copper pricing occurred in 2003. Production ramped up in 2007 after Nord introduced better processing techniques via a new crushing and conveyor system, as well as expanding and upgrading the JCM. Again, production ceased due to economics, however, during the 11 years the Johnson Camp property was owned and operated by Nord, over 25 million tons were produced with an approximate 30% CuT recorded in the years of 2009 and 2010 (Curtis Associates, 2011).

Excelsior Mining Corporation (Excelsior) 2015-Present Excelsior purchased all assets of Nord Resources as they related to the Johnson Camp Mine, through a court appointed receiver, in December of 2015 (Zimmerman et. al 2016). No production at the Johnson Camp Mine has occurred since Excelsior purchased the property in 2015. Processing facilities have been upgraded and re-purposed for Excelsior's adjacent Gunnison Project.

24.6.3 Historical Mineral Resource and Reserve Estimates

A number of estimations of mineralized materials at the Johnson Camp Mine were carried out by historical operators, only a few of which are summarized herein.

The classification terminology is presented as described in the original references. It is not known if this terminology conforms to the meanings ascribed to the Measured, Indicated, and Inferred mineral resource classifications, or the Proven and Probable reserve classifications of the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines" ("CIM Standards"). The presentation of the historic mineral resources and mineral reserves does not imply that these are current or that there is a mineral reserve at Johnson Camp. The term 'ore' is used in the historic sense only.

In March 2000, the Winters Company prepared a Feasibility Study for Nord Copper Corporation (subsequently Nord Resources Corporation). The mineral resources were summarized in the report at various cut-off grades for both the Burro and Copper Chief pits. At a 0.2% total copper cut-off, the Burro Pit deposit was estimated to contain 67.2 million tons of material at a grade 0.397% total copper equating to 532.8 million pounds of contained copper. According to the report, 59% of the contained copper pounds were classified as measured, 29% of the contained copper pounds were classified as indicated, and 12% of the contained copper pounds were classified as inferred at the Burro Pit. At Copper Chief, the mineral resources reported at a 0.2% total copper cut-off were 68.9 million tons at a grade of 0.344% total copper equating to 474.1 million pounds of contained copper. The Copper Chief mineral resources were classified as follows: 35% of contained copper pounds were classified as measured, 18% of the contained copper pounds were classified as indicated, and 47% of the contained copper pounds were classified as inferred. The estimate was summarized from a block model containing 50-foot by 50-foot by 20-foot-high blocks. The block grades were estimated in two distinct methods, according to the report: a nearest neighbor estimation for any blocks pierced by drill holes and a kriged estimate for all other blocks. The kriging consisted of five separate runs constrained by rock type. The feasibility study reported a proven and probable ore reserve for both pits of 33.3 million tons at a total copper grade of 0.426%.

In October of 2005, Winters, Dorsey & Company, LLC prepared a Feasibility Study for Nord Resources Corporation. According to the report, the Feasibility Study used the same resource estimate as presented in the 2000 Feasibility Study. The study reported a proven and probable ore reserve for both pits of 35.1 million tons at a total copper grade of 0.393%.

Another feasibility study was prepared in September of 2007 by Bikerman Engineering & Technology Associates, Inc. (“BETA”) for Nord Resources Corporation. The report indicated that the resources were summarized from the study and estimation by the Winters Company in March of 2000. BETA used the mineral resource estimation to report proven and probable mineral reserve estimates based on total copper assays and recoveries for the Burro and Copper Chief deposits. From the historical data available, BETA concluded the reserve estimates to be conceptual in nature in that the reported ore reserves would reflect an over-estimation based on the use of total copper assays rather than acid soluble copper assays. The total of proven and probable reserves based on the methodology and approach of BETA was 73.4 million tons of ore with a total copper grade of 0.335%. Also noted by [Bikerman et al., 2007] was historical estimates for the JCM Burro copper oxide deposit reported by Cyprus, as a 22-million-ton mineral reserve with a total copper grade of 0.85%. The mineral reserve was defined from a drilling program that led to open pit mining in 1975.

In 2010, Mincom, Inc generated a resource estimate for Nord Resources. According to the associated report, the new estimate was intended to address the problem of calculated acid soluble copper assays influencing previous estimates. It is unclear to the author if the estimate was ever released into the public domain. The existing internal technical report for the estimate provides several tabulations of resources with various estimation methods and sources of data.

In 2011, Curtis Associates updated the mineral reserves previously reported by BETA in 2007. The new approach for defining the mineral reserves was based on acid soluble copper instead of total copper. This method generated a proven and probable reserve of 111.3 million tons at a total copper grade of 0.29%. The report does not provide the acid soluble grade in the mineral reserve. The report does mention that the new reserve estimate contained 17% less recoverable acid soluble copper pounds than what was originally reported by BETA in 2007.

These historical estimates are relevant only for historical completeness and are not considered reliable. A qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. Excelsior is not treating these historical estimates as current mineral resources or mineral reserves and these estimates should not be relied on.

24.6.4 Cochise District Past Production

Production from the Johnson Camp Mine, as summarized by Curtis Associates (2011) for Cyprus, Arimetco, and Nord, is given below in Table 24-10 through Table 24-12, respectively. Note: the production tables use “CuAs” to describe acid soluble copper. This description is equivalent to “ASCu” which is used elsewhere in this report.

Table 24-10: Cyprus Production at Johnson Camp by Year

(modified from Curtis Associates, 2011)

Year	Ore to Pad	Contained CuAs (%)	Contained CuAs	Lbs of Cu Shipped
1975	2,132,260	0.496	21,152,019	6,143,024
1976	1,821,476	0.357	13,005,339	10,059,807
1977	1,563,030	0.399	12,472,979	10,327,424
1978	1,202,500	0.426	10,245,300	10,205,142
1979	1,588,400	0.522	16,582,896	10,032,003
1980	1,499,600	0.411	12,326,712	10,320,407
1981	1,551,500	0.470	14,584,100	10,693,485
1982	1,894,700	0.322	12,201,868	9,702,272
1983	1,962,600	0.504	19,783,008	9,717,616
1984	52,100	0.713	742,946	8,803,361
1985	0	0	0	6,200,836
1986	0	0	0	4,854,796
Sub	15,268,166	0.436	133,097,167	107,060,173

Table 24-11: Arimetco Production at Johnson Camp by Year

(modified from Curtis Associates, 2011)

Year	Ore to Pad	Contained CuT (%)	Contained CuT	Lbs of Cu Shipped
1991	750,100	0.340	5,100,680	5,549,725
1992	2,516,320	0.480	24,156,672	8,156,435
1993	3,259,320	0.340	22,163,376	7,386,504
1994	2,719,690	0.290	15,774,202	5,618,012
1995	2,995,592	0.290	17,374,434	6,345,518
1996	3,084,254	0.350	21,589,778	9,921,576
1997	1,254,971	0.379	9,286,785	4,747,995
1998	0	0	0	2,181,304
Sub	16,580,247	0.348	115,445,927	49,907,069

Table 24-12: Nord Production at Johnson Camp by Year
(modified from Curtis Associates, 2011)

Year	Tons of Ore to Pad	Contained CuAs (%)	Contained CuAs	Lbs of Cu Shipped	Estimated Accum. CuAs Recovery % - New Ore Incl. Inventory
1999	0	---	---	672,004	---
2000	0	---	---	1,632,245	---
2001	0	---	---	1,133,914	---
2002	0	---	---	495,494	---
2003	0	---	---	556,388	---
2004-2007	0	---	---	0	---
2008	0	---	---	2,436,588	---
2009	4,553,275*	0.154	14,000,000	8,407,421	75%
2010	2,344,762*	0.160	7,500,000	9,338,000	84%
2011 (6 mths)	0	---	---	2,083,196	90%
Sub	6,898,037	0.157 avg.		26,755,250	
	*Ore Crushed to -1 inch				

The Johnson Camp Mine lies within the Cochise mining district. Production by mine for the entire district is summarized in Table 24-13 with data from [Cooper et al., 1964], [Curtis Associates, 2011], and [Zimmerman, 2016] for the years 1902-2010.

Table 24-13: Historical Copper and Zinc Production, Cochise Mining District

Operation Name	Production Period	ktons of Ore	Commodity
Johnson Camp Mine	1975-2010	39,000	Copper
Moore Mine	1951-1954	250	Copper, Zinc
Republic/Mammoth Mine	1882-1952	550	Copper, Zinc
Copper Chief Mine	1905-1919	24.1	Copper, Silver
Peabody Mine	1907-1918	14.2	Copper, Silver
Black Prince Mine	1902-1918	1.4	Copper, Silver
Keystone Mine	1916-1937	1.8	Copper
Centurion Mine	1908-1944	1.5	Copper, Silver, Gold
Texas Arizona Mine	1910-1928	0.7	Copper, Lead, Silver, Gold
Total	1902-2010	39,844	

Note: data for 1902 through 2010 compiled from Cooper and Silver (1964), Curtis Associates (2011) and Zimmerman (2016).

In addition to the operations listed in Table 24-13, several small-scale production operations with poorly preserved production records existed in the district in the late 1800s to early 1900s. This included tungsten production from vein systems in the Texas Canyon Quartz Monzonite (Cooper et al., 1964).

24.7 GEOLOGICAL SETTING AND MINERALIZATION

The information presented in this section of the report is derived from multiple sources, as cited. Mr. Bickel has reviewed this information and believes this summary accurately represents the Johnson Camp Mine area geology and mineralization as it is presently understood.

24.7.1 Regional Geologic Setting

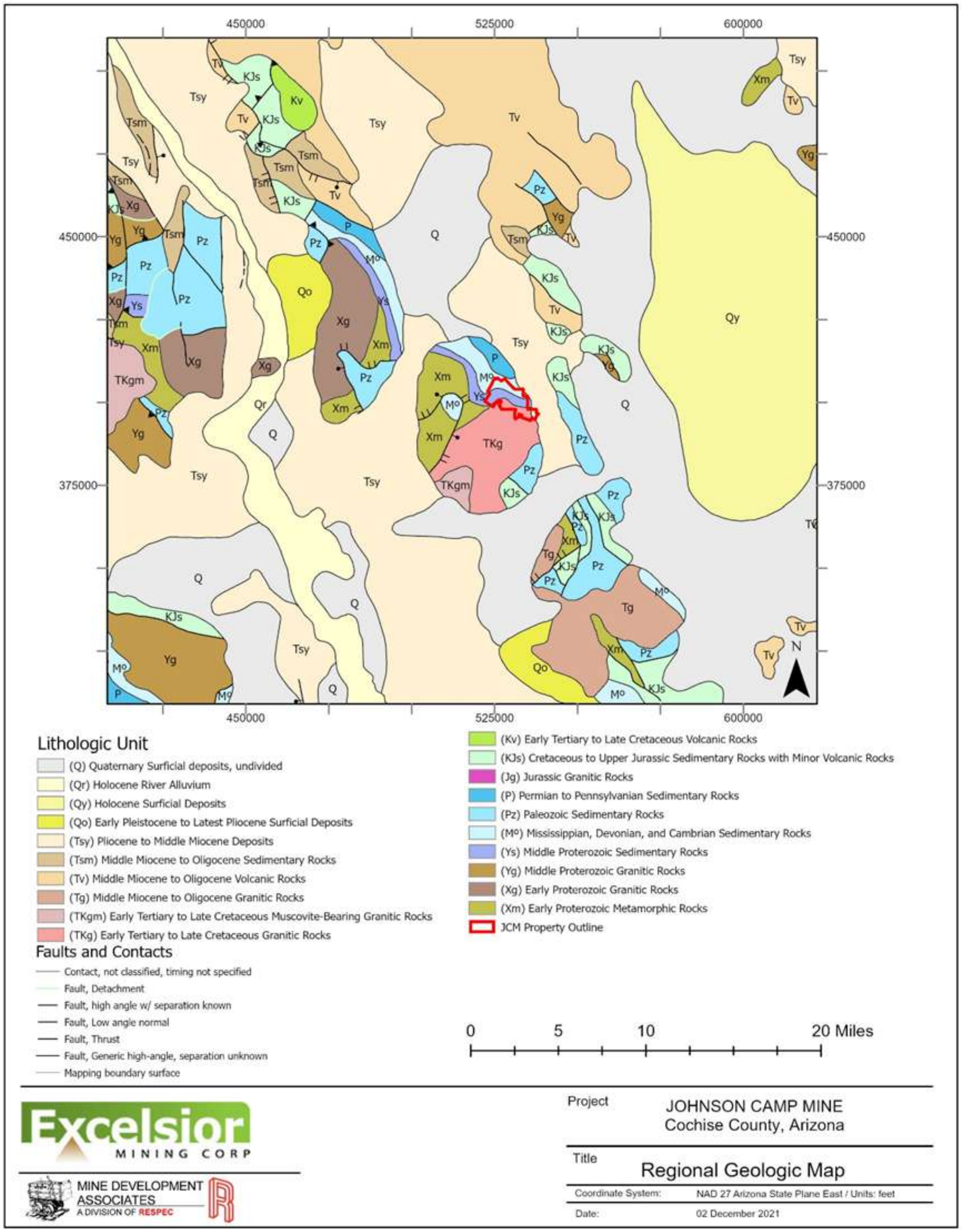
The Johnson Camp Mine is located within the Mexican Highland region of the Basin and Range province. The region is characterized by fault-bounded mountain ranges, typically with large intrusions forming the cores of the ranges. The ranges are separated by extensional grabens containing thick sequences of Tertiary and Quaternary volcanic and alluvial deposits that overlie a basement of Precambrian through Mesozoic rocks.

The project lies on the eastern edge of the Little Dragoon Mountains shown in Figure 24-5 within the Cochise mining district. The Little Dragoon Mountains are an isolated, fault bounded horst block comprised of rocks spanning from 1.4 billion years ago (Ga) Pinal Group schists to Holocene sediments. The southern portion of the Little Dragoon Mountains consists predominately of the Texas Canyon Quartz Monzonite of Tertiary age, whereas the Pinal Group schists and a sequence of Paleozoic sedimentary units dominate the northern half of the range.

The oldest rocks in the area, the Pinal Group schists, are composed of sandstones, shales and volcanic flows that have been metamorphosed to greenschist and amphibolite facies. The Precambrian Apache Group unconformably overlies the Pinal Group schists and is composed of conglomerates, shales and quartzite that were subsequently intruded by diabase sills. The Apache Group is then unconformably overlain by Paleozoic rocks that host most of the mineralization in the district. At Johnson Camp, the important Paleozoic hosts is the Cambrian Abrigo Formation.

The Texas Canyon Quartz Monzonite is porphyritic with large potassium feldspar phenocrysts from 1 to 10 cm in length. Livingston et. al. [1967] determined the age to be 50.3 ± 2.5 Ma (not recalculated to current decay constants). Reynolds et. al. [1986] listed eight determinations ranging from 49.5 to 55.0 Ma. The intrusion crops out to the southwest of the Burro Pit at the Johnson Camp Mine.

Several deformations have occurred in the area with the most recent being the latest Cretaceous-Paleocene Laramide Orogeny compression, followed by Miocene and younger Basin and Range extension that has modified the topography to its current appearance. Proterozoic, pre-Apache Group deformation of the Pinal Schist Group included isoclinal folding with steep to overturned fold axes with a general northeastern structural trend. Minor deformations took place in late Precambrian and post-Paleozoic but pre-Cretaceous times. The post Paleozoic-pre-Cretaceous deformation is characterized by steep northeast to easterly striking faults with displacements up to hundreds of feet.



(modified from Richard et al., 2000)

Figure 24-5: Regional Geology Little Dragoon Mountains

The Laramide deformation produced structures striking in a northwesterly direction and was more or less perpendicular to the Pre-Apache Group deformation. Pre-late Cretaceous faults were reactivated and modified, and folds and thrust faults are common features of the Laramide.

Two episodes of block faulting have created the Basin and Range topography that dominates the current landscape and postdates the mineralization in the region.

24.7.2 Property and Deposit Geology

The Johnson Camp property's geology is characterized by a package of upper Precambrian and lower Paleozoic rocks striking to the northwest and dipping moderately to the northeast. Copper mineralization is hosted throughout the geologic section although it is most abundant in the Cambrian Abrigo formation, which is sub-divided into upper, middle, and lower units at the property. The Abrigo formation is underlain by the Cambrian Bolsa quartzite and finally the Apache Group rocks of the upper Precambrian, specifically the Precambrian diabase and Pioneer Shale which are underlain by Precambrian Pinal Schist. A description of relevant geological units is given in Table 24-14. Note that the Bolsa quartzite at the Johnson Camp property is inappropriately applied to the formation directly below it, the Dripping Springs quartzite, which is part of the Precambrian Apache Group. There is no angular discordance between the two formations, however the basal unit of the Dripping Springs quartzite, the Barnes Conglomerate, can be easily identified in the open pits at Johnson Camp. Despite some subtleties, the two units are largely indistinguishable and historical operators at Johnson Camp lumped them together. For the purposes of this report, the "Bolsa quartzite" will refer also to the Dripping Springs quartzite and Barnes Conglomerate.

Table 24-14: Geologic Descriptions of Relevant Johnson Camp Mine Formations

Rock or Formation	Age	Approximate Thickness	Geologic Description
Upper Abrigo formation	Upper Cambrian	150 feet	Thin sandy dolomite beds with minor quartzite altered to white and green calc-silicate hornfels
Middle Abrigo formation	Upper Cambrian	300 feet	Thin bedded crenulated limestone with minor shale altered to distinct brown garnet-rich skarn
Lower Abrigo formation	Upper Cambrian	250 feet	Shale with interbedded limestone and dolomite altered to a dark grey/black calc-silicate hornfels
Bolsa Quartzite/ Dripping Springs Quartzite	Middle Cambrian/ Upper Precambrian	200 feet	Quartzite with minor shale beds altered to calc-silicate hornfels. Distinct red-brown color A 10-foot-thick conglomerate (Barnes Conglomerate) marks the base of the unit
Diabase	Upper Precambrian	30 feet	Thin sills of metadiabase intruding the Pioneer Shale. Two distinct sills (upper and lower) and defined in the Johnson Camp area
Pioneer Shale (Apache Group)	Upper Precambrian	150 feet	Shale with quartzite interbeds. Distinct purple-maroon and white color and reduction marks throughout with cubic pyrite.

The stratigraphic package is rotated generally thirty degrees to the northeast. Figure 24-6 shows a cross section of the Johnson Camp Mine geology through the Burro Pit area.

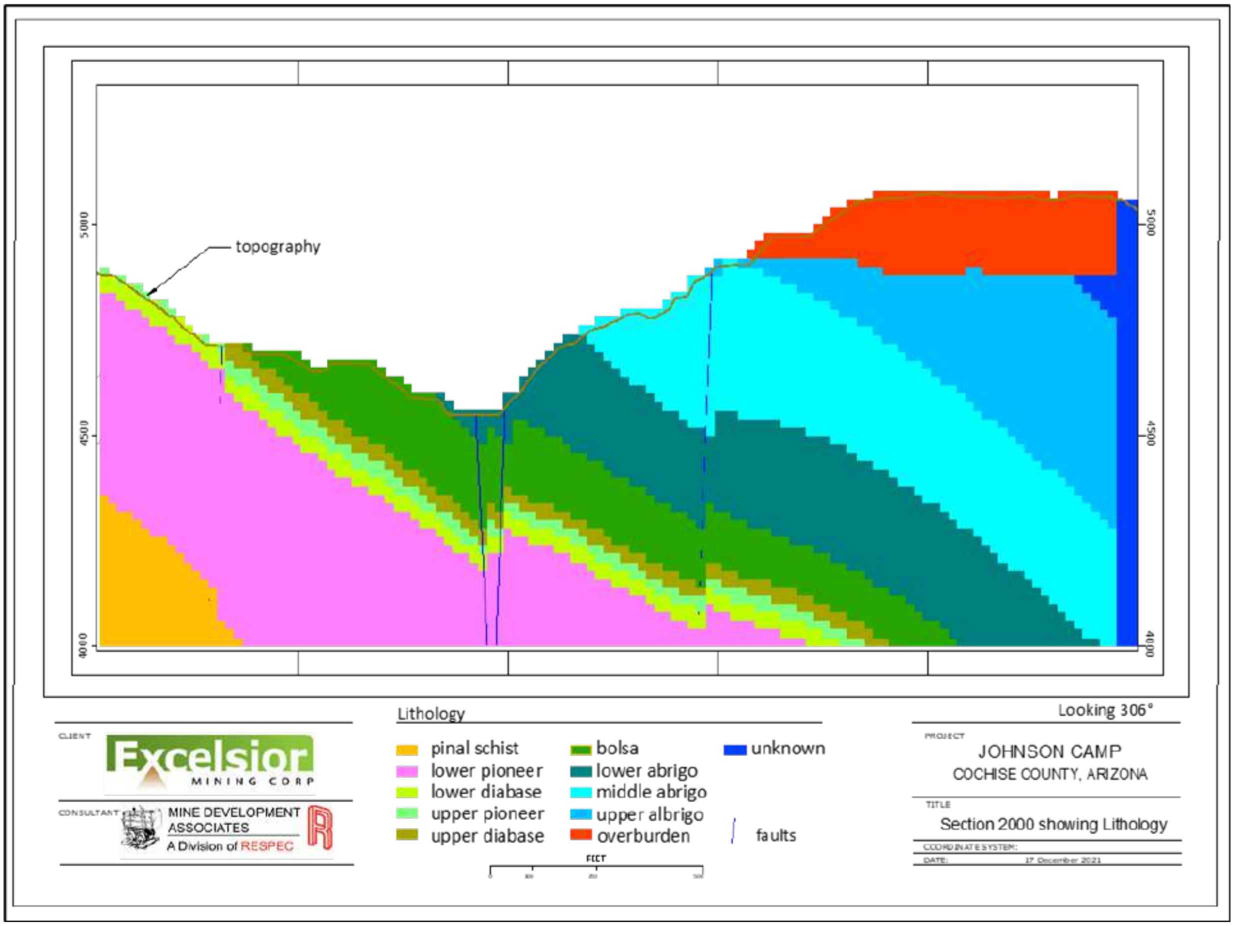
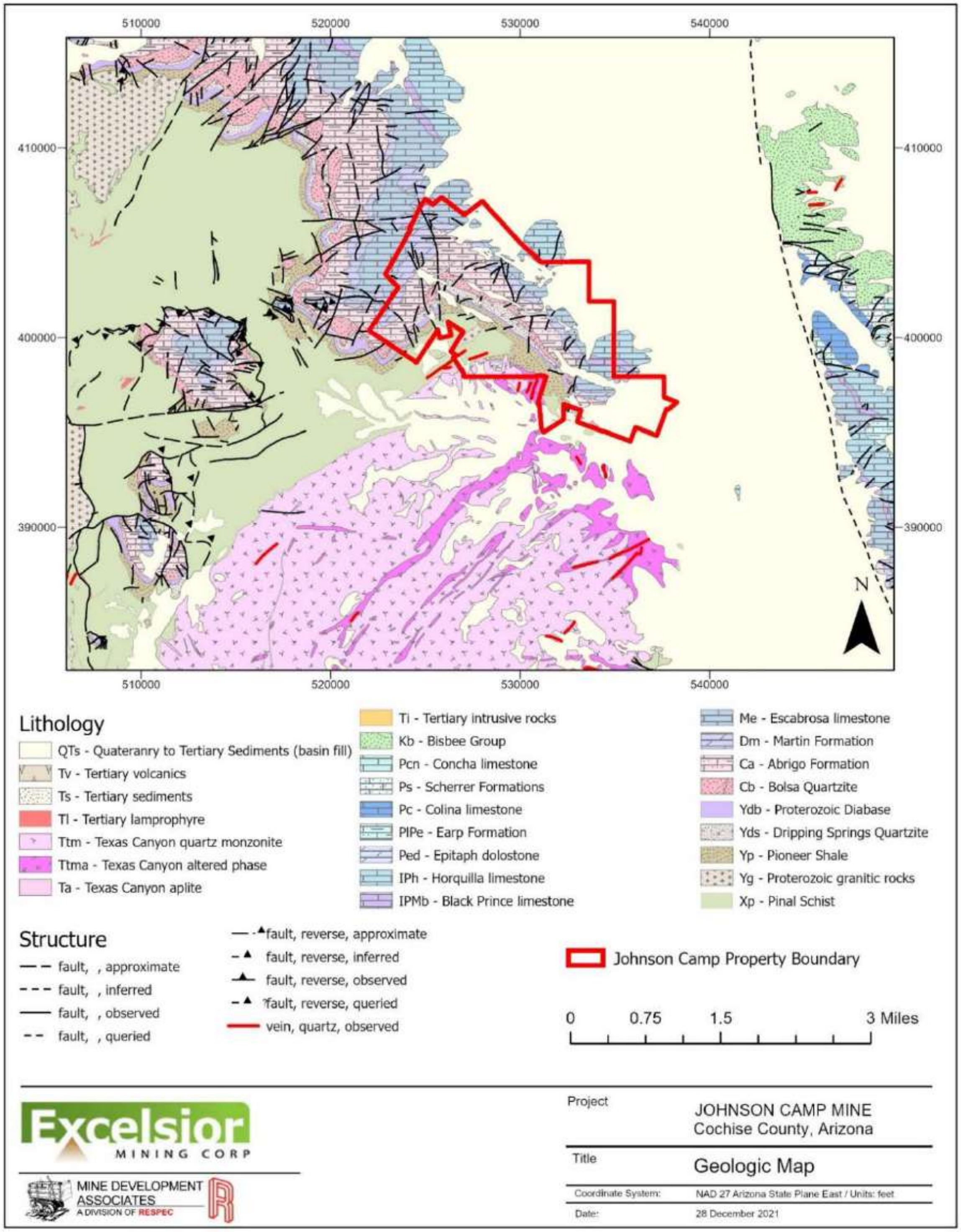


Figure 24-6: Cross Section Through the Burro Pit at the Johnson Camp Mine

To the southwest of the property, the surface geology is dominated by outcrops of Precambrian Pinal schist and the Tertiary Texas Canyon quartz monzonite stock, as mapped by Cooper. To the northeast of the property, the Devonian Martin formation and Mississippian Escabrosa limestone overlie the Abrigo formation. The Texas Canyon quartz monzonite stock is thought to be the source of metallization in the district and at Johnson Camp. Roughly 750 feet to the southwest of the Burro Pit, the "altered phase" of the stock (as mapped by Cooper) crops out in a road cut on the south end of the old leach pad and the unit continues to the south and west of the property. The altered phase in this area and generally district-wide contains cm-scale quartz or quartz-orthoclase veins with coarse muscovite halos. These vein sets can be followed in a south-southwesterly direction from the outcrop where weak mineralization can be observed in and around the veins. No quartz monzonite is recognized in the Burro or Copper Chief pit nor recognized in any drilling at Johnson Camp. However, similar styles of veining observed in the stock can be identified at the Johnson Camp property.

According to historical interpretations, three sets of faults are recognized at Johnson Camp: north-northeast striking faults with apparent right-lateral displacement and steep dips usually to the southeast, east-northeast striking faults with apparent right-lateral displacement and moderate to steep dips to the south-southeast, and northwest-striking faults with apparent reverse or normal displacement and steep dips variably to either the northeast or southwest. It is likely that the northwest-striking set covers two separate tectonic events but is grouped together for the sake of this report. In all sets of faults, displacements are relatively minor (usually less than 200 feet of vertical or lateral displacement). Immediately to the south of the property, the Keystone fault, which is another north-northeast-striking fault with right-lateral displacement, shows significant throw juxtaposing the Mississippian section against the

Cambrian. None of the faults in the Johnson Camp area, however, show displacement close to the magnitude of the Keystone Fault.



(from Excelsior, December 2021)

Figure 24-7: Property Geologic Setting for the Johnson Camp Mine

24.7.3 Alteration

Moderate to intense calc-silicate alteration of the stratigraphic units at Johnson Camp is widespread throughout the property, especially in the Cambrian Abrigo formation. Calc-silicate minerals including garnet, epidote, and diopside are common in various assemblages which are primarily controlled by protolith mineralogy and secondarily controlled by proximity to structures. The Middle and Lower sub-units of the Abrigo formation contain the most intense calc-silicate alteration in the area. In the Middle Abrigo Formation, brown-tan garnet and epidote alteration is consistent and diagnostic of the unit. In the Lower Abrigo Formation, the rock is dominantly altered to dark grey to black hornfels with interbedded sub-units containing diopside-rich and/or garnet-epidote lenses similar to alteration in the Middle Abrigo Formation. Pervasive quartz veining occurs in both the Abrigo Formation and underlying Bolsa Quartzite throughout the Johnson Camp Mine area. Quartz vein orientations are typically sub-parallel to the stratigraphic units. Distinctive vein halos containing coarse muscovite can be observed in certain areas of the property, especially in the Bolsa Quartzite at the Copper Chief Pit.

24.7.4 Mineralization

Primary copper mineralization at Johnson Camp is dominantly found along bedding planes or in veins and replacements as chalcopyrite along with quartz and pyrite, closely associated with skarn and calc-silicate alteration in the rock. The presence of copper mineralization at JCM is generally within the Bolsa Quartzite, Diabase Units, Lower and Middle Abrigo Formations. The deposit has been oxidized in a general profile that decreases with depth although structural features such as faults and stratigraphic contacts have a strong influence in the oxidation of the primary mineralization. Oxide copper consists primarily of chrysocolla, malachite, copper limonite, and manganiferous wad. Supergene chalcocite and occasional native copper occur generally below the oxidized zone. Further below the supergene zone, the mineralization transitions to primary sulfides despite local zones of supergene mineralization. North-northwest and north-northeast fault sets appear to have had some influence on mineralizing fluids although the structural zones themselves are not significant in terms of bulk mineralization. Locally in the diabase sills and Bolsa quartzite, copper has been observed as exotic accumulations on fractures, presumably derived from dissolution of copper in the immediately overlying lower Abrigo Formation. This is especially apparent in the Copper Chief pit, where mobilized and precipitated copper oxides can be easily observed on joint planes and fractures (Curtis Associates, 2011).

24.8 DEPOSIT TYPES

The Johnson Camp Mine copper deposit is a sub-type of or related to a classic copper skarn (Einaudi et al.,1982) and (Meinert et al., 2005). Skarn deposits range in size from a few million to 500 million tonnes and are globally significant, particularly in the southwestern US. They can be stand-alone copper skarns, which are generally small, or can be spatially and temporally closely associated with porphyry copper deposits, in which case they tend to be very large. The skarn at Johnson Camp and collectively in the Cochise mining district is presumably related to the Texas Canyon Quartz Monzonite, despite the intrusive itself hosting very little-known economic mineralization. Mineralization in the quartz monzonite would require more specialized conditions involving the metal and volatile content of the magma, depth of emplacement, or other factors (Burt, 1977).

Copper skarns generally form in calcareous shales, dolomites, and limestones peripheral or adjacent to the margins of diorite to granite intrusions that range from dikes and sills to large stocks or phases of batholithic intrusions, and frequently are associated with mineralized intrusions. Copper mineralizing hydrothermal fluids are focused along structurally complex and fractured rocks and convert the calcareous shales and limestones to andradite-rich garnet assemblages near the intrusive body, and to pyroxene and wollastonite rich assemblages at areas more distal to the intrusive. Retrograde evolution of the hydrothermal fluids produces actinolite-tremolite-talc-quartz-epidote-chlorite assemblages that overprint earlier garnet and pyroxene. Mineralization at Johnson Camp occurs approximately 500 ft northeast of known occurrences of the Texas Canyon Quartz Monzonite intrusion in the Cochise mining district, which is thought to be the source of mineralizing hydrothermal fluids. Therefore, Johnson Camp can be sub-categorized as proximal skarn related to a porphyry copper system. This assumption is supported by the high abundance of garnet-epidote alteration in the mineralized zones, and the characterization of the deposits in numerous historical publications. The anatomy of a telescoped porphyry copper system model illustrated in Figure 24-8 by (Sillitoe, 2010) can be used as a conceptual model to understand the spatial relationship of proximal skarns in the district.

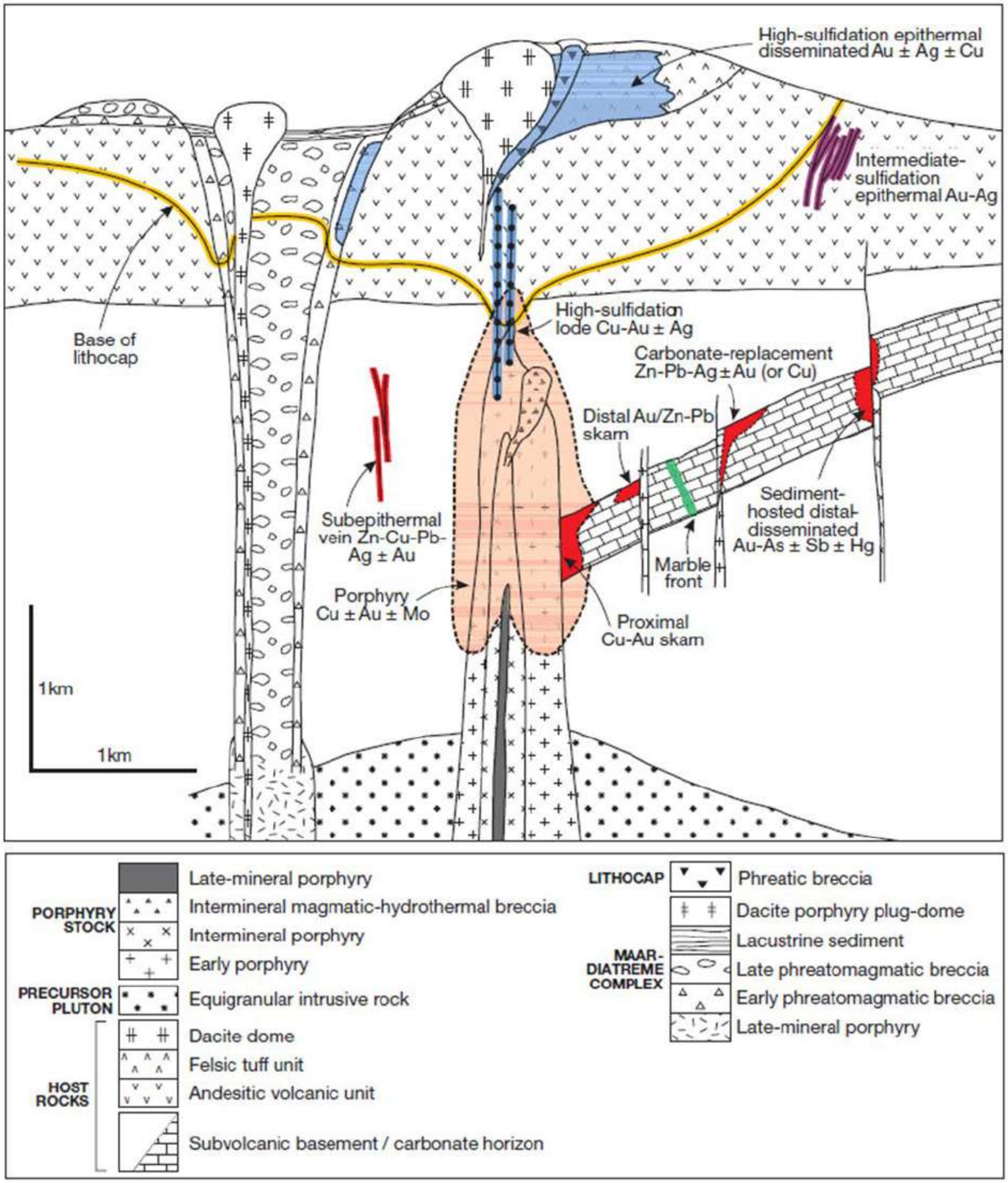


Figure 24-8: Schematic Model

24.8.1 Discussion of Resources and Recommendations

Future drilling, exploration, and resource definition at Johnson Camp should focus on increasing the understanding of the distribution of soluble copper mineralization. Though Johnson Camp has a long history of drilling, exploration, and mining, collection of soluble copper assay data is not consistent throughout the property history and therefore the current understanding of soluble copper mineralization could be improved. Infill drilling in key areas to increase drill density, and drill-testing of the unconstrained limits of the deposit, particularly down-dip from known mineralization, are also notable areas of focus for future development of the property. The author recommends collection of more structural data for the purposes of bolstering current geological understanding of the deposit and mineralization controls. Drilling angle holes to test structures is recommended for this purpose.

As the date of this report, Mr. Bickel is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the Johnson Camp mineral resources and that are not otherwise discussed in this report. The impact of taxation was taken into consideration when establishing cut-off grade and further details are provided in Section 24.22: Economic Analysis.

24.9 EXPLORATION

This section summarizes the exploration work carried out at the Johnson Camp Mine. Mr. Bickel has reviewed the information and believes it is an accurate representation.

24.9.1 Historical Exploration

Exploration in the JCM area has taken place since the late 1800s and during that time, mining at Johnson Camp was considered archaic (Curtis Associates, 2011). Open pit mining commenced in 1975 by Cyprus and replaced the underground mining operations following the completion of an exploratory drilling program that defined the reserve of the Burro deposit. Cyprus and Arimetco collectively drilled 254 holes within both the Burro and Copper Chief pits. In 1999, Nord focused drilling exploration efforts on prospective targets outside of the pits such as the North and Keystone-Walnut areas. As a result of the four-phase exploration drilling program, 43 holes were drilled in the North area and 17 in the Keystone-Walnut area. Of the 60 drillholes, it was determined no copper mineralization could be classified as reserves (Curtis Associates, 2011). Geological mapping was conducted by Nord in 2005 throughout the Burro and Copper Chief pit areas to identify and update existing geological maps. In 2008, Nord completed 25 drillholes that were placed at the extents of the Burro and Copper Chief pits to further delineate the resource. The drillholes confirmed mineralization to the north and south of the respective pits. An additional 6 drillholes were completed in 2010 by Nord that confirmed the geological and mineralogical continuity between the Burro and Copper Chief pits. The exploration programs carried out by Nord further defined the copper resources at the Burro and Copper Chief pits, along with indicating potential target areas for future development.

24.9.2 Excelsior Exploration

From 2016-2017, Excelsior catalogued and evaluated all data, drill core, pulp, and coarse reject material in the core shed inherited from Nord and subsequently commenced a re-logging and re-sampling program focused on soluble copper mineralization and assays.

In 2018, RESPEC began evaluating data provided by Excelsior for the purposes of building a new database to eventually create a new mineral resource estimate. These activities included:

- Review of historical documentation prepared by Excelsior that discuss the overall project geology, details on historical soluble-copper analyses, and historical modeling of total copper and soluble copper.
- Review of the Nord “10a” total copper and soluble copper resource modeling methodologies.

- Detailed statistical analysis of historical data vs Excelsior results from 2016 and 2017 resampling of historical drill core and re-analyses of historical pulps.
- Implementation of the transformation of certain project data from original mine-grid coordinates (JCM Grid) into NAD1927 State Plane Arizona East FIPS coordinates using a 2-point rotation determined in 2016 by Darling Geomatics.
- The re-creation of a version of the Nord “10a” block model in the new project coordinates.
- Creation of project topography by contouring DEM, topography, and USGS 10 m NED data.
- Creation of a RESPEC project drill-hole database.

In 2022, Excelsior completed a drilling program at Johnson Camp in which 44 drill-holes were completed totaling 15,313 feet.

24.10 DRILLING

All of the drilling summarized in this section was conducted by historical operators from the 1960s through 2010. Excelsior completed a drilling program in 2022 at the property focused on the Burro pit area.

This section summarizes the historical drilling and the information presented in this section of the report is derived from multiple sources, as cited. The author has reviewed this information and believe this summary accurately represents drilling done at the Johnson Camp Mine.

24.10.1 Summary

The Johnson Camp Mine database contains 357 drill holes total 121,536 feet of drilling. Several drilling campaigns and operators span the contents of the database. Based on RESPEC’s current knowledge, historical operators of the campaigns include Cyprus Mining (187 drill holes), Arimetco (83 drill holes), Nord (31 drill holes), Sumitomo (12 drill holes), and 16 drill holes were completed by an operator unknown to RESPEC. 44 drill holes were drilled by Excelsior in 2022. Drilling is concentrated in and immediately around the historically producing open pits. Figure 24-9 below shows the collar locations for the drill holes in the database and Table 24-15 is a breakdown of the drilling and operators in the Johnson Camp Mine area.

Table 24-15: Summary of Johnson Camp Drilling

Operator	Year	Holes	Feet
Cyprus Mining	1960 – 1986	187	61,417
Arimetco	1989 - 1997	83	24,638
Summo USA Corp.	1998	12	5,800
Nord Resources Corp.	2008-2010	31	14,368
Excelsior	2022	44	15,313
	Totals	357	121,536

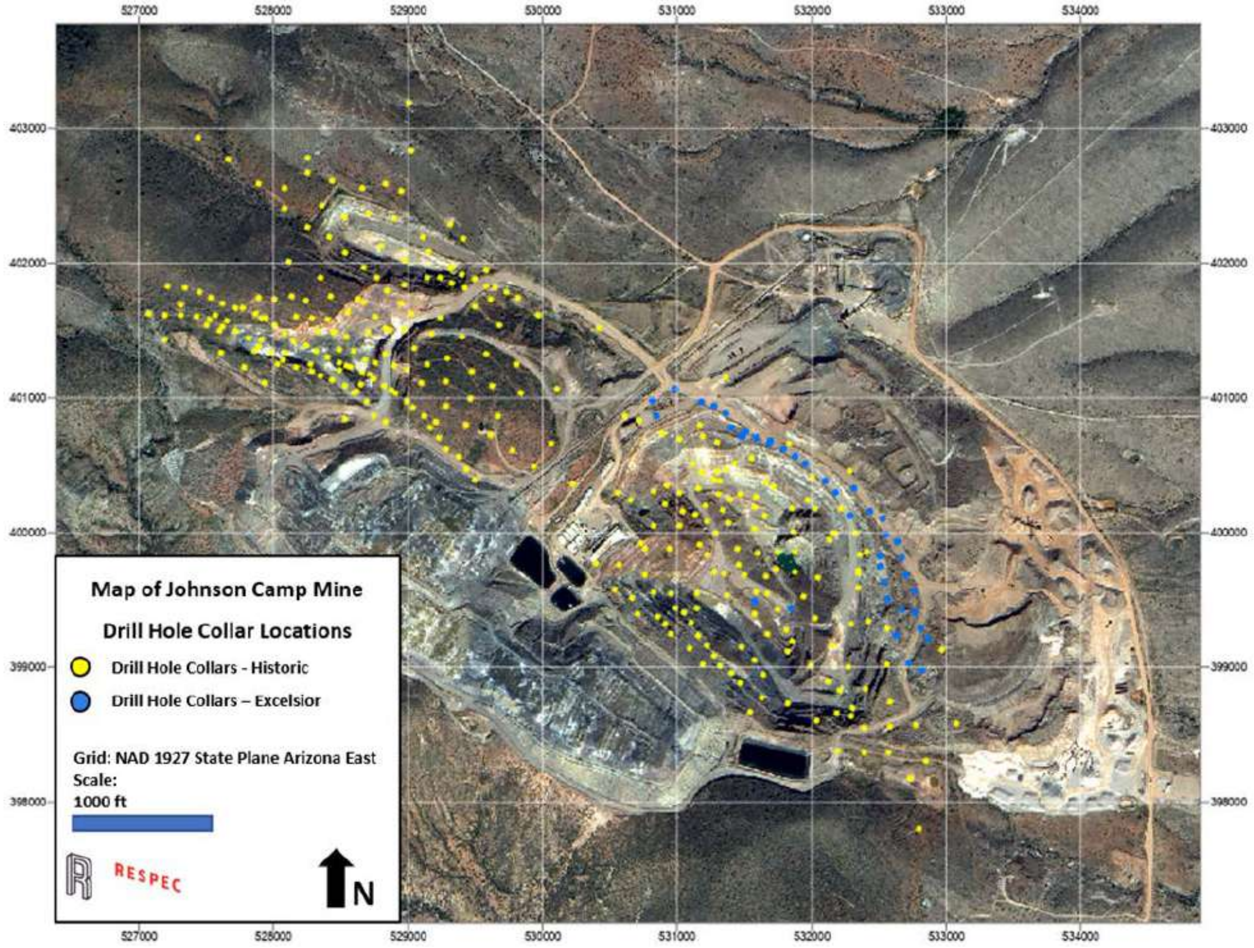


Figure 24-9: Map of Johnson Camp Drill Holes

24.10.2 1960-1986 Historical Drilling by Cyprus Mining

Cyprus Mining ("Cyprus") drilled a total of 61,417 feet in 187 holes, of which 10 were drilled vertically and 3 were angle holes. These holes were drilled generally in the period of 1960-1986, although exact dates of each hole are not known to RESPEC. The drilling was done on approximately 100-foot centers [Curtis Associates, 2013]. Cyprus drilled NQ size (1.8-inch core diameter) holes. No information is available regarding the drill contractor(s), rig type(s) or methods and procedures for collar and down-hole surveys, if any were conducted. The current drill hole database does not include downhole surveys for the holes.

24.10.3 1989-1997 Historical Drilling by Arimetco

Arimetco drilled a total of 24,638 feet in 83 holes, of which 180 were drilled vertically and 7 were angle holes. These holes were drilled generally in the mid-1990s [Curtis Associates, 2013]. Exact drilling dates of each hole are not known to RESPEC. Arimetco drilled primarily reverse circulation holes with some core drilling [Bikerman et al., 2007]. No information is available regarding the drill contractor(s), rig type(s) or methods and procedures for collar and down-hole surveys, if any were conducted. The current drill hole database does not include downhole surveys for the holes.

24.10.4 1998 Historical Drilling by Summo USA Corp.

Summo USA Corp. ("Summo") drilled a total of 5,800 feet in 12 holes, of which 7 were drilled vertically and 5 were angle holes. These holes were drilled in 1998. Summo drilled them all by reverse circulation methods. No information is available regarding the drill contractor(s), rig type(s) or methods and procedures for collar and down-hole surveys, if any were conducted. The current drill hole database does not include downhole surveys for the holes.

24.10.5 2008-2010 Historical Drilling By Nord Resources Corp.

Nord Resources Corp. ("Nord") drilled a total of 14,368 feet in 31 holes, of which 27 were drilled vertically and 4 were angle holes. Twenty-five of these holes were drilled in 2008 and the remainder were drilled in 2010. Nord drilled by reverse circulation methods. No information is available regarding the drill contractor(s), rig type(s) or methods and procedures for collar and down-hole surveys, if any were conducted. The current drill hole database does not include downhole surveys for the holes.

24.10.6 2022 Drilling by Excelsior Mining Corp.

Excelsior completed an infill and metallurgical core drilling program in the Burro pit area. 36 HQ core size (2.5-inch core diameter) infill holes and 8 PQ core size (3.3-inch core diameter) metallurgical holes were completed. The drill contractor for the program was Godbe Drilling. Downhole surveys were conducted on all but 12 core holes of which two of the holes were redrilled with the original hole lacking a survey.

24.10.7 Summary Statement

Mr. Bickel believes that the drilling sampling procedures provided samples that are representative and of sufficient quality for use in the resource estimations discussed in Section 14. The author is unaware of any sampling or recovery factors that materially impact the mineral resources discussed in Section 14.

There is a general lack of down-hole deviation survey data for the historical holes in the Johnson Camp Mine area. While the paucity of such data is not unusual for drilling done prior to the 1990s, the lack of deviation data contributes a level of uncertainty as to the exact locations of drill samples at depth. However, these uncertainties are mitigated to a significant extent by the vertical orientation of nearly all drill holes, and the open-pit nature of any potential future mining operation that is based in part on data derived from the historical holes.

24.11 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section summarizes all information known to Mr. Bickel relating to sample preparation, analysis, and security, as well as quality assurance/quality control procedures and results, that pertain to the Johnson Camp Mine Project. The information has either been compiled by Mr. Bickel from historical records as cited or provided by Excelsior.

24.11.1 Historical Sample Preparation and Analysis

Mr. Bickel is unaware of any information on the methods and procedures used by Nord for the preparation of their drilling samples. Incomplete records indicate that all samples were analyzed for copper and some were analyzed for soluble copper, but the analytical methods are not known.

According to Bikerman et al. (1974), all Cyprus drilling was NQ size, and general procedures involved delivering the core from the rig to a core facility on site where it was marked and split by geologists. Core splitting was completed on a concrete pad to avoid contamination, and that the samples were shipped to a certified independent certified lab. The samples were subject to QA/QC standards, and Cyprus did perform check assays through multiple labs. Cyprus also used performed a QA/QC procedure of compositing sample pulps of a given intersection and comparing the composited assay to the original analyses (Bikerman et al., 2007). Based on copies of assay certificates provided by Excelsior, the majority of the samples submitted by Cyprus were assayed at Southwestern Assayers and Chemists ("SWAC") in Tucson, Arizona. The samples were analyzed for total copper and soluble copper. The author has no information on the analytical methods and procedures used. The author infers that SWAC was independent of Cyprus.

Arimetco drilling, which was primarily by reverse-circulation methods, was sampled using a sample cone and Jones splitter (Bikerman et al., 2007). Arimetco use certified labs to perform the analyses. No information is available on the methods and procedures used for sample preparation and analysis.

Samples from the Summo USA Corp drill-holes were analyzed by Actlabs-Skyline, another predecessor to what Skyline Assayers and Laboratories is now. Samples were analyzed for total copper and sequential analysis of acid-soluble and cyanide-soluble copper. No information is available on the methods and procedures used for sample preparation and analysis.

Nord samples from their 2008 and 2010 drilling campaigns were analyzed by Skyline Assayers and Laboratories. Samples were analyzed for total copper and sequential analysis of acid-soluble and cyanide-soluble copper. No information is available on the methods and procedures used for sample preparation and analysis.

24.11.2 Excelsior Re-Sampling Procedures

Following Excelsior's purchase of the Johnson Camp Mine, a detailed inventory of historical drill core and sample pulp from the existing storage site near at the mine was undertaken. The core and pulp material at the Johnson Camp core shed was found to be well-organized. However, the physical state of the core shed itself was in poor condition. The facility had been exposed on one side by a broken bay door, and some core boxes and pulp containers were dilapidated or destroyed by rodents. Excelsior salvaged what material remained in-tact and transported it to their core facility in Casa Grande. Drill core and pulp remaining from historical drilling was inspected and selected intervals were re-sampled by Excelsior in 2016 and 2017. The core was logged, photographed, and inspected by Excelsior staff. Samples were selected based on criteria developed by Excelsior for the purposes of data investigations. The criteria were limited by core and pulp availability. Samples existed as half core (originally split by historical operators). These samples were split to ¼ core. Pulps were transferred into new bags, shaken up, and a minimum of 20 grams was separated for re-sample. All core samples were mechanically split and placed in bags. Internal QA/QC samples (standards, blanks, and ¼ core duplicates) were inserted approximately every tenth sample in the sequence.

The Excelsior samples were prepared and analyzed at Skyline Laboratories ("Skyline") in Tucson, Arizona. Skyline is an independent commercial laboratory that holds ISO 9001:2015 and ISO/IEC 17025:2017 accreditations.

The samples were crushed to plus 75% passing -10 mesh, then split and pulverized with standard steel to plus 95% passing -150 mesh.

The analytical methods for the assays are as follows:

Total Cu (CuT) analyses: Samples are digested in a mixture of hydrochloric, nitric and perchloric acids. This solution is heated and taken to dryness. The contents are treated with concentrated hydrochloric acid and the solution is brought to a final volume of 200 mL with de-ionized water. This solution is read by Atomic Absorption using Standard Reference Materials made up in 5% hydrochloric acid.

Sequential Analysis of Acid-Soluble Cu (CuAs) and Cyanide-Soluble Cu (CuCN) analyses: Samples are digested in 5% sulfuric acid and supernatant solution is diluted to 100 mL with de-ionized water. The residue is digested in 10% sodium-cyanide solution and diluted to 100 mL. The CuAs samples are read on Atomic Absorption units using 0.5% H_2SO_4 calibration standards. The CNCu samples are read on Atomic Absorption units using 1% NaCN calibration standards.

24.11.3 Excelsior 2022 Sample Preparation and Analysis

In 2022, Excelsior completed an infill and metallurgical core drilling program. The core was logged, photographed, and inspected by Excelsior staff. Samples were selected based on a suggested 10-foot length with flexibility to sample on geologic contacts and mineralization boundaries. All core samples were mechanically split and placed in bags. Internal QA/QC samples (standards, blanks, and ¼ core duplicates) were inserted approximately every tenth sample in the sequence. Metallurgical holes were sampled by cutting an approximately 1/8 core slice down the core's long axis for each interval. Slice locations were chosen by the logging geologist to ensure representative mineralization from the core was selected for each slice.

The Excelsior samples were prepared and analyzed for CuAs and CuCN analyses (described in Section 24.11.2) at Skyline Laboratories ("Skyline") in Tucson, Arizona. Skyline is an independent commercial laboratory that holds ISO 9001:2015 and ISO/IEC 17025:2017 accreditations

24.11.4 Sample Security

The authors have no information on the sample security methods and procedures used by historical operators. Drill core remaining from the historical drill campaigns has been stored at the Excelsior core facility in Casa Grande, AZ Excelsior's samples were selected and stored in bags at the Excelsior core facility. The bags were placed into large mobile bins and made available for direct pickup by Skyline labs. Upon pickup by Skyline, Chain of Custody sheets were filled out and signed by Excelsior and Skyline.

For the 2022 drilling program, drill core was stored at the Excelsior core facility in Casa Grande, AZ. Excelsior's samples were collected and stored in bags at the Excelsior core facility. The bags were placed into large mobile bins and made available for direct pickup by Skyline labs. Upon pickup by Skyline, Chain of Custody sheets were filled out and signed by Excelsior and Skyline.

24.11.5 Quality Assurance/Quality Control

24.11.5.1 Historical QA/QC Results

Little information is provided in the historical records pertaining to the results of historical QA/QC programs. According to Bikerman et al., (2007), a QA/QC procedure whereby Cyprus composited sample pulps and re-submitted the composite for assay as a comparison to the composite grade of original assays was practiced during exploration of the property. Some original assay certificates for this procedure are available, and those that exist compare assays from Southwestern Assayers and Chemists to Union. Others compare results from Hazen labs to an unknown source of assays.

24.11.5.2 Excelsior QA/QC Methods and Results

CRMs for resampling program. Excelsior purchased commercial certified reference materials ("CRMs") for use in the 2016-2017 re-sampling program. The CRMs were inserted into the re-sample stream and analyzed with the core samples for total copper. The results were used to evaluate the analytical accuracy and precision of the analyses in Excelsior's samples.

In the case of normally distributed data, 95% of the CRM analyses are expected to lie within the two standard-deviation limits of the certified value, while only 0.3% of the analyses are expected to lie outside of the three standard-deviation limits. Note, however, that most assay datasets from metal deposits are positively skewed. Samples outside of the three standard-deviation limits are typically considered to be failures. As it is statistically unlikely that two consecutive analyses of CRMs would lie between the two and three standard-deviation limits, such samples are also considered to be failures unless further investigations suggest otherwise. All potential failures should trigger investigation, possible laboratory notification of potential problems, and possible reanalysis of all samples included with the failed standard result.

Table 24-16 lists the CRMs used by Excelsior.

Table 24-16: Certified Reference Materials for 2016-2017 Assays

Reference Material	Certified Value (%Cu)	2 Std Dev (%Cu)	No of Skyline Analyses
AMIS 0249	0.37	0.01	147
AMIS 0370	0.70	0.05	16

The Skyline copper analyses of the Excelsior CRMs returned excellent results, with generally good precision and accuracy for both AMIS 0249, shown in Figure 24-10, and AMIS 0370, shown in Figure 24-11. Only one technical 'failure' occurred in standard analyses, shown on the chart for AMIS 0370. The value reported from the lab, 0.64, is slightly below the 'failure' threshold of 0.6418. Excelsior considered that the reporting precision from the lab is two decimal places, and that the value is likely within the threshold limits if reported with higher precision. Therefore, the sample was not considered an actual failure. The author considers this conclusion reasonable.

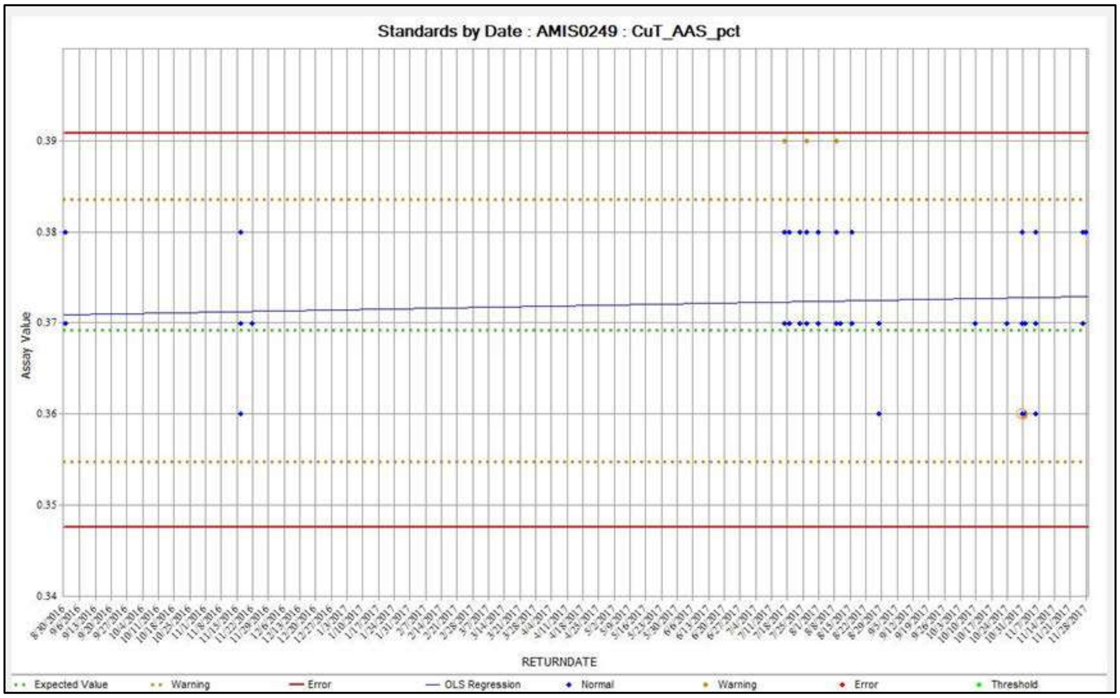


Figure 24-10: AMIS 0249 Total Copper Analyses

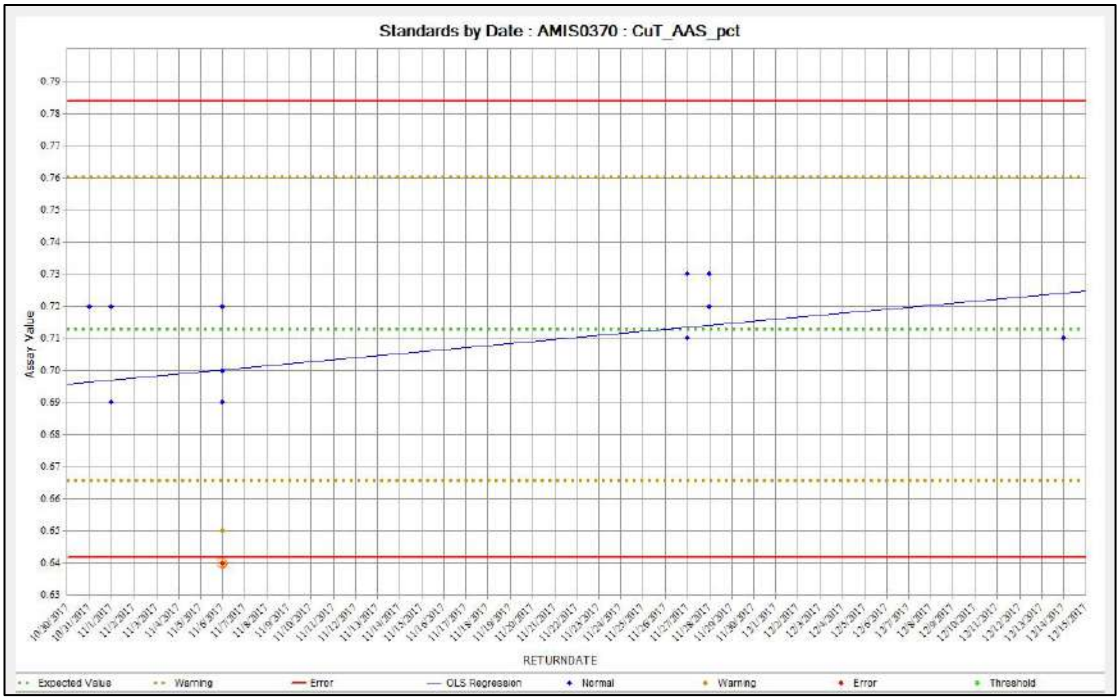


Figure 24-11: AMIS 0370 Total Copper Analyses

<u>Coarse Blanks for resampling program</u>. Coarse blanks are samples of barren material that are used to detect possible contamination in the laboratory, which is most common during sample preparation stages. In order for analyses of blanks to be meaningful, they must be sufficiently coarse to require the same crushing and pulverizing stages as the drill samples. It is also important for a significant number of the blanks to be placed in the sample stream within, or immediately following, a set of mineralized samples, which would be the source of most contamination issues. In practice, this is much easier to accomplish with core samples than RC.

Blank results that are greater than five times the lower detection limit of the relevant analyses are typically considered failures that require further investigation and possible re-assaying of associated drill samples. The detection limit of the Skyline analyses was 0.01% for total copper. Blank samples assaying in excess of five times these detection limits (0.005%) are considered to be failures. A chart of the Skyline analyses of the coarse blanks is shown in Figure 24-12. There were no coarse blank failures among the samples analyzed.

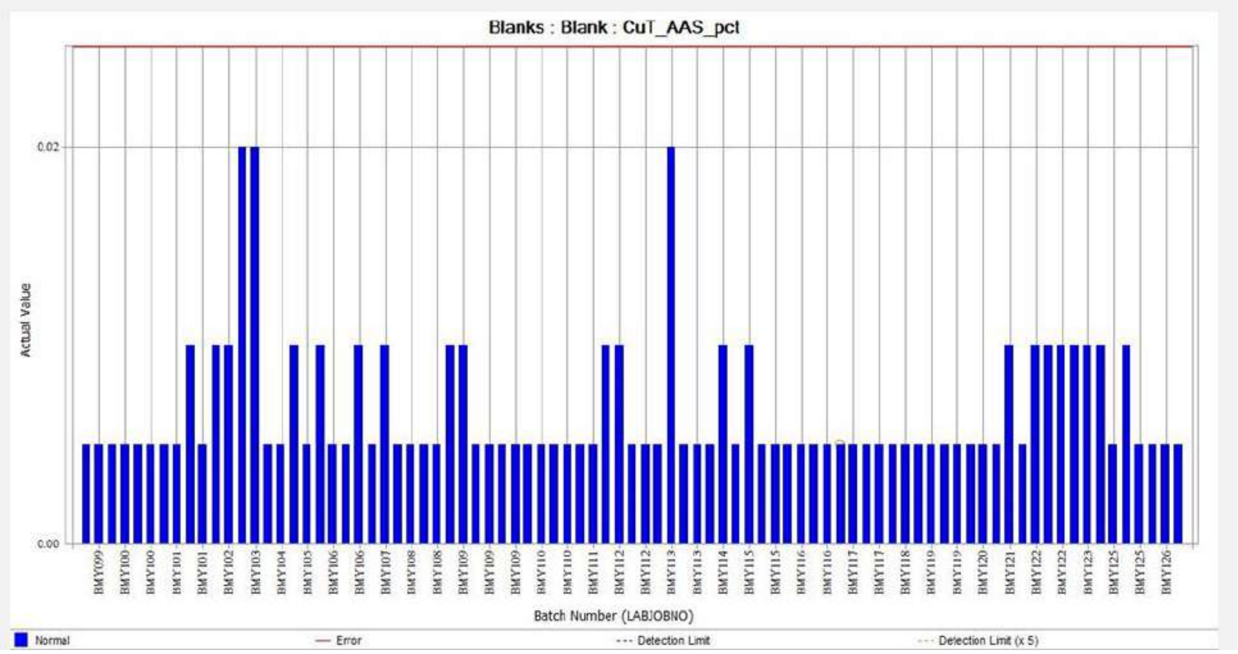


Figure 24-12: Coarse Blank Copper Values

Core-Duplicates for resampling program. Core field duplicates are secondary splits of original core samples collected simultaneously with the primary sample splits. One half split core is quartered to create the duplicate. Core duplicates are used to evaluate the total variability introduced by subsampling, including in the laboratory as well as the variability in the analyses. Core-duplicates should therefore be analyzed by the primary analytical laboratory.

Excelsior's resampling program included a total of 32 pairs of total copper analyses from core-duplicate samples. Figure 24-13 is a scatter plot of the core-duplicate results.

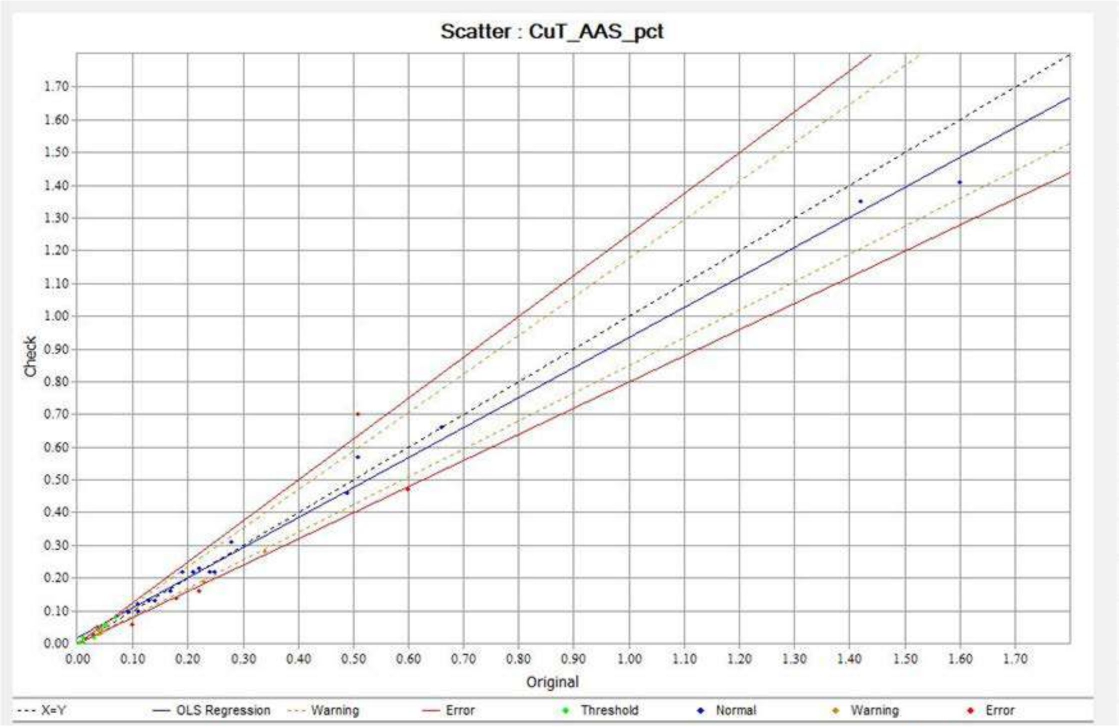


Figure 24-13: Core-Duplicate Total Copper (“CuT”) Results Relative to Primary Sample Assays

Excelsior’s resampling program included a total of 44 pairs of total copper analyses from pulp-duplicate samples. Figure 24-14 is a scatter plot of the pulp-duplicate results.

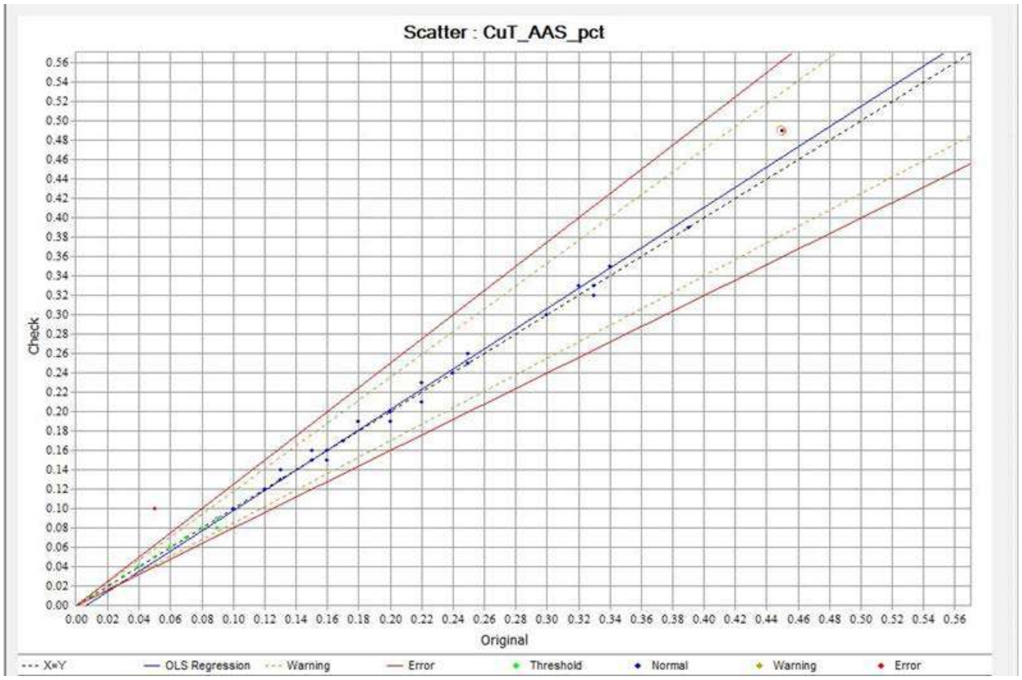


Figure 24-14: Pulp-Duplicate Total Copper (“CuT”) Results Relative to Primary Sample Assays

There is no obvious bias in the duplicate sample results. The average assay duplicate assay values for copper are within 10% of the average original values. No outliers were removed.

24.11.5.3 Excelsior 2022 QA/QC Methods

CRMs. Excelsior purchased commercial certified reference materials ("CRMs") for use in the 2022 core drilling program. The CRMs were inserted into the sample stream and analyzed with the core samples for total copper. The results were used to evaluate the analytical accuracy and precision of the analyses in Excelsior's samples.

Table 24-17 lists the CRMs used by Excelsior for the 2022 core drilling program.

Table 24-17: Certified Reference Materials for 2022 Assays

Reference Material	Certified Value (%Cu)	2 Std Dev (%Cu)	No. of Skyline Analyses
A106009X	0.136	0.02	39
AMIS0358	0.7568	0.0396	4
CDN-ME-2001	1.06	0.04	45

The Skyline copper analyses of the Excelsior CRMs returned excellent results, with generally good precision and accuracy, shown in Figure 24-15 to Figure 24-17. A106009X and AMIS0358 returned no values outside of 2 standard deviations. CDN-ME-2001 had a slight low bias and returned 5 values below 2 standard deviations.

Figure 24-15: A106009X Total Copper Analyses

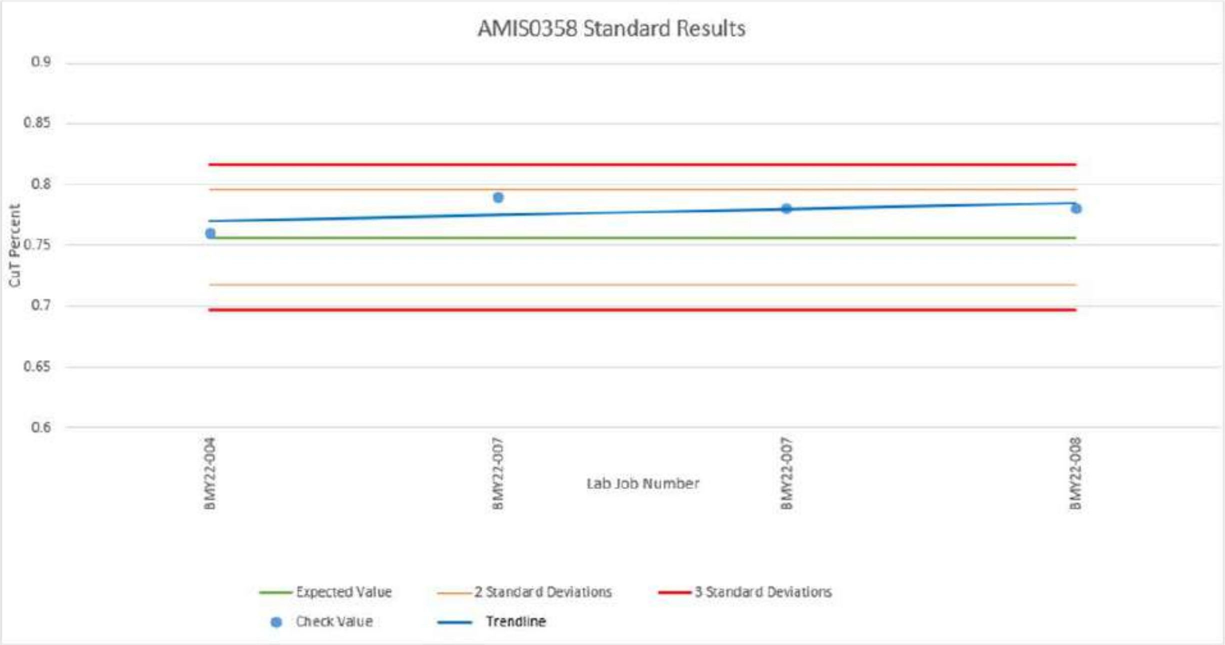


Figure 24-16: AMIS 0358 Total Copper Analyses

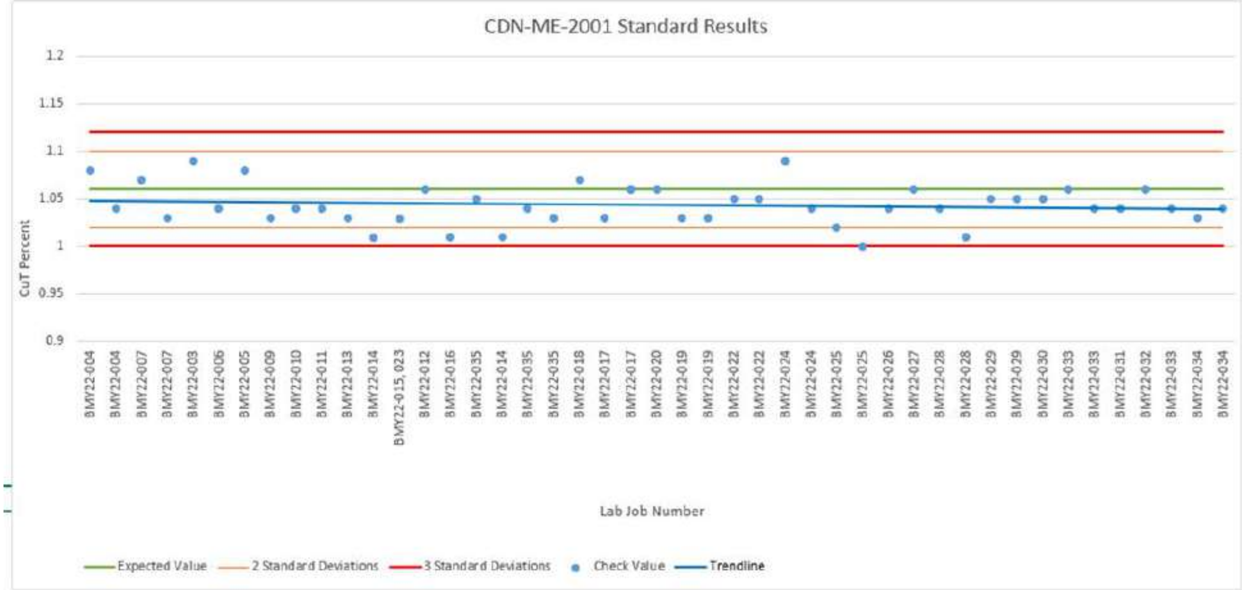


Figure 24-17: CDN-ME-2001 Total Copper Analyses

<u>Coarse Blanks for 2022 drill program.</u> Blank results that are greater than five times the lower detection limit of the relevant analyses are typically considered failures that require further investigation and possible re-assaying of associated drill samples. The detection limit of the Skyline analyses was 0.01% for total copper. Blank samples assaying in excess of five times these detection limits (0.005%) are considered to be failures. A chart of the Skyline analyses of the coarse blanks are shown in Figure 24-18. There was one blank failure that was determined to be a sample swap. Four samples around the failure were re-run to confirm the swap. The failure was removed from the blank figure below.

Figure 24-18: Coarse Blank Copper Values

Core-Duplicates. Field-duplicates are taken by quartering the core and submitting two quarter core samples with different sample numbers and are collected at a rate of approximately 1 in 40 samples. Excelsior's 2022 core drilling program included a total of 33 pairs of total and acid-soluble copper analyses from core-duplicate samples. Figure 24-19 and Figure 24-20 are scatter plots of the core-duplicate results for CuT and CuAS. If one outlier and the lower grade values below a threshold of 0.9 percent CuT are removed, then there is no bias in the data. If one outlier and the lower grade values below a threshold 0.05 percent CuAS are removed, then there is a 3% bias in CuAS. There is no obvious bias in the field-duplicate sample results. The average assay duplicate assay values for copper are within 10% of the average original values. Assay results for the CuT and CuAS comparisons but are included in the figures below.

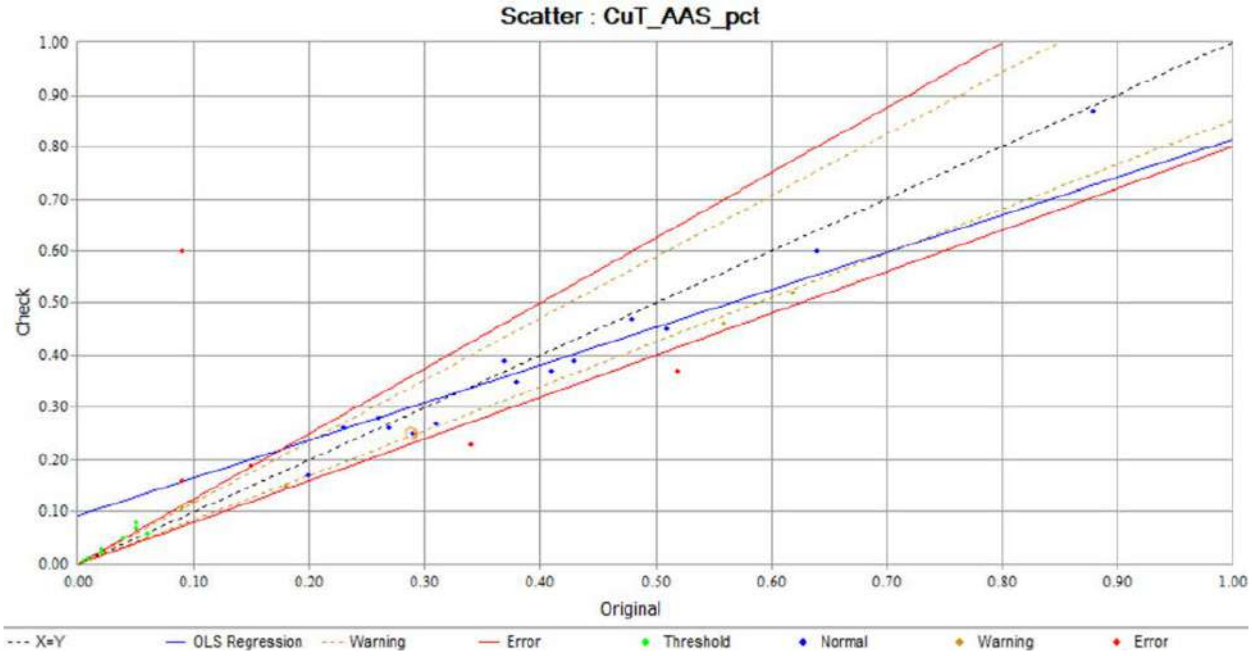


Figure 24-19: Field-Duplicate Total Copper ("CuT") Results Relative to Primary Sample Assays

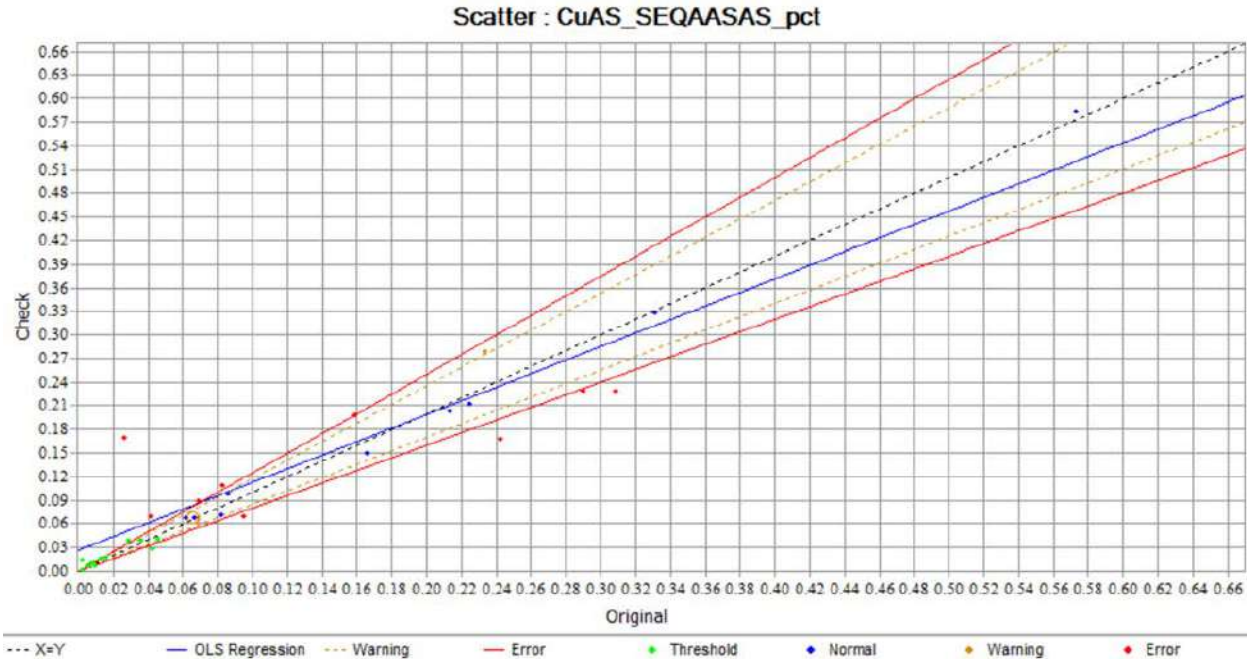


Figure 24-20: Field-Duplicate Acid-Soluble Copper ("CuAS") Results Relative to Primary Sample Assays

A crush-duplicate is a split of the reject prior to pulverization and is done by the lab at a rate of approximately 1 in 40 samples.

Excelsior's 2022 program included a total of 23 pairs of total and acid-soluble copper analyses from crush-duplicate samples. The results for CuT and CuAS are in Figure 24-21 and Figure 24-22. There is no obvious bias in the crush-duplicate sample results. The average assay duplicate assay values for copper are within 10% of the average original values. No outliers were removed.

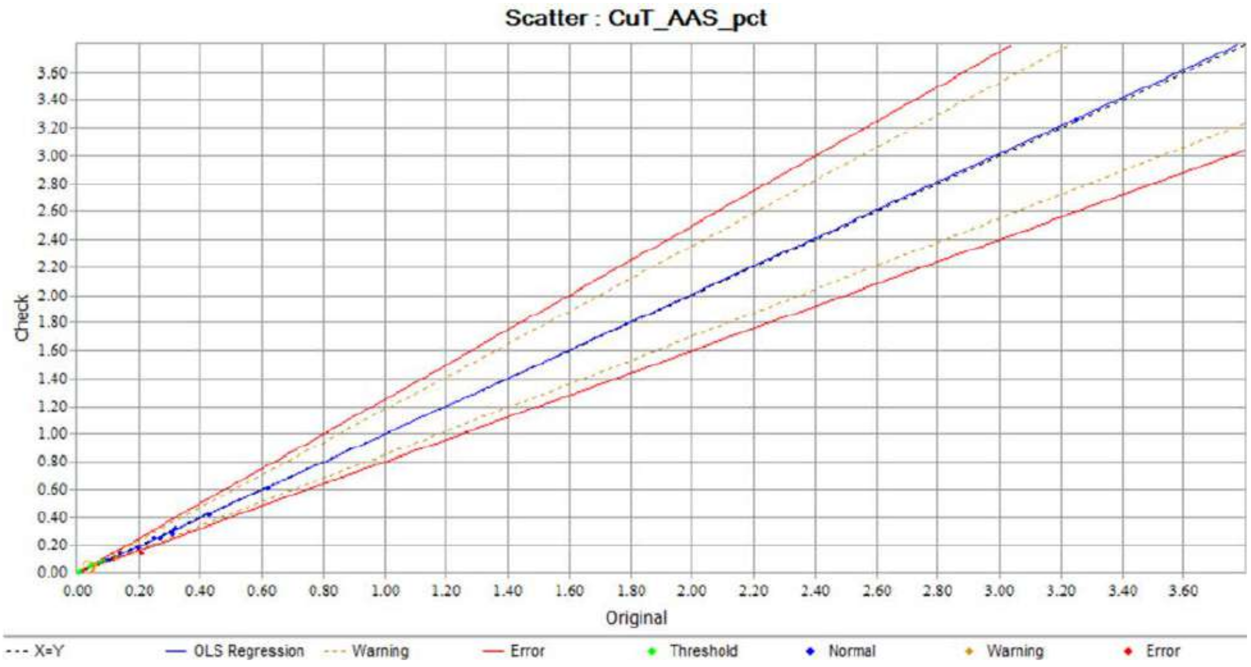


Figure 24-21: Crush-Duplicate Total Copper (“CuT”) Results Relative to Primary Sample Assays

Figure 24-22: Crush-Duplicate Acid-Soluble Copper (“CuAS”) Results Relative to Primary Sample Assays

A pulp duplicate is collected by the lab by producing two pulps from the sample and is done at a rate of approximately 1 in 20 samples. Excelsior’s 2022 program included a total of 48 pairs of total and acid-soluble copper analyses from pulp-duplicate samples. There is no obvious bias in the pulp-duplicate sample results. The average assay duplicate assay values for copper are within 10% of the average original values. No outliers were removed.

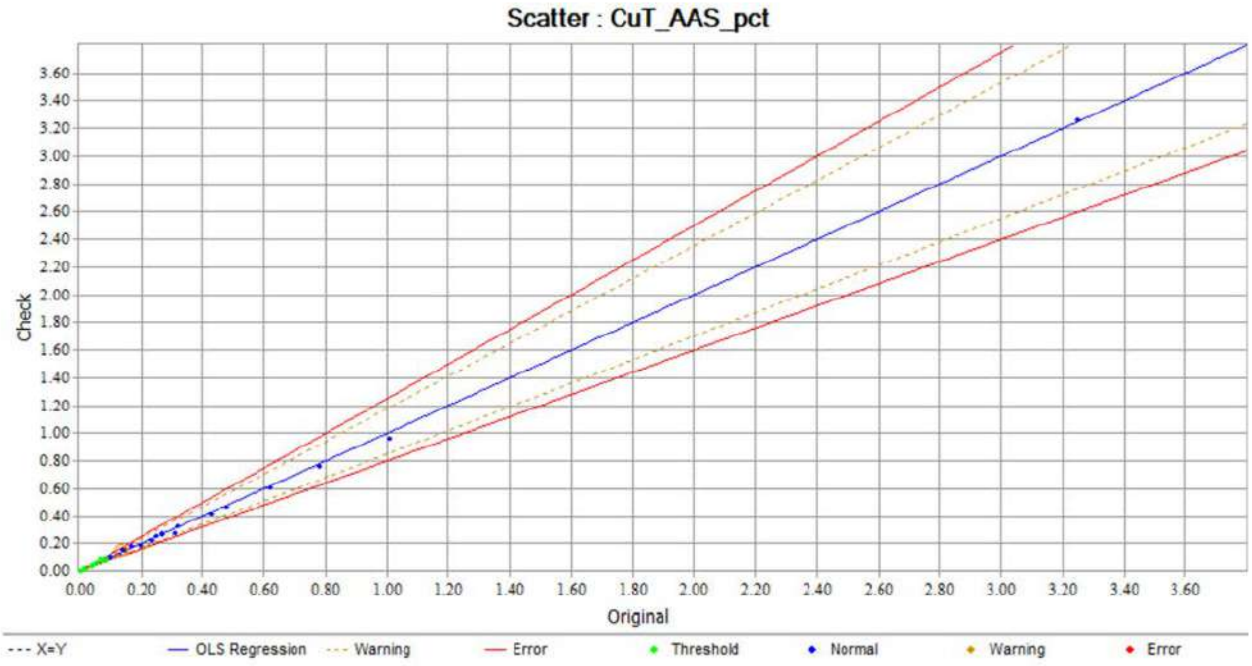


Figure 24-23: Pulp-Duplicate Total Copper (“CuT”) Results Relative to Primary Sample Assays

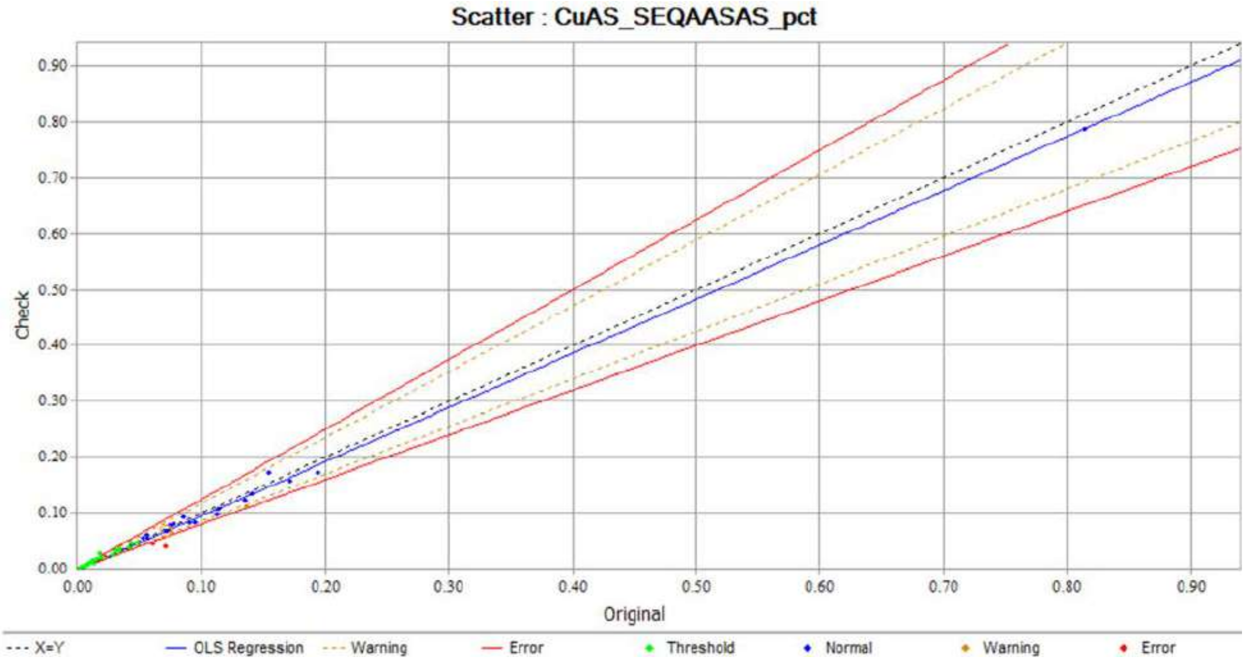


Figure 24-24: Pulp-Duplicate Acid-Soluble Copper (“CuAS”) Results Relative to Primary Sample Assays

24.11.6 Summary Statement

The certification status of some of the historical analytical laboratories is not known. Southwestern Assayers and Chemists is the predecessor to Skyline. Mr. Bickel believes the historical labs were independent commercial laboratories that were widely recognized and used by the mining industry at that time.

Documentation of the methods and procedures used for historical sample preparation, analyses, and sample security, as well as for quality assurance/quality control procedures and results, is incomplete and in many cases not available. Despite this, some of the historical assay certificates have been preserved and Excelsior was able to reasonably duplicate and/or verify the original results through re-sampling of historical core (described in section 24.12.2.4) and new drilling. Mr. Bickel is therefore satisfied that the historical analytical data are adequate to support the current resources, interpretations, conclusions, and recommendations summarized in this report.

Excelsior's sample preparation and analyses were performed at a well-known certified laboratory, and the sample security and QA/QC procedures are adequate to support the current resources, interpretations, conclusions, and recommendations summarized in this report.

24.12 DATA VERIFICATION

Mr. Bickel has verified the Johnson Camp project database and compiled and analyzed available quality QA/QC data collected by Excelsior. Data verification, as defined in NI 43-101, is the process of confirming that data has been generated with proper procedures, has been accurately transcribed from the original source and is suitable to be used. There were no limitations on, or failure to conduct, the data verification for this report other than those discussed later in this section. Additional confirmation on the drill data's suitability for use are the analyses of the Johnson Camp Mine project QA/QC procedures and results as described in Section 24.11.4.

24.12.1 Site Visit

Mr. Bickel visited the Johnson Camp Mine project site on May 3, 2021, and the Excelsior core facility in March of 2021. During the site visits Mr. Bickel inspected the surface geology of the Johnson Camp Mine open pit areas; reviewed historical drill data; and carried out discussions of the current geologic interpretations with Excelsior personnel.

Mineralization verification procedures were conducted, and core was inspected. Mr. Bickel has also maintained a relatively continual line of communication through telephone calls and emails with Excelsior personnel in which the project status, procedures, and geologic ideas and concepts have been discussed. The result of the site visits and communications is that the author has no significant concerns with the project procedures.

RESPEC personnel managed and oversaw the initial setup and startup of the standard operating procedures (SOP) for the core drilling and sampling program, and RESPEC is not aware of Excelsior changing any material aspects of the SOP after the company took control of management of the program.

24.12.2 Database Verification

The current drill-hole database, which supports the resource estimation of the Johnson Camp project area, was provided to RESPEC along with available original historical paper records and reports in the possession of Excelsior. This drill-hole information was then supplemented with Excelsior's sampling data and results through May 3, 2021. The historical information was subjected to various verification measures, the primary one consisting of the core re-sampling campaign conducted by Excelsior in 2016-2017 which was subsequently analyzed and evaluated by RESPEC. Analysis of soluble copper assay data has been a particular focus of RESPEC's analysis. Historical operators of the Johnson Camp Mine have taken multiple approaches with respect to soluble copper assay values in the database including variable analysis methods and calculated soluble copper assay values. RESPEC's analysis isolated only those soluble copper values in the database which could be verified and considered suitable for resource estimation and discarded those that did not meet those criteria. The Johnson Camp Mine has historically produced copper through heap leaching and solvent-extraction/electrowinning processing methods. As such, the reliability of the soluble copper data upon which resource estimates are generated is critical to future mining of the resource. RESPEC's analysis and conclusions are described herein. RESPEC also reviewed the historical cyanide soluble copper assays for use in resource estimation. Historical cyanide soluble copper assays were from the Nord drilling programs completed in 2008 through 2010 and there are assay certificates available to confirm the results. The other two sources of cyanide soluble copper data include the Excelsior's 2016-2017 resampling program and the 2022 drilling program.

24.12.2.1 Drill-Collar Verification

The Johnson Camp Mine database contains 313 historical drill-holes and 44 Excelsior drill-holes. Based on data availability, historical drill-hole-collar coordinates and hole orientations in the database were compared to original paper documentation in the possession of Excelsior. The database was found to reasonably match the historical paper documents with the exception of hole BP-50, for which historical coordinates could not be located and location could not be verified. RESPEC excluded hole BP-50 from the resource area as a result of this finding.

All historical drill-holes were transformed from local coordinates to NAD1927 State Plane Arizona East FIPS coordinates using a two-point rotation determined in 2016 by Darling Geomatics, a land surveyor company. Control points used in determining the coordinate transformation included historical collars from Nord Resources Corp drill-holes.

Collar locations for the 2022 drill program were collected by Darling Geomatics using a Trimble Global Positioning System ("GPS"), which can be accurate to 0.05 ft horizontally and 0.2 ft vertically.

24.12.2.2 Down-Hole Survey Verification

Down-hole deviation data does not exist for any of the historical Johnson Camp Mine drill-holes. All but 19 of the drill-holes (6%) were drilled vertically. Based on the vertical nature of the holes, Excelsior's recent drilling campaign, and the open-pit mining method planned for the resource, Mr. Bickel considers the lack of down-hole deviation data in historical drilling to be immaterial to the mineral resources reported herein.

Down-hole deviation data was collected on 34 of 44 holes for the 2022 Excelsior core drilling program. The holes were surveyed using a magnetic deviation survey tool.

24.12.2.3 Assay Database Verification

Historical Assays: RESPEC completed an analysis of the original drill-hole database provided by Excelsior containing historical assays and used it to build a 'final' database to be used in RESPEC's resource estimation. The final build excluded some assays from the original based on the judgement of the author.

Historical paper records, including copies of original assay certificates were reviewed and compared to the Excelsior database under the supervision of Mr. Bickel. Assay data from the original lab certificates were generally available for review. RESPEC conducted an audit on the database using select historical assay certificates and found them to match except for four discrepancies. One error was found for a CuT historical assay, which was corrected in the RESPEC database. Three historical sample intervals were excluded from the RESPEC database due to historical CuAs values that are significantly higher than the associated historical CuT values.

The existence of calculated, "untrusted" soluble copper assays in the original database were known to RESPEC, based on concerns raised from Excelsior. RESPEC found that a 1,512 of the soluble copper assays were calculated by on a set of linear equations. In the author's opinion, calculated soluble copper values are not appropriate to use in mineral resource estimations. The calculated values were excluded from the RESPEC database.

Arimetco soluble copper data have also been excluded from RESPEC the database due to the high-temperature nature of the analyses, which differs from the ambient temperature of the remaining analyses of soluble copper in the database. These are two distinct methods of analysis and are not appropriate to use mixed together in a resource estimation.

No soluble copper data from the eight Summo holes drilled in the Copper Chief area were accepted in the RESPEC database, due to their anomalously high soluble copper values compared to adjacent holes; the mean of the CuT/CuAs ratios is 0.90 for these holes. The four drilled in the Burro area, which were apparently drilled in a different program and potentially analyzed by different methods, do not appear to have anomalous values compared to adjacent holes and therefore these CuAs results were accepted. Although all Summo CuAs analyses are labelled as "not trusted" in the original database provided by Excelsior due to suspicious CuAs values, RESPEC could not identify reasons for the label upon further investigation.

Excelsior Assays: RESPEC received electronic records directly from the assay lab with the results from Excelsior's 2016-2017 re-sampling program. These data were prioritized in the RESPEC database according to the hierarchy discussed in Section 24.12.2.4. For the 2022 core drill program, RESPEC received electronic records from the client. 99% of electronic records from the lab were checked against the Excelsior database. One minor discrepancy was found due to a re-assay, and the Excelsior accepted result was updated in the database.

24.12.2.4 Excelsior 2021 Re-Samples

Excelsior re-sampled selected intervals of historical drill core and pulp and submitted them to Skyline for analysis. The samples were selected from a spatial distribution of drill holes throughout the deposit, as well as a distribution of drill holes from the various historical operators who originally drilled and explored the property. The program was limited by availability of either core or pulp. Excelsior was able to make comparisons to Arimetco, Cyprus, and Nord samples based on availability.

Results from the re-sampled intervals of pulps represent pulp-duplicate analyses, and re-sampled intervals of ¼ core represent core-duplicate analyses. Mr. Bickel compared the 2016 and 2017 pulp-duplicate and core-duplicate analyses with the historical analyses by operator in the RESPEC database and conducted a mean of pair ("MOP") analysis for each respective duplicate type by year of re-sample and historical operator.

The MOP analysis for 2017 total copper ("CuT") pulp-duplicate samples in holes drilled by Arimetco is provided in Figure 24-25. A total of 1,449 samples were submitted to Skyline for analysis from 48 Arimetco drill-holes. The average relative difference between the new data and historical data is -1%. No outliers were removed. The assays pair compare well and show expected variability.

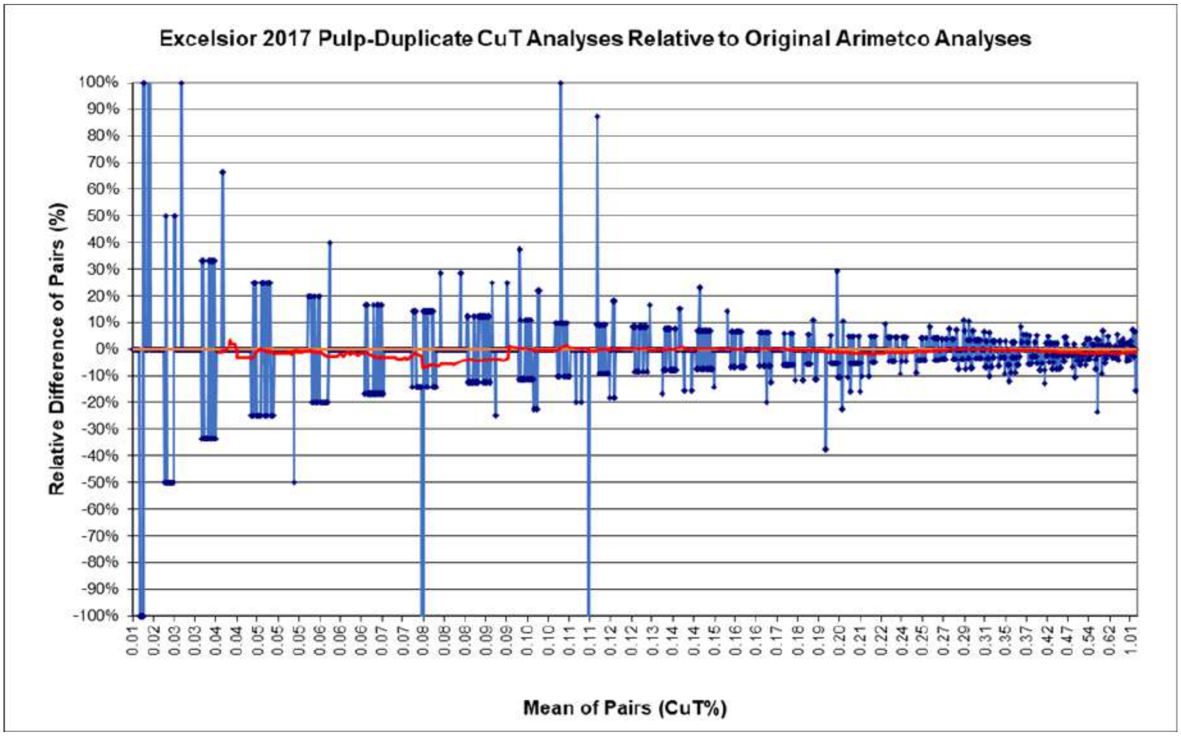


Figure 24-25: 2017 Total Copper (“CuT”) Pulp-Duplicate Analyses Relative to Historical Arimetco Analyses

The MOP analysis for 2016 total copper (“CuT”) pulp-duplicate samples in holes drilled by Arimetco is provided in Figure 24-26. A total of 150 samples were submitted to Skyline for analysis from 2 Arimetco drill-holes. The average relative difference between the new data and historical data is -5%. No outliers were removed. The assays pair compare reasonably well and perhaps show a slight low bias in the new data versus the old.

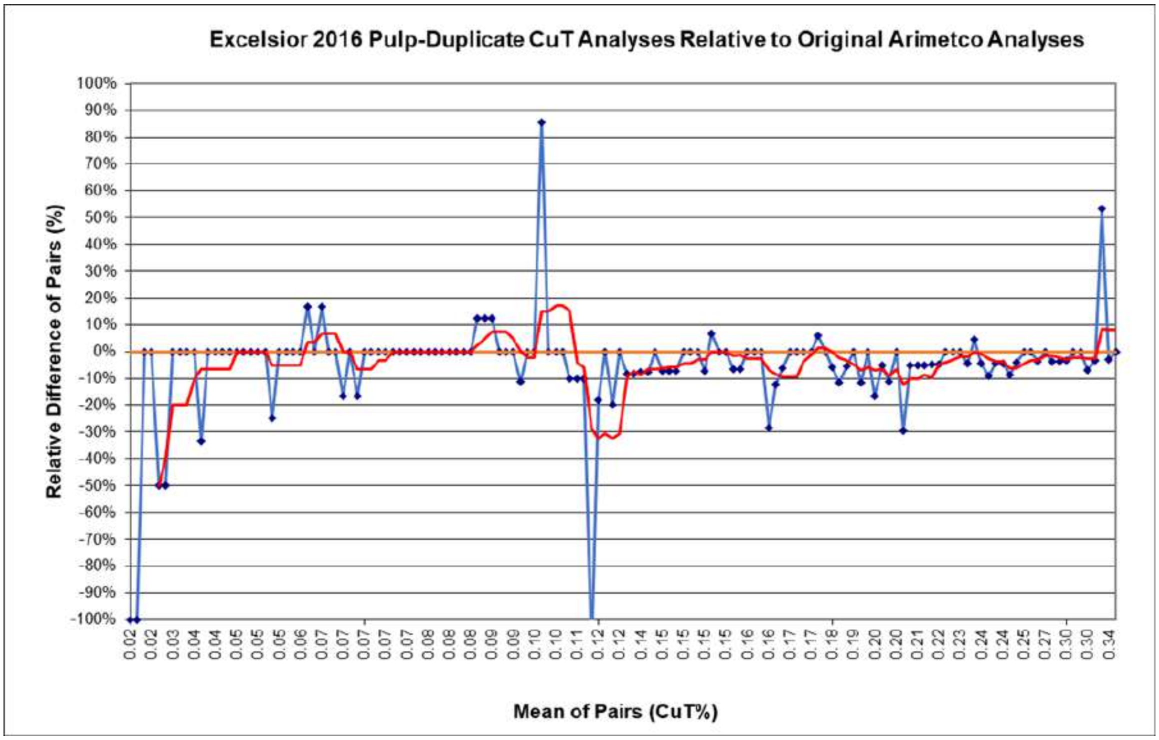


Figure 24-26: 2016 Total Copper (CuT) Pulp-Duplicate Analyses Relative to Historical Arimetco Analyses

The MOP analysis for 2017 total copper ("CuT") core-duplicate samples in holes drilled by Arimetco is provided in Figure 24-27. A total of 115 samples were submitted to Skyline for analysis from 7 Arimetco drill-holes. The average relative difference between the new data and historical data is -2%. Four outlier pairs were removed. The assays pair compare reasonably well.

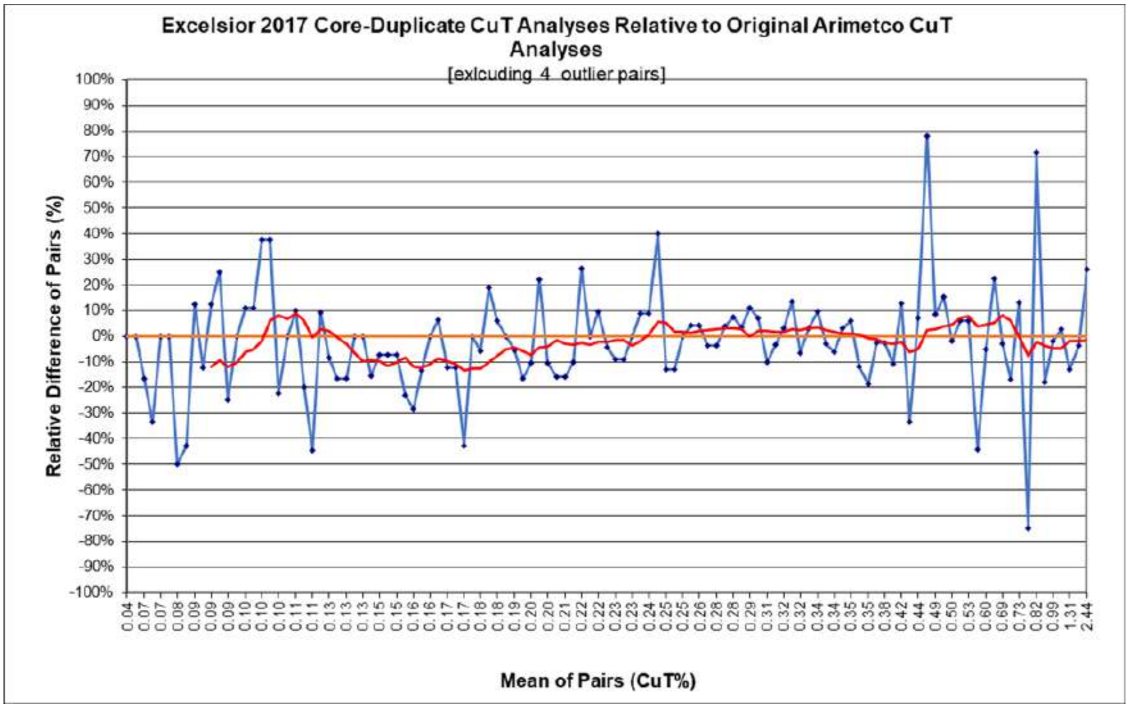


Figure 24-27: 2017 Total Copper (CuT) Core-Duplicate Analyses Relative to Historical Arimetco Analyses

The MOP analysis for 2016 total copper ("CuT") core-duplicate samples in holes drilled by Arimetco is provided in Figure 24-28. A total of 113 samples were submitted to Skyline for analysis from 2 Arimetco drill-holes. The average relative difference between the new data and historical data is -3%. Six outlier pairs were removed. The assays pair compare reasonably well.

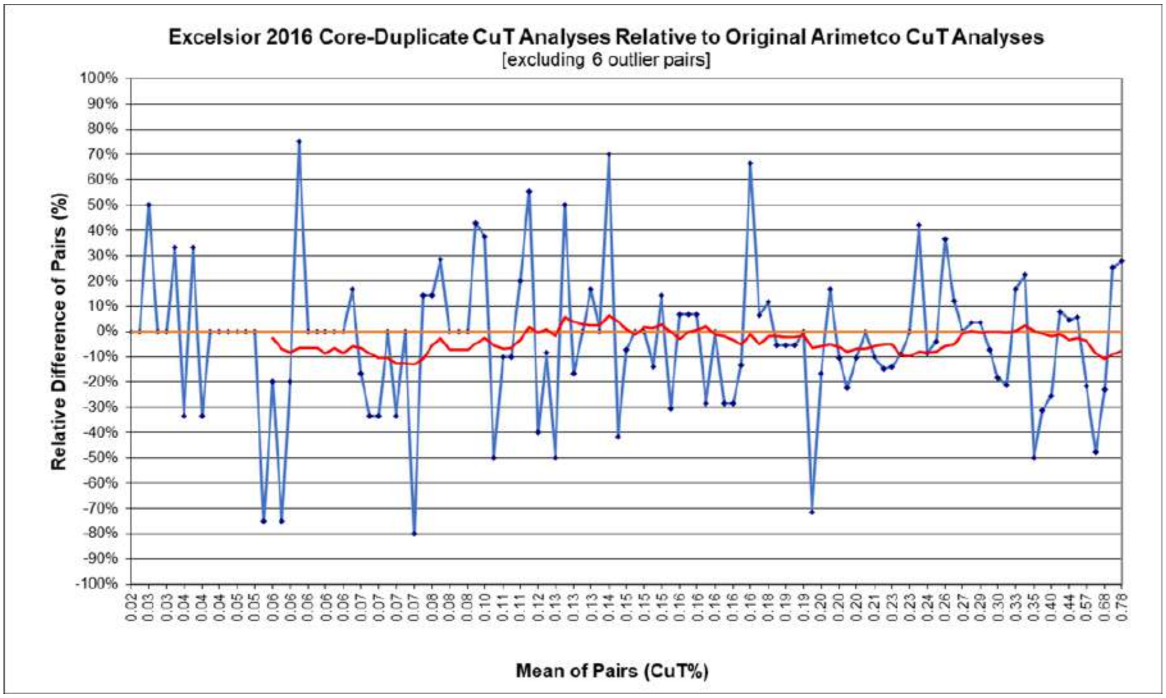


Figure 24-28: 2016 Total Copper (CuT) Core-Duplicate Analyses Relative to Historical Arimetco Analyses

Soluble copper assays from the Arimetco drill-holes are not considered appropriate for resource estimation because of the analysis method used to generate the values. This conclusion is discussed in Section 24.12.2.3. Therefore, no comparison was made to the Excelsior CuAs values and the Arimetco CuAs values. Excelsior CuAs analyses for the Arimetco holes have been used in the RESPEC database for the 1,572 samples that were analyzed. For those samples, Excelsior CuT values also replaced the Arimetco CuT values in the RESPEC database to maintain consistency and ensure that appropriate ratios are used in the estimation.

The MOP analysis for 2017 total copper (CuT) pulp-duplicate samples in holes drilled by Cyprus is provided in Figure 24-29. A total of 127 samples were submitted to Skyline for analysis from 5 Cyprus drill-holes. The average relative difference between the new data and historical data is 2%. Ten outlier pairs were removed. Although the average difference suggest that the assay data pairs compare reasonably well, the author notes that the variability is high for pulp-duplicates. This might be explained by the low-grade nature of the assays showing the most variability.

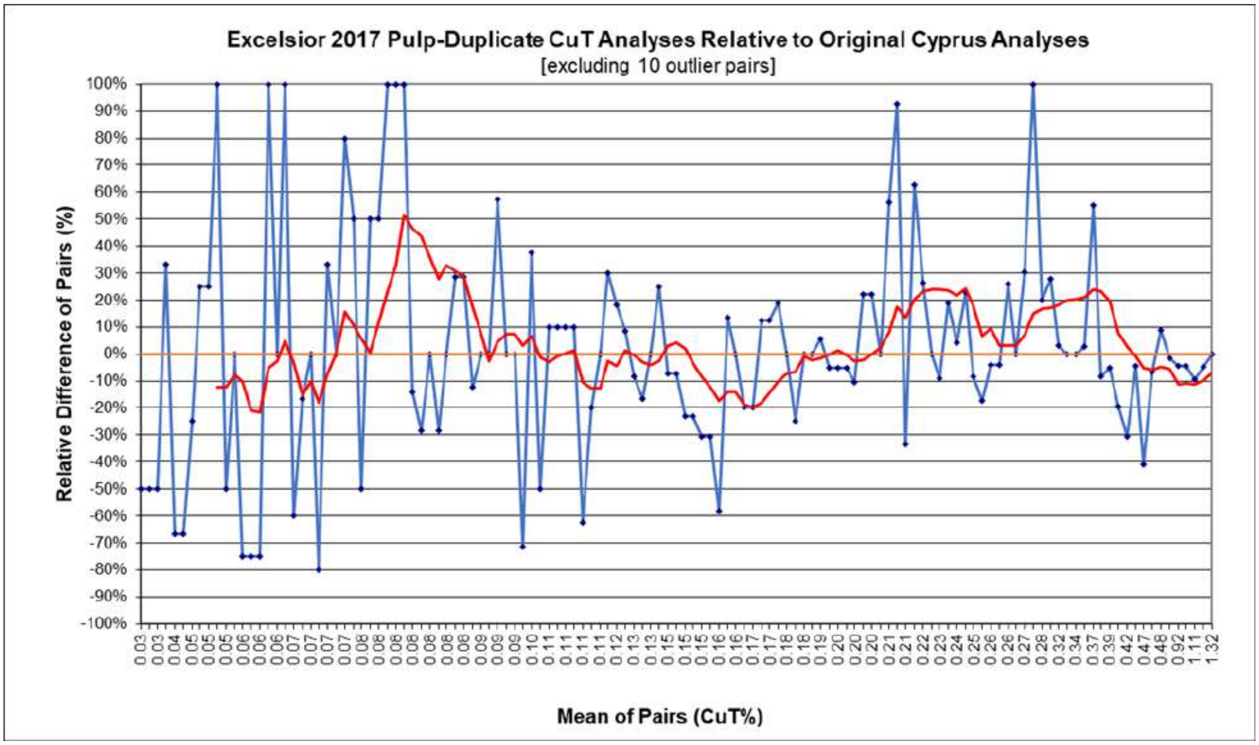


Figure 24-29: 2017 Total Copper (CuT) Pulp-Duplicate Analyses Relative to Historical Cyprus Analyses

The MOP analysis for 2017 total copper (CuT) core-duplicate samples in holes drilled by Cyprus is provided in Figure 24-30. A total of 524 samples were submitted to Skyline for analysis from 17 Cyprus drill-holes. The average relative difference between the new data and historical data is -29%, showing a consistent low bias in the Excelsior samples compared to the original Cyprus assays. No outlier pairs were removed. The data suggest that significant core loss occurred in the Cyprus drill-holes, which is unsurprising given the age of the core.

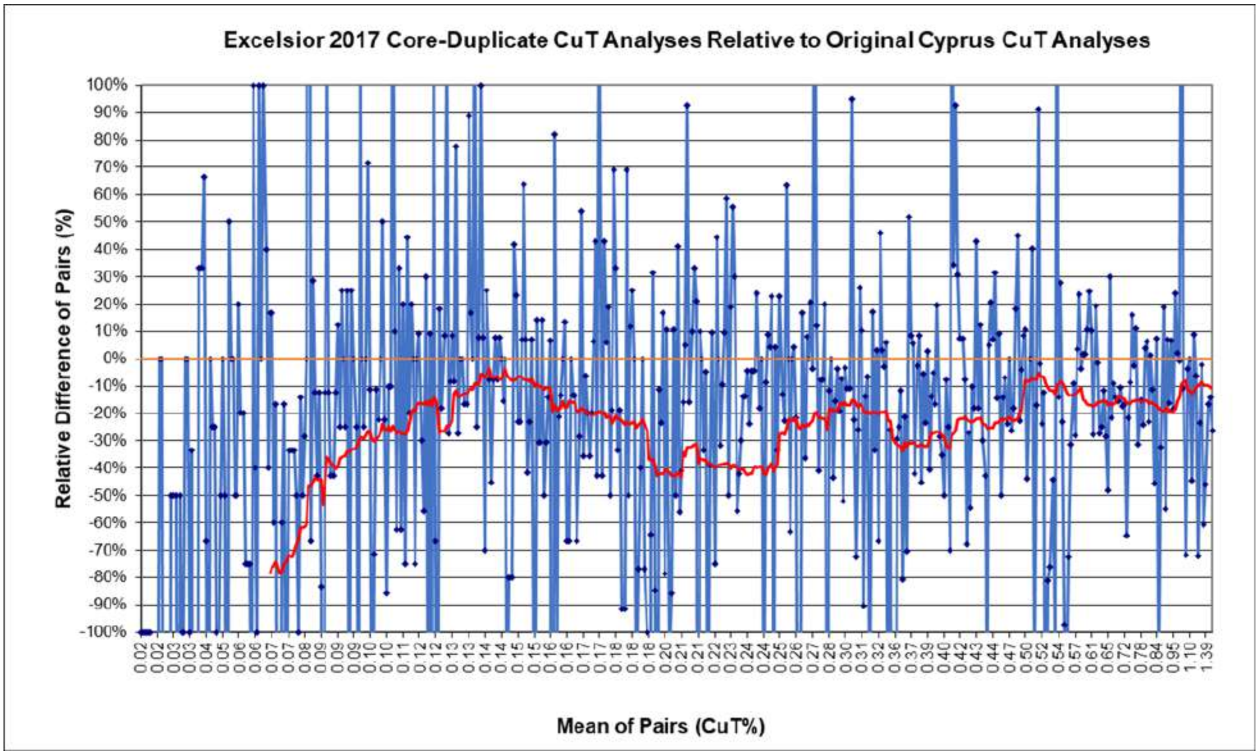


Figure 24-30: 2017 Total Copper (CuT) Core-Duplicate Analyses Relative to Historical Cyprus Analyses

The MOP analysis for 2017 total copper (CuT) core-duplicate samples in holes drilled by Cyprus is provided in Figure 24-31. A total of 114 samples were submitted to Skyline for analysis from 5 Cyprus drill-holes. The average relative difference between the new data and historical data is -122%, showing a consistent low bias in the Excelsior samples compared to the original Cyprus assays, with variability almost exclusively on the low side. No outlier pairs were removed. The data suggest that significant core loss occurred in the Cyprus drill-holes, which is unsurprising given the age of the core.

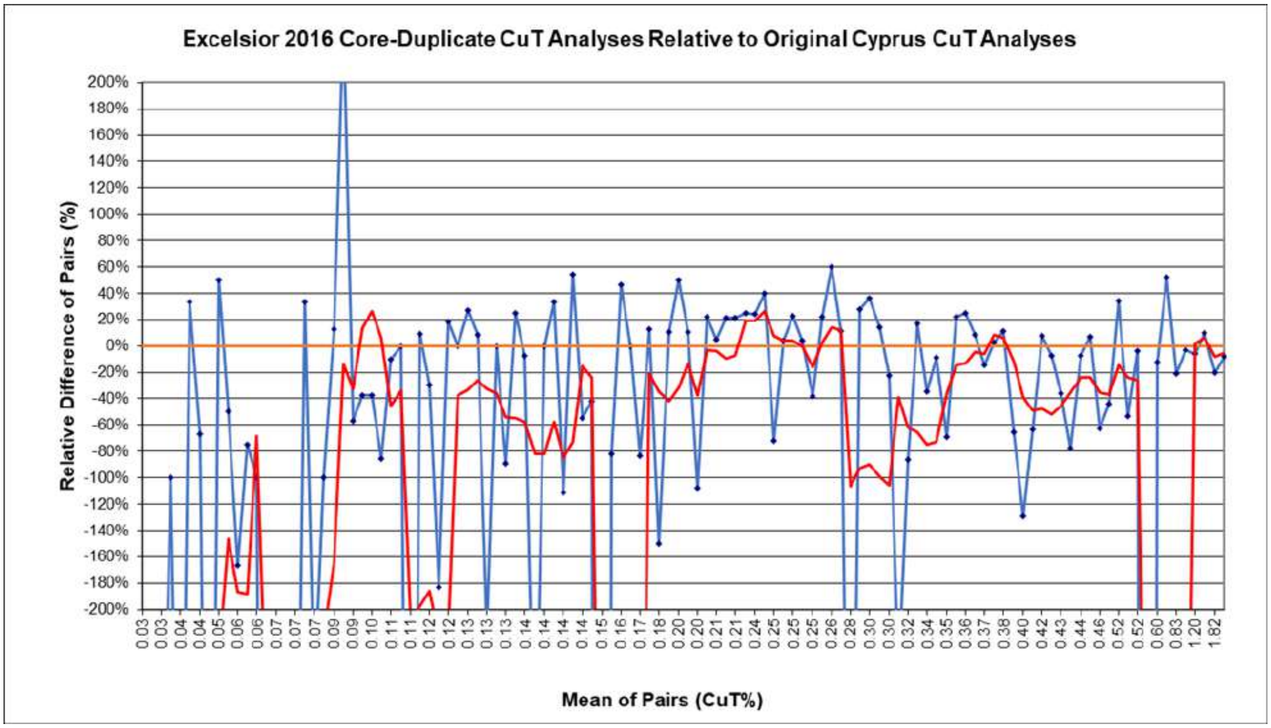


Figure 24-31: 2016 Total Copper (CuT) Core-Duplicate Analyses Relative to Historical Cyprus Analyses

The MOP analysis for 2017 soluble copper (“CuAs”) core-duplicate samples in holes drilled by Cyprus is provided in Figure 24-32. A total of 124 samples were submitted to Skyline for analysis from 5 Cyprus drill-holes. The average relative difference between the new data and historical data is -155%, showing a consistent low bias in the Excelsior samples compared to the original Cyprus assays, with variability almost exclusively on the low side. No outlier pairs were removed. The data are consistent with and slightly lower than the total copper core-duplicate data, suggesting that soluble copper was preferentially lost in the Cyprus core.

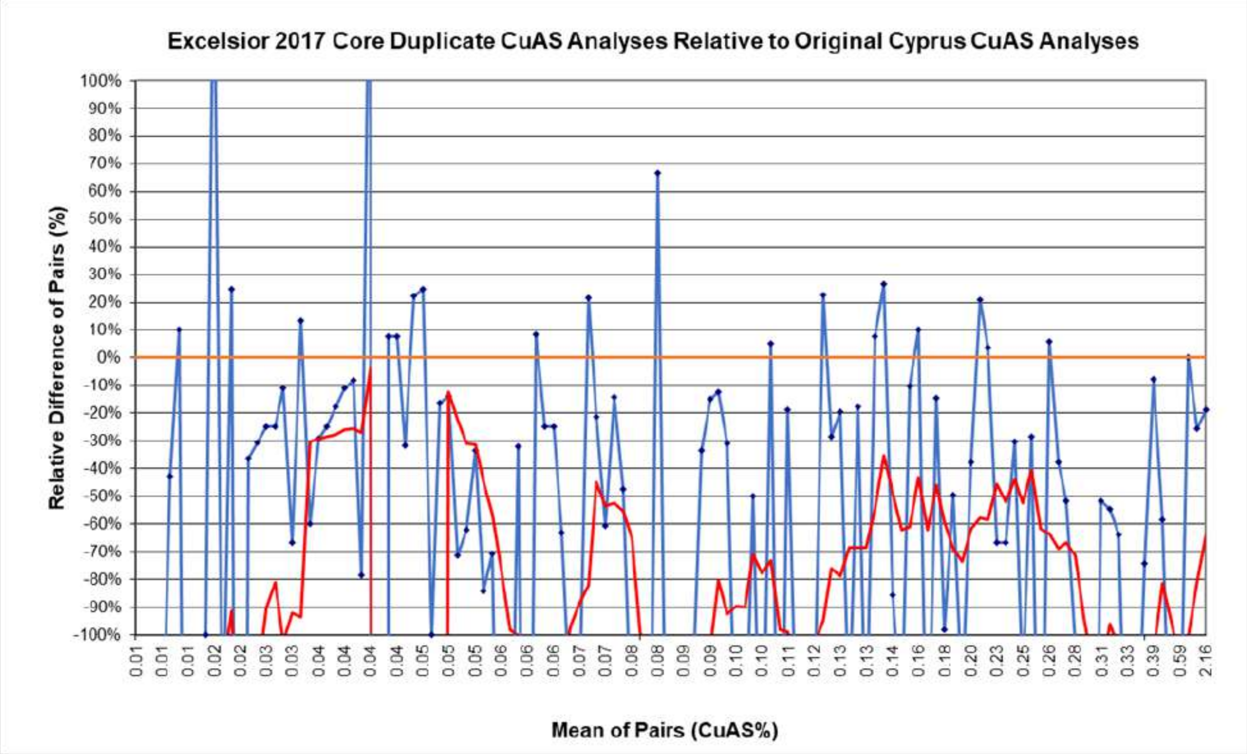


Figure 24-32: 2017 Soluble Copper (“CuAs”) Core-Duplicate Analyses Relative to Historical Cyprus Analyses

The MOP analysis for 2016 total copper (CuT) pulp-duplicate samples in holes drilled by Nord is provided in Figure 24-33. A total of 13 samples were submitted to Skyline for analysis from 1 Nord drill-hole. The average relative difference between the new data and historical data is less than 1%. No outlier pairs were removed. The data, while limited, show good reproducibility from the original samples.

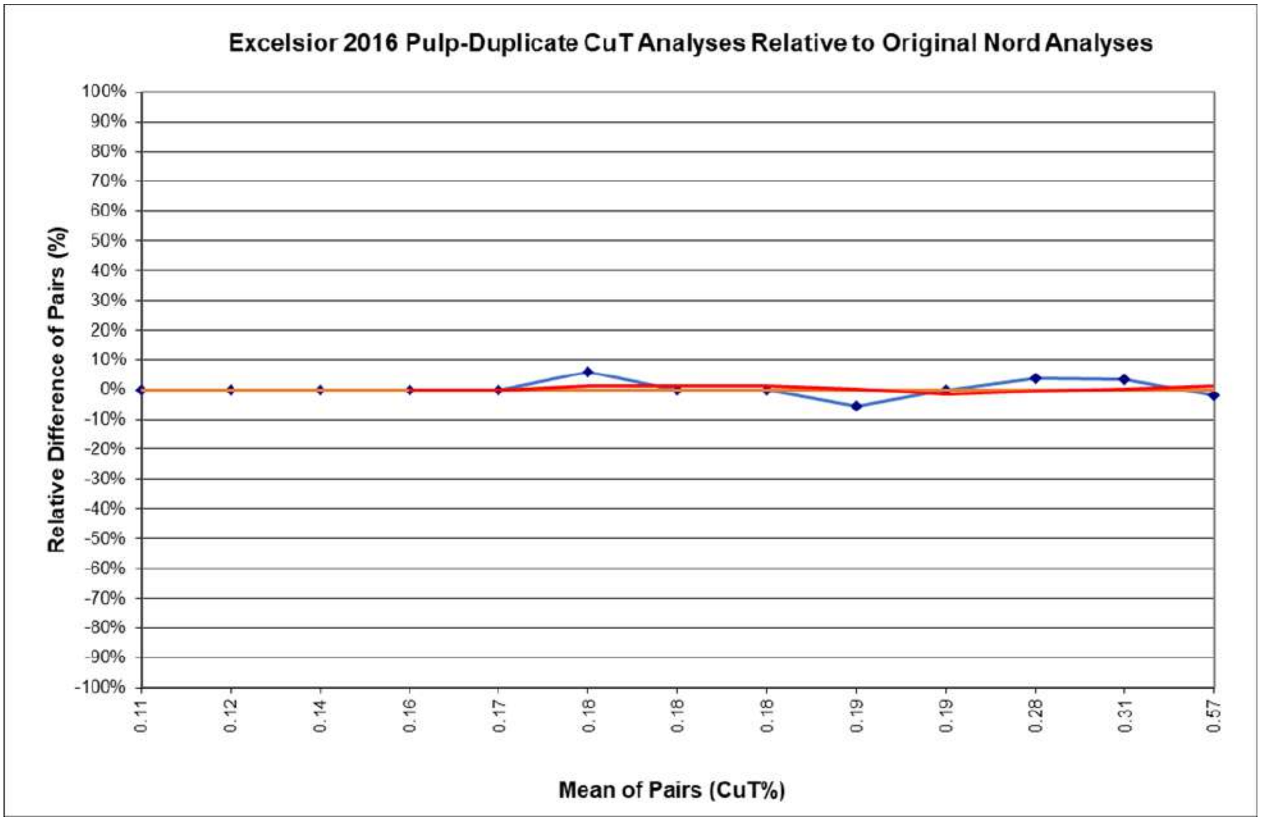


Figure 24-33: 2016 Total Copper (“CuT”) Pulp-Duplicate Analyses Relative to Historical Nord Analyses

The MOP analysis for 2016 soluble copper ("CuAs") pulp-duplicate samples in holes drilled by Nord is provided in Figure 24-34. A total of 13 samples were submitted to Skyline for analysis from 1 Nord drill-hole. The average relative difference between the new data and historical data is less than 1%. No outlier pairs were removed. The data, while limited, show good reproducibility from the original samples.

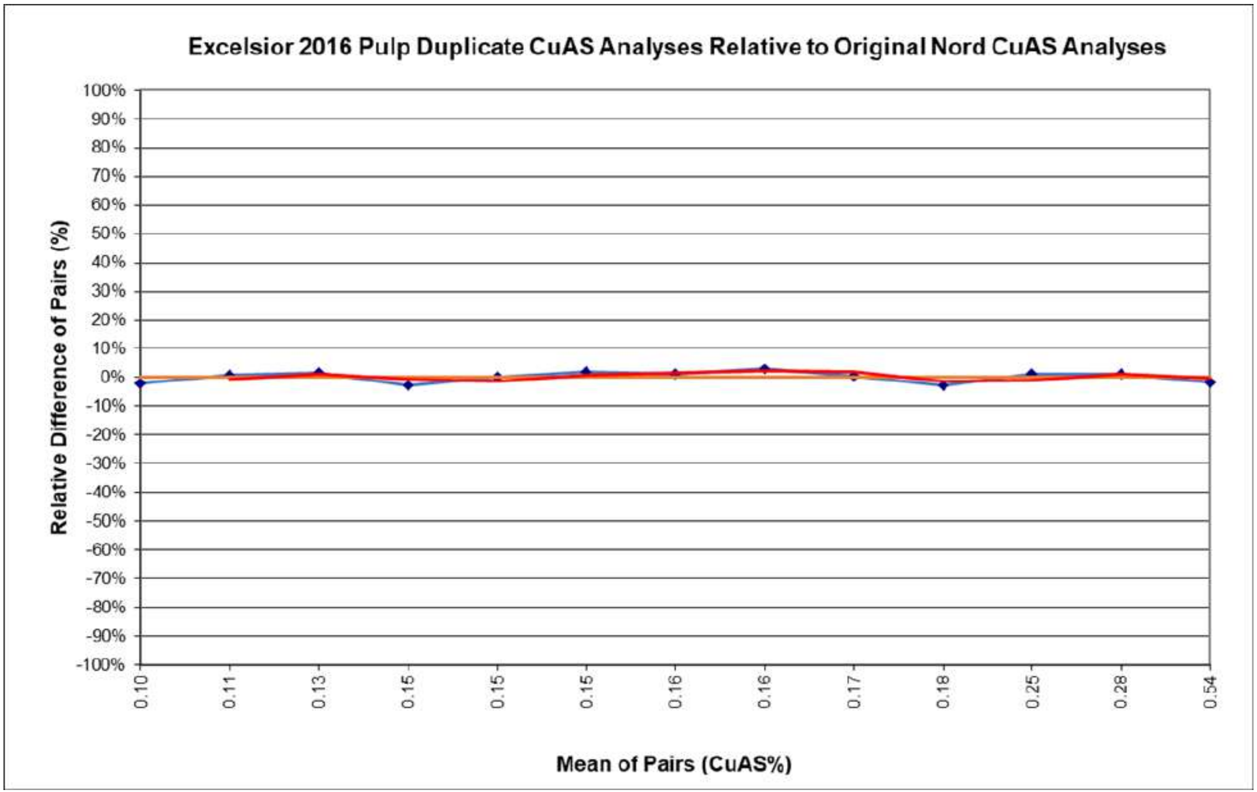


Figure 24-34: 2016 Soluble Copper ("CuAs") Core-Duplicate Analyses Relative to Historical Nord Analyses

<u>Prioritization of Excelsior Data for Importing into RESPEC Database</u>

When multiple sets of CuT and CuAs are available for any single historical interval, the following hierarchy was followed (from highest to lowest priority):

2017 pulp checks > 2016 pulp checks > 2017 core resamples > 2016 core resamples

Pulps were prioritized over core resamples due to the evidence of loss of CuT/CuAs from historical core. 2017 analyses were chosen over 2016 because (1) there are many more analyses from the 2017 resampling program than from 2016; and (2) the 2016 CuT analyses tend to be lower than those from 2017, and the 2017 CuT and CuAs agree well with matched historical data.

Note that the choice of any prioritization would be very unlikely to lead to significantly different results in a resource estimation. For example, prioritization of pulp over core affects only 119 historical sample intervals (all from Arimetco holes ACC06 and AJ63).

In Mr. Bickel's opinion, a more important rule to follow is that if a certain CuAs dataset is chosen for use in the project database, the corresponding CuT values from the same set of analyses should also be used.

Table 24-18 summarizes the source of analyses chosen to be included in the RESPEC database, by historical operator.

Table 24-18: Summary of Analyses in RESPEC Database

Data	Sample Intervals	Comments
Arimetco CuT	978 Arimetco + 1,572 Excelsior = 2,550	1,572 Excelsior values used
Arimetco CuAS	0 Arimetco + 1,572 Excelsior = 1,572	1,572 Excelsior values used
Arimetco CuCN	0 Arimetco + 1,572 Excelsior = 1,572	1,572 Excelsior values used
Nord CuT + CuAS	2,336 Nord + 0 Excelsior = 2,338	13 Excelsior pulp checks not used
Nord CuCN	2,278 Nord + 13 Excelsior = 2,291	
Cyprus CuT	6,691 Cyprus + 363 Excelsior = 7,055	1,001 Excelsior values available
	140 Excelsior pulp checks	140 Excelsior values used
	638 Excelsior core duplicates	0 Excelsior values used
	223 Excelsior core samples not sampled by Cyprus	223 Excelsior values used
Cyprus CuAS	3,013 Cyprus + 363 Excelsior = 3,376	Excelsior values used as per CuT
Cyprus CuCN	0 Cyprus + 1,001 Excelsior = 1,001	1,001 Excelsior values used
Summo CuT	499 Summo + 0 Excelsior = 499	0 Excelsior values used
Summo CuAS	176 Summo + 0 Excelsior = 176 271 from holes 1-7 w/ anomalously high CuAS/CuT 176 from holes 8-12	all Summo CuAS is "not trusted" 0 Summo values used 176 Summo values used
Summo CuCN	0 Summo + 0 Excelsior = 0	
Unknown CuT	80 unknown + 0 Excelsior = 80	Quintana?
Unknown CuAS	0 unknown + 0 Excelsior = 0	
Unknown CuCN	0 unknown + 0 Excelsior = 0	
Excelsior 2022 CuT	1,541 Excelsior	
Excelsior 2022 CuAs	1,541 Excelsior	
Excelsior 2022 CuCN	1,541 Excelsior	
RESPEC DB CuT	10,587 historical + 3,476 Excelsior = 14,063	
RESPEC DB CuAS	5,524 historical + 3,476 Excelsior = 9,000	
RESPEC DB CuCN	2,278 historical + 4,139 Excelsior = 6,417	

24.12.3 Independent Verification of Mineralization

Verification of mineralization was conducted during Mr. Bickel's visits to Excelsior's properties in March and May of 2021. During these site visit, drill core was examined pit faces with visible copper were observed at the property. The existence of the Johnson Camp Mine has been widely known in the industry for many years prior to Excelsior's involvement and there is a documented production history of the mine from several companies (Cyprus, Arimetco, and Nord) that were well-known and reputable operators.

24.12.4 Summary Statement on Data Verification

Mr. Bickel has undertaken extensive verification of the historical data. The core-duplicate analyses performed in 2016-2017 as well as the Excelsior drilling campaign in 2022 allowed Mr. Bickel to verify that the historical assay data in the Johnson Camp Mine database is of sufficient quality for use in the estimations of the current resources.

Explicit modeling of the copper mineralization was the most critical component to the estimation of the project mineral resources. This 'hands-on' approach provided meaningful verification of the historical data, whereby continuity and sensibility of meaningful geological variables, and the assays in the context of those variables, were carefully evaluated and considered.

Mr. Bickel experienced no limitations with respect to data verification activities related to the Johnson Camp Mine project other than limited availability of some of the historic data. In consideration of the information summarized in this and other sections of this report, Mr. Bickel has verified that the project data are adequate as used in this report, most significantly to support the estimation and classification of the mineral resources reported herein.

24.13 MINERAL PROCESSING AND METALLURGICAL TESTING

24.13.1 Introduction

The current PEA focuses on continued heap leaching of the JCM oxide copper zones, supplemented by a new pad on which deeper transition and primary sulfide mineralization will be heap leached. Since oxide leaching will continue, this section on Mineral Processing and Metallurgical Testing will be included from past studies. Additional text specific to heap leaching of the deeper resource is presented in Section 24.13.3.

The Johnson Camp District was an historic producer of copper, gold, silver, lead, zinc, and tungsten beginning in 1881. The current Johnson Camp Mine (JCM) was developed by Cyprus Minerals and commissioned in 1975 at a capital cost of $3.3 million (in 1975 dollars) for the SX-EW plant[5]. The plant was modeled on the original Cyprus Bagdad, AZ, SX-EW circuit that was built in the early-1970s and is still in operation. JCM was either the third or fourth domestic heap leach/SX-EW operation after Ranchers Bluebird, AZ and Cyprus Bagdad and has undergone few changes during its long life. However, the basic design has stood the test of time. The plant was modernized in 2019-2020.

The total design PLS flowrate is nominally 3,880 USGPM to two parallel SX circuits comprising 2 extraction stages in series and one strip stage with an SX copper recovery of 92 percent from a PLS grade of 1.63 gpl copper. The electrowinning section was designed with a cathode capacity of 25 million pounds annually and consists of two blocks of polymeric concrete cells with 56 cells in one block and 32 in the second block.

From 1975 through 1984, the operation produced about 100 million pounds of copper from 15 million tons of material assaying 0.8% TCu. A prolonged depression in copper prices forced closure of JCM, as well as most other Arizona copper properties, including Morenci, Bagdad, and Ajo in the early-to-mid 1980s. In 1984, Arimetco acquired the property for $1 million and began operating JCM in 1991, leaching about 3 million tons of ROM material annually for several years, followed by crushing the material for a 2-year period to improve copper recovery. However, JCM was closed again in 1997 in response to low copper prices. Mineralized material placement on the heap pad by Cyprus and Arimetco totaled 31.8 million tons and yielded 157 million pounds of copper, averaging 4.94 pounds of cathode recovered per ton of material leached.

Nord Resources acquired JCM in 2008 and mined and stacked crushed and ROM material on the original heap leach pad from 2009 through June 2010. Most of this material came from Lower Abrigo and Upper Diabase mineralized rock from both the Copper Chief and Burro pits, although a small amount of Bolsa Quartzite was included in the mixture. Grades averaged 0.32% TCu and 0.15% ASCu by the standard ambient assay procedure. No additional material was mined after 2010.

In 2015, Excelsior acquired the JCM assets for $ 8.4 million, which was a very strategic investment, as it provided Excelsior with an operating mine and a 25 million pound per year SX-EW facility.

---

[5] “Mining Directory 1994/95”, Randol International, page 207.

24.13.2 Laboratory Metallurgical Tests for General Leaching Response

24.13.2.1 Column Leaching Tests

24.13.2.1.1 2010-2012 Column Tests for Nord Resources

Column leaching tests performed between 2010 and 2012 were reported by Dr. Ronald J. Roman. The tests were conducted in 8.5-inch diameter columns for the minus 1 inch sample sizes and 20-inch diameter columns for the minus 6 inch sample sizes. Columns were approximately 20 feet high and used samples from the JCM active mining operation in pits named Copper Chief ("CC") and Burro Pit ("BP"). Some samples were crushed and screened to minus 1-inch fragment size, blended, agglomerated, cured, and loaded into the columns which were all leached concurrently. Other samples were crushed and screened to minus 6-inch fragment size. These received no cure or agglomeration.

For the minus 1-inch columns the samples varied significantly in breakage characteristics with the minus 6-mesh fraction ranging from 20 to 47 percent of the total sample weight. Agglomeration of fines and curing of the samples with dilute aqueous sulfuric acid was done by mixing the samples and solution in a portable cement mixer to a target of 8% moisture. The amount of 100% sulfuric acid in the curing solution that was added to the samples varied from 9.8 to 14.3 lb/ton of sample and averaged 12.2 lb/ton. This quantity was added to the eventual net acid consumption estimate.

The 8.5-inch diameter columns were then charged with the minus 1-inch agglomerated and cured samples and the 20-inch diameter columns were charged with the minus 6-inch samples. Both were irrigated with a lixiviant consisting of acidified JCM SX raffinate. The recorded flowrates of approximately 13 liters per day for the 8.5-inch diameter columns and approximately 71 liters per day for the 20-inch diameter columns were somewhat variable and resulted in average solution application rates between 0.0054 and 0.0062 gpm/ft^2. Head assays were calculated from residue and solution weights or volumes and assays. Acid consumptions were average values at the copper extractions shown in Table 24-19. The assays shown were conducted after hot acid digestion because that procedure gave results that correlated most closely to column copper extractions. However, they overstate ASCu. Excelsior uses a more industry-standard ambient acid soluble assay technique. Nord also assayed several columns using an ambient assay technique. ASCu copper extraction for those columns using the ambient technique is shown in the far right-hand column in Table 24-19.

Table 24-19: 2010-2012 Column Leaching Tests

Column #	Size	Pit	Formation Name	Assayed Head (HOT)		Leach	Acid Consumption		Copper Extraction		
				%TCu	%ASCu	Days	lbs/ton	lb/lb	%TCU	%ASCu HOT	%ASCu AMBIENT
1	-1"	CC	Bolsa Quartzite	0.49	0.47	79	19	3.9	67	70	95
2	-1"	CC	Pioneer Shale	1.23	1.21	111	11	2.1	82	84	92
3	-1"	CC	Lower Abrigo	0.24	0.20	70	45	29.6	48	58	177
4	-1"	CC	Diabase	0.47	0.44	102	33	5.9	73	79	233
5	-1"	BP	Pioneer Shale	0.26	0.24	102	24	7.0	74	81	120
6	-1"	BP	Bolsa Quartzite	0.22	0.20	62	15	4.2	76	83	152
8*	-1"	BP	Diabase	0.36	0.33	95	37	6.4	76	82	161
9	-6"	BP	Bolsa Quartzite	0.25	0.16	111	29	18.9	33	52	
10	-6"	BP	Lower Abrigo	0.26	0.24	137	9	8.9	49	53	
11	-6"	CC	Bolsa Quartzite	0.67	0.48	155	29	4.6	71	98	
12	-6"	CC	Diabase	0.51	0.17	155	66	15.9	45	133	
13	-6"	BP	Mid/Up Abrigo 1	0.34	0.32	165	56	18.3	49	51	
14	-6"	BP	Lower Abrigo	0.63	0.55	162	44	9.6	43	49	
15	-6"	BP	Mid/Up Abrigo 3	0.31	0.27	74	40	28.0	24	28	
16	-6"	BP	Mid/Up Abrigo 2	0.40	0.37	126	36	13.8	37	40	
17	-1"	BP	Mid/Up Abrigo 1	0.29	0.28	87	43	9.9	48	50	
18	-1"	BP	Lower Abrigo	0.91	0.85	164	77	6.5	58	63	
19	-1"	BP	Mid/Up Abrigo 3	0.24	0.22	87	39	52.1	16	17	

Column #	Size	Pit	Formation Name	Assayed Head (HOT)		Leach Days	Acid Consumption		Copper Extraction		
				%TCu	%ASCu		lbs/ton	lb/lb	%TCU	%ASCu HOT	%ASCu AMBIENT
20	-1"	BP	Mid/Up Abrigo 2	0.38	0.37	87	39	9.9	43	44	
21	-1"	CC	Lower Abrigo	0.24	0.20	93	39	30.1	26	34	194
22	-1"	CC	Lower Abrigo	0.24	0.20	93	41	25.6	30	37	245
23	-1"	CC	Lower Abrigo	0.24	0.20	91	43	36.8	24	29	174
24	-1"	CC	Lower Abrigo	0.24	0.20	91	57	47.9	24	30	179

*Note: Column 7 was discontinued due to plugging

There were 35 tests, but some results were inconclusive, and the laboratory daily reporting sheets are missing for some. Results from the 23 reliable tests and well-documented tests are summarized in Table 24-19.

It is important to note that the far right-hand column, summarizing results for Column Leaching Tests numbered 1-6, 8*, 21 and 22, presents ASCu leaching extractions well in excess of 100 percent. This probably is because transitional minerals like chalcocite, which do not report to the ambient ASCu assay technique, will dissolve over the longer period of time in a column leach test, thus contributing more extracted copper than indicated by the ambient assay. This interpretation bears directly on Sections 24.13.2.1 and 24.13.3 that discuss predicted sulfide leaching extractions.

The data presented in Table 24-19 require a few additional comments and tentative conclusions. Questionable values are highlighted in red. The % ASCu extraction for column number 12 appears suspicious and a likely error was the low ASCu head assay. Also, the acid consumption appears high for diabase and the minus 6-inch fragments surely would not consume more acid that the fine minus 1-inch crushed product, especially with a shorter leach retention time. Samples of the Abrigo formation show some variability in acid consumption, but the *lb acid/ton* and *lb acid/lb* figures reported for Column 24 appear too high and may have been incorrect calculations.

It is important to note that acid consumptions and copper extractions obtained from column tests do not faithfully predict acid consumptions or copper extractions that will be obtained in commercial heaps, as both will depend on leach cycle time, as well as various factors including care taken during heap construction and operation. Also, the original reports expressed copper recovery, which is misleading. It is more correct to use copper extraction. Copper recovery should apply to commercial cathode production and is always somewhat lower than the leaching extraction during column or heap leaching. For example, this difference can be attributed partially to the residual copper in the acidified raffinate (typically 5-10 percent of the copper in the PLS), which may not be completely recovered in subsequent leaching cycles.

There were only a few comparisons between fine and coarse column feeds, but they do not make a strong case for converting JCM from ROM to crushing and agglomeration. Nonetheless, a minus 6-inch fragment population probably does not represent ROM very faithfully, so it is quite possible that ROM underperforms a finer heap feed sufficiently to consider reactivating the crushing and screening plant.

24.13.2.1.2 Recent Column Leaching Tests on Transition Mineralization

Two column tests are underway on samples that contain transition zone copper minerals, a well as possible primary sulfides. The test conditions have been aggressive, beginning with curing by addition of the full acid requirement at a concentration of 200 gram/liter H_2SO_4, as practiced during Nord Resources' operation. Results indicate that extractions of oxide copper and transition minerals were complete and that primary sulfide mineralization dissolved to some extent. The high extractions of secondary sulfides and the more refractory oxides, e.g., chalcocite and lattice substitutions in clays, respectively, are not surprising, since the initial addition of strong acid could have oxidized and dissolved sufficient iron-bearing minerals to generate enough ferric ion to oxidize and dissolve chalcocite and other transition minerals.

Table 24-20: Column Leaching Tests on Transition Mineralization

Test and Sample Description				Assayed Heads (Ambient)			Time	Acid Consumption		Copper Extraction	
Column #	Size	Pit	Formation Name	%TCu	%ASCu	%CNCu	Leach Days	lb/ton	lb/lb	%TCu	%ASCu
13	-3”	Burro	Lower Abrigo	0.49	0.14	0.27	160	56	9.6	60	209
26	-3”	Burro	Lower Abrigo	0.44	0.14	0.21	232	52	10.6	56	175

The +100% ASCu leach extractions shown in the far right-hand column of Table 24-2 indicate that a significant fraction of slower-leaching copper sulfides that were not represented by the ASCu head assay have leached over the longer time period in the column. The recent tests were not monitored for oxidation potential, and curing with strong sulfuric acid, rather than dilute acid, may have been unrealistic, but the high extractions of oxide and transition copper minerals remain favorable indicators of leachability. Although acid curing is a common initial step in column simulation of heap leaching, it may usually be preferable to add a dilute solution, typically 10-20 gpl H_2SO_4. Applying a much stronger solution will increase the attack of gangue minerals, leading potentially to over-statement of gangue acid consumption. These interim column leaching results may change when final residues have been assayed and metallurgical balances calculated.

24.13.2.1.3 Predicted JCM Oxide Heap Leaching Performance

Lacking any recent laboratory testing and comparison of results with current heap performance, a meaningful prediction of near-term operating results is not a realistic expectation. However, it is not unreasonable to expect 95% average ASCu extraction and net acid consumptions in pounds per ton of leach pad material as follows: Upper Abrigo, 45; Middle Abrigo, 55; Lower Abrigo, 40; Bolsa Quartzite, 25; and Martin/Escabrosa, 70.

24.13.3 Heap Leaching of Sulfide Copper with Accelerated Pyrite Oxidation

The subject of this PEA is a copper sulfide and transitional mineralization resource that has been core drilled, but has not been fully evaluated metallurgically, except for the two incomplete column tests discussed in Section 24.13.2.1. However, it is likely that the very high ambient ASCu extraction rates from the prior Nord column tests shown in Table 24-19 are likely the result of dissolution (leaching) of transitional minerals. With the exception of transition sulfides, mainly chalcocite, the dominant primary sulfide mineral apparently is chalcopyrite.

Heap leaching of chalcocite/digenite ($Cu_2S/Cu_{1.8}S$) mineralization has been carried out successfully at many locations, including Inspiration, Miami, and Morenci in Arizona, Cananea in Sonora, and a number of mines in Chile. However, chalcopyrite ($CuFeS_2$) is very resistant to oxidation and subsequent leaching in dilute sulfuric acid raffinate, and its maximum extraction during conventional heap leaching is typically in the range 5-10 percent for run-of-mine (“ROM”) heap feed and slightly higher, commonly 10-15 percent, for mineralized rock that has been crushed and agglomerated. Metallurgists have struggled with this challenge for at least 60 years, and the prevalent explanation for such refractory behavior has been the formation of a diffusion-limiting sulfur coating on the surfaces of partially reacted chalcopyrite grains.

The universe of technical literature describes literally hundreds of investigations involving at least twenty lixiviants. A thorough summary of recent studies was published by Isabel Barton and Brent Hiskey at the University of Arizona in 2019 (Barton and Hiskey, 2019). Briefly, their survey considered a baseline of ferric sulfate and ferric chloride and compared those reagent systems with glycine, methanesulfonic acid, and various ionic liquids. As has been recognized for decades, ferric chloride provides about four times faster leaching kinetics than ferric sulfate, but reagent cost and the necessity for expensive corrosion-resistant materials of construction have always led process developers to rule out the chloride system as impractical and uneconomical.

Barton and Hiskey concluded that “Despite promising reaction rates and total extraction(s) by some of these novel extractants, comparison with baseline ferric chloride and ferric sulfate makes it doubtful that any will be commercially viable in the near term.”

Jetti Resources have reported “improved” ROM heap leaching of chalcopyrite at the Pinto Valley, AZ, operation of Capstone Resources, but initial chalcopyrite concentration may have been difficult to quantify. Jetti also published a paper describing laboratory evaluation of their proprietary reagent(s) (Rebolledo, Zarate, and Mora, 2019). This work was not considered by Barton and Hiskey, presumably because it had not yet been published.

It is noteworthy that none of the work cited by Barton and Hiskey was based on column simulation of heap leaching. Instead, nearly all of the investigations began with chalcopyrite concentrates or core samples that had been pulverized prior to testing. Only the Jetti paper discussed column testing on crushed, but not pulverized, samples. The greatest value of the Jetti paper may be revelation of the variability in leaching response of sulfide samples from 13 mines in Arizona and Chile. This issue will be addressed in more detail at the end of this Section.

The primary copper sulfide zone contains abundant pyrite with an approximate volumetric ratio of 3.5-to-1 pyrite-to-chalcopyrite. Assuming relative specific gravities of 5.01 and 4.35, this corresponds to a mass ratio of 4.03 to 1. This is unusually high and makes the JCM sulfide resource a good candidate for generation of ferric iron as an oxidant, a feature that encouraged Phelps Dodge to conduct commercial-scale accelerated ROM heap leaching at Bagdad, AZ, then at Morenci, AZ. These experiments were reported by Freeport after completion of a 3-year program at Morenci (Ekenes and Caro, 2013).

The experiment at Morenci involved two sub-cutoff grade waste dumps that had been created during accumulation of feed for the modernized flotation concentrator. Rather than separating the performances of the two dumps, it is simpler to discuss results obtained from the Lower Chase Creek stockpile, which received material that contained less acid-soluble copper with mostly chalcopyrite mineralization.

Construction and instrumentation of the so-called “engineered heaps” followed extensive column testwork and modelling at the PD/Freeport Process Technology Center in Safford, AZ. Results confirmed three fundamental and long-understood facts that (1) chalcopyrite leaches more rapidly at elevated temperatures; (2) aeration accelerates pyrite oxidation, and (3) bacteria also accelerate pyrite oxidation. As to the effect of temperature, the kinetic equation derived by Gustav Arrhenius in the late-1800s predicts that most chemical reaction rates double for every 10°C (18°F) temperature rise. (This is generally true of bacterial activity, as well, up to the limiting temperature that kills the particular species of bacteria.) Pilot-scale tests were then conducted at Bagdad, AZ, to establish process design criteria, which allowed engineering of the Morenci heaps.

A biological reactor was built to supplement the native bacterial population that apparently was similar to that found in other sulfide deposits. Encouragement of the most effective biological population was done with experimentally-determined aeration and acidity parameters. In this manner, native acidophilic bacteria, including *Acidithiobacillus thio-oxidans* and *Acidithiobacillus caldus* (both for oxidizing sulfur), *Acidithiobacillus ferro-oxidans* for oxidizing both sulfur and iron, and *Leptospirrilum ferro-oxidans* for oxidizing iron, were reinforced by other catalyzing mesophilic, moderately thermophilic, and extreme thermophilic bacteria, as well as non-bacterial *archea.*

Column simulation of heap leaching also disclosed that optimum conditions required a rest cycle, and the final design criteria embodied doubling the heap height to 20 meters from the usual 10 meters and reducing the customary solution application rate from a constant 5 L/t/day to applying that rate for seven leaching days, followed by a 14-day rest period. The overall average irrigation rate was therefore 1.67 L/t/day, which is equivalent to about 0.0014 GPM/ft^2, or roughly one-third of the industry standard application rate for a fully settled oxide copper heap. Column tests established a minimum first-cycle leaching time of 150 days. Three lifts were ultimately leached.

Although exploration drilling in the Morenci mine indicated an average pyrite content of 4 percent by weight, subsequent drilling of the waste stockpile found only about 2 percent. The discrepancy was not explained in the paper.

The initial footprint of the two heaps was 560,000 m^2 and air was supplied through perforated HDPE pipe by twelve blowers rated at 10,000 standard cubic meters per hour apiece. The air distribution grid was duplicated at the beginning of each new lift. Blower discharge pressure was not specified, but calculated flow resistance through 20 meters of compacted rock suggests approximately 10-15 psig.

The augmented heap leaching program provided the following conclusions:

- Heat from oxidation of the pyrite elevated the rock temperature to about 45-50°C from an ambient temperature of 15-20°C, and the PLS temperature was in the range of 30-35°C. Clearly, higher temperatures would accelerate kinetics. The limiting factor probably will be bacterial mortality.
- Aeration of each heap lift is necessary to supply oxygen to bacteria and pyrite.
- A leach/rest cycle was recommended by the Phelps Dodge Process Technology Center, so it probably is beneficial, and is consistent with experimental work published during the last three decades on investigations into heap leaching of secondary copper sulfides and refractory pyritic gold mineralization.
- Of direct importance to the JCM Sulfide project is the generation of sulfuric acid from oxidation of sulfides. Assuming 0.3 weight % Cu as chalcopyrite and a pyrite/chalcopyrite ratio of 3.5/1.00, the oxidation of those minerals will generate approximately 120 pounds of sulfuric acid per ton of heap feed. In the Morenci experiment, the net acid consumption dropped from 15 pounds per ton to zero within about 18 months of the initial application of forced air. The sulfide content of the JCM resource may be considerably higher than it was at Morenci, so zero acid generation could be achieved sooner. Creation of abundant free acid in the JCM sulfide heap would be a valuable supplement to the parallel oxide heap leaching operation.

Current information as provided by Rio Tinto/Nuton™:

During 2022, Rio Tinto and its wholly-owned subsidiary, Nuton LLC, have started commercialising their process for enhanced heap leaching of primary copper sulfides, especially, chalcopyrite. Nuton™ is an innovative new venture that aims to help grow Rio Tinto's copper business. At the core of Nuton™ is a portfolio of proprietary copper leach- related technologies and capabilities – a product of almost 30 years of research and development. The Nuton™ technologies offer the potential to economically unlock known low-grade copper sulphide resources, copper sulphides with high-amounts of deleterious elements (such as arsenic), and copper-bearing waste and tailings, and also achieve higher copper recoveries from primary copper sulphide material, allowing for a significantly improved copper production outcome. One of the associated key differentiators of Nuton™ is the potential to deliver leading environmental performance, including more efficient water usage, lower carbon emissions, and the ability to reclaim mine sites by reprocessing mine waste. The current work is based on studies that were originally started at Kennecott in Utah during the 1990s and have been further progressed at Rio Tinto's Australia based, Bundoora Technical Development Centre. It appears that the degree of augmentation may be increased by reliance on microbes that are unusually tolerant of very high temperatures. (These organisms may be similar to those found in geothermal hot springs.)

24.13.3.1 Predicted Heap Leaching Performance from JCM Transition and Sulfide Mineralization

It is the opinion of the QP for this section that the following copper heap leaching results in Table 24-21 can be obtained at Johnson Camp, given reasonable care and adherence to design operating conditions. This prediction recognizes the need for conservatism in this PEA, while assuming that significant near-term progress by major copper producers will be made on heap leaching of chalcopyrite.

Table 24-21: Predicted Copper Sulfide Heap Leaching Extractions

	ASCu	CNCu	S"Cu*
ROM without augmentation	60%	40%	10%
ROM with augmentation	80%	80%	30%
Crushed, agglomerated, & augmented	95%	95%	70%
* S"Cu denotes chalcopyrite.			

24.13.4 Risks

Implementation of new technology may be risky, but only if (1) proper care is not taken to establish basic chemistry on samples that faithfully represent the resource; (2) sufficient small-scale testing has not been done to define critical process design criteria; (3) commercialization is attempted before all pertinent data have been collected and analyzed; and (4) project owners are not fully engaged throughout the development program. These are all conditions that can be met if adequate time and finances are made available.

Technically, the risk probably is confined to (1) ensuring that bactericides, e.g., mercury, fluoride, and chloride, are either absent or controlled.; and (2) the variability of response to oxidation by chalcopyrite from different localities. This variability is evident from the following Table 24-22 published by Jetti Resources and affiliates (Rebolledo, Zarate, and Mora, 2019).

Table 24-22: Sulfide Copper Column Leaching Results

Sample Origin	Copper Grade, %TCu	Leaching Days	%TCu Recovery (Un-Catalyzed)	%TCu Recovery (Catalyzed)
CHILE	0.83	954	28	69
CHILE	2.23	667	33	50
USA	0.33	334	6	18
USA	0.19	537	7	29

Several conclusions can be drawn from these results provided by Jetti:

- Even with augmentation, copper sulfide leaching kinetics are much slower than for non-sulfides;
- Chalcopyrite has a highly variable response to oxidation, which is consistent with the findings of many solid-state physical chemists since about 1965, and this variability is partly due to crystal lattice substitutions by traces of di-valent and tri-valent elements such as arsenic, bismuth, selenium, tellurium, etc.
- Part of the apparent variability may result from uncertainties in mineralogical characterization of the samples themselves, since “sulfide copper” minerals include chalcopyrite, bornite, enargite, chalcocite, digenite, and various complex sulfosalts. Also, there are many cyanide-soluble minerals.

24.13.5 Opportunities

Oxidation and leaching of pyrite and copper sulfides, especially chalcopyrite, will generate sulfuric acid, so these reactions will cause the net acid requirement for heap leaching to diminish during the first year of operation and to quickly drop below zero. Net generation should then rise to approximately 120 lb/ton during the second year, as demonstrated during the Phelps Dodge experiment. This will create a necessary “acid sink” for the sulfide leach, while eliminating part, or possibly all, of the acid addition requirement for the on-going oxide heap leach.

24.13.6 Recommendations for Future Process Development

Although augmentation of chalcopyrite oxidation and leaching with the aid of biological oxidation of pyrite, while supplying low-pressure aeration, now appears to be within reach after decades of effort by hundreds of metallurgists and chemists, the need for careful testwork on representative samples of the resource must be recognized. Arrangements are being made for column testing under the supervision of Rio Tinto/Nuton™. The Nuton™ evaluation will include quantitative mineralogical characterization by QEMSCAN. The response of various minerals to leaching in dilute solutions of sodium cyanide and sulfuric acid has been summarized by Bazzanella (Bazzanella, 2000), who cited earlier publications by Hedley and Tabachnick (American Cyanamid, 1950) and Bhappu (SME, 1995). A few examples based on tests at room temperature are as follows in Table 24-23.

Table 24-23: Approximate Solubilities of Copper Minerals in Dilute Aqueous Sodium Cyanide and Sulfuric Acid Solutions

Copper Mineral	Solubility in Dilute NaCN	Solubility in Dilute H_2SO_4
Azurite	95	100
Malachite	90	100
Chrysocolla	12	100
Chalcocite	90	3
Bornite	70	2
Native Copper	90	5
Chalcopyrite	6	2

In the event that heap leaching of the JCM sulfide resource does not prove to be feasible, a logical alternative is conventional grinding and flotation. The original Johnson Camp Mining District produced copper, silver, gold, tungsten, lead, and zinc. Molybdenum was practically unknown at the time and a market was not available. However, it is possible that silver, gold, and molybdenum are present in economically recoverable concentrations, so a limited amount of assaying, followed by a modest laboratory flotation program, is recommended.

For example, potential increased revenues per ton milled with resource grades of 1.0 oz/ton Ag, 0.02 oz/ton Au, and 0.015% Mo, and with respective recoveries of 90%, 90%, and 50%, would total approximately US$60. Obviously, the added revenues would be offset by increased CAPEX and OPEX for fine grinding, flotation, concentrate dewatering, and tailings management.

24.14 MINERAL RESOURCE ESTIMATES

24.14.1 Introduction

The mineral resource estimation for the Johnson Camp Mine project was completed for a PEA adhering to NI 43-101 standards. The modeling and estimation of the copper mineral resources were completed in September 2022 under the supervision of Jeffrey Bickel. The Effective Date of the mineral resource estimate is July 13, 2022. Mr. Bickel is independent of Excelsior by the definitions and criteria set forth in NI 43-101 as of the Effective Date of this report. There is no affiliation between Mr. Bickel and Excelsior except that of independent consultant/client relationships. Mr. Bickel is not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the Johnson Camp Mine mineral resources as of the date of this report. No mineral reserves have been estimated for the Johnson Camp Mine Project.

The Johnson Camp Mine mineral resources are classified in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories in accordance with the "CIM Definition Standards - For Mineral

Resources and Mineral Reserves" (2014) and therefore NI 43-101. CIM mineral resource definitions are given below, with CIM's explanatory text shown in italics:

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.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated, or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing, and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

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.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate *sampling techniques from locations such as outcrops, trenches, pits, workings, and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.*

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

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.

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

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.

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

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.

The mineral resources are reported herein at cut-offs that are reasonable for deposits of this nature given anticipated mining methods and plant processing costs, while also considering economic conditions, because of the regulatory requirements that a resource exists "in such form and quantity and of such a grade or quality that it has reasonable prospects for eventual economic extraction".

24.14.2 Data

The Johnson Camp Mine copper resources were modeled and estimated using information provided by Excelsior under Mr. Bickel's supervision. The information is derived from historical core holes drilled by Cyprus Mining, Arimetco, Summo USA Corp., and Nord Resources Corp. Excelsior completed 44 diamond drill holes in the Burro Pit area in 2022. The drill hole database also includes analyses performed by Excelsior on the historical core. This data, as well as digital topography of the project area, were provided to RESPEC by Excelsior in a digital database in Arizona State Plane, East Zone coordinates in US Survey feet using the NAD27 datum.

Modeling of the Johnson Camp Mine mineral domains and estimation of the mineral resources were performed using GEOVIA Surpac mining software as well as proprietary software developed at RESPEC, which was acquired by RESPEC. Lithologic and oxidation models were built in Leapfrog. The oxidation model was used to constrain the estimation of total copper ("CuT"), the acid-soluble ("CuAs") ratio (CuAs/CuT), and the cyanide-soluble ("CuCN") copper ratio (CuCN/CuT). The Johnson Camp Mine resource block model extents and dimensions are provided in Table 24-24.

Table 24-24: Block Model Extents and Dimensions

In Feet	X	Y	Z
Min Coordinates	532,624	397,419.6	4,000
Max Coordinates	536,624	405,019.6	6,000
Block Size	20	20	20
Rotation	0	-54	0

24.14.3 Deposit Geology Pertinent to Resource Block Model

The copper mineralization at the Johnson Camp Mine occurs primarily in lower Paleozoic and upper Precambrian sedimentary units and upper Precambrian intrusive diabase sills. The primary controls on mineralization are (i) favorable stratigraphic units within geologic formations; (ii) diabase sills; (iii) the intersection of favorable units with important structures; and (iv) oxidation of primary mineralization. Geologic factors critical to the grade domain modeling of Johnson Camp copper mineralization therefore include lithology, structure, and oxidation.

24.14.4 Geologic and Oxidation Models

RESPEC created three-dimensional geologic and oxidation models using Leapfrog software. The geologic model interpretations were mainly based on previous models generated by Nord Resources Corp and were updated incorporating new drilling conducted by Excelsior. The geologic interpretations included solidified wireframes of geologic formations and three-dimensional fault surfaces. The oxidation model interpretations were based on the acid-soluble and cyanide-soluble to total copper ratios primarily projected along bedding following the geologic model. Four oxidation groups were modeled including oxide, mixed, transition, and sulfide. The criteria for the oxidation groups are summarized below in Table 24-25.

Table 24-25: Oxidation Group Modeling Criteria

Oxidation Group	Criteria
Oxide	>60% CuAs/CuT ratio
Mixed	<60% and ~>10% CuAs/CuT ratio and CuCN/CuT ratio ~<10%
Transition	<60% and ~>10% CuAs/CuT ratio and CuCN/CuT ratio ~>10%
Sulfide	~<10% CuAs/CuT ratio and ~<10% CuCN/CuT ratio

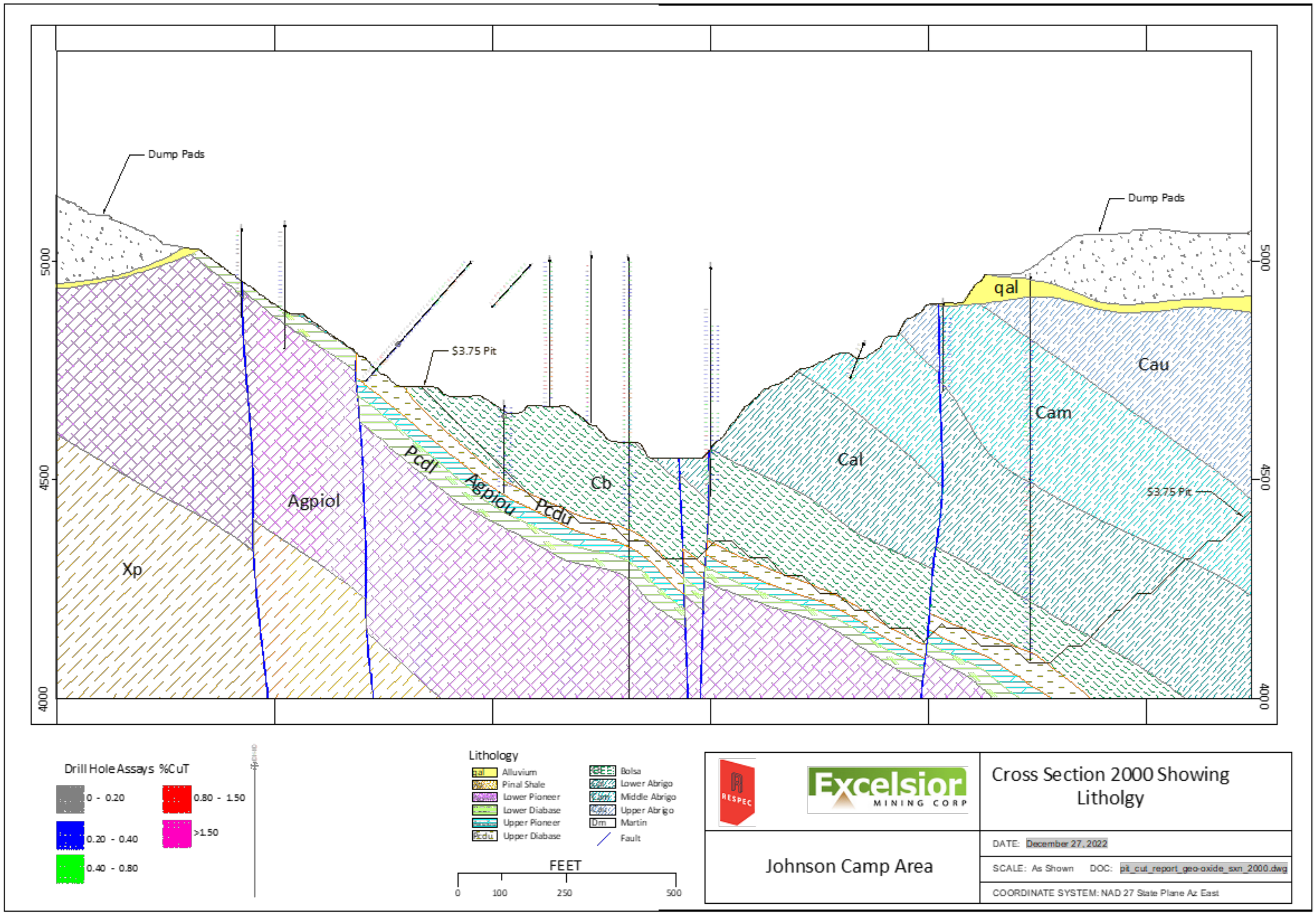


Figure 24-35: Geologic Cross Section with Geologic Model Burro Pit Area

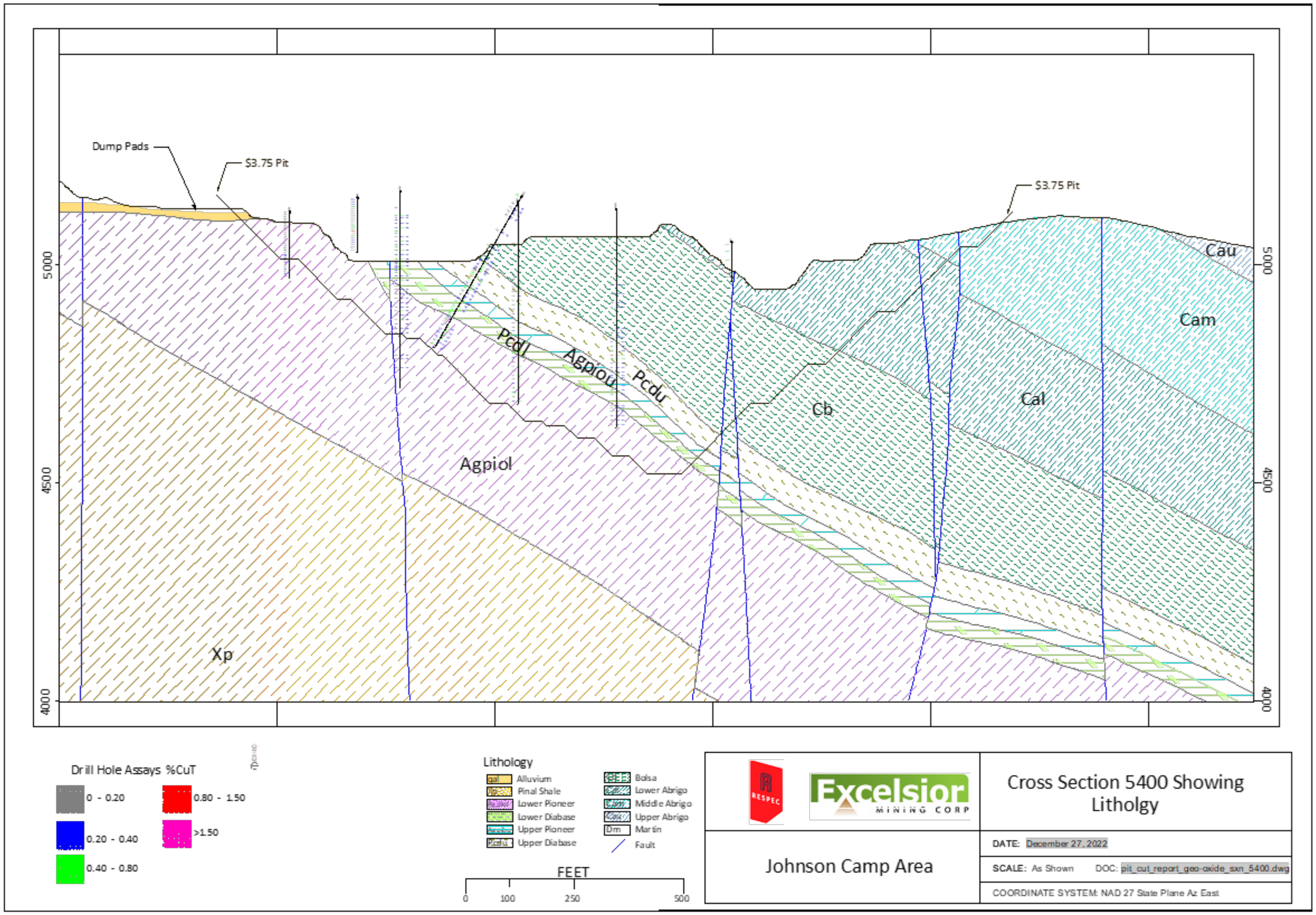
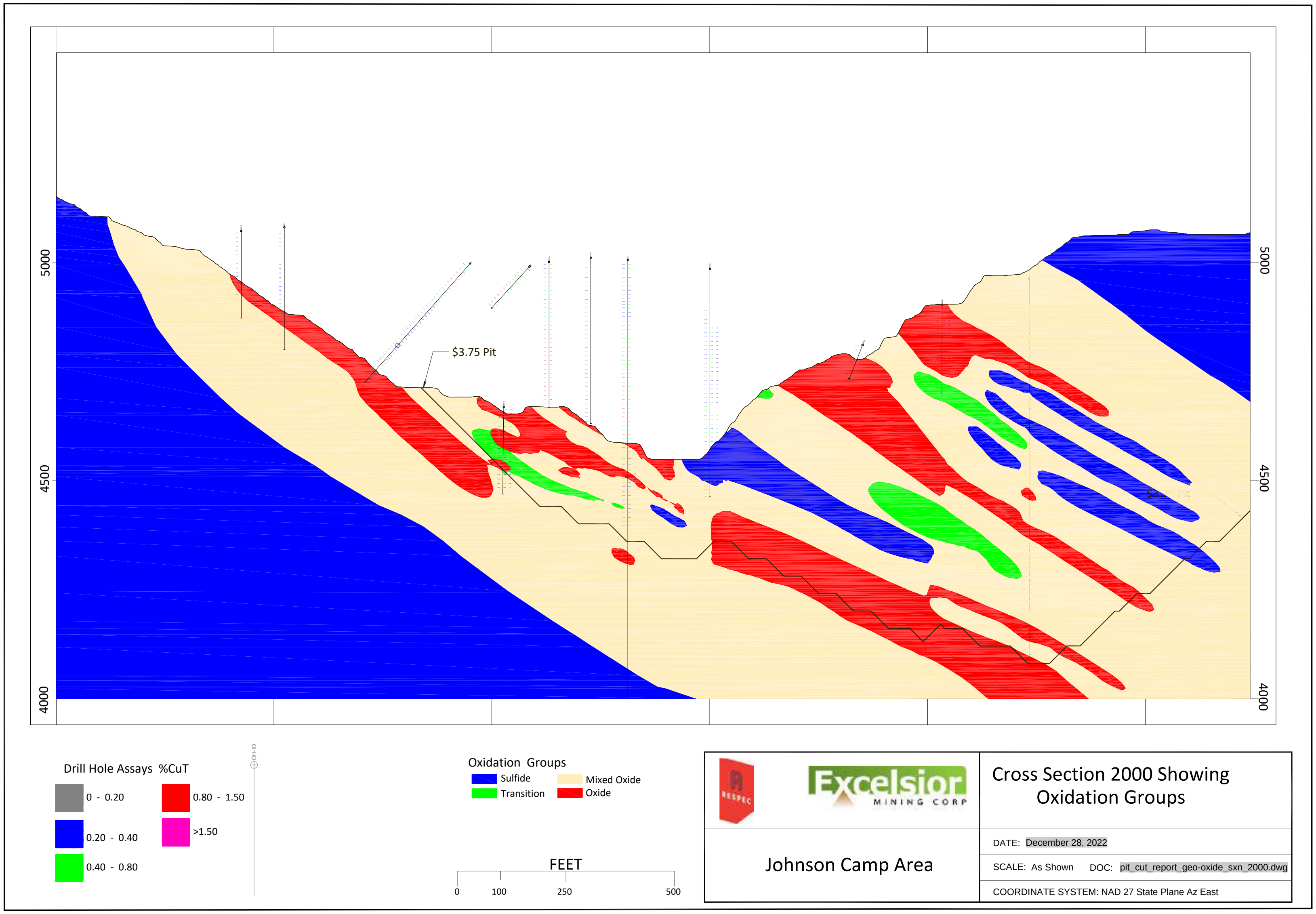


Figure 24-36: Geologic Cross Section with Geologic Model Copper Chief Pit Area



Cross Section 2000 Showing Oxidation Groups

Johnson Camp Area

DATE: December 28, 2022

SCALE: As Shown DOC: pit_cut_report_geo-oxide_sxn_2000.dwg

COORDINATE SYSTEM: NAD 27 State Plane Az East

Figure 24-37: Geologic Cross Section with Oxidation Model Burro Pit Area

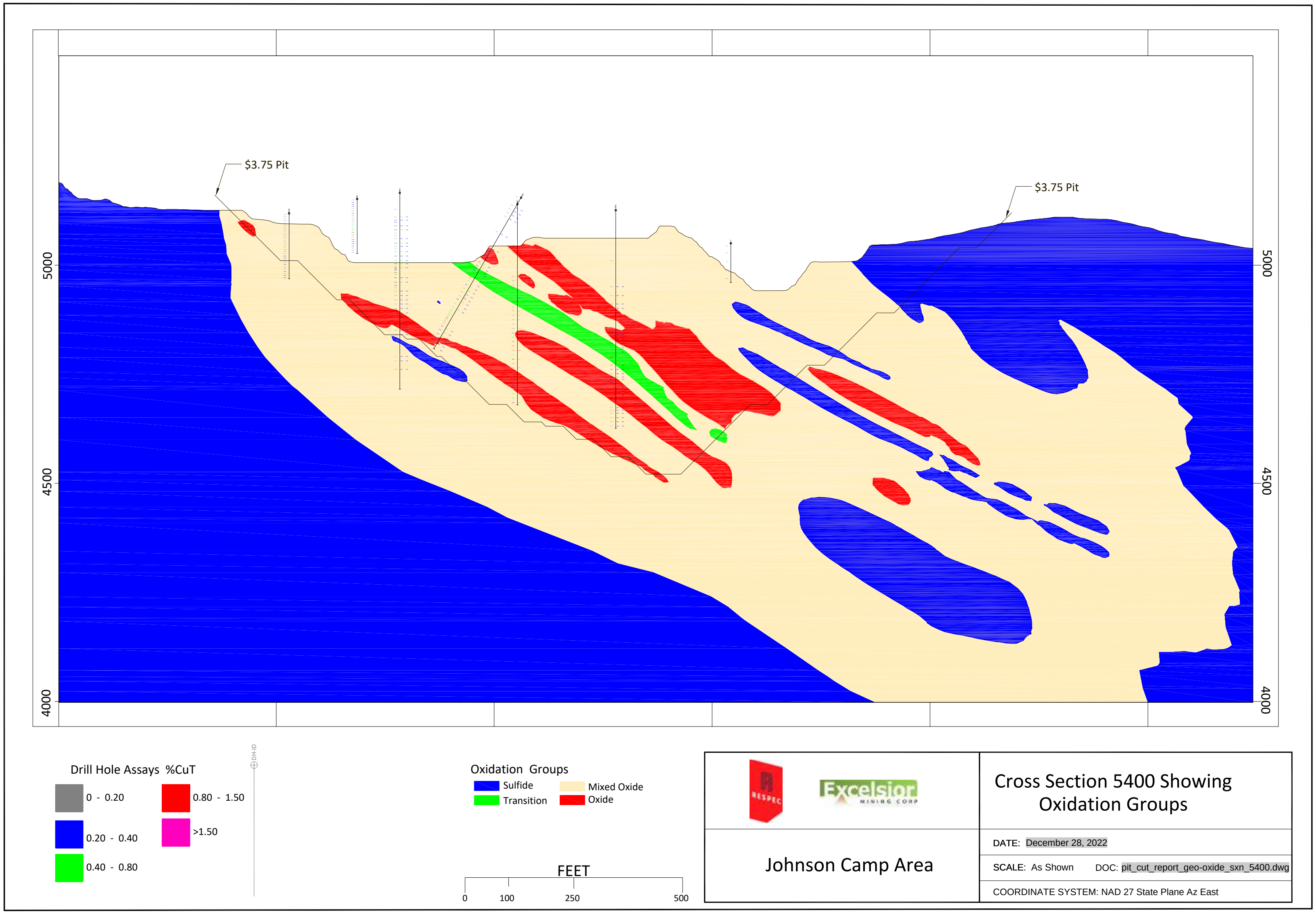


Figure 24-38: Geologic Cross Section with Oxidation Model Copper Chief Pit Area

24.14.5 Density

Density values from previous models by Nord Resources Corp were used to code the RESPEC model. The values were based on bulk test work by previous operators. The density values correspond to geologic formations, as summarized in Table 24-26.

Table 24-26: Average SG and Tonnage Factors by Copper Domain

Lithologic Unit	Lithologic Code	Tonnage Factor – Ft^3 per Ton
Martin Formation	1	12.52
Upper Abrigo	2	12.46
Middle Abrigo	3	12.46
Lower Abrigo	4	12.46
Bolsa Quartzite	5	12.62
Upper Diabase	6	11.33
Upper Pioneer Shale	7	12.00
Lower Diabase	8	11.33
Lower Pioneer Shale	9	12.00
Pinal Schist	10	12.52
OB (Alluvium)	11	16.27
Dump	12	16.27

24.14.6 Mineral Domain Modeling

A mineral domain encompasses a volume of rock that is ideally characterized by a single, natural population of metal grades that occurs within a specific geologic environment. Mineral domains were modeled by RESPEC to respect the lithologic and structural interpretations of the deposit. Following statistical evaluation of the drillhole data, mineral domains were modeled on cross sections for total copper. Low-, mid-, and high-grade domains were modeled for total copper and were numbered 100, 200, and 300, respectively. Material outside the 100, 200, and 300 domains was assigned to the 0 domain. These grade domains were based on assay data populations. Soluble copper and cyanide-soluble domains were not explicitly modeled; instead, the soluble copper to total copper ratio and the cyanide-soluble to total copper ratio was used in the block model to calculate the grade for soluble-copper and cyanide-soluble copper, described in detail below.

24.14.6.1 Copper Domain Modeling

In order to define the mineral domains at the Johnson Camp Mine, the natural populations of total copper grades were identified on population-distribution graphs for all drillhole samples in the deposit area. The analysis led to identification of distinct populations. Ideally each of these populations can be correlated with geologic characteristics which then can be used in conjunction with the grade populations to interpret the bounds of each of the mineral domains. The approximate grade ranges of the domains are listed in Table 24-27.

Table 24-27: Grade Domain Ranges

Domain	Total Copper (%)
100	~0.025 to ~0.15
200	~0.15 to 0.7
300	> ~0.7

Using these grade populations in conjunction with lithologic and structural interpretations, grade domains were independently modeled within the Johnson Camp Mine deposit by interpreting mineral domain polygons on a set of

100 ft-spaced cross sections oriented along the approximate direction of dip (036° azimuth). Representative cross sections showing the copper mineral domains in the Burro and Copper Chief areas are shown in Figure 24-39 and Figure 24-40, respectively.

The final cross-sectional mineral-domain polygons were projected horizontally to the drill data in each sectional window, and these three-dimensional polygons were then sliced vertically along 20-foot planes that are orthogonal to the cross sections. These slices, along with similar slices lithologic and structural surfaces, were used to guide the final rectification of the copper mineral domains on the long sections. The 20-foot long-section plane locations coincide with resource-model block centroids along y-axis columns within the rotated model. Long sections were chosen over level plans for rectification purposes due to the generally gently dipping nature of the mineralization. The product of this work is a set of 20-foot-spaced long sectional copper domain polygons that span the full extents of the drilled mineralization.

Figure 24-41 is a representative long section long section showing the mineral domain polygons that were used to code the model.

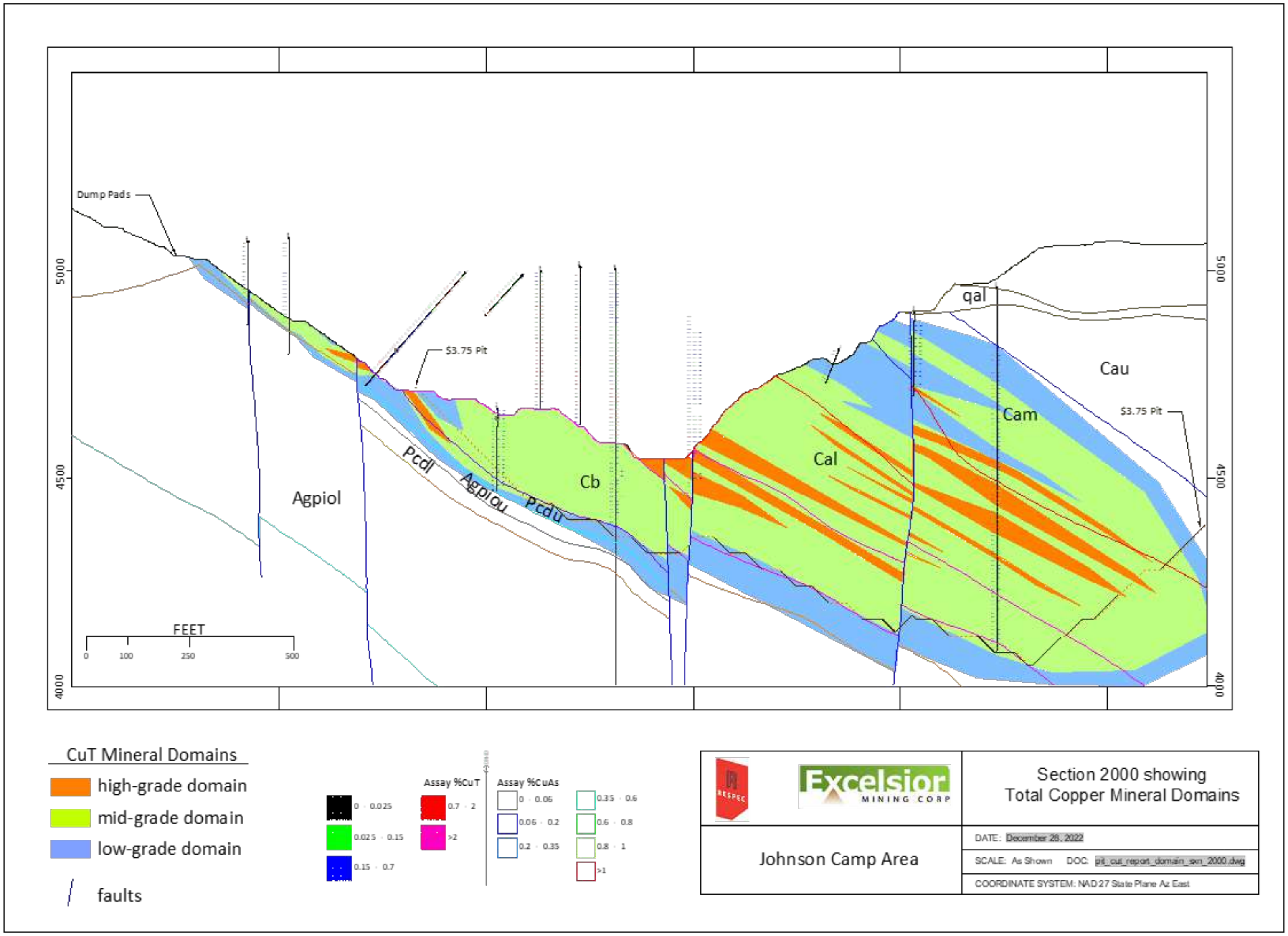


Figure 24-39: Geologic Cross Section with Copper Domains Burro Area Mineralization and $3.75/lb Cu Pit Shells

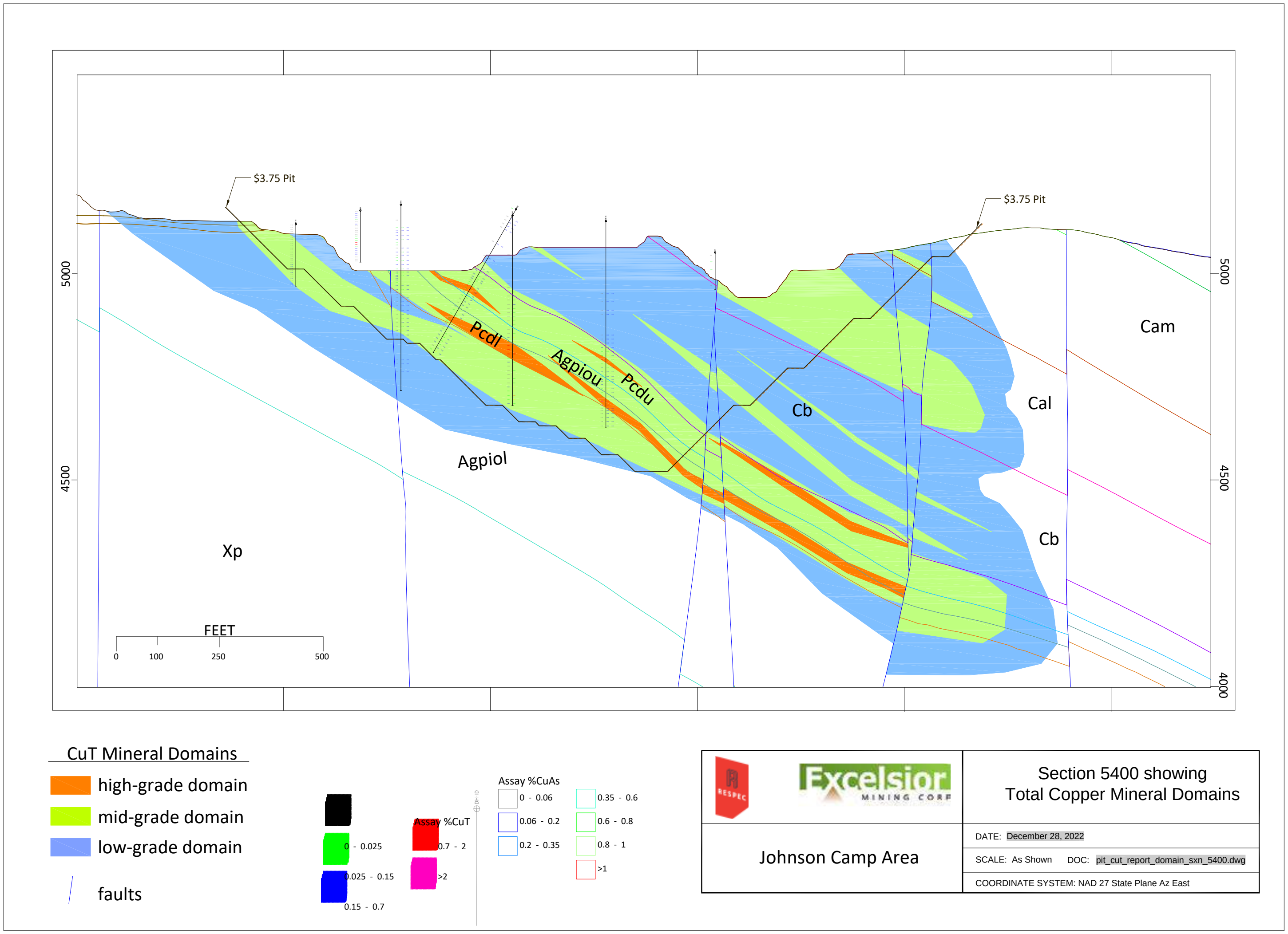


Figure 24-40: Geologic Cross Section with Copper Domains Copper Chief Area Mineralization and $3.75/lb Cu Pit Shells (December 2022)

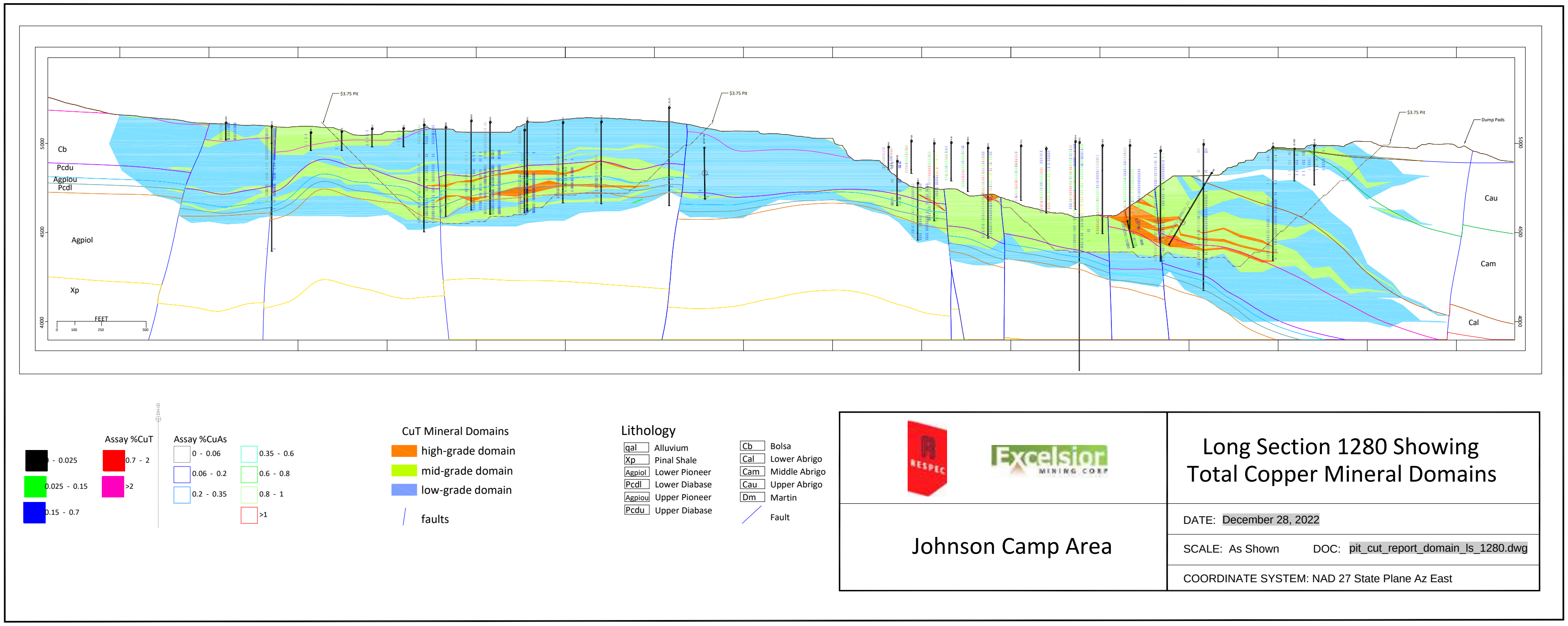


Figure 24-41: Representative Long-Section Through Johnson Camp Mineralization and $3.75/lb Cu Pit Shells (December 2022)

24.14.6.2 Copper Ratios

There are two methods for estimating acid-soluble copper or cyanide-soluble copper: directly, using composites of the soluble copper or cyanide soluble copper analyses from the database; or indirectly, by estimating the acid-soluble copper to total copper ratios (“CuAs Ratio”) or the cyanide-soluble copper to total copper ratios (“CuCN Ratio”). In the latter case, the ratios are determined for each drill interval that has both acid-soluble- and total copper analyses or cyanide-soluble and total copper analyses, and these ratios are then coded, composited, and used to estimate the ratios into the model blocks. The estimated acid-soluble copper or cyanide-soluble copper model values are then derived by multiplying the estimated ratio by the estimated copper value in each block.

Remobilization of supergene copper is not ubiquitous at Johnson Camp. It is evident locally, especially in the certain geologic units at the Copper Chief pit where it can be found as exotic accumulations on fractures in the Bolsa Quartzite and diabase sills [Curtis Associates, 2013]. Where remobilization does exist, the distance of transport is typically inconsequential, and the remobilized copper has the same geologic controls on mineralization as the primary mineralization (along favorable stratigraphic units). Based on geological and statistical analyses, the author has concluded that the oxidation of copper minerals at Johnson Camp is strongly stratigraphically controlled and does not adhere to more classical oxidation profile that is influenced primarily by elevation as observed in other deposits in the district. As such, the modeled oxidation groups to control the estimate of various copper ratios are mostly influenced by stratigraphy, with elevation being a secondary control on oxidation.

The estimation of ratios for soluble-copper or cyanide-soluble copper can negate possible biases created by intervals that were selectively analyzed for total copper but not acid-soluble copper or cyanide-soluble copper. In the Johnson Camp database, 68% of the total copper samples have acid-soluble copper analyses and 29% of the total copper samples have cyanide-soluble copper analyses.

RESPEC used estimated ratios to code the Johnson Camp block model with acid-soluble copper and cyanide-soluble copper values. The ratio estimation was confined to blocks with estimated total copper values. The ratios of the blocks coded to the oxidation model and to the geologic units were estimated independently.

The sulfide copper (“CuSu”) grade was calculated by taking the total copper grade and subtracting the estimated acid-soluble copper grade and estimated cyanide-soluble copper grade. The remaining value is considered the residual sulfide copper grade. However, residual copper in the oxide zone, especially near-surface, can occasionally exist in non-sulfide minerals. Because of this possibility, a factor was applied within the modeled oxide and mixed groups for elevations between 4500 and 5000 feet. To apply the factor, the residual copper grade was multiplied by 0 at 4950 to 5000 feet. For each 50 feet descending from 5000 feet, 10% was added to the multiplication factor. Table 24-28 below shows how the factor was applied. The author considers this a conservative approach to copper that is hosted in insoluble minerals near surface.

Table 24-28: Residual Sulfide Calculation

Elevation (feet)	Residual Sulfide Formula
> 4,950	Residual Sulfide =–(CuT -–CuAs - CuCN)*0
4,900 to 4,950	Residual Sulfide =–(CuT -–CuAs - CuCN)*10%
4,850 to 4,900	Residual Sulfide =–(CuT -–CuAs - CuCN)*20%
4,800 to 4,850	Residual Sulfide =–(CuT -–CuAs - CuCN)*30%
4,750 to 4,800	Residual Sulfide =–(CuT -–CuAs - CuCN)*40%
4,700 to 4,750	Residual Sulfide =–(CuT -–CuAs - CuCN)*50%
4,650 to 4,700	Residual Sulfide =–(CuT -–CuAs - CuCN)*60%
4,600 to 4,650	Residual Sulfide =–(CuT -–CuAs - CuCN)*70%
4,550 to 4,600	Residual Sulfide =–(CuT -–CuAs - CuCN)*80%
4,500 to 4,550	Residual Sulfide =–(CuT -–CuAs - CuCN)*90%

24.14.7 Assay Coding, Capping, and Compositing

The cross-sectional mineral-domain polygons described in Section 24.14.6 were used to code drillhole assay intervals to their respective copper mineral domains. The polygons were coded 10 feet either side of the section plane from which they were created. Acid-Soluble copper and cyanide-soluble copper ratios were coded to the oxidation model and the modeled geologic units. Assay caps were determined by domain to identify high-grade outliers that might be appropriate for capping. Visual reviews of the spatial relationships concerning possible outliers and their potential impacts during grade interpolation were also considered in the assay cap definitions. Table 24-29 provides the caps used by each domain for total copper.

Table 24-29: Grade Caps

Copper	Cap (% CuT)
0	0.3
100	0.8
200	1.5
300	3

Descriptive statistics of the coded assays of capped and uncapped copper analyses are provided in Table 24-30 and Table 24-31. All soluble copper ratios were capped at 1. Soluble copper ratio statistics are provided in Table 24-31.

Table 24-30: Coded Total Copper (CuT) Assay Statistics

Domain	Assays	Count	Mean	Median	Std. Dev.	CV	Min.	Max.
0	CuT	896	0.03	0.01	0.07	2.38	0	0.78
	CuT Cap	896	0.03	0.01	0.05	1.85	0	0.3
100	CuT	5,886	0.08	0.07	0.06	0.78	0	1.76
	CuT Cap	5,886	0.08	0.07	0.06	0.75	0	0.8
200	CuT	6,316	0.30	0.26	0.17	0.57	0.01	4
	CuT Cap	6,316	0.30	0.26	0.16	0.53	0.01	1.5
300	CuT	977	0.97	0.87	0.43	0.45	0.06	4.21
	CuT Cap	977	0.96	0.87	0.41	0.42	0.06	3
100+200+300	CuT	13,179	0.26	0.16	0.29	1.13	0	4.21
	CuT Cap	13,179	0.26	0.16	0.29	1.1	0	3

Table 24-31: Coded Acid-Soluble (CuAS) Copper Ratio and Cyanide-Soluble (CuCN) Ratio Statistics (Capped)

Domain	Assays	Count	Mean	Median	Std. Dev.	CV	Min.	Max.
Oxide	CuAs Ratio Cap	2,065	0.73	0.73	0.15	0.2	0.05	1
Sulfide	CuAs Ratio Cap	646	0.19	0.14	0.21	1.09	0	1
Transition	CuAs Ratio Cap	1,098	0.33	0.31	0.17	0.53	0.02	1
Mixed	CuAs Ratio Cap	4,944	0.37	0.37	0.16	0.43	0	1
Oxide	CuCN Ratio Cap	1,158	0.11	0.05	0.19	1.75	0	1
Sulfide	CuCN Ratio Cap	421	0.28	0.17	0.28	1.01	0	1
Transition	CuCN Ratio Cap	1,039	0.29	0.2	0.22	0.75	0.02	1
Mixed	CuCN Ratio Cap	3,240	0.21	0.08	0.3	1.43	0	1

The capped assays were composited at 10-foot down-hole intervals, respecting the mineral domain boundaries. Descriptive statistics of the composites for each metal are given in Table 24-32.

Table 24-32: Composite Statistics

Total Copper Composites by Domain								
Domain	Hole Count	Comp.	Mean	Median	Std. Dev.	CV	Min.	Max.
0	99	714	0.03	0.01	0.05	1.80	0.00	0.30
100	333	4,831	0.08	0.07	0.06	0.72	0.00	0.80
200	343	5,453	0.30	0.27	0.16	0.52	0.01	1.50
300	175	898	0.96	0.88	0.39	0.41	0.06	3.00
all	356	11,182	0.26	0.17	0.28	1.09	0.00	3.00
Soluble Copper Ratio and Cyanide Soluble Ratio Composites by Oxidation								
Domain	Hole Count	Comp.	Mean	Median	Std. Dev.	CV	Min.	Max.
oxide	157	1,894	0.73	0.73	0.14	0.19	0.05	1.00
sulfide	71	476	0.16	0.14	0.13	0.79	0.00	0.55
transition	93	909	0.32	0.31	0.16	0.49	0.02	1.00
mixed	210	4,237	0.37	0.37	0.15	0.40	0.00	1.00
oxide	103	981	0.07	0.05	0.07	1.00	0.00	0.78
sulfide	44	248	0.20	0.14	0.14	0.73	0.00	0.75
transition	88	849	0.28	0.20	0.19	0.70	0.02	1.00
mixed	146	2,287	0.11	0.07	0.10	0.96	0.00	1.00

24.14.7.1 Variography

Using all total copper composites, variogram ranges of 500 feet along the strike of the sedimentary units (305°) and 300 feet in the dip direction (-30° at 125°) were obtained. These ranges were used as a check for reasonableness for the search ellipsoids used in the estimate. Additionally, a kriged estimate was performed purely for the purposes of statistical checking and the variography was used to define the kriging parameters in the grade interpolations.

24.14.8 Block Model Coding

The 100-foot-spaced cross-sectional mineral-domain polygons were used to code 20 x 20 x 20 (x, y, z)-foot blocks that comprise a digital model rotated to a bearing of 306°. The percentage volume of each mineral domain, as coded directly by the cross-sections, is stored within each block as a “partial percentage”, as is the partial percentage of the block that lies outside of the modeled metal domains (domain 0). In other words, each block stores the partial percentage of each of the four domains for total copper. The oxidation model was used to domain the acid-soluble copper ratio and cyanide-soluble copper ratio estimates.

The Johnson Camp geologic formations were coded to each block to a single lithology on a ‘majority wins’ basis. The Johnson Camp digital topographic surface was used to code the block model on a partial percentage basis. The specific gravity values shown in Table 24-26 were assigned to the model blocks based on the geologic formation codes in each model block.

The mineralization has a variety of orientations. Wireframe solids were therefore created to encompass model areas with similar mineral domain orientations, and the solids were used to code the model blocks to these areas on a block-in/block-out basis. This coding was then used to control search-ellipse orientations during copper interpolations. The orientations given in Table 24-33 were applied to all domains for total copper, acid-soluble copper and cyanide-soluble ratios.

Table 24-33: Estimation Area Orientations

Area	Bearing	Plunge	Tilt
301	306	0	-30
302	306	0	-40

24.14.9 Grade Interpolation

Total copper grades, as well as acid-soluble copper and cyanide-soluble copper ratios, were interpolated using inverse distance, ordinary kriging, and nearest-neighbor methods. The mineral resources reported herein were estimated by inverse distance interpolation as this method led to results that most appropriately respected the drill data and geology of the deposit. This is particularly true with respect to the estimation of the lowest-grade areas in the model, where potential overestimation of volumes could materially impact the resource estimation at grades close to potential open-pit mining cut-offs. The nearest-neighbor estimation was completed for the purposes of statistical checking of the various estimation iterations. The parameters applied to the grade estimations at Johnson Camp Mine are summarized in Table 24-34.

Table 24-34: Estimation Parameters

Estimation	Search Ranges (feet)			Composite Constraints		
Pass	Major	Semi-Major	Minor	Min	Max	Max/Hole
Pass 1	350	350	175	2	15	3
Pass 2	650	650	325	2	15	3
Pass 3	1000	1000	1000	1	15	3

Grade interpolations were completed using 10-foot composites. The estimation passes were performed independently for each of the mineral domains, so that only composites coded to a particular domain were used to estimate grade into blocks coded to that domain. Blocks coded as having partial percentages of more than one domain had multiple grade interpolations, one for each domain coded into the block. The estimated grades for each of the metal domains 0, 100, 200, and 300 coded to a block were coupled with the coded partial percentages of those domains to enable the calculation of a single volume-weighted grade of each of the metal species for each block. These resource block grades are therefore diluted to the full block volumes using this methodology.

24.14.10 Mineral Resources

The Johnson Camp Mine project mineral resources have been estimated to reflect potential open-pit extraction and potential processing by heap leaching. To meet the requirement of the resources having reasonable prospects for eventual economic extraction, a pit optimization was completed in 2022 using the parameters summarized in Table 24-35.

Table 24-35: Pit Optimization Parameters

Parameter	Value	Unit
Copper Price	$3.75	$/lb sold
Contract Mine Cost	$2.30	$/ton Mined
Technical Services	$0.25	$/ton Processed
Heap Management	$0.30	$/ton Processed
Heap Capital Cost	$0.80	$/ton Processed
Crushing Cost	$1.10	$/ton Processed
G&A Cost	$0.05	$/lb Cu Produced
SX-EW Cost	$0.25	$/lb Cu Produced
Recovery	95%	Acid Soluble Cu
Recovery	95%	Cyanide Soluble Cu
Recovery	70%	Sulfide Cu
Royalty	17.90%	NSR
Acid Cost	$150	$/ ton
Acid Consumption and Costs by Formation		
Formation	Acid Cons. lb/ton	Acid Cost $/ton Processed
Pioneer Shale	20	$1.20
Bolsa Quartzite	25	$1.50
Diabase	30	$1.80
Middle Abrigo	55	$3.30
Upper Abrigo	45	$2.70
Lower Abrigo	40	$2.40
Martin	70	$4.20

The pit shells created using these optimization parameters were used to constrain the project resources for comparison purposes. An exclusion line (Figure 24-42) was used to limit the pit optimization on the west side of the Burro Pit to prevent the pit optimizations from encroaching on the process plant and the leach pad. The in-pit resources were further constrained by the application of a cut-off of 0.1% Cu to all model blocks within the optimized pits.

Figure 24-42: Map of Johnson Camp Pit Optimization Exclusion Line

The Johnson Camp Mine project mineral resource comparison is summarized in Table 24-36. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 24-36: Johnson Camp Mineral Resources
(0.1% CuT cut-off)

Classification	Tons	% Cu	% CuAs	% CuCN	% CuSu	lbs CuT	lbs CuAs	lbs CuCN	lbs CuSu
Measured	20,771,000	0.31	0.13	0.05	0.09	127,545,000	54,762,000	22,564,000	37,551,000
Indicated	87,166,000	0.32	0.13	0.05	0.11	550,118,000	218,657,000	82,380,000	184,432,000
Inferred	50,998,000	0.32	0.12	0.04	0.12	322,656,000	119,614,000	45,377,000	122,781,000

1. The Effective Date of the mineral resources is July 13, 2022.
2. The project mineral resources are shown in bold and are comprised of all model blocks at a 0.1 % CuT cut-off that lie within optimized resource pits.
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
4. The estimate of mineral resources may be materially affected by geology, environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
5. Rounding as required by reporting guidelines may result in apparent discrepancies between tons, grade, and contained metal content.

24.14.11 Mineral Resource Classification

The Johnson Camp mineral resources were classified as Inferred before Excelsior's 2022 drilling campaign. New drilling has upgraded a portion of the Burro Pit area to Measured and Indicated, based primarily on the 2022 drilling campaign's confirmation of historical data. Measured and Indicated were classified using two drill holes with at least one sample from one Excelsior hole. Distances chosen for Measured and Indicated were influenced by geological confidence as well as ranges identified in variography. Measured is reported at a distance of 100 feet from two samples with one being from an Excelsior hole and Indicated is reported at a distance of 350 feet with one being from an Excelsior hole. Additional Measured and Indicated resources were classified based on the positive results from the data validation section for the Arimetco drilling campaigns. A similar classification scheme was used as the Excelsior data with half the distances for Measured and Indicated. Measured is reported at a distance of 50 feet from two samples with one being from an Arimetco hole and Indicated is reported at a distance of 175 feet with one being from an Arimetco hole. A summary of the classification parameters is in Table 24-37.

Table 24-37: Resource Classification Parameters

Classification	Criteria
Measured	Minimum of 2 holes contributing composites, including at least 1 drilled by Excelsior, that lie within an average distance of 100 feet from the block or minimum of 2 holes contributing composites, including at least 1 drilled by Arimetco, that lie within an average distance of 50 feet from the block
Indicated	Minimum of 2 holes contributing composites, including at least 1 drilled by Excelsior, that lie within an average distance of 350 feet from the block or minimum of 2 holes contributing composites, including at least 1 drilled by Arimetco, that lie within an average distance of 175 feet from the block
Inferred	all other blocks that meet the resource constraints

The Johnson Camp in-pit resources cover an aerial extent of over 1.2 miles along strike with two distinct spatial areas: The Burro Pit and the Copper Chief Pit. Figure 24-43 through Figure 24-46 are representative cross sections through the block model along section line 2000 in the Burro Pit Area. Figure 24-47 through Figure 24-50 are representative cross sections through the block model along section line 5400 in the Copper Chief Pit area.

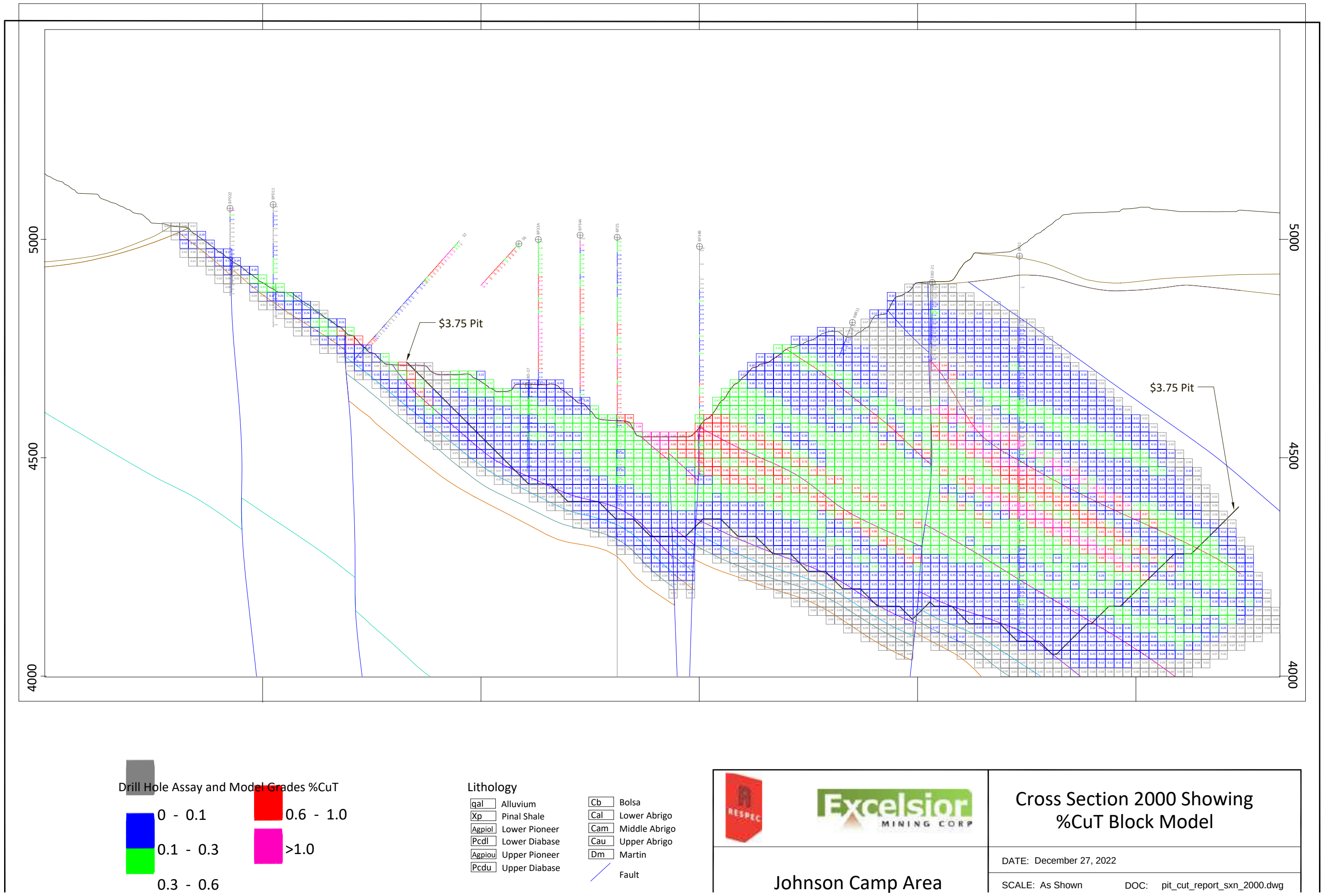


Figure 24-43: Geologic Cross Section 2000 with Total Copper ("CuT") Block Model Grades

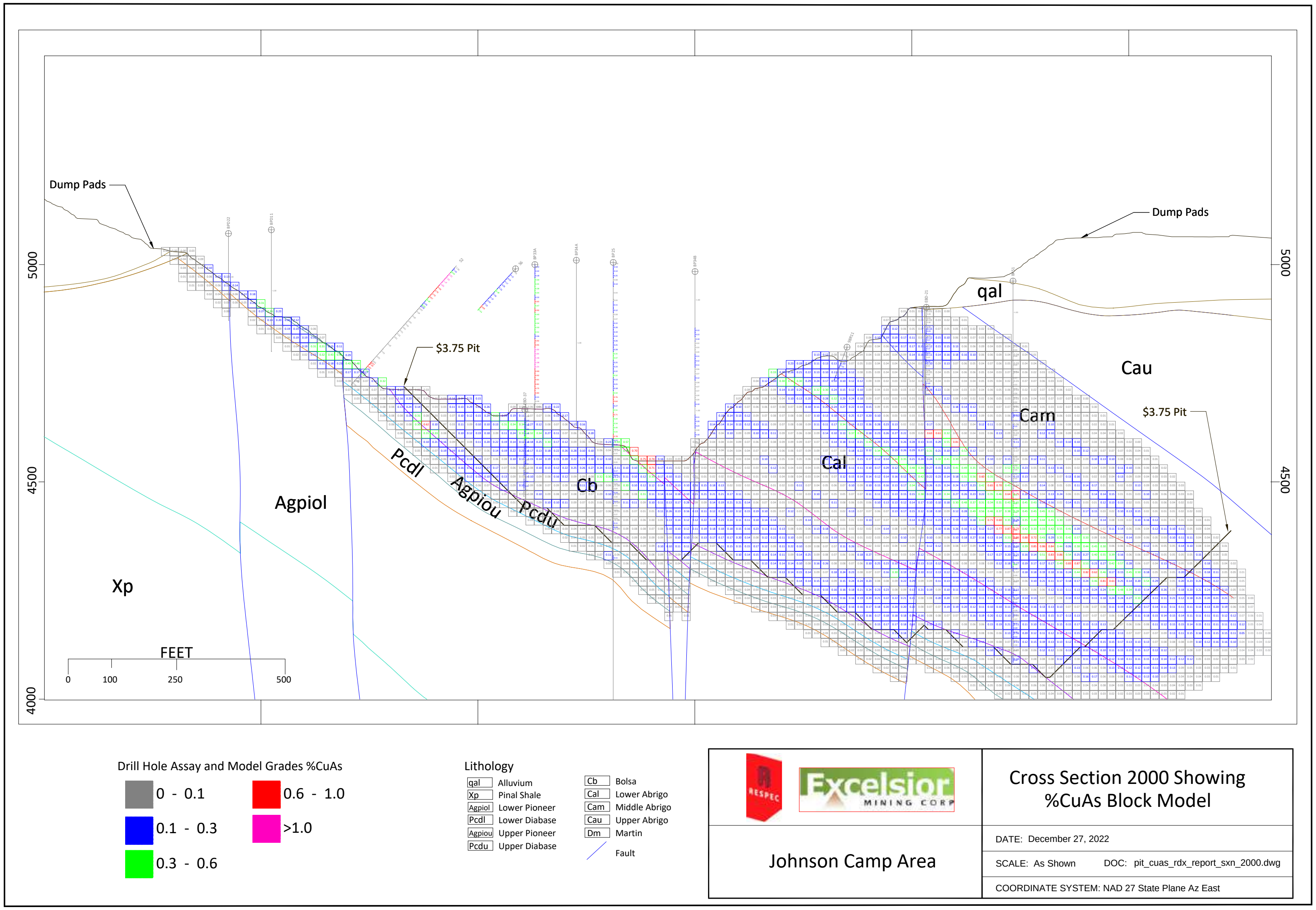


Figure 24-44: Geologic Cross Section 2000 with Acid-Soluble Copper ("CuAs") Block Model Grades

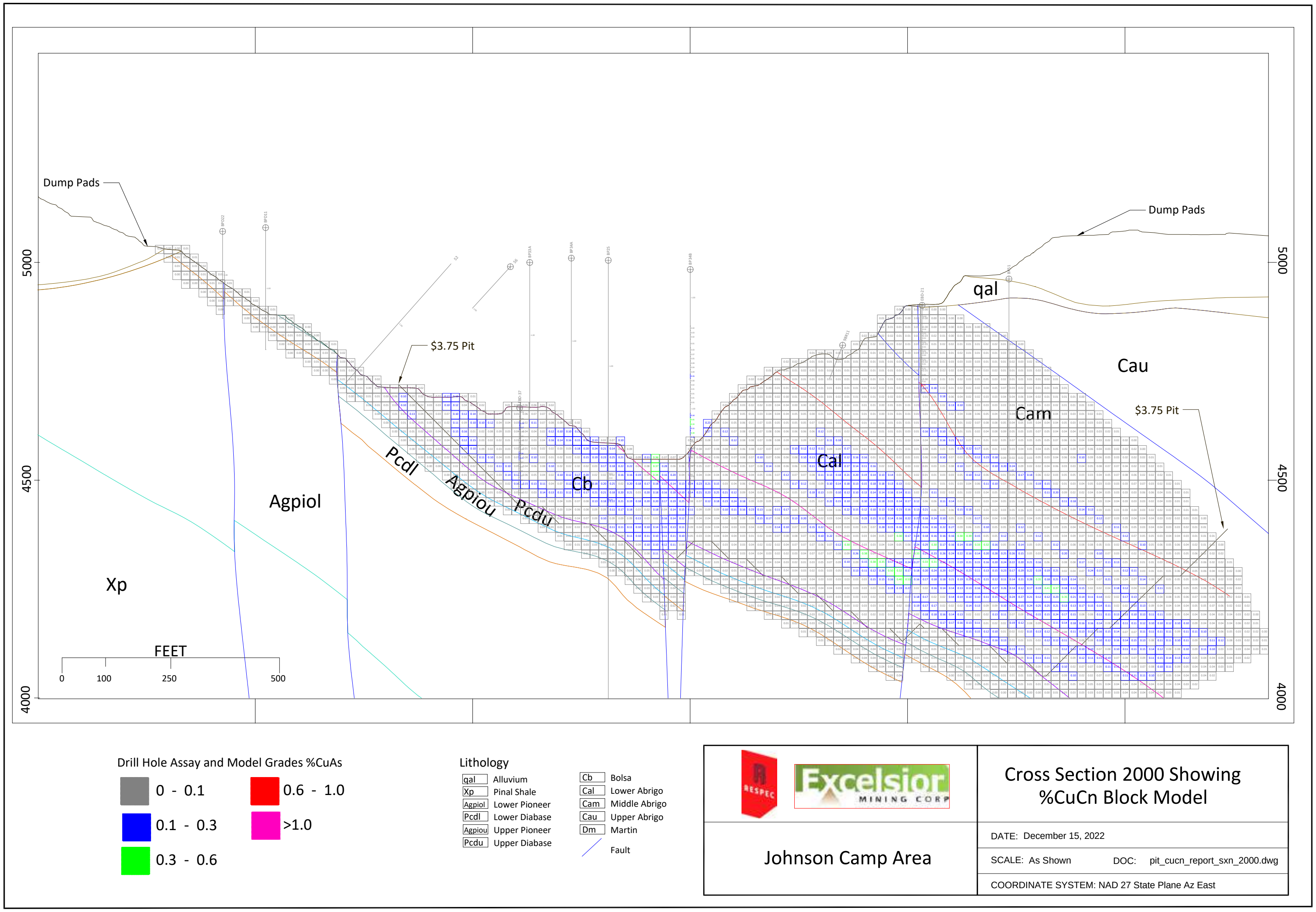


Figure 24-45: Geologic Cross Section 2000 with Cyanide-Soluble Copper ("CuCN") Block Model Grades

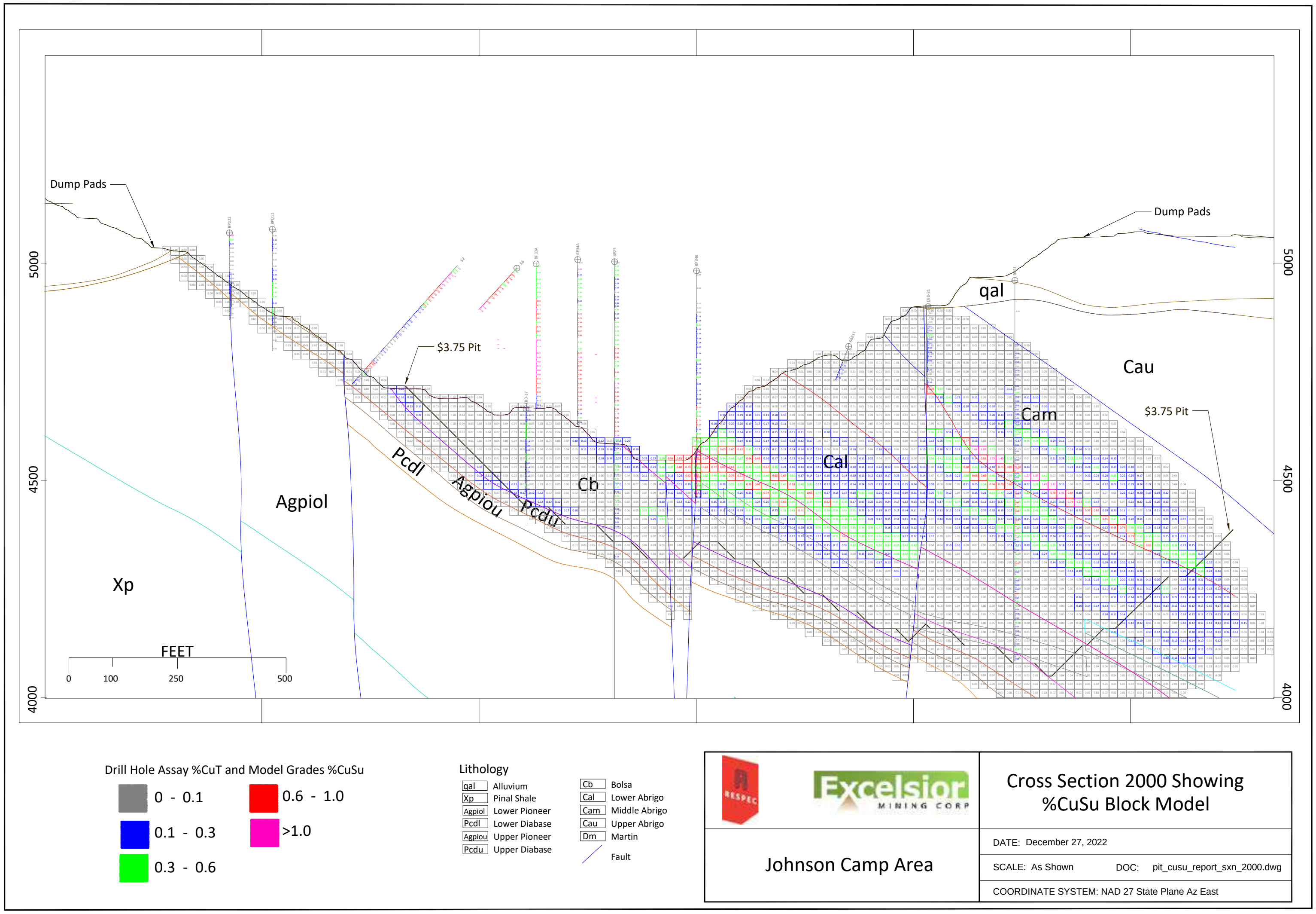


Figure 24-46: Geologic Cross Section 2000 with Sulfide Copper ("CuSu") Block Model Grades

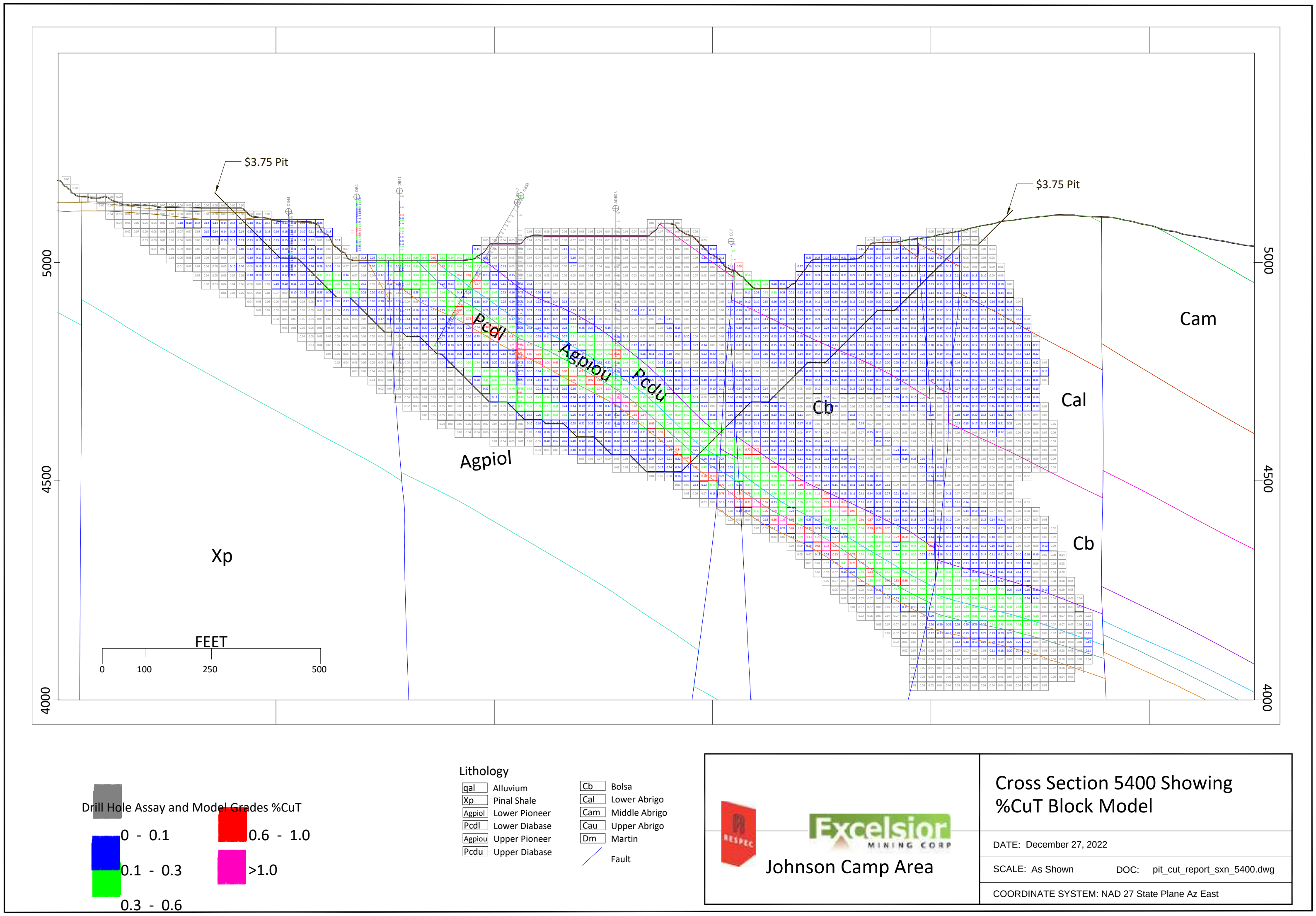


Figure 24-47: Geologic Cross Section 5400 with Total Copper ("CuT") Block Model Grades

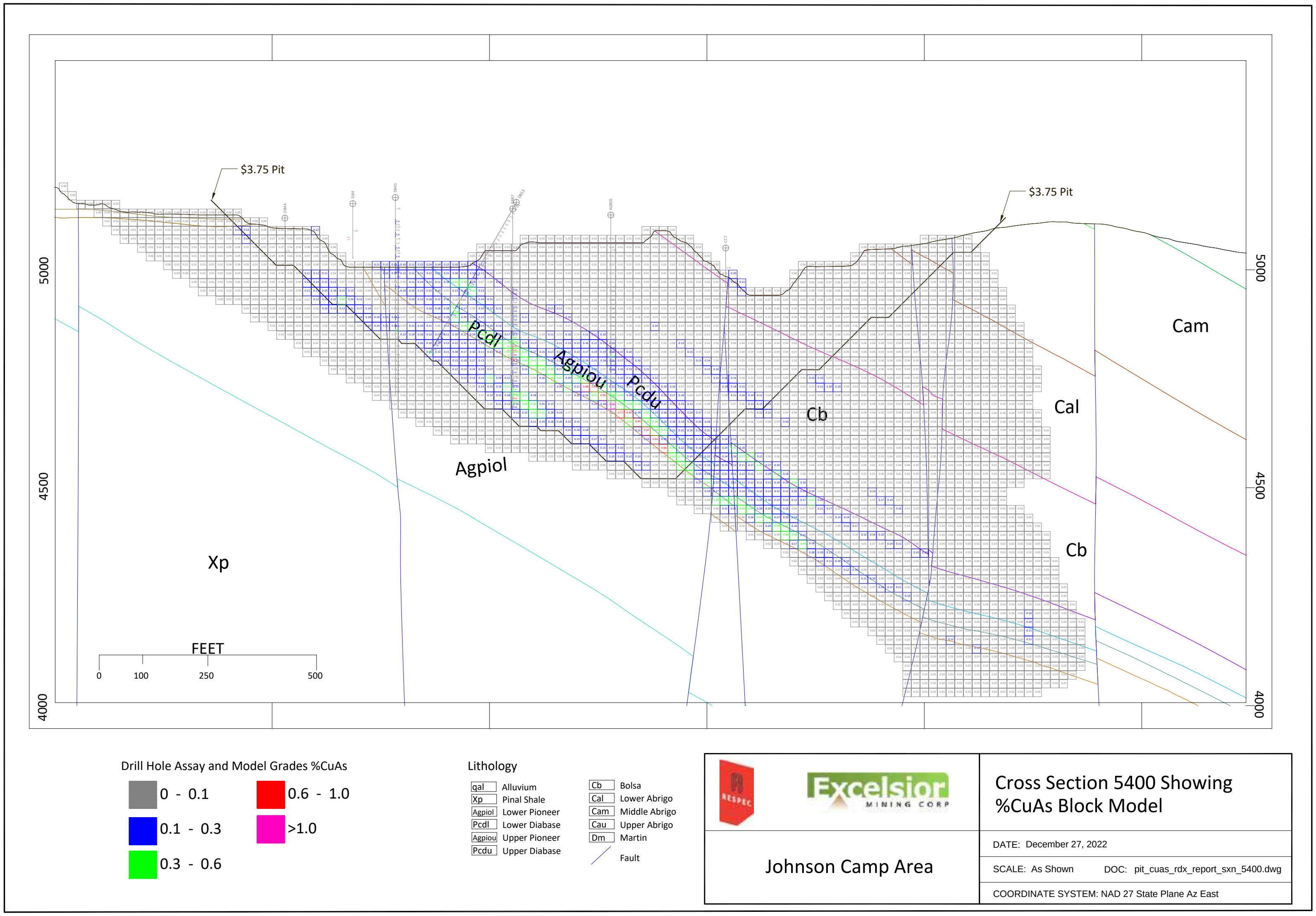


Figure 24-48: Geologic Cross Section 5400 with Acid-Soluble Copper ("CuAs") Block Model Grades

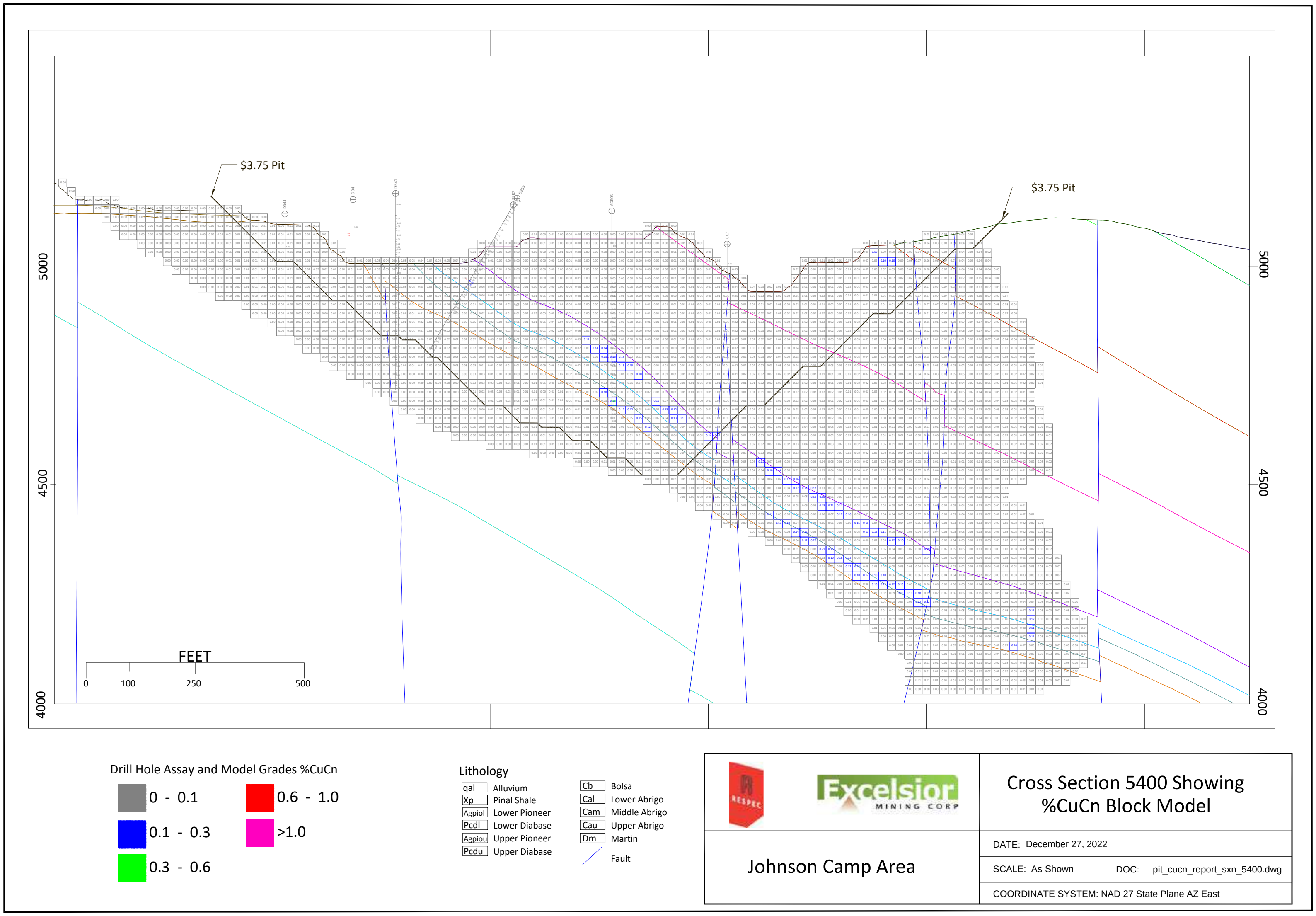


Figure 24-49: Geologic Cross Section 5400 with Cyanide-Soluble Copper ("CuCN") Block Model Grades

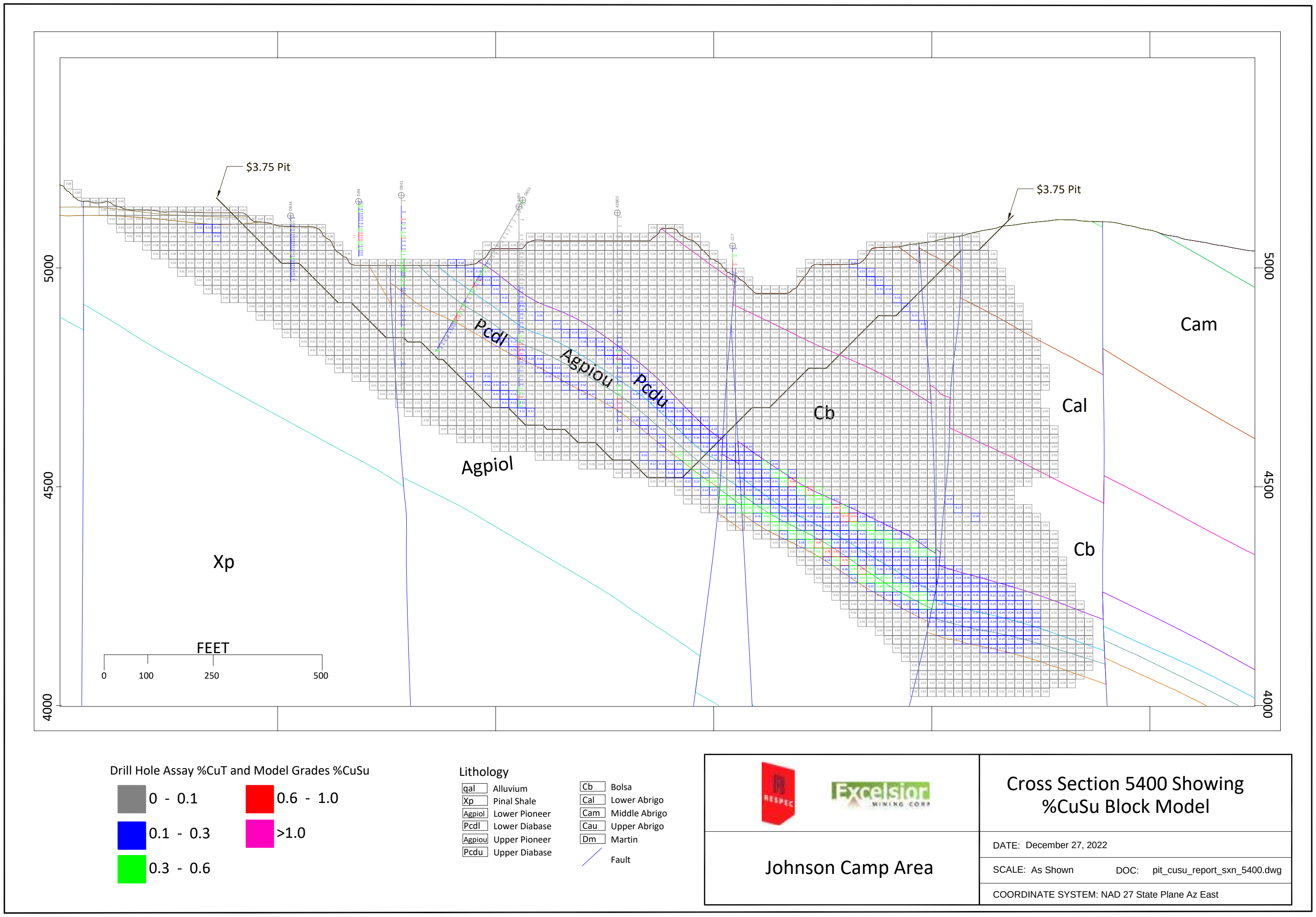


Figure 24-50: Geologic Cross Section 5400 with Sulfide Copper ("CuSu") Block Model Grades

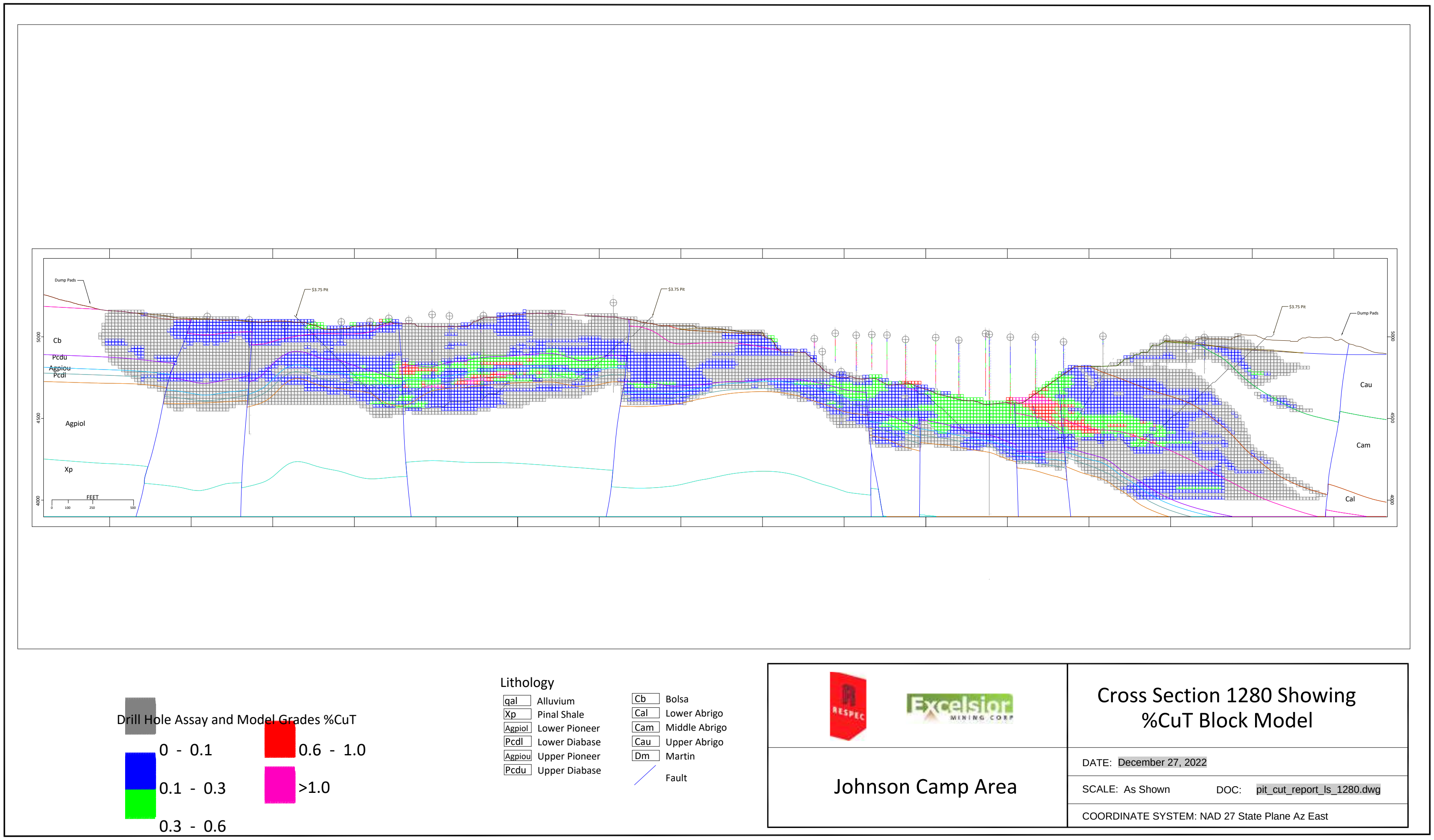


Figure 24-51: Geologic Long Section 1280 with Total Copper ("CuT") Block Model Grades

Figure 24-51 through Figure 24-54 are representative long sections through the block model showing the estimated copper grades.

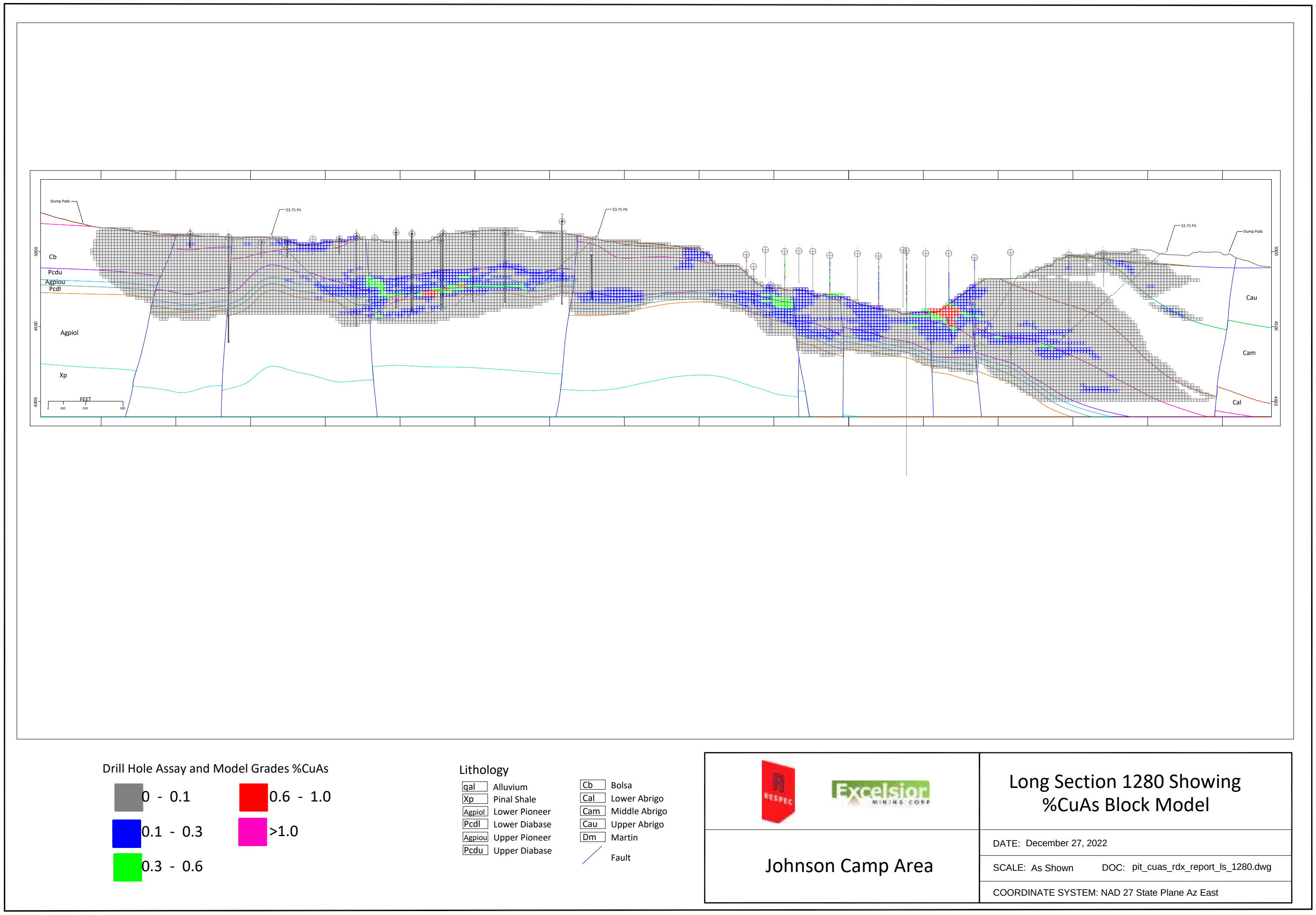
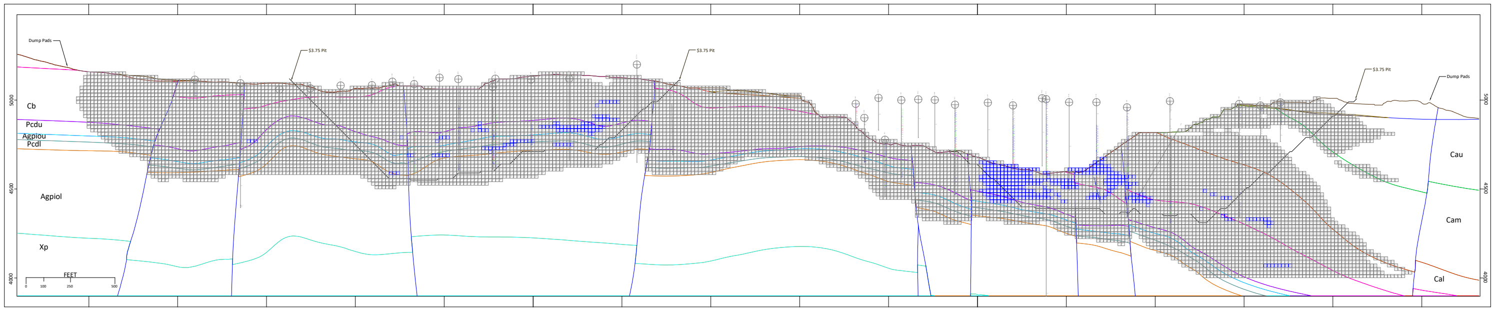


Figure 24-52: Geologic Long Section 1280 with Acid-Soluble Copper (“CuAs”) Block Model Grades



Drill Hole Assay and Model Grades %CuAs

▨ 0 - 0.1	🟥 0.6 - 1.0
🟦 0.1 - 0.3	🟪 >1.0
🟩 0.3 - 0.6	

Lithology

qal	Alluvium	Cb	Bolsa
Xp	Pinal Shale	Cal	Lower Abrigo
Agpiol	Lower Pioneer	Cam	Middle Abrigo
Pcdl	Lower Diabase	Cau	Upper Abrigo
Agpiou	Upper Pioneer	Dm	Martin
Pcdu	Upper Diabase	/	Fault

RESPEC

Excelsior MINING CORP

Johnson Camp Area

Cross Section 1280 Showing
%CuCn Block Model

DATE: December 27, 2022

SCALE: As Shown DOC: pit_cucn_report_ls_1280.dwg

COORDINATE SYSTEM: NAD 27 State Plane Az East

Figure 24-53: Geologic Long Section 1280 with Cyanide-Soluble Copper (“CuCN”) Block Model Grades

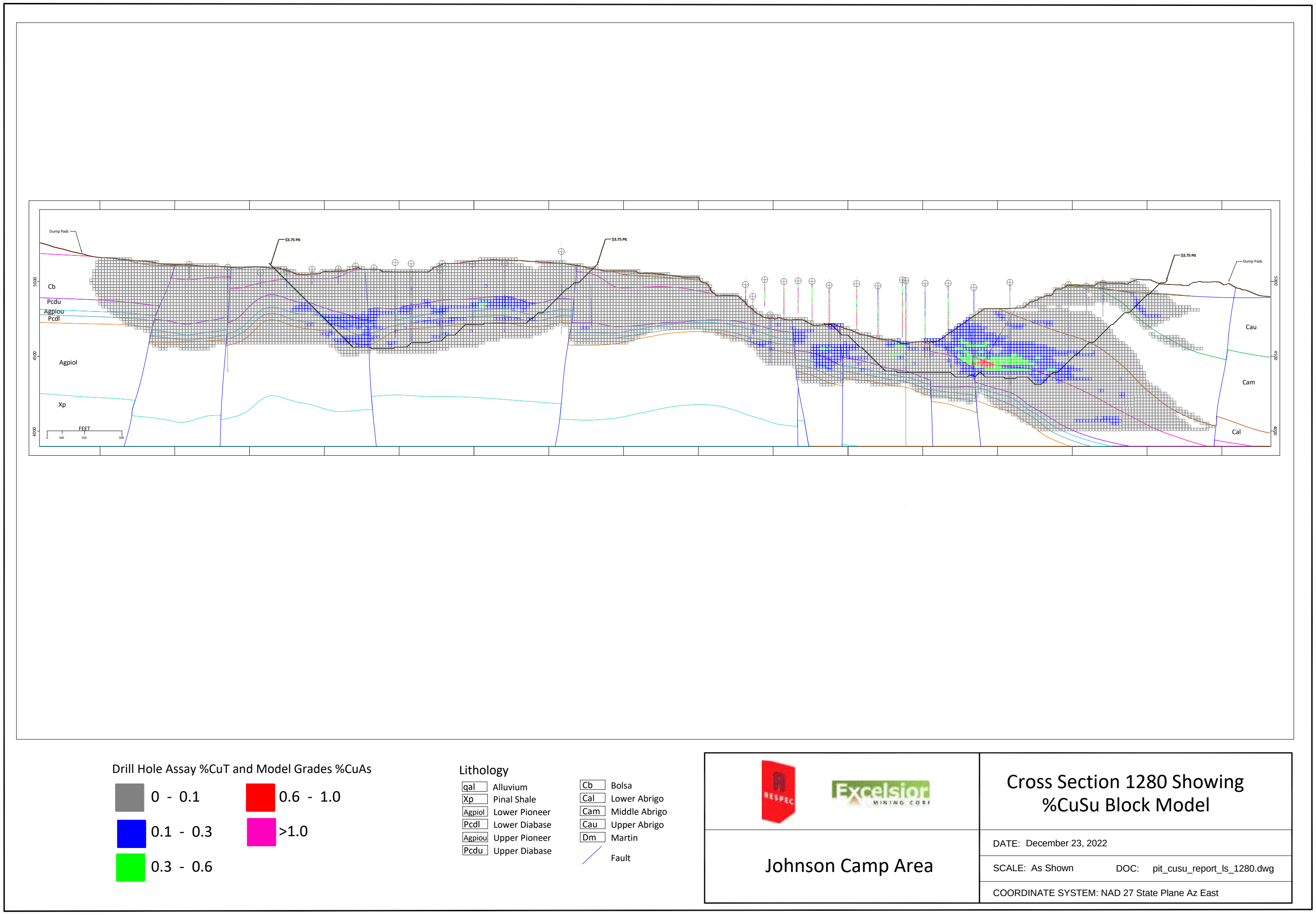


Figure 24-54: Geologic Long Section 1280 with Sulfide Copper ("CuSu") Block Model Grades

The Johnson Camp Mine in-pit resources are categorized by the two separate pit areas that make up the property (Burro and Copper Chief). The in-pit resources are broken-down by pit in Table 24-38, by lithology in Table 24-39, and by oxidation in Table 24-40.

Table 24-38: Johnson Camp Pit-Constrained Resources by Pit Area
(0.1% CuT cut-off)

Classification	Pit	tons	% Cu	% CuAs	% CuCN	% CuSu	lbs Cu	lbs CuAs	lbs CuCN	lbs CuSu
Measured	Burro	20,544,000	0.31	0.13	0.05	0.09	126,304,000	54,013,000	22,513,000	37,546,000
Indicated	Burro	69,039,000	0.33	0.12	0.05	0.12	449,205,000	171,894,000	71,542,000	167,899,000
Inferred	Burro	34,598,000	0.34	0.12	0.05	0.16	235,942,000	82,288,000	31,926,000	111,533,000
Measured	Copper Chief	227,000	0.27	0.16	0.01	0	1,241,000	749,000	52,000	5,000
Indicated	Copper Chief	18,127,000	0.28	0.13	0.03	0.05	100,913,000	46,763,000	10,838,000	16,533,000
Inferred	Copper Chief	16,400,000	0.26	0.11	0.04	0.03	86,713,000	37,326,000	13,451,000	11,248,000

Table 24-39: Johnson Camp Pit-Constrained Resources by Lithology
(0.1% CuT cut-off)

Classification	Lithology	tons	% CuT	% CuAs	% CuCN	% CuSu	lbs CuT	lbs CuAs	lbs CuCN	lbs CuSu
Measured	Upper Abrigo	2,024,000	0.21	0.11	0.02	0.05	8,595,000	4,518,000	895,000	1,968,000
Indicated	Upper Abrigo	2,855,000	0.22	0.08	0.03	0.07	12,719,000	4,692,000	1,551,000	4,138,000
Inferred	Upper Abrigo	799,000	0.23	0.05	0.03	0.12	3,604,000	794,000	465,000	1,919,000
Measured	Middle Abrigo	7,473,000	0.29	0.12	0.03	0.10	43,573,000	18,025,000	4,880,000	15,154,000
Indicated	Middle Abrigo	15,889,000	0.30	0.13	0.03	0.10	95,773,000	41,202,000	9,897,000	31,283,000
Inferred	Middle Abrigo	12,422,000	0.29	0.12	0.02	0.13	73,046,000	29,717,000	6,025,000	31,225,000
Measured	Lower Abrigo	6,609,000	0.34	0.13	0.07	0.11	44,996,000	16,763,000	9,115,000	14,050,000
Indicated	Lower Abrigo	35,101,000	0.36	0.12	0.05	0.15	249,736,000	85,316,000	37,394,000	102,100,000
Inferred	Lower Abrigo	21,901,000	0.35	0.11	0.06	0.16	152,331,000	47,816,000	24,240,000	70,090,000
Measured	Bolsa	4,304,000	0.33	0.17	0.09	0.07	28,500,000	14,351,000	7,491,000	6,212,000
Indicated	Bolsa	20,813,000	0.27	0.12	0.06	0.08	113,590,000	49,556,000	25,242,000	32,588,000
Inferred	Bolsa	8,432,000	0.24	0.10	0.05	0.07	41,202,000	16,434,000	8,327,000	12,199,000
Measured	Upper Diabase	158,000	0.24	0.14	0.04	0.05	768,000	429,000	136,000	165,000
Indicated	Upper Diabase	5,759,000	0.31	0.13	0.05	0.07	35,511,000	14,672,000	5,186,000	8,333,000
Inferred	Upper Diabase	3,312,000	0.35	0.14	0.05	0.05	23,097,000	9,566,000	3,342,000	3,551,000
Measured	Upper Pioneer	10,000	0.18	0.05	0.02	0.00	34,000	9,000	3,000	-
Indicated	Upper Pioneer	2,022,000	0.28	0.14	0.02	0.04	11,199,000	5,836,000	959,000	1,745,000
Inferred	Upper Pioneer	1,224,000	0.35	0.18	0.04	0.04	8,646,000	4,388,000	1,024,000	1,074,000
Measured	Lower Diabase	38,000	0.39	0.16	0.02	0.00	290,000	122,000	14,000	-
Indicated	Lower Diabase	2,069,000	0.45	0.24	0.03	0.06	18,577,000	9,837,000	1,433,000	2,686,000
Inferred	Lower Diabase	1,551,000	0.48	0.25	0.05	0.05	14,930,000	7,727,000	1,515,000	1,694,000
Measured	Lower Pioneer	155,000	0.25	0.17	0.01	0.00	789,000	543,000	30,000	-
Indicated	Lower Pioneer	2,658,000	0.24	0.14	0.01	0.03	13,013,000	7,546,000	718,000	1,559,000
Inferred	Lower Pioneer	1,357,000	0.21	0.12	0.02	0.04	5,800,000	3,173,000	439,000	1,028,000

Table 24-40: Johnson Camp Pit-Constrained Resources by Oxidation Group

(0.1% CuT cut-off)

Classification	Oxidation Group	tons	% CuT	% CuAs	% CuCN	% CuSu	lbs CuT	lbs CuAs	lbs CuCN	lbs CuSu
Measured	sulfide	1,257,000	0.29	0.02	0.03	0.24	7,245,000	600,000	727,000	5,918,000
Indicated	sulfide	6,784,000	0.42	0.04	0.04	0.34	56,881,000	5,392,000	5,652,000	45,836,000
Inferred	sulfide	5,876,000	0.35	0.04	0.05	0.26	41,455,000	5,038,000	5,514,000	30,902,000
Measured	transition	6,049,000	0.32	0.09	0.10	0.12	38,593,000	11,092,000	12,648,000	14,853,000
Indicated	transition	8,440,000	0.31	0.10	0.09	0.12	52,354,000	16,043,000	15,657,000	20,654,000
Inferred	transition	2,130,000	0.28	0.09	0.09	0.10	11,902,000	3,665,000	3,896,000	4,342,000
Measured	mixed	7,595,000	0.30	0.11	0.04	0.08	45,486,000	16,302,000	6,825,000	12,524,000
Indicated	mixed	55,824,000	0.30	0.11	0.05	0.09	338,947,000	123,230,000	54,370,000	103,121,000
Inferred	mixed	38,438,000	0.30	0.11	0.04	0.11	229,387,000	82,314,000	33,219,000	81,145,000
Measured	oxide	5,870,000	0.31	0.23	0.02	0.04	36,220,000	26,768,000	2,364,000	4,255,000
Indicated	oxide	16,118,000	0.32	0.23	0.02	0.05	101,935,000	73,991,000	6,700,000	14,821,000
Inferred	oxide	4,555,000	0.44	0.31	0.03	0.07	39,912,000	28,598,000	2,748,000	6,392,000

Table 24-41 presents the Johnson Camp Mine mineral resources compared to subsets of mineralized material tabulated with increasing cut-off grades. This is presented to provide grade-distribution data that allows for detailed assessment of the project resources. All of the tabulations are constrained as lying within the same optimized pit shells used to constrain the current mineral resources, which means the tabulations at cut-offs higher than the resource cut-off grade of 0.1% CuT represent subsets of the current resources.

Table 24-41: Johnson Camp Pit-Constrained Resources at Various Cut-offs

Classification	% CuT Cut-off	tons	% CuT	% CuAs	% CuCN	%CuSu	lbs CuT	lbs CuAs	lbs CuCN	lbs CuSu
Measured	**0.1**	**20,771,000**	**0.31**	**0.13**	**0.05**	**0.09**	**127,545,000**	**54,762,000**	**22,564,000**	**37,551,000**
	0.2	15,859,000	0.36	0.15	0.07	0.11	113,358,000	48,290,000	20,756,000	33,823,000
	0.3	9,384,000	0.43	0.19	0.09	0.13	81,494,000	35,147,000	15,977,000	23,724,000
	0.4	4,083,000	0.56	0.25	0.11	0.16	45,485,000	20,126,000	8,734,000	12,881,000
	0.5	1,981,000	0.69	0.32	0.12	0.19	27,145,000	12,505,000	4,844,000	7,568,000
	0.6	1,163,000	0.79	0.37	0.13	0.22	18,323,000	8,625,000	3,109,000	5,160,000
	0.7	720,000	0.88	0.42	0.14	0.26	12,665,000	6,067,000	2,084,000	3,687,000
	0.8	461,000	0.96	0.46	0.16	0.28	8,822,000	4,241,000	1,458,000	2,624,000
	0.9	246,000	1.06	0.5	0.18	0.33	5,200,000	2,461,000	880,000	1,615,000
	1	125,000	1.17	0.54	0.18	0.4	2,921,000	1,359,000	455,000	998,000
Indicated	**0.1**	**87,166,000**	**0.32**	**0.13**	**0.05**	**0.11**	**550,118,000**	**218,657,000**	**82,380,000**	**184,432,000**
	0.2	66,281,000	0.37	0.15	0.06	0.13	491,318,000	193,611,000	74,674,000	169,243,000
	0.3	37,675,000	0.46	0.18	0.07	0.17	350,056,000	134,938,000	54,363,000	128,588,000
	0.4	17,911,000	0.6	0.23	0.09	0.24	216,565,000	80,853,000	32,152,000	86,649,000
	0.5	10,322,000	0.73	0.26	0.1	0.31	150,050,000	54,282,000	20,995,000	64,035,000
	0.6	6,706,000	0.83	0.3	0.11	0.37	110,895,000	39,566,000	14,821,000	48,974,000
	0.7	4,366,000	0.93	0.33	0.12	0.41	80,846,000	29,134,000	10,326,000	35,963,000
	0.8	2,898,000	1.02	0.38	0.13	0.45	59,097,000	21,755,000	7,352,000	26,092,000
	0.9	1,794,000	1.13	0.44	0.14	0.47	40,488,000	15,920,000	4,848,000	17,014,000
	1	1,077,000	1.25	0.55	0.14	0.48	27,002,000	11,787,000	3,071,000	10,315,000
Inferred	**0.1**	**50,998,000**	**0.32**	**0.12**	**0.04**	**0.12**	**322,656,000**	**119,614,000**	**45,377,000**	**122,781,000**
	0.2	37,933,000	0.38	0.14	0.05	0.15	286,470,000	106,473,000	40,358,000	112,668,000
	0.3	22,331,000	0.47	0.17	0.06	0.19	208,011,000	77,035,000	28,889,000	85,586,000
	0.4	10,146,000	0.62	0.23	0.09	0.26	125,610,000	47,579,000	17,481,000	52,130,000
	0.5	5,822,000	0.75	0.29	0.1	0.32	87,463,000	33,404,000	11,863,000	36,681,000
	0.6	3,509,000	0.89	0.36	0.12	0.35	62,141,000	25,406,000	8,249,000	24,850,000
	0.7	2,460,000	0.99	0.42	0.13	0.39	48,689,000	20,743,000	6,472,000	19,180,000
	0.8	1,820,000	1.08	0.48	0.14	0.41	39,157,000	17,347,000	5,203,000	15,020,000
	0.9	1,314,000	1.17	0.55	0.16	0.42	30,641,000	14,383,000	4,165,000	11,061,000
	1	933,000	1.26	0.63	0.17	0.42	23,431,000	11,674,000	3,157,000	7,895,000

1. The project mineral resources are shown in bold and are comprised of all model blocks at a 0.1% CuT cut-off that lie within optimized resource pits.
2. Tabulations at higher cutoffs than used to define the mineral resources represent subsets of the mineral resource.
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
4. Rounding as required by reporting guidelines may result in apparent discrepancies between tons, grade, and contained metal content.

24.14.12 Discussion of Resources and Recommendations

Future drilling, exploration, and resource definition at Johnson Camp Mine should focus on increasing the understanding of the distribution of cyanide soluble and primary sulfide copper mineralization and detailed mineralogical characterization of the various modeled oxidation groups. Though Johnson Camp Mine has a long history of drilling, exploration, and mining, collection of cyanide soluble copper assay data is limited throughout the property and therefore the current understanding of cyanide soluble copper mineralization could be improved. Infill drilling in key areas of lower classification to increase drill density, and drill-testing of the unconstrained limits of the deposit, particularly down-dip from known mineralization, are also notable areas of focus for future development of the property. The author recommends collection of more structural data for the purposes of bolstering current geological understanding of the deposit and mineralization controls. Drilling more angle holes to test structures is recommended for this purpose.

The Johnson Camp copper resources have complex oxidation profile groups. The groups were modeled based on available data, mainly from assay ratios, grades, and lithologies. Continued modeling of these oxidation groups based on new data should be considered in future resource estimate iterations to ensure the best possible estimate of copper ratios into the model and accurate mineralogical characterization of the resources.

Sulfide copper has been estimated based on the residual copper value from total and soluble copper estimates. The author believes that this assumption is valid based on knowledge of deposit mineralogy. Near-surface sulfide copper values in the oxide zone have been factored to be conservative and account for copper in silicates and oxides which don't report to the various soluble copper assays. Analytical work, focused on the mineralogy of residual copper in the oxide zone, is recommended to confirm these assumptions and improve the overall mineralogical understanding of various geological and spatial zones in the deposit.

As the date of this report, Mr. Bickel is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that may materially affect the Johnson Camp Mine mineral resources and that are not otherwise discussed in this report. The impact of taxation was taken into consideration when establishing cut-off grade and further details are provided in Section 24.22: Economic Analysis.

24.15 MINERAL RESERVE ESTIMATES

The Johnson Camp Mine does not currently have any mineral reserves.

24.16 MINING METHODS

RESPEC has completed a PEA for the Johnson Camp project with the Burro and Chief deposits which anticipates mining using conventional open pit truck and loader methods. This assessment assumes that waste material would be loaded into 100-ton haul trucks and hauled to waste rock facilities. Leach material would be mined from the pit and placed on a heap leach pad for leaching of copper. RESPEC assessed the economic impact of different acid consumption costs for ROM Leach processing. Ultimate pit limits were developed using pit optimization techniques, and preliminary pit designs have been created. Production schedules have been developed using the resources from these pit designs.

The following sections discuss the methodology used to define the pit designs, waste dump designs, and the production schedule with relation to the PEA.

24.16.1 Pit Optimization

Pit optimization was completed using Whittle software (version 7.3). Economic and geometrical parameters were input into Whittle to complete the work. The economic parameters were developed assuming a processing method of crushing and leaching and single throughput rate of 20,000 tons per day.

Whittle pit shells for varied metal prices and acid consumption costs were used to determine pit phases and ultimate pits for each scenario. Whittle was then used to generate production schedules and preliminary cash-flows for each scenario.

24.16.1.1 Economic Parameters

Economic parameters were developed for each scenario and included contract mining cost, incremental mining cost, crushing cost, process cost, process capital cost, General and Administrative ("G&A") costs, acid cost, SX-EW cost, net smelter return royalty, and metallurgical recoveries. These are shown in Table 24-42.

Table 24-42: Economic Parameters

Run of Mine Leaching		
Contract Mine Cost	$2.30	$/ton Mined
Incremental Mining Cost	$0.01	$/ton/20ft bench below 5000'
Tech Service Mine Cost	$0.25	$/ton Mined
Crushing Cost	$1.10	$/ton Processed
Heap Management	$0.30	$/ton Processed
Heap Capital Cost	$0.80	$/ton Processed
G&A Cost	$0.05	$/lb Cu Produced
Acid Cost	$150	$/ton
SX-EW Cost	$0.25	$/lb Cu Produced
Recovery	95%	CuAs
Recovery	95%	CuCN
Recovery	70%	CuSu
Royalty	17.90%	NSR

The 20,000 ton per year throughput scenario assumes contract mining using costs from a previous study. All the scenarios assumed that leaching would be done north of the Copper Chief and Burro pits.

Process and G&A costs were provided by other consultants using an $0.05 per pound of copper produced.

Recoveries were provided by Excelsior and their consultants and were applied by the estimated copper solubility. Acid Soluble copper (CuAs), cyanide soluble copper (CuCN) and sulfide soluble copper (CuSu) were all estimated into the resource model so that the total copper (CuTot) is a combination of the three estimate without overlap. A 95%, 95%, and 70% recovery of CuAs, CuCN, and CuSu respectively was used.

Acid costs of $120/ton through $225/ton were investigated as cost sensitivities. A final cost of $150/ton was used to guide the pit design process. Acid consumption varies by lithology in both the Copper Chief and Burro pits and was used in the pit optimization shown in Table 24-43. The acid cost was calculated into the resource model and then past into Whittle so that the proper overall costs were considered. Note that the acid cost is above and beyond the cost for processing as listed in Table 24-42.

Table 24-43: Acid Consumption by Lithology

Acid Consumption by Lithology (lb/ton)	
Pioneer	20
Bolsa	25
Diabase	30
Lower Abrigo	40
Middle Abrigo	55
Upper Abrigo	45
Martin	70

Copper recoveries were provided by Excelsior and their consultants.

While various metal prices were considered in the pit optimizations, base metal prices of $3.00 per pound of copper was used to determine the ultimate pit limits.

A single royalty factor of 17.90% was imposed on the entire Whittle model as a net smelter return (“NSR”), which means that the royalty percentage has been multiplied by the recovered metal and metal prices.

24.16.1.2 Geometrical Parameters

Geometrical parameters will often include property and royalty boundaries as well as pit slope parameters. The mineral resources are all within current property boundaries, so none were considered as a restriction to the pit optimization. However, a facility boundary constraint was applied to the pit optimization to prevent optimizations from mining in the area to the south. This maintained the current historical mining highwall of the Burro pit to the south. All Burro pit expansion is to the north.

There are no recent pit slope stability studies, and pit slopes were assumed to be a constant 45-degree slopes in all sectors.

24.16.1.3 Pit Optimization Results

Pit optimizations used both Indicated and Inferred resources. Note that Canadian NI 43-101 guidelines define a PEA as follows:

> *A preliminary economic assessment is preliminary in nature, and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as mineral reserves, and there is no certainty that the preliminary assessment will be realized*.

Pit optimizations were run to determine appropriate pit phasing and ultimate limits for each scenario. Whittle was then used to generate preliminary production and cash-flows for each scenario.

Optimized pits were generated for various metal prices ranging from \$1.00/lb Cu to \$5.00/lb Cu using \$0.05/lb Cu increments. Results of the pit optimization are shown in Table 24-44 in \$0.25/lb Cu increments. The \$3.00/lb Cu result is highlighted in the table as the base case pit for the Whittle analysis.

A second step of the process to select the ultimate pit limits was to use the Pit by Pit ("PbP") analysis tool in Whittle to generate a discounted operating cash flow (note that capital is not included). This used a rough scheduling by pit phase for each pit shell to generate the discounted value for the pit. The program develops three different discounted values: best, worst, and specified. The best-case value uses each of the pit shells as pit phases or pushbacks. For example, when evaluating pit 20, there would be 19 pushbacks mined prior to pit 20, and the resulting schedule takes advantage of mining more valuable material up front to improve the discounted value. Evaluating pit 21 would have 20 pushbacks; pit 22 would have 21 pushbacks and so on. Note that this is not a realistic case as the incremental pushbacks would not have enough mining width between them to be able to mine appropriately, but this does help to define the maximum potential discounted operating cash flow.

The worst case does not use any pushbacks in determining the discounted value for each of the pit shells. Thus, each pit shell is evaluated as if mining a single pit from top to bottom. This does not get the advantage of mining more valuable material up front, so it generally provides a lower discounted value than that of the best case.

The specified case allows the user to specify pit shells to be used as pushbacks and then schedules the pushbacks and calculates the discounted cash flow. This is more realistic than the base case as it allows for more mining width, though the final pit design will have to ensure that appropriate mining width is available. The specified case has been used for each mine to determine the ultimate pit limits to design to, as well as to specify guidelines for designing pit phases.

The PbP analysis results are listed in Table 24-45 showing results for pits 20 through 60 with pit 40 highlighted as the best specified pit case. The PbP results are shown graphically in Figure 24-55.

The PbP results shown in Table 24-44 provide the basis for determining the Whittle ultimate pit limits. The best discounted value for the specified case was obtained with pit shell 40. Pit phasing for the PbP analysis used pit shell 29 as an initial mining phase. This pit is mostly in the Burro deposit area. Thus, for pit design guidance, pit shell 29 was used for phase 1 in the Burro area and pit 40 was used for guidance in of both Burro and Chief ultimate pit designs.

Table 24-44: Pit Optimization Results

		Material Processed									Waste	Total	Strip	
Pit	$/lb Cu	ktons	CuAs %	K Lbs CuAs	CuCN %	K Lbs CuCN	CuSu %	K Lbs CuSu	CuT %	K Lbs CuT	P_CSt	K $P_CSt	ktons	ktons	Ratio
1	$1.00	55	0.57	628	0.21	229	0.29	322	1.16	1,279	$2.17	120	14	69	0.26
6	$1.25	187	0.50	1,851	0.15	573	0.24	894	0.97	3,619	$2.15	402	79	266	0.42
11	$1.50	452	0.43	3,858	0.13	1,182	0.20	1,832	0.83	7,497	$2.14	969	359	812	0.80
16	$1.75	1,585	0.27	8,473	0.12	3,798	0.19	6,030	0.62	19,787	$1.99	3,148	1,265	2,850	0.80
21	$2.00	4,013	0.20	15,967	0.10	8,253	0.16	13,063	0.50	40,344	$1.90	7,637	2,269	6,282	0.57
26	$2.25	8,871	0.18	31,961	0.09	15,737	0.14	25,627	0.45	79,556	$2.05	18,203	6,128	14,999	0.69
31	$2.50	27,368	0.17	91,741	0.08	41,713	0.15	79,462	0.42	229,845	$2.32	63,437	29,197	56,565	1.07
36	$2.75	50,223	0.16	161,857	0.07	69,704	0.15	152,335	0.41	410,127	$2.38	119,604	62,931	113,155	1.25
41	$3.00	86,106	0.15	266,920	0.06	103,566	0.14	233,701	0.38	661,403	$2.28	196,690	112,852	198,958	1.31
46	$3.25	99,952	0.15	297,846	0.06	115,189	0.13	260,174	0.37	740,648	$2.28	227,547	123,602	223,554	1.24
51	$3.50	117,560	0.14	335,687	0.06	129,384	0.13	294,033	0.36	837,862	$2.27	266,568	142,306	259,865	1.21
56	$3.75	131,808	0.14	362,095	0.05	139,179	0.12	318,708	0.34	909,179	$2.26	298,507	154,065	285,873	1.17
61	$4.00	159,008	0.13	406,393	0.05	159,602	0.12	372,704	0.33	1,046,229	$2.26	359,367	187,469	346,477	1.18
66	$4.25	176,415	0.12	434,557	0.05	172,104	0.11	401,945	0.32	1,127,668	$2.26	398,780	206,981	383,395	1.17
71	$4.50	191,688	0.12	456,780	0.05	181,797	0.11	423,338	0.31	1,189,601	$2.25	431,123	220,545	412,234	1.15
76	$4.75	212,888	0.11	485,630	0.05	196,732	0.11	452,167	0.30	1,274,681	$2.23	474,869	244,170	457,057	1.15
81	$5.00	239,752	0.11	520,029	0.04	212,732	0.10	483,896	0.29	1,369,125	$2.20	527,824	271,019	510,770	1.13

Table 24-45: Pit by Pit Analysis Results

	Material Processed									Waste	Total	Strip	Disc. Op Cash Flow (M USD)			LoM
Pit	ktons	CuAs %	K Lbs CuAs	CuCN %	K Lbs CuCN	CuSu %	K Lbs CuSu	CuT %	K Lbs CuT	ktons	ktons	Ratio	Best	Specified	Worst	Years
20	4,795	0.18	17,259	0.09	8,515	0.15	13,925	0.46	43,834	1,231	6,026	0.26	$34.16	$34.16	$34.16	0.67
21	5,018	0.18	17,954	0.09	8,787	0.14	14,369	0.45	45,391	1,264	6,282	0.25	$35.09	$35.09	$35.09	0.70
22	5,472	0.18	19,275	0.09	9,514	0.14	15,412	0.45	48,717	1,427	6,899	0.26	$36.96	$36.96	$36.96	0.76
23	5,897	0.17	20,562	0.09	10,033	0.14	16,654	0.44	52,005	1,616	7,513	0.27	$38.65	$38.65	$38.65	0.82
24	6,536	0.17	22,584	0.08	10,891	0.14	18,518	0.44	57,152	2,024	8,560	0.31	$41.09	$41.09	$41.09	0.91
25	7,856	0.17	27,057	0.08	13,004	0.13	20,497	0.43	66,810	2,694	10,550	0.34	$45.57	$45.57	$45.57	1.09
26	10,463	0.17	35,251	0.08	16,557	0.13	27,389	0.42	87,169	4,536	14,999	0.43	$54.56	$54.14	$54.14	1.45
27	12,486	0.16	41,076	0.08	19,504	0.13	33,451	0.41	102,998	6,242	18,728	0.50	$60.93	$59.88	$59.88	1.73
28	13,386	0.17	44,240	0.08	20,877	0.13	34,756	0.41	109,516	6,879	20,265	0.51	$63.43	$62.11	$62.11	1.86
29	15,545	0.17	51,810	0.08	23,857	0.13	39,190	0.41	126,173	8,916	24,461	0.57	$69.06	$67.18	$67.18	2.16
30	20,264	0.16	66,298	0.07	30,249	0.13	51,710	0.40	162,085	13,626	33,890	0.67	$80.09	$76.49	$76.49	2.81
31	30,286	0.16	97,209	0.07	43,099	0.14	83,249	0.40	243,182	26,279	56,565	0.87	$100.58	$92.47	$92.47	4.21
32	40,476	0.16	131,618	0.07	57,269	0.15	118,731	0.41	330,126	44,744	85,219	1.11	$120.15	$112.11	$104.09	5.62
33	42,137	0.16	135,583	0.07	59,498	0.15	124,654	0.41	342,773	46,903	89,039	1.11	$122.40	$114.02	$105.05	5.85
34	44,318	0.16	140,892	0.07	62,181	0.15	131,902	0.41	359,032	49,536	93,854	1.12	$125.03	$116.10	$105.82	6.16
35	51,188	0.16	162,111	0.07	69,632	0.15	152,141	0.40	411,253	59,624	110,812	1.16	$132.40	$120.86	$105.85	7.11
36	52,332	0.16	165,088	0.07	70,681	0.15	155,414	0.40	419,211	60,823	113,155	1.16	$133.37	$121.46	$105.75	7.27
37	59,094	0.16	183,732	0.07	78,559	0.15	179,275	0.40	471,386	72,500	131,594	1.23	$138.22	$122.37	$101.69	8.21
38	61,869	0.15	190,939	0.07	80,911	0.15	186,632	0.40	490,049	75,419	137,288	1.22	$139.30	$123.48	$99.39	8.59
39	65,335	0.15	200,059	0.06	84,379	0.15	196,142	0.39	514,264	80,051	145,386	1.23	$140.23	$124.44	$95.60	9.07
40	78,804	0.16	246,150	0.06	96,368	0.14	213,515	0.39	610,242	100,803	179,607	1.28	$141.94	$125.37	$82.09	10.95
41	86,106	0.15	266,920	0.06	103,566	0.14	233,701	0.38	661,403	112,852	198,958	1.31	$142.13	$123.98	$71.35	11.96
42	89,349	0.15	275,540	0.06	107,000	0.14	241,292	0.38	682,995	117,892	207,241	1.32	$141.78	$123.60	$65.82	12.41
43	91,746	0.15	281,966	0.06	109,163	0.13	246,671	0.38	698,631	121,419	213,166	1.32	$141.23	$123.07	$61.41	12.74
44	93,879	0.15	286,980	0.06	111,811	0.13	252,243	0.38	713,079	125,167	219,046	1.33	$140.57	$122.43	$57.23	13.04
45	95,195	0.15	290,281	0.06	112,870	0.13	254,944	0.38	720,924	126,821	222,016	1.33	$140.08	$121.89	$54.86	13.22
46	95,828	0.15	291,806	0.06	113,540	0.13	256,304	0.38	724,653	127,726	223,554	1.33	$139.78	$121.55	$53.73	13.31
47	98,867	0.15	299,267	0.06	115,952	0.13	262,311	0.38	742,775	131,719	230,586	1.33	$138.12	$119.55	$47.15	13.73
48	100,282	0.15	302,121	0.06	117,574	0.13	266,463	0.37	751,812	134,309	234,591	1.34	$137.19	$118.40	$43.73	13.93
49	104,011	0.15	312,594	0.06	121,407	0.13	275,866	0.37	776,720	142,642	246,653	1.37	$133.95	$114.62	$33.31	14.45
50	105,086	0.15	314,890	0.06	122,232	0.13	278,057	0.37	782,707	144,079	249,164	1.37	$133.11	$113.61	$30.55	14.60
51	108,659	0.15	323,399	0.06	125,912	0.13	286,062	0.37	804,686	151,206	259,865	1.39	$129.70	$109.22	$19.23	15.09
52	110,985	0.15	328,623	0.06	127,573	0.13	290,905	0.37	817,900	154,535	265,520	1.39	$127.52	$106.49	$12.80	15.41
53	112,008	0.15	330,716	0.06	128,608	0.13	293,617	0.37	824,233	156,565	268,572	1.40	$126.42	$105.07	$ 9.86	15.56
54	112,967	0.15	332,764	0.06	129,567	0.13	295,407	0.37	829,735	158,193	271,160	1.40	$125.40	$103.75	$ 7.02	15.69
55	115,461	0.15	338,204	0.06	131,908	0.13	301,446	0.37	844,803	163,384	278,845	1.42	$122.37	$99.74	$ (1.72)	16.04
56	117,767	0.15	343,571	0.06	133,852	0.13	306,717	0.36	858,449	168,107	285,873	1.43	$119.42	$95.99	$ (9.34)	16.36
57	119,291	0.15	346,671	0.06	135,141	0.13	309,826	0.36	866,803	170,664	289,955	1.43	$117.57	$93.54	$ (14.53)	16.57
58	123,841	0.14	356,901	0.06	139,756	0.13	320,274	0.36	894,660	181,828	305,669	1.47	$110.80	$84.52	$ (32.32)	17.20
59	125,893	0.14	361,537	0.06	141,426	0.13	324,758	0.36	906,425	186,908	312,801	1.48	$107.61	$80.23	$ (41.64)	17.49
60	128,718	0.14	363,765	0.06	143,548	0.13	338,516	0.36	926,405	194,005	322,723	1.51	$102.93	$74.69	$ (52.93)	17.88

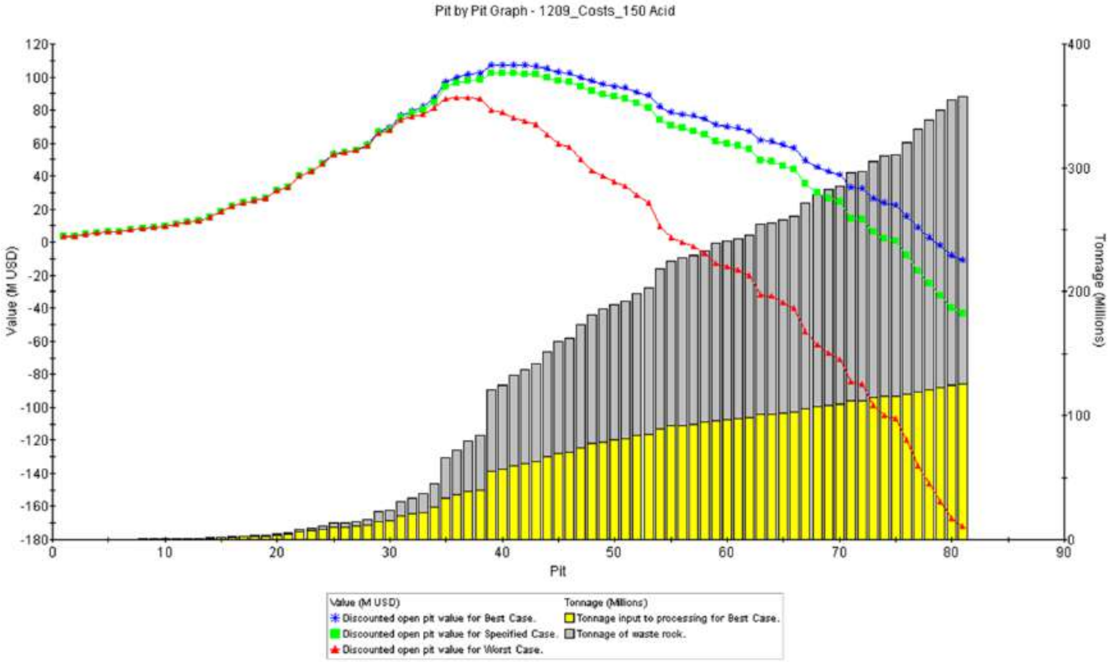


Figure 24-55: Johnson Camp PbP Graph

Acid cost sensitivities were calculated at the $3.00/lb Cu base case. The sensitivity analysis was used to determine the best-case scenario for the cost of acid consumption. Acid costs varied from $120/ ton to $225/ton. $150/ton acid cost was selected to be used for the final PEA reporting based on Excelsior and their consultants' inputs. Table 24-46 shows the resulting acid cost sensitivity analysis.

Table 24-46: Acid Cost Sensitivity

			Material Processed													
Scenario	Pit	$/lb Cu	ktons	CuAs %	K Lbs CuAs	CuCN %	K Lbs CuCN	CuSu %	K Lbs CuSu	CuT %	K Lbs CuT	P_CSt	K $P_CSt	Waste ktons	Total ktons	Strip Ratio
120 Acid	41	$3.00	95,663	0.14	277,379	0.06	106,695	0.12	223,438	0.36	682,379	$2.22	212,588	99,433	195,096	1.04
150 Acid	41	$3.00	80,444	0.15	246,257	0.06	96,717	0.12	198,407	0.37	601,745	$2.21	177,434	96,485	176,930	1.20
175 Acid	41	$3.00	66,337	0.16	212,447	0.06	83,859	0.12	164,054	0.39	512,007	$2.17	144,032	83,808	150,145	1.26
200 Acid	41	$3.00	60,925	0.16	200,028	0.07	79,235	0.12	152,125	0.39	479,045	$2.16	131,558	81,354	142,279	1.34
225 Acid	41	$3.00	51,629	0.17	174,349	0.07	69,714	0.13	131,622	0.40	416,809	$2.13	110,033	73,389	125,018	1.42

24.16.2 Pit Designs

Detailed pit designs were completed for Johnson Camp mine. The Copper Chief pit was designed as an individual pit, and the Burro pit was designed with a total of three pit phases. All pit designs were completed in Surpac software (version 2020). Burro pit design is shown in Figure 24-57 through Figure 24-59, and the ultimate pit design for both Copper Chief and Burro pits is shown in Figure 24-60. The following sections discuss the parameters used to determine the resources inside of the pit designs. The Copper Chief pit design is shown in Figure 24-56.

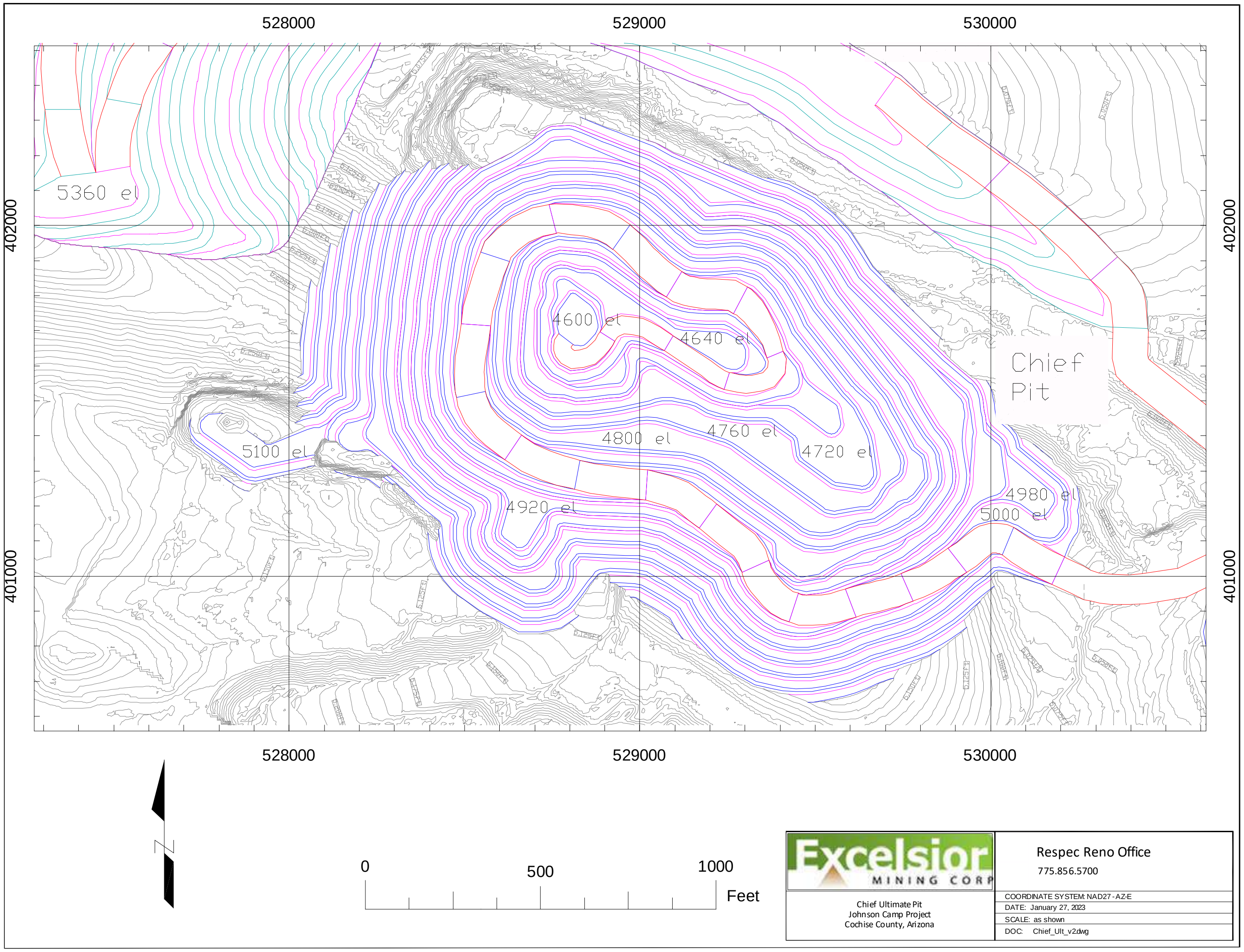


Figure 24-56: Copper Chief Ultimate Pit Design

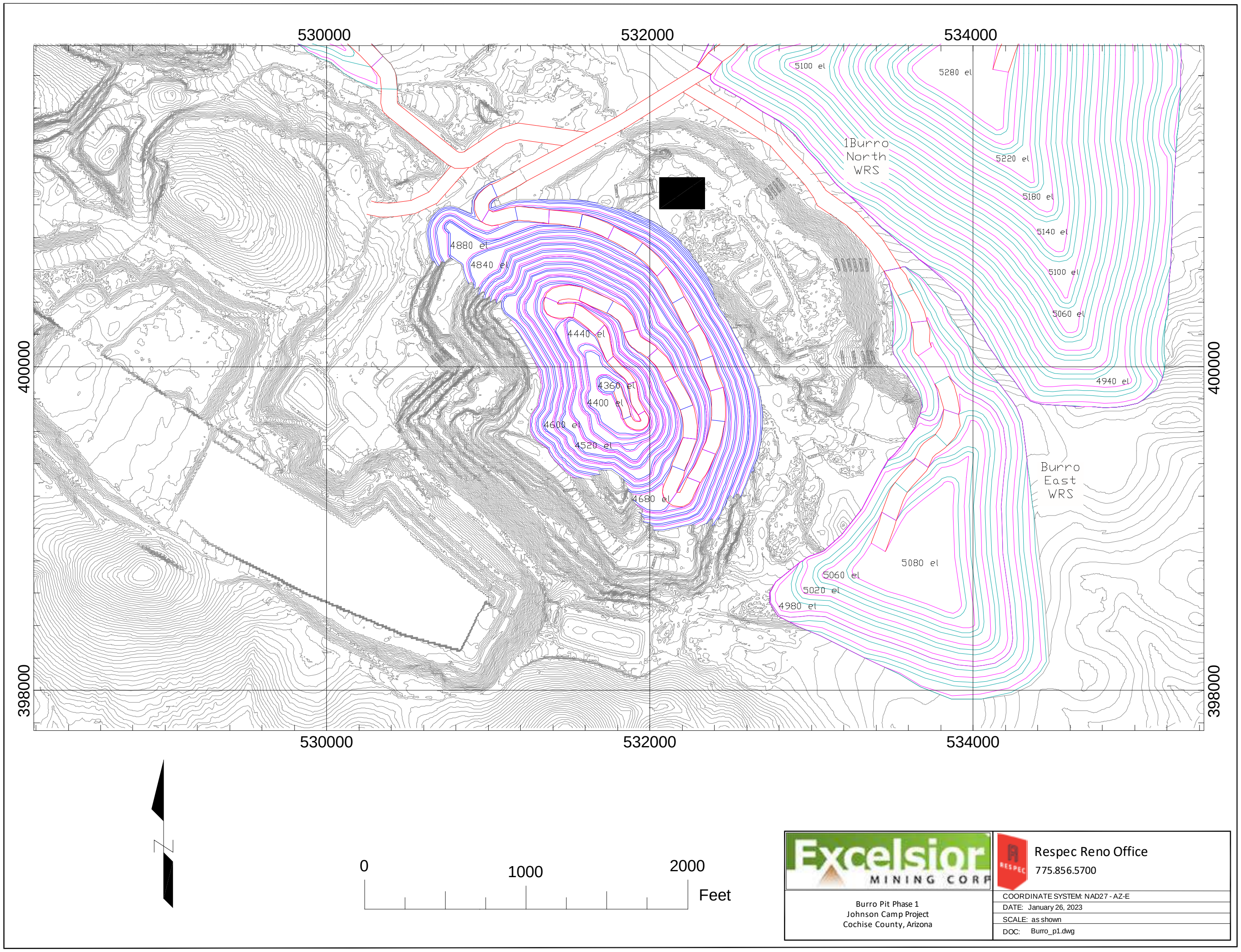


Figure 24-57: Burro Phase 1 Pit Design

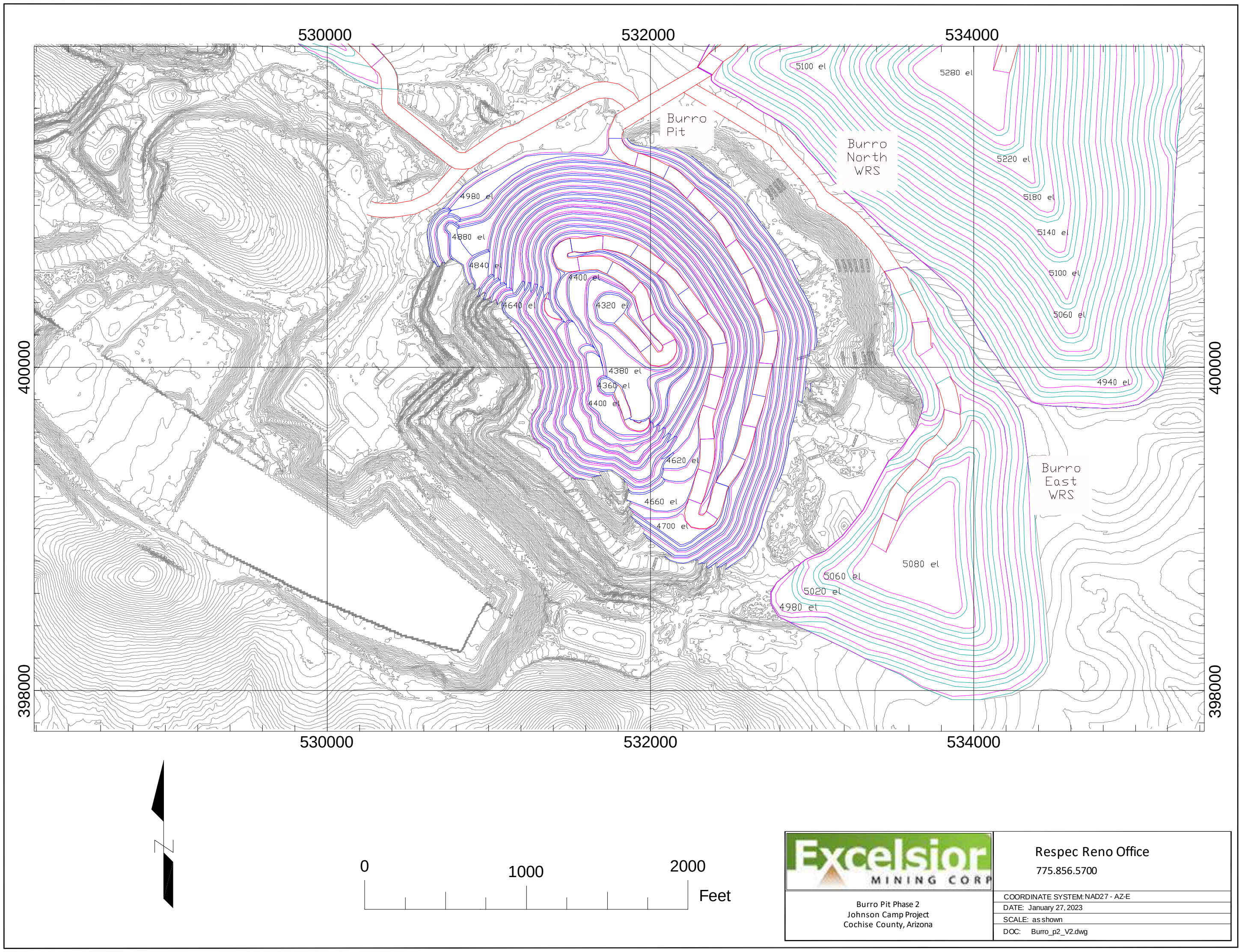


Figure 24-58: Burro Phase 2 Pit Design

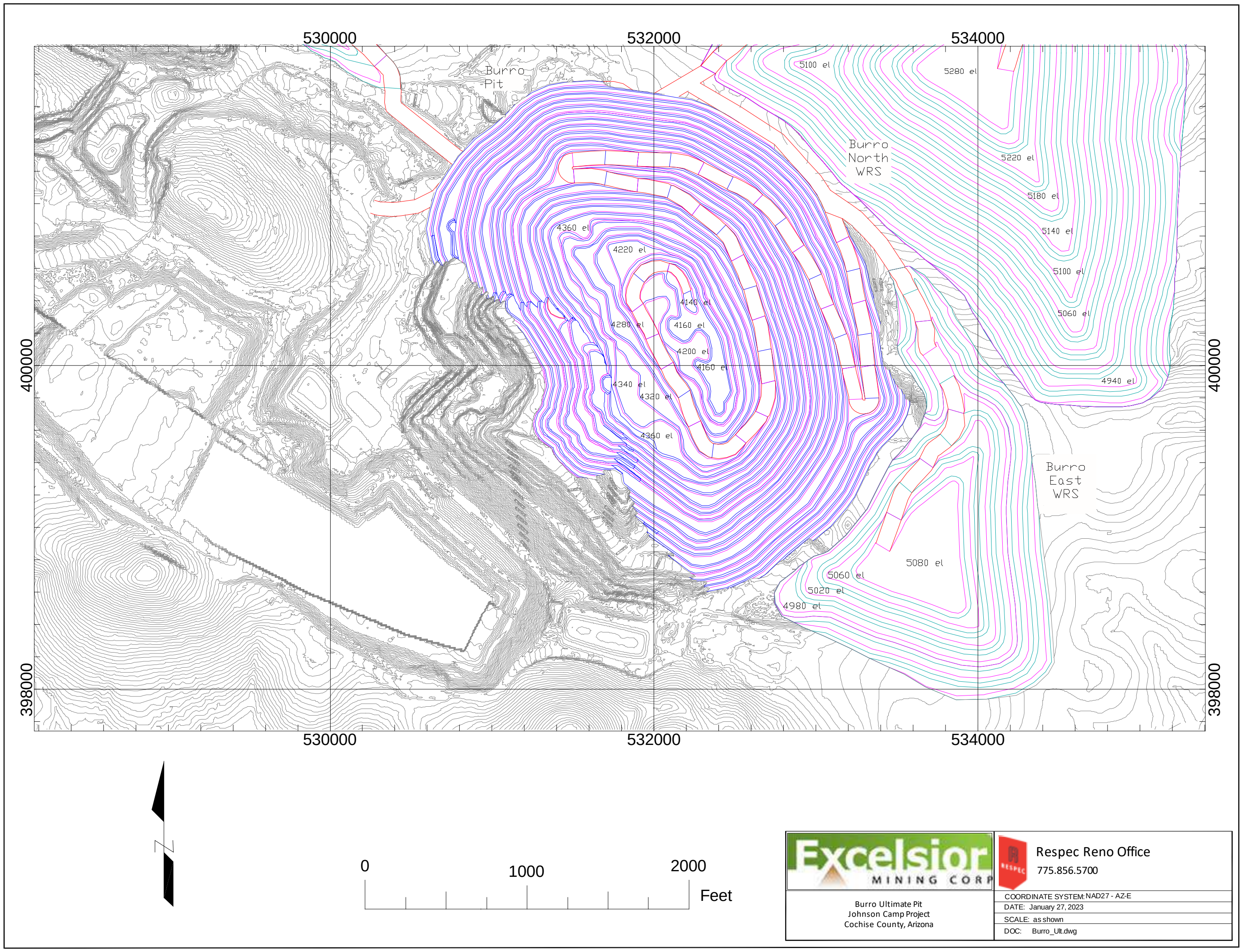
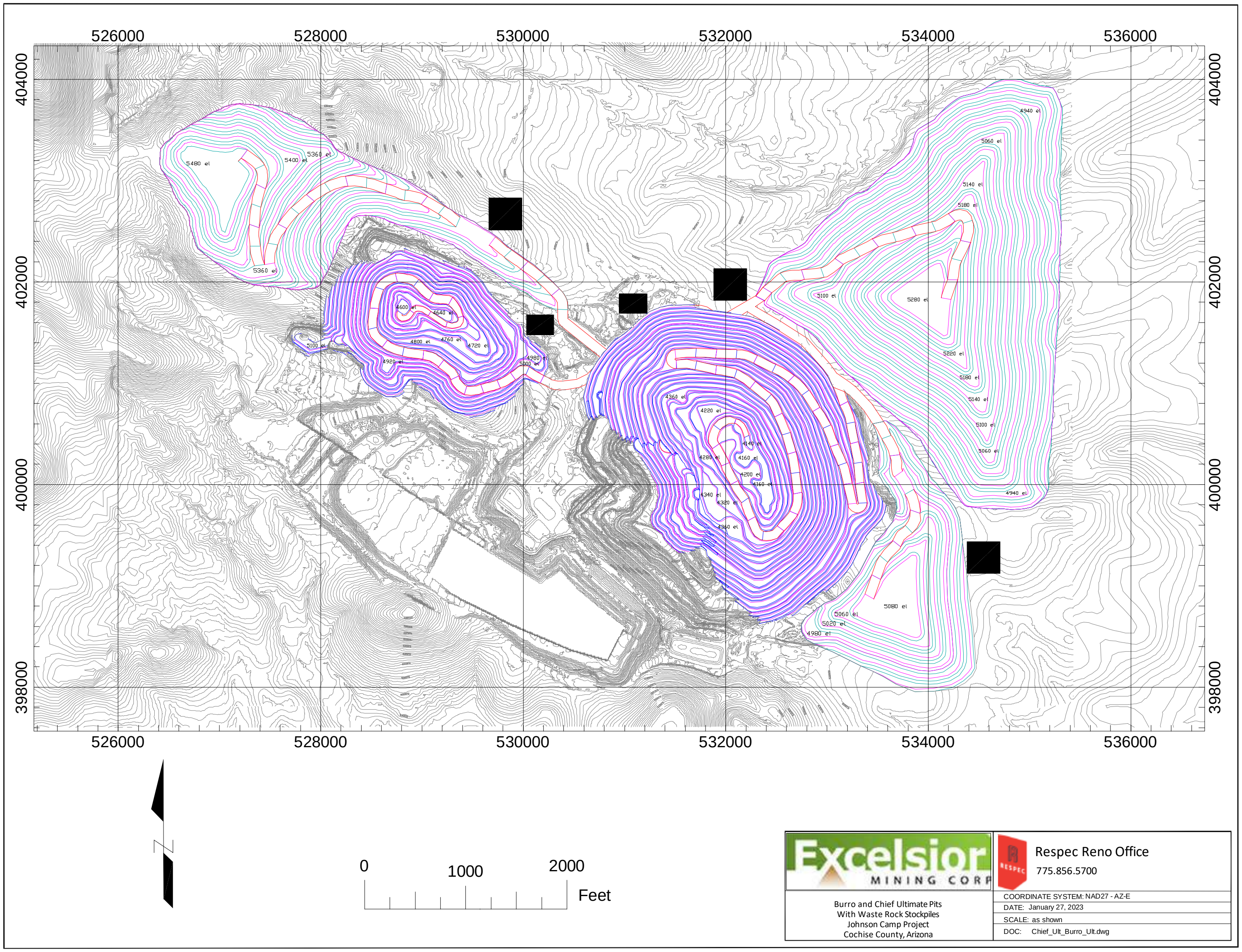


Figure 24-59: Burro Ultimate Pit Design



[redacted]
[redacted]
[redacted]
[redacted]
[redacted]
[redacted]

Figure 24-60: Chief and Burro Ultimate Pit Designs

24.16.2.1 Bench Height

Pit designs were created to use 20ft bench heights. This corresponds to the resource model block heights, and RESPEC believes this to be reasonable with respect to dilution and equipment anticipated to be used in mining.

24.16.2.2 Pit Design Slope Parameters

While no definitive geotechnical study has been provided to RESPEC. RESPEC has designed pits targeting an inner-ramp angle of 45-degrees until such time that geotechnical studies can be completed.

Pit slopes use definition of height between catch benches, bench face angle, and catch bench width. Mineralized material and most waste material will be mined on 20ft benches. Every other bench will have a catch bench 21ft wide. A bench face angle of 65º has been assumed, providing an inner-ramp slope of 45º. The slope design parameters are shown in Figure 24-61.

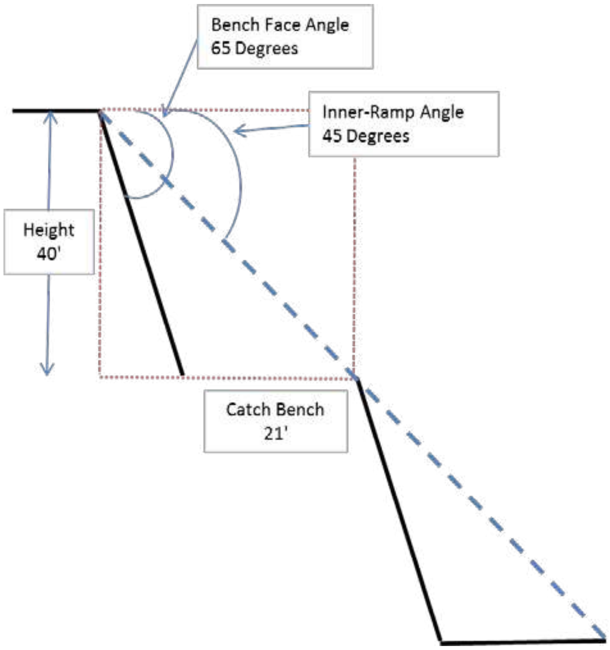


Figure 24-61: Pit Design Slope Parameters

24.16.2.3 Haul Roads

In-pit ramps and haul roads were designed to allow safe operation of haul trucks while allowing for two-way traffic. A ramp width of 85ft was used in the pit and allows for 3 times the running width of a 777 CAT truck and a safety berm of 15.33ft. Ramps use a maximum design gradient of 10%; however, some steeper sections may exist on the inside of curves for short distances. Haulage outside of the pit is required to deliver material to the waste rock stockpiles and heap leach pad. In cases where these roads require a berm on each side, the road design width is 100ft. This allows for 69ft for running width.

24.16.2.4 Dilution

The resource block model is 20ft by 20ft by 20ft high and contains grades that are diluted to this block size. The equipment that has been selected will provide reasonable selectivity with respect to these block sizes. As the resource estimate has been diluted to the block size, RESPEC believes that appropriate dilution has been accounted for in the resource modeling and has not added any additional dilution factors.

24.16.2.5 In-Pit Resources

Resources inside of the final pit designs were calculated using Surpac software. The PEA resources to be processed were Oxide, Mixed, Transition, and Sulfide. The in-pit resources are shown in Table 24-47.

Table 24-47: In-Pit Resources and Associated Waste Material

Measured	ktons	CuAs%	K Lbs CuAs	CuCN%	K Lbs CuCN	CuSu%	K Lbs CuSu	CuTot%	K Lbs CuTot
Oxide	4,963	0.24	23,854	0.02	2,158	0.04	3,586	0.32	32,219
Mixed	4,263	0.14	12,117	0.06	5,069	0.11	9,389	0.38	32,094
Transition	4,403	0.10	8,968	0.12	10,365	0.14	12,383	0.36	31,716
Sulfide	1,034	0.02	516	0.03	651	0.26	5,378	0.32	6,544
Total	14,663	0.16	45,455	0.06	18,243	0.10	30,736	0.35	102,573
Indicated									
Oxide	12,850	0.24	62,152	0.02	5,621	0.05	12,236	0.33	85,736
Mixed	30,006	0.14	85,361	0.07	39,532	0.12	74,179	0.38	230,860
Transition	6,585	0.10	13,155	0.10	13,645	0.13	17,134	0.33	43,933
Sulfide	5,551	0.04	4,644	0.05	5,023	0.38	42,341	0.47	52,007
Total	54,993	0.15	165,311	0.06	63,820	0.13	145,889	0.38	412,535
Inferred									
Oxide	2,842	0.32	17,978	0.03	1,714	0.06	3,261	0.44	24,896
Mixed	9,984	0.14	28,228	0.05	10,400	0.10	20,886	0.38	75,607
Transition	992	0.09	1,826	0.10	1,941	0.12	2,313	0.31	6,080
Sulfide	1,770	0.05	1,707	0.06	2,050	0.36	12,650	0.46	16,408
Total	15,588	0.16	49,739	0.05	16,105	0.13	39,110	0.39	122,990
Waste	ktons								
Overburden	19,875								
Oxide	6,364								
Mixed	60,973								
Transition	6,800								
Sulfide	16,836								
Total Waste	110,848								
Total Tons	196,092								
Strip Ratio	1.30								

24.16.3 Mine-Waste Facilities

Three waste rock stockpiles (“WRS”) were designed. The Chief West WRS is located to the north of Chief pit. The Burro North and Burro East WRS are located to the north and east of the Burro pit, respectively.

The WRS’s were designed using an assumed angle of repose of 34º. The design was completed using 40ft lift-heights. Catch benches of 40ft were used on each lift providing an overall design slope of 2.5H:1V. This allows for final reclamation at the overall slope.

The Chief West WRS will be used primarily for waste from the Chief pit. Once the Chief West WRS is filled, waste will be hauled to the Burro East WRS. After Burro ultimate pit is complete, waste from Chief pit will be backfilled into Burro ultimate pit.

The Burro North WRS will be used primarily for waste from the Burro pit followed with placement of waste into the Burro Est WRS.

The total dump capacity is 90.3 million tons assuming a swell factor of 1.3 and a loose density of 0.055 tons per ft^3. The waste dump capacities are shown in Table 24-48 along with the capacity of the heap leach pad. The additional waste of about 177 million cubic feet will be placed into the Burro pit as backfill.

Table 24-48: Waste Rock Stockpiles and Heap Leach Pad Capacities

	Cubic Feet (millions)	Tonnage (millions)
Chief West WRS	352.2	19.4
Burro North WRS	1,183.5	65.1
Burro East WRS	105.9	5.8
Total WRS Capacity	1,641.6	90.3
Heap Leach Pad Capacity	1,778.7	97.8

24.16.4 Production Scheduling

Mine production scheduling was done using MineSched software. Scheduling targets 25 million lbs of Cu per year processed. Constraints on processed pounds of copper and number of benches mined per period prohibited the mine from producing to full capacity.

Waste material was modeled as either fill waste or rock waste to estimate contractor mining costs. Fill waste is material mined from historical dumps which is assumed to not require drilling and blasting. Rock waste is all other waste material mined and is assumed to require drilling and blasting.

The production schedule was created using monthly periods so that appropriate lag times for copper recovery could be used for the process production schedule. The schedule was then summarized in yearly periods as shown in Table 24-49. The "Yr-1" is used to represent pre-production. Note that while some material is sent to the leach pad during pre-production, no metal production is attributed to this material until Year 1.

Table 24-49: Mine Production Schedule

		Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Yr 21	Total
West_pit	Total Leach	ktons	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	823	4,723	6,222	5,402	1,762	-	18,932
		CuAS Cu %	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.090	0.125	0.135	0.180	0.242	-	0.153
		CuAS K Lbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1,482	11,824	16,795	19,460	8,533	-	58,094
		CuCN Cu %	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.02	0.04	0.04	0.04	0.04	-	0.04
		CuCN K Lbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	318	3,496	4,722	4,631	1,577	-	14,744
		CuSu Cu %	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.02	0.02	0.04	0.07	0.07	-	0.044
		CuSu K Lbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	353	1,733	4,988	7,311	2,312	-	16,698
		CuTot %	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.22	0.28	0.30	0.36	0.39	-	0.32
		CuTot K Lbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3,576	26,595	37,740	38,858	13,728	-	120,497
	ovb_wst	ktons	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	59	220	-	-	-	-	278
	ox_wst	ktons	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	46	241	671	151	3	-	1,112
	su_wst	ktons	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	222	723	356	9	-	-	1,310
	trans_wst	ktons	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	764	1,133	913	331	39	-	3,179
	wox_wst	ktons	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3,418	6,167	4,816	1,127	86	-	15,614
	Total Waste	ktons	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4,508	8,483	6,757	1,618	128	-	21,494
	Total Mined	ktons	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5,330	13,207	12,979	7,020	1,890	-	40,426
	Strip Ratio	W:O																	5.48	1.80	1.09	0.30	0.07		1.14
East_p2_pit	Total Leach	ktons	-	-	3	2,104	2,694	2,100	2,067	3,431	2,264	768	903	-	-	-	-	-	-	-	-	-	-	-	16,334
		CuAS Cu %	-	-	0.148	0.147	0.159	0.169	0.212	0.173	0.138	0.111	0.116	-	-	-	-	-	-	-	-	-	-	-	0.161
		CuAS K Lbs	-	-	8	6,176	8,578	7,111	8,753	11,871	6,243	1,703	2,095	-	-	-	-	-	-	-	-	-	-	-	52,539
		CuCN Cu %	-	-	0.01	0.03	0.03	0.05	0.07	0.07	0.08	0.10	0.09	-	-	-	-	-	-	-	-	-	-	-	0.06
		CuCN K Lbs	-	-	1	1,134	1,662	2,110	2,902	5,074	3,811	1,505	1,588	-	-	-	-	-	-	-	-	-	-	-	19,786
		CuSu Cu %	-	-	-	0.07	0.10	0.13	0.14	0.17	0.21	0.21	0.13	-	-	-	-	-	-	-	-	-	-	-	0.143
		CuSu K Lbs	-	-	-	3,064	5,512	5,258	5,798	11,971	9,648	3,163	2,411	-	-	-	-	-	-	-	-	-	-	-	46,825
		CuTot %	-	-	0.19	0.32	0.35	0.39	0.46	0.43	0.44	0.41	0.34	-	-	-	-	-	-	-	-	-	-	-	0.40
		CuTot K Lbs	-	-	10	13,326	18,812	16,306	18,880	29,717	19,702	6,371	6,095	-	-	-	-	-	-	-	-	-	-	-	129,220
	ovb_wst	ktons	-	-	4,328	3,225	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	7,553
	ox_wst	ktons	-	-	6	1,177	372	139	31	2	-	-	2	-	-	-	-	-	-	-	-	-	-	-	1,729
	su_wst	ktons	-	-	-	934	114	52	23	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1,124
	trans_wst	ktons	-	-	-	516	412	330	63	22	35	12	8	-	-	-	-	-	-	-	-	-	-	-	1,399
	wox_wst	ktons	-	-	51	6,647	3,415	1,471	504	132	6	1	5	-	-	-	-	-	-	-	-	-	-	-	12,232
	Total Waste	ktons	-	-	4,386	12,498	4,314	1,992	621	156	41	13	15	-	-	-	-	-	-	-	-	-	-	-	24,037
	Total Mined	ktons	-	-	4,388	14,602	7,008	4,092	2,688	3,587	2,305	781	918	-	-	-	-	-	-	-	-	-	-	-	40,370
	Strip Ratio	W:O			1,715.10	5.94	1.60	0.95	0.30	0.05	0.02	0.02	0.02												1.47
East_p3_pit	Total Leach	ktons	-	-	-	-	-	-	227	481	1,579	4,746	3,915	3,395	3,331	3,577	3,395	3,760	4,274	995	-	-	-	-	33,672
		CuAS Cu %	-	-	-	-	-	-	0.153	0.149	0.125	0.107	0.120	0.209	0.178	0.158	0.123	0.134	0.131	0.175	-	-	-	-	0.143
		CuAS K Lbs	-	-	-	-	-	-	693	1,434	3,959	10,202	9,429	14,212	11,846	11,305	8,346	10,062	11,207	3,484	-	-	-	-	96,177
		CuCN Cu %	-	-	-	-	-	-	0.03	0.02	0.02	0.03	0.04	0.05	0.06	0.06	0.08	0.09	0.09	0.10	-	-	-	-	0.06
		CuCN K Lbs	-	-	-	-	-	-	114	148	565	2,688	2,911	3,502	4,004	4,129	5,222	6,588	8,109	2,011	-	-	-	-	39,991
		CuSu Cu %	-	-	-	-	-	-	0.05	0.03	0.07	0.13	0.16	0.20	0.22	0.22	0.24	0.20	0.11	0.10	-	-	-	-	0.170
		CuSu K Lbs	-	-	-	-	-	-	219	269	2,113	12,512	12,601	13,269	14,806	15,554	16,365	15,366	9,590	1,978	-	-	-	-	114,643
		CuTot %	-	-	-	-	-	-	0.34	0.34	0.29	0.31	0.34	0.47	0.46	0.43	0.44	0.43	0.34	0.38	-	-	-	-	0.39
		CuTot K Lbs	-	-	-	-	-	-	1,551	3,282	9,078	29,716	26,778	31,628	30,813	30,988	29,933	32,016	28,905	7,473	-	-	-	-	262,163
	ovb_wst	K Tons	-	-	-	-	-	-	8,635	1,745	10	-	-	-	-	-	-	-	-	-	-	-	-	-	10,390
	ox_wst	K Tons	-	-	-	-	-	-	49	22	330	958	102	16	3	1	-	5	15	5	-	-	-	-	1,506
	su_wst	K Tons	-	-	-	-	-	-	167	10,530	2,507	479	285	86	90	3	-	8	35	-	-	-	-	-	14,190
	trans_wst	K Tons	-	-	-	-	-	-	4	1	261	943	465	176	22	8	14	12	17	-	-	-	-	-	1,924
	wox_wst	K Tons	-	-	-	-	-	-	292	5,534	6,321	6,351	4,467	1,335	1,425	570	76	205	194	114	-	-	-	-	26,884
	Total Waste	K Tons	-	-	-	-	-	-	9,147	17,831	9,429	8,732	5,319	1,613	1,539	582	90	231	260	119	-	-	-	-	54,893
	Total Mined	K Tons	-	-	-	-	-	-	9,374	18,313	11,008	13,478	9,234	5,007	4,870	4,159	3,484	3,990	4,534	1,114	-	-	-	-	88,565
	Strip Ratio	W:O							40.33	37.07	5.97	1.84	1.36	0.48	0.46	0.16	0.03	0.06	0.06	0.12					1.63

		Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Yr 21	Total
Total	Total Leach	K Tons	782	4,958	4,251	4,329	4,246	4,088	2,846	3,912	3,843	5,514	4,818	3,395	3,331	3,577	3,395	3,760	5,096	5,718	6,222	5,402	1,762	-	85,243
		CuAS Cu %	0.215	0.159	0.171	0.159	0.164	0.158	0.188	0.170	0.133	0.108	0.120	0.209	0.178	0.158	0.123	0.134	0.124	0.134	0.135	0.180	0.242	-	0.153
		CuAS K Lbs	3,370	15,788	14,564	13,782	13,897	12,889	10,726	13,304	10,202	11,905	11,524	14,212	11,846	11,305	8,346	10,062	12,689	15,308	16,795	19,460	8,533	-	260,505
		CuCN Cu %	0.01	0.04	0.07	0.07	0.05	0.09	0.07	0.07	0.06	0.04	0.05	0.05	0.06	0.06	0.08	0.09	0.08	0.05	0.04	0.04	0.04	-	0.06
		CuCN K Lbs	213	3,603	6,315	5,691	4,559	7,128	4,061	5,222	4,375	4,194	4,499	3,502	4,004	4,129	5,222	6,588	8,427	5,507	4,722	4,631	1,577	-	98,168
		CuSu Cu %	0.01	0.07	0.12	0.13	0.12	0.14	0.12	0.16	0.15	0.14	0.16	0.20	0.22	0.22	0.24	0.20	0.10	0.03	0.04	0.07	0.07	-	0.13
		CuSu K Lbs	84	7,398	9,791	11,416	10,205	11,815	6,710	12,240	11,762	15,675	15,012	13,269	14,806	15,554	16,365	15,366	9,943	3,711	4,988	7,311	2,312	-	215,735
		CuTot %	0.30	0.33	0.40	0.40	0.38	0.41	0.41	0.42	0.37	0.33	0.34	0.47	0.46	0.43	0.44	0.43	0.32	0.30	0.30	0.36	0.39	-	0.37
		CuTot K Lbs	4,622	33,014	33,608	34,689	31,986	33,738	23,449	32,999	28,781	36,088	32,872	31,628	30,813	30,988	29,933	32,016	32,481	34,068	37,740	38,858	13,728	-	638,099
	ovb_wst	K Tons	1,645	9	4,328	3,225	-	-	8,635	1,745	10	-	-	-	-	-	-	-	59	220	-	-	-	-	19,875
	ox_wst	K Tons	490	1,504	24	1,178	376	139	80	24	330	958	104	16	3	1	-	5	60	246	671	151	3	-	6,364
	su_wst	K Tons	0	208	3	934	114	52	190	10,531	2,507	479	285	86	90	3	-	8	257	723	356	9	-	-	16,836
	trans_wst	K Tons	16	237	21	522	427	334	67	23	296	956	473	176	22	8	14	12	780	1,133	913	331	39	-	6,800
	wox_wst	K Tons	992	4,483	774	6,678	3,420	1,478	796	5,666	6,327	6,352	4,472	1,335	1,425	570	76	205	3,612	6,281	4,816	1,127	86	-	60,973
	Total Waste	K Tons	3,143	6,442	5,150	12,537	4,339	2,003	9,769	17,988	9,471	8,746	5,334	1,613	1,539	582	90	231	4,768	8,602	6,757	1,618	128	-	110,848
	Total Mined	K Tons	3,925	11,400	9,401	16,866	8,584	6,091	12,614	21,900	13,313	14,259	10,152	5,007	4,870	4,159	3,484	3,990	9,864	14,320	12,979	7,020	1,890	-	196,092
	Strip Ratio	W:O	4.02	1.30	1.21	2.90	1.02	0.49	3.43	4.60	2.46	1.59	1.11	0.48	0.46	0.16	0.03	0.06	0.94	1.50	1.09	0.30	0.07		1.30

24.16.4.1 Mine Equipment Requirements

The PEA mining is based on contract mining, and equipment requirements will be the responsibility of the contractor to maintain production. However, for the purpose of estimating the equipment and personnel requirements, 100-ton CAT 777 trucks and CAT 992 wheeled loaders were assumed to be used as the primary production equipment.

24.16.4.2 Mine Operations Personnel

It is assumed that the Johnson Camp mine will be in operation along with other excelsior operations at the same time. Thus, the mine management, engineering, and geological services will be shared services. Accordingly, the cost of these services will be somewhat variable with the amount of mining activity within the property. For this reason, mine operations general costs were estimated based on a $0.03/ton mined. This cost was provided by Excelsior from previous studies.

24.16.5 Pit Dewatering

A groundwater model developed for the Gunnison Project site[6] was used to evaluate the dewatering requirements for the proposed expansion of two open pits at the Johnson Camp Mine (JCM) site. The Gunnison Project model was developed using the U.S. Geological Survey (USGS) MODFLOW-NWT model code. The JCM site is an open pit mining operation, located north of the Excelsior Gunnison Project (Figure 24-62).

The model domain includes the town of Dragoon and the entire drainage basin north of the town. There is limited groundwater pumping in this groundwater basin. Because there is limited groundwater pumping in the basin and stable groundwater levels, the area is simulated in a steady-state flow condition. To simulate the development of the open pit, MODFLOW drain cells were added to the model to represent the discharge of groundwater from the ultimate pits. Figure 24-63 shows the drain cells (color coded by model layers) based upon the final, end of mining, pit elevation contours. The drain elevations and layer designations are set based on the contours and the elevation of the model layers.

Figure 24-64 shows the simulated groundwater drawdown at the site as predicted by the steady-state simulation for the deepest model layer intersected (layer3). The maximum drawdown is predicted to be 320 feet at the deepest point of the pit. The steady state model simulation predicts the dewatering rate will reach about 126 gallons per minute (gpm).

To analyze the development of the JCM pits over time, a second model run was completed transiently for a period of 21 years. The mine plan (based on 2022_JohnsonCamp_Sched_V7.xlsx[7]) assumes that the East Pit (Figure 24-63) is excavated over the first 17 years of the project, while the West Pit starts excavation in Year 16 ending in Year 21. In addition to the final pit contours, contour maps were also consulted for an intermediate period for the East Pit from Year 6. An intermediate period contour map was also consulted for the West Pit.

Figure 24-65 shows an example of Year 6 drain cell locations as compared to elevation contours for the East Pit. The same process was used to set Year 17 elevations, using the ultimate East Pit contours. Intervening periods were smoothly decreased based on the changes estimated from the starting and ending periods (6 to 17). The initial land surface was based upon a regional 40-foot contour map. Transitions using annual stress periods result in unrealistic sudden drops in mining elevations which result in dewatering “spikes” at the beginning of each year. To smooth this effect, monthly time steps were used. Figure 24-66 shows the drawdown after Year 21, representing the ultimate pit depth. Figure 24-67 illustrates the projected flows into all drain cells during the simulation. As the pits are deepened,

---

[6] Groundwater Modeling Report – Gunnison Project Site (2016, Appendix I of Aquifer Protection Permit application)

[7] A newer version of the Mine Plan (v8) is now available, but the overall schedule of pit excavation is about the same.

flows increase at each (annual) step of the simulation, with recovery over the subsequent monthly time steps. A 15-point running average was used to smooth the dewatering rate which is more realistic (Figure 24-67). Flows begin to increase in Year 4 as the pit depth is deepened below the simulated water table. Flows rise to over 100 gallons per minute (gpm) by Year 6, and 200 gpm by Year 17. When the pits reach their ultimate depth in Year 21, the dewatering rate is about 400 gpm.

24.16.5.1 Limitations

This analysis has a number of limitations which need to be considered when interpreting the results. These include:

- The model has not been re-calibrated to address water level data for the Johnson Camp Mine, and represents the last model calibration conducted in 2021, after a year of operation at the Gunnison site.
- The hydraulic conductivity values represent a good match with regional values, but may not represent local conditions at the JCM site. The regional zones used may not reflect local variations specific to JCM.
- Specific yield at JCM is set similar to the Gunnison site. This may be excessive since fracturing is abundant at the Gunnison site, but limited at the JCM site.
- The model cell dimensions are relatively coarse in the vicinity of JCM (300-by-300 feet). At Gunnison, the model cells are more refined (75-by75 feet) to accommodate the spacing of injection and recovery wells.

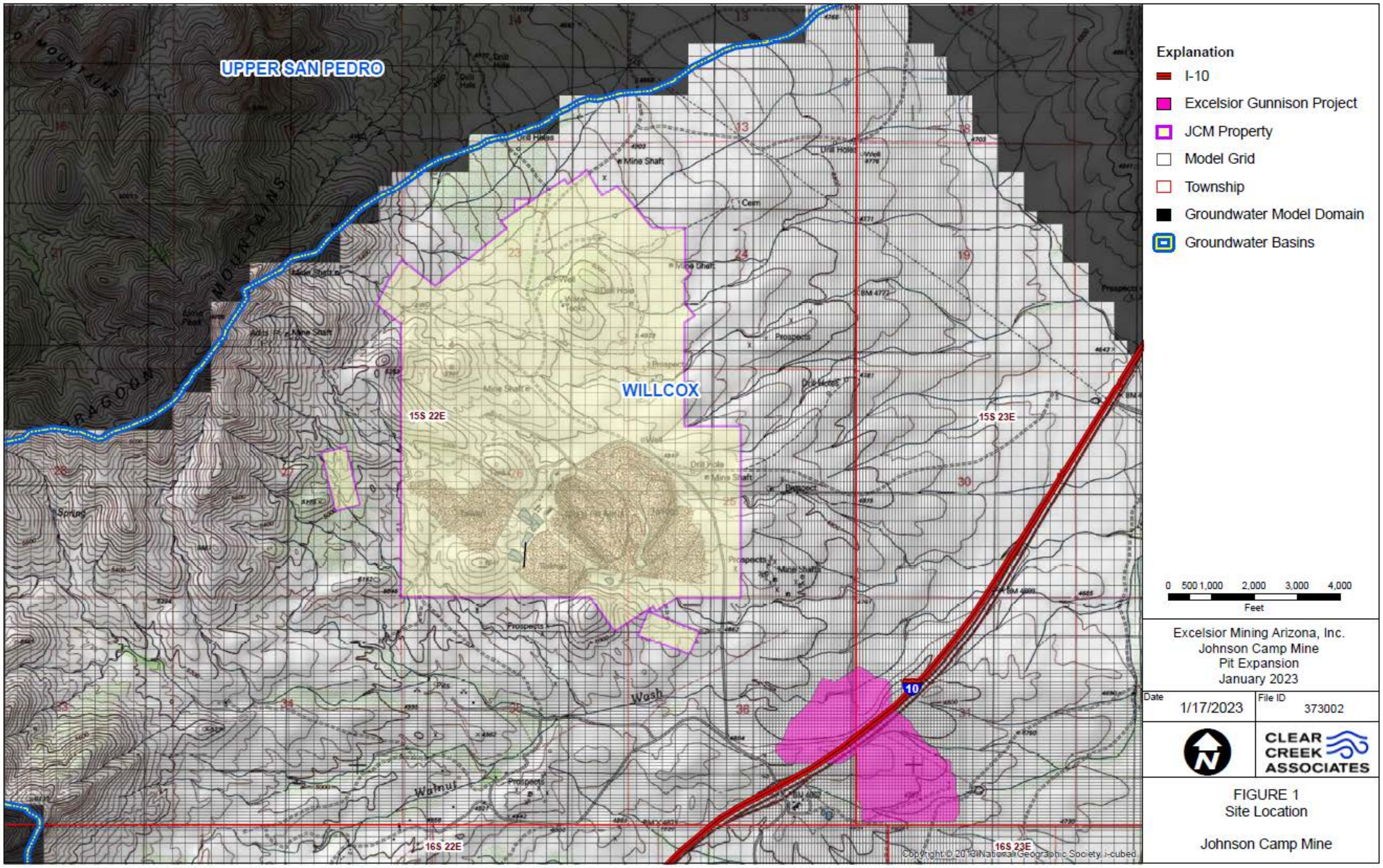


Figure 24-62: Site Location

Figure 24-63: Ultimate Pit Contours and Model Drain Cells

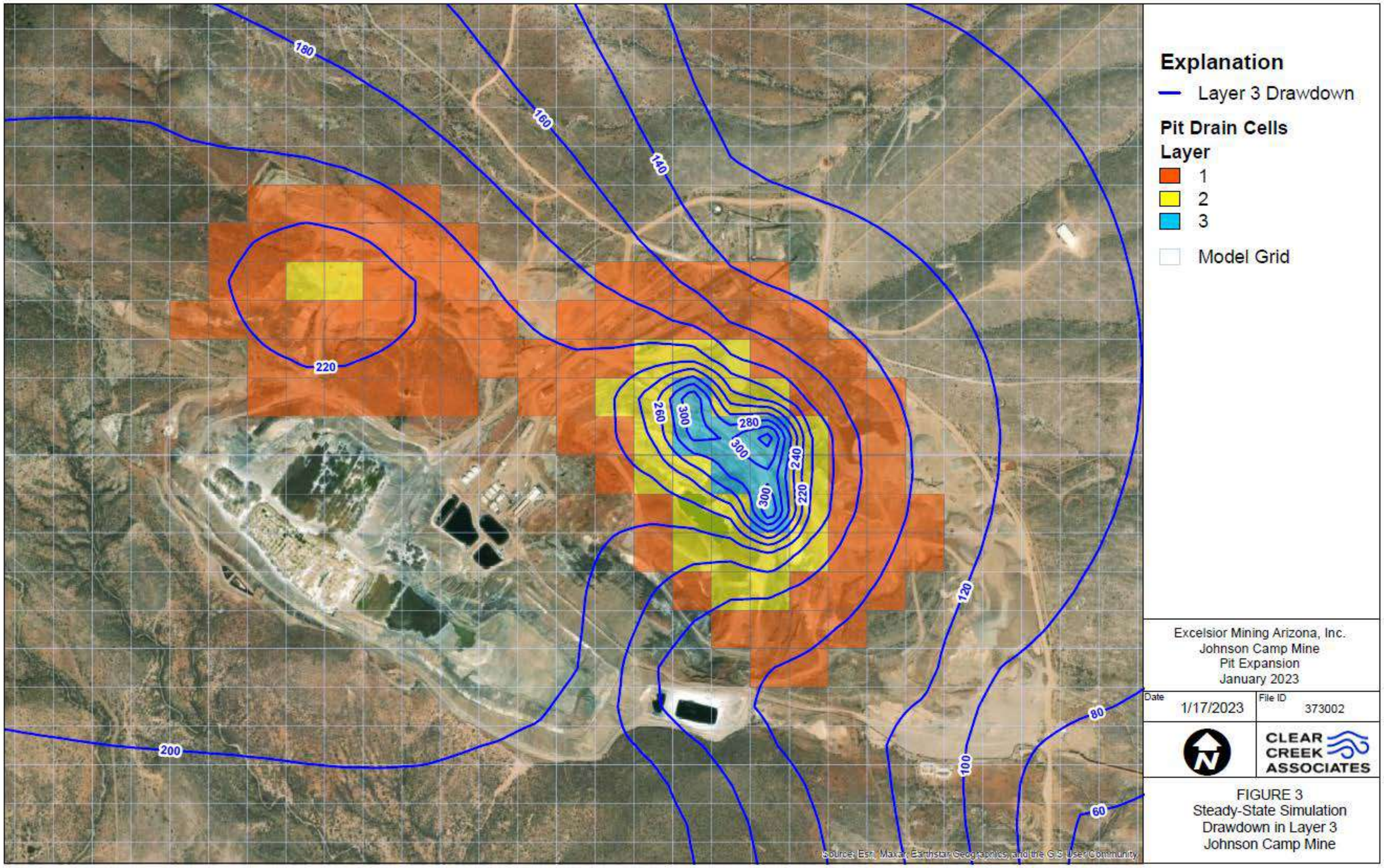


Figure 24-64: Steady-State Simulation Drawdown in Layer 3

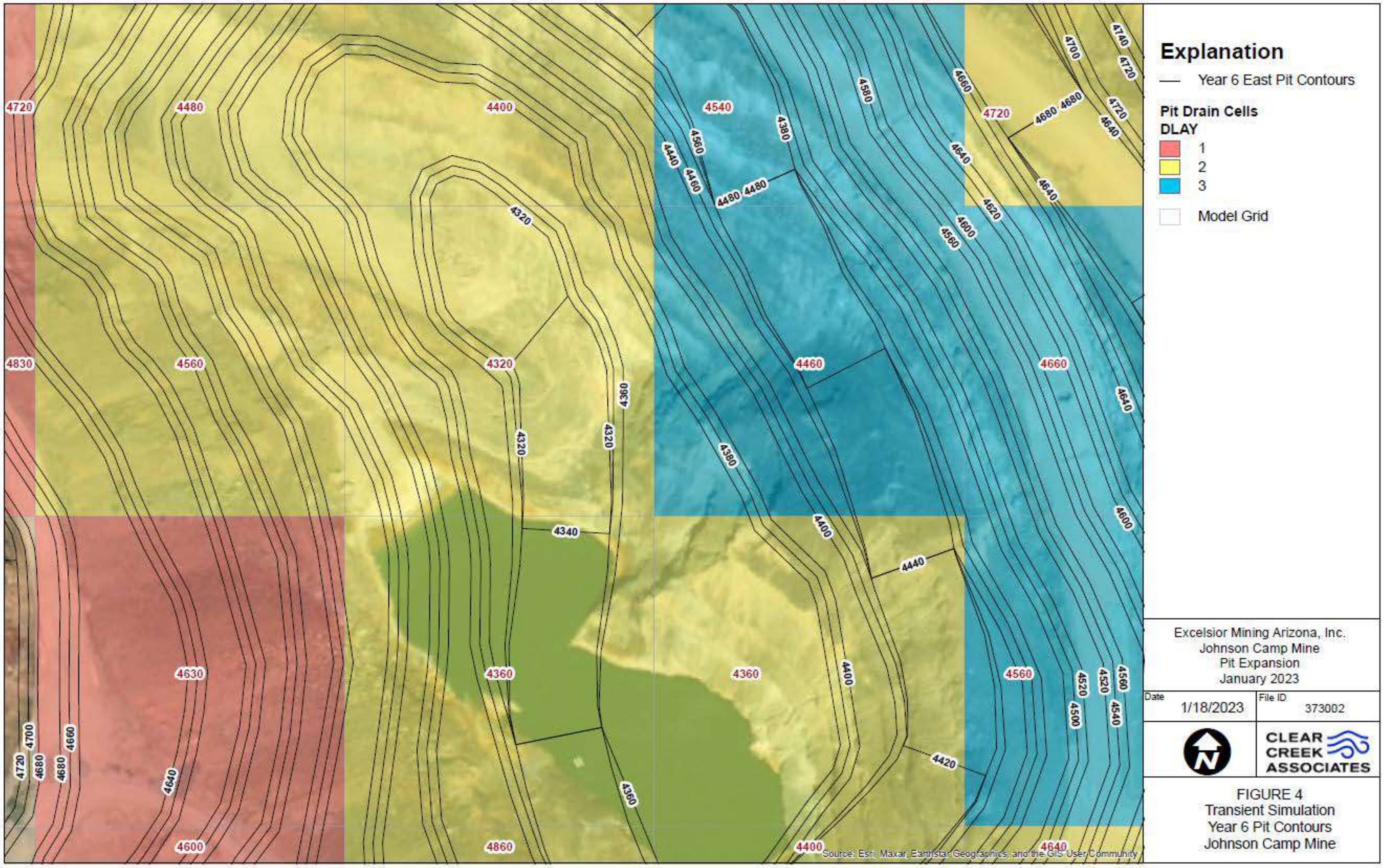


Figure 24-65: Transient Simulation Year 6 Pit Contours

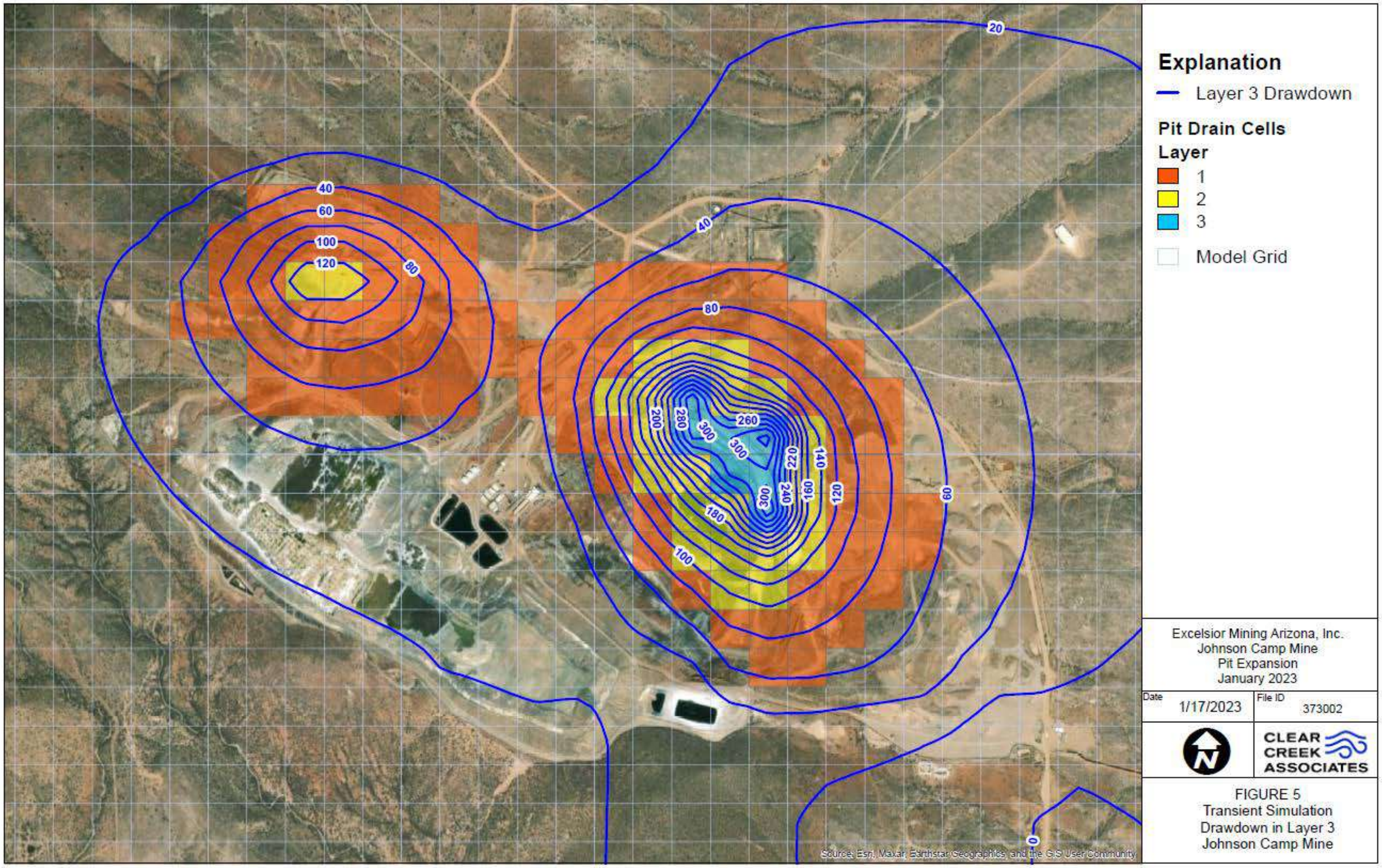


Figure 24-66: Transient Simulation Drawdown in Layer 3

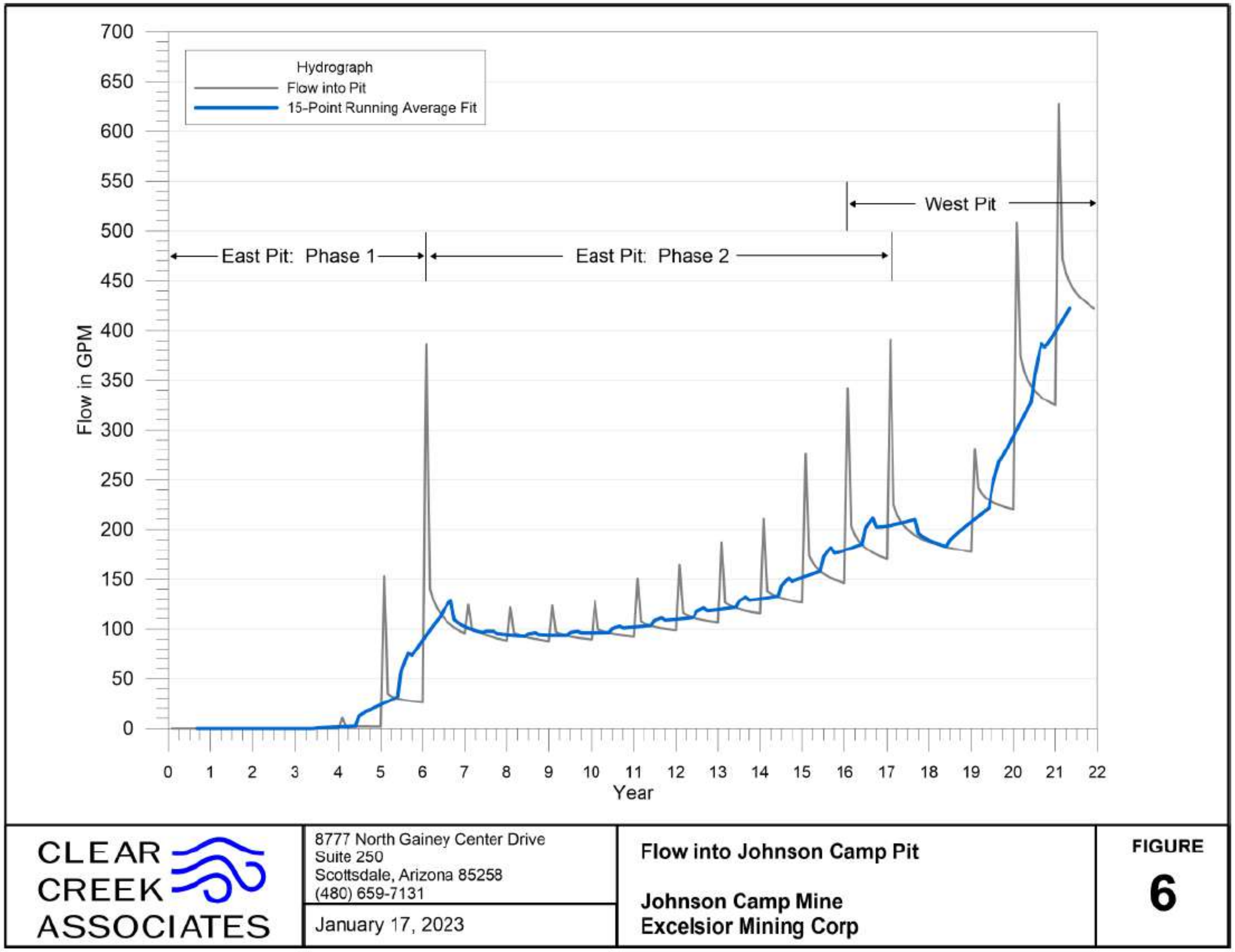


Figure 24-67: Flow into JCM Pit

24.17 RECOVERY METHODS

The existing leach pads, Pads 1, 2, and 3, have been abandoned for future mining of any new material extracted from the Burro and Copper Chief Pits. The new leach pad area, Pad 5, is to be located northeast of the existing plant facility and is to be designed such that leach solutions flow by gravity into the new combined ILS-PLS (Intermediate Leach Solution and Pregnant Leach Solution from Pad 5) pond located down slope of the new leach pad. The combined ILS-PLS solution will be pumped back to the existing Johnson Camp Mine SX-EW plant. A storm water pond is also provided.

24.17.1 Leach Pad 5

The location of the leach pad has been determined by Glasgow Engineering, based on an analysis of topographic, geologic, and hydrologic characteristics, and is not within the scope of this study. The proposed leach pad and ancillary facilities are to be located in a drainage northeast of the Copper Chief Pit expansion.

The leach pad design is approximately 8,000,000 ft^2 in area and oriented to match existing topography so that it allows gravity drainage of solutions down to the eastern toe of the pad at 4,900 ft elevation for collection and transport by pumping system back up to the PLS storage pond at approximately 5,030 ft elevation. The pad will be constructed on a prepared base that has been cut from within the pad area and filled with borrow materials from within the pad or from elsewhere on the mine site. It is anticipated that cut and fill volumes will be approximately 440,000 yd^3 net. About 275,0000 yd^3 of soil liner (a clayey alluvial material) will be taken from within the pad perimeter and shall be screened then graded to form a 12-inch layer beneath the HDPE liner. After installing an HDPE liner over the entire pad, a system of perforated leachate collection pipes will be installed upon the liner (and in some cases upon a pipe bedding material).

Approximately half of the designed leach pad will be constructed for the initial operation, the cost of which is included as initial CAPEX. The leach pad will require two expansions, the cost of which are included in sustaining CAPEX. The first expansion entail construction of the western half of the designed leach pad in Year 4 of mine operation. The second expansion is planned as an extension to the northeast of the designed pad. The second expansion has not been designed or permitted, but will not be required until approximately Year 14 of operation. The surface area required for the second expansion is approximately 35 percent of the designed leach pad (Initial and first expansion). Additional area for leach pad construction is available north of the legacy heaps (1-3) and could be used as the second expansion, but would require design and permitting prior to construction.

The collection pipe system will be buried in a course of overliner material consisting of minus 1 ½” to plus ¾” material (also referred to as Liner Protection material). This material will be placed to a depth of not more than 36 inches above the HDPE liner. It is anticipated that this material, totalling approximately 512,000 yd^3, will be taken from on-site stockpiles of Bolsa Quartzite and crushed using the mine’s existing crushing facilities. The overliner will meet standard specifications for hydraulic conductivity.

The proposed leaching will be on crushed, agglomerated, and stacked material. Crush size and the need to agglomerate will be evaluated during the sulfide and transition material testing program to be completed during the first half of 2023.

Based on column testwork reports described in Section 24.13.2, the material to be leached ranges in sulfuric acid consumption from 20 lbs/ton for the Pioneer shale to 70 lbs/ton for the Martin formation. The average acid consumption for the mineral resources in the conceptual mine plan is approximately 40 lb/ton of material. Approximately 30 percent of the acid demand will be added during the agglomeration process prior to placement on the leach pad.

Lifts are planned to be 65 feet (20 meters) high. Sulfide materials are planned to be placed on the leach pad for the recovery of copper, as presented in Section 24.13.3. To accomplish the recovery of sulfide copper, the sulfides must be oxidized. The process requires aeration of the leach pad to provide the oxygen necessary for the oxidation process. The process is also enhanced by the presence of bacteria and elevated temperatures in the heap. Aeration is planned to be facilitated by a network of perforated pipe at the bottom of each lift. Perforated pipe 6 inches in diameter are planned for placement at 20-foot intervals connected to a 12-inch diameter manifold. Air will be injected by blowers at a pressure of approximately 10-15 pounds per square inch. Additional testing of transition and sulfide materials from JCM is planned to test the efficacy of this method and refine the parameters of the design and the associated copper recovery kinetics.

The presence of abundant pyrite in the sulfide-bearing leach materials is projected to enhance the oxidation of copper in the form of chalcopyrite, as suggested in Section 24.13.3. The oxidation of pyrite will generate ferric iron to enhance the oxidation of copper sulfides. The leaching plan anticipates that the acid addition to the heap can be reduced to the amount being added in the agglomeration process and that acid addition to the raffinate can be greatly reduces or eliminated when the leach heap has reached maturity.

To achieve maximum metal extraction, several leaching parameters must be optimized in concert, based on an appropriate testing program. These include the irrigation rate, acid concentration, bacteria, and leach cycle time. The irrigation system will be laid out on the heap surface and the drip lines should be fairly closely spaced in order to wet the entire lift of material.

The PLS coming from various leach cells will comingle and be pumped to the existing JCM PLS pond. This mixing will help maintain a uniform PLS grade going to the solvent extraction plant. The PLS pond will promote settling of any solids entrained in the PLS due to a precipitation event or a broken leach line. However, it may be advantageous to settle solids or to filter them at the Pad 5 collection sump prior to pumping solution down to the JCM PLS pond, which is not included in the current plans.

Figure 24-68 is an adaptation from a 2014 report by Hydrogeologica (2014) showing that the solution balance for Pad 5 includes approximately 3,078 gpm of PLS draining from the leach pad into JCM Pond 3 (PLS pond).

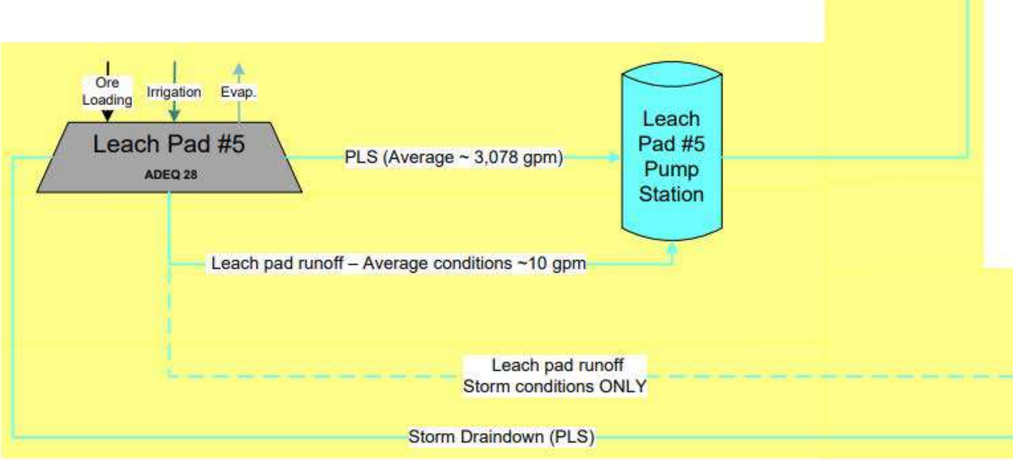


Figure 24-68: Pad 5 Solution Management

24.17.2 Solvent Extraction

The existing JCM SX circuit consists of two trains of mixer-settlers that strip copper from the PLS and transfer it to the lean electrolyte solution. Each train has two extraction settlers and one strip settler. The extraction settlers use an extractant dissolved in a petroleum-based diluent (collectively called the “organic”) to extract copper from the aqueous phase. The strip settlers (one in each train) use a high-acid solution (lean electrolyte) to strip copper from the organic phase. The aqueous phase (strong electrolyte) is then pumped to the existing JCM tankhouse for recovery by electrowinning.

The SX trains for the JCM plant are operated in series such that the entire PLS flow through each train passes through both extraction settlers in the train. The organic passes counter-current through both extraction settlers, transferring copper from the PLS and becoming “loaded organic”. The copper-bearing loaded organic is mixed with lean electrolyte in the strip pumper mixers to transfer the copper from the extractant in the organic phase to the aqueous electrolyte solution. The strip settler allows the immiscible liquids to separate in laminar flow. The rich electrolyte then flows to the Electrolyte Filter Feed Tank.

Stripped organic is sent to the extraction pumper mixers where intimate contact between the organic and PLS solutions promotes exchange of copper ions by the extractant in the organic phase. The extraction settlers allow the immiscible liquids to separate in laminar flow so that the aqueous phase (raffinate) and organic phase can be collected in separate launders at the end of the settler. Raffinate is re-acidified in the aqueous launder of the second extraction settler and flows by gravity to the Raffinate Pond. The partially loaded organic from the second extraction settler flows to the pumper mixers of the first extraction settler and exchanges copper from the other half of the PLS stream. Fully loaded organic from the first extraction settler flows to the Loaded Organic Tank. The SX process is designed to extract 92% of the copper contained within the PLS at an incoming copper grade of up to 1.50 grams per liter (g/L).

24.17.3 Electrowinning

Rich electrolyte solution advances from the solvent extraction area and flows by gravity to the Electrolyte Filter Feed Tank. Electrolyte is pumped from this tank through two electrolyte filters to remove entrained organic emulsion and particulates from electrolyte prior to electrowinning. The filters are backwashed periodically with water (or lean electrolyte solution) and air from a blower. In Stage 1, filter backwash solution flows by gravity to the JCM Raffinate Pond. In the Stage 2 and 3 plant, the filters are backwashed with lean electrolyte and the backwash solution is pumped to the PLS Pond.

Filtered electrolyte solution is pumped to an electrolyte recirculation tank through the electrolyte heat exchangers. The filtered rich electrolyte flows through one heat exchanger and is warmed by lean electrolyte returning to solvent extraction from electrowinning. Rich electrolyte is heated in the trim heater, when required, with supplemental heat from a hot water heating system to the final temperature, typically 45°C, for electrowinning. When supplemental heat is not required, lean electrolyte flows through the trim heater, countercurrent to the flow of rich electrolyte being heated.

After returning by gravity from the SX stripper to the rich electrolyte tank, rich electrolyte is pumped through a series of filters and heat exchangers to the commercial electrolyte recirculation tank. The commercial solution, a blend of the rich and lean electrolyte solution, is pumped to the EW cells. The solution exits the cells overflowing by gravity to the Lean Electrolyte Tank. A portion of the lean electrolyte is pumped back to the strip stages in the SX for copper recovery with the balance of lean electrolyte overflowing through a cross connection pipe into the commercial tank.

Copper is plated onto stainless steel cathode blanks in the EW cells. The copper cathodes are harvested on a weekly basis. The tankhouse has an overhead bridge crane for transporting cathodes (and anodes) to and from the cells using a lifting “strongback” frame. Harvested cathodes are washed in the Cathode Wash Tanks using circulation pumps. Washed cathodes are removed from the stainless-steel blanks, sampled, weighed, and banded using a semi-automatic stripping machine. Copper produced by this process is LME Grade A for sale on the world market in 2-to-3-ton packages.

The electrowinning operation will also require small electrolyte bleeds to control the buildup of impurities. This bleed stream can either be returned to the extraction stage or to the Raffinate pond.

24.17.4 Tank Farm

The tank farm contains tanks, pumps, and filters for handling solutions needed for the SX-EW process. The primary process function of the tank farm is storage and transfer of solutions. However, there are two process functions that take place in the tank farm: electrolyte filtration and crud treatment.

Electrolyte filters in the tank farm remove impurities from the rich electrolyte returning from SX to prevent contamination of the tankhouse and electrolyte system. Rich electrolyte flows by gravity to the Electrolyte Filter Feed Tank and is pumped through one or more anthracite-garnet filters to remove entrained organic and particulates that could interfere with electrowinning. Filtered rich electrolyte flows to the Electrolyte Recirculation Tank. The filters are periodically backwashed to remove impurities and to maintain design flow rates through the filter media.

Crud is a mixture of solids, organic liquid, and aqueous solution that (a) accumulates at the organic/aqueous interface in the settlers or (b) may be any mixture of aqueous and organic liquids that requires separation. Crud is removed by suction from the settlers and needs to be treated to separate the three phases for reuse in the process or, in the case of the solids, for disposal. Crud also comes from the mixture of aqueous, organic, and solids that accumulates in the electrolyte filters. The crud treatment system consists of the following major equipment.

- Crud Holding Tank
- Crud Treatment Tank
- Crud Centrifuge (“Tri-canter”)
- Recovered Organic Tank

Crud from the Crud Holding Tank will be pumped to the Crud Treatment Tank, an agitated, cone-bottom tank. Amendments including clay and diatomaceous earth can be added to the Crud Treatment Tank to assist in separation of the phases. The Crud Centrifuge is a horizontal-axis centrifuge that separates the crud into its three component phases, allowing aqueous and organic liquids to be returned to the process, while solids are collected in a container for offsite disposal.

24.18 PROJECT INFRASTRUCTURE

The Johnson Camp Mine is an existing and operating copper hydrometallurgical plant. Figure 24-69 shows the location of the open pits, waste dumps, SX-EW plant facilities and mine infrastructure that will be used when mine operations in the Burro and Copper Chief pits resume.

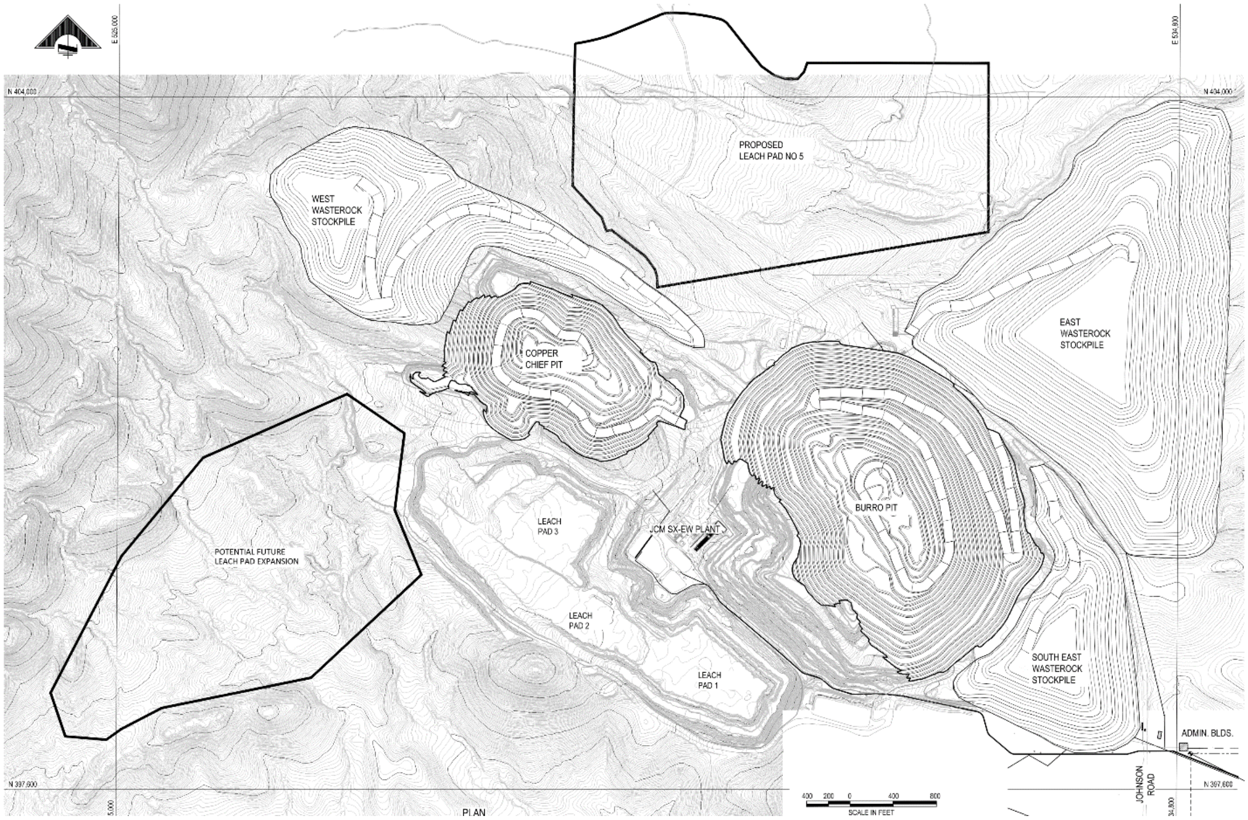


Figure 24-69: Johnson Camp Mine Facilities

24.18.1 Access

The JCM site is accessed from the North Johnson Road exit from the I-10 freeway, by traveling approximately 1 mile north. The Stage 1 JCM plant area is approximately 1.6 miles from the main entrance.

Inert waste rock will be deposited on the east and southeast waste rock stockpiles in the beginning years of operation. An area north of the Copper Chief has been designated for future years of operation to expand waster rock storage capacity. Waste rock samples will be tested to confirm their inert behavior at regular production rate intervals.

24.18.2 Power

JCM has an existing 69 kV power line that lands at the JCM substation at the plant where it is transformed to 5 kV for distribution around the property. The existing power line is owned by the Sulfur Springs Valley Electric Cooperative Inc. located in Willcox, Arizona. Previously, there was a substation located at the JCM crushing facility, but that facility was relocated to the Gunnison wellfield in 2020. To power the raffinate and PLS pumps at Pad 5, a 5 KV power line will have to be run to the pump station where there is planned a 1500 kVA step down transformer to 480V.

24.18.3 Water Supply & Distribution

Fresh water is supplied from existing wells on the JCM property and pumped to an existing process/fire water storage tank. The lower portion of the storage tank is reserved for fire water. Process water for plant use is taken from the storage tank above the fire water reserve level. Potable water for the JCM site is provided by the existing section 19 well, chlorinator building, and potable water tank.

24.18.4 Sanitary Waste Disposal

Sanitary wastes from sinks, lavatories, toilets, and showers are handled by septic systems that are dedicated to individual buildings or groups of ancillary facilities that share a septic tank or leach field. The septic systems have been designed and permitted in accordance with Cochise County regulations.

Sinks and drains where chemical handling operations are taking place will either drain to the tank farm sump and ultimately report to the Raffinate Pond, or in dedicated in a chemical containment tank. Containment tanks are serviced by licensed hazardous materials handling contractors in accordance with federal, state, and local regulations.

24.18.5 Waste Management

Solid wastes are collected in approved containers, removed from site by a solid waste contractor, and disposed in accordance with federal, state, and local regulations. Excess construction materials and construction debris will be removed from site by the generating contractor.

Recyclable materials that are non-hazardous, such as scrap metal, paper, used oil, batteries, wood products, etc., will be collected in suitable containers and recycled with appropriate vendors.

Hazardous materials, such as contaminated greases, chemicals, paint, and reagents, will be collected and recycled, whenever possible, or shipped off-site for destruction, treatment, or disposal.

24.19 MARKET STUDIES AND CONTRACTS

Please refer to Section 19 of this Report for relevant Market Studies and Contracts.

24.20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

24.20.1 Introduction

The Johnson Camp Mine (JCM) is an inactive open pit mine. A processing (SX-EW) plant and associated ponds located at JCM are used to process pregnant leach solutions (PLS) from the Gunnison Project. A pipeline under I-10 connects Gunnison with JCM. JCM plans to resume mining of the open pit and process the material in a new heap leach pad. Existing permits will be modified to address resumption of mining at JCM.

24.20.2 Environmental Studies and Permitting

This section identifies applicable key environmental permits. Federal, state, and local government existing environmental permits are listed in Table 24-50.

Table 24-50: JCM Environmental Permits

Agency	Permit	Description	Citation	When Required/ Permit No.
Federal				
US Fish & Wildlife Service (USFW)	Incidental Take Permit	Mining activities that may affect species listed as endangered or threatened need to conduct studies to identify any targeted species and to apply for a permit to conduct their activities. Any identified threatened or endangered species identified in pre-mining surveys would need to be mitigated before mining could proceed.	50 CFR Sections 7 and 10	None previously identified. New studies may be required prior to disturbing new ground
State of Arizona				
Arizona Department of Environmental Quality (ADEQ)				
Air Quality Division	Air Quality Control Permit	Ensures air pollutants from any source do not exceed the National Ambient Air Quality Standards	ARS §49-402	AQP-71633; covers the Gunnison Project and JCM
Groundwater Section	Aquifer Protection Permit	Covers surface impoundments, solid waste disposal facilities, mine tailings piles and ponds, heap leaching operations. This permit requires designs for the proper management of process facilities, ponds, tailings impoundments, and includes monitoring requirements to ensure compliance with the permit.	AAC R18-9 Articles 1 – 4	P-100514; JCM has amended the APP to include a new leach pad. It may require an amendment at a later date for expansion.
	APP Closure Plan and Bonding for APP Facilities	Closure strategy and estimated cost of closure, post closure monitoring, and surety bond. Bonding estimate must be approved by the agencies and the bond must be posted prior to commencement of construction.	AAC R18-9 Articles 1 – 4	Closure costs for the new leach pad have been provided with the APP amendment application.
Waste Management Division	EPA ID Number	Generators of hazardous waste must have an EPA ID prior to offering the waste for shipment.	ARS §49-922	Covers JCM
	Pollution Prevention Plan	Plan identifying opportunities to reduce waste.	ARS §49-961 thru 973	Report to be submitted annually
	Toxic Release Inventory	Submit Form R for quantity of copper in waste rock.	40 CFR 372	Report to be submitted annually
Arizona State Mine Inspector	Mined Land Reclamation Plan and Bond	Exploration and mining activities on private land with greater than 5 acres disturbance. Does not include facilities covered in Aquifer Protection Permit.	AAC R11-2-101 thru 822	Approved April 2018; may require updating for future modifications.
Arizona Department of Agriculture	Notice of Intent to Clear Land	Ensures enforcement of Arizona Native Plant Laws	ARS §3-904	60 days prior to new disturbance
Arizona Game and Fish Department		Ascertain whether or not the mining operation would endanger fish and game habitat, etc.	AAC Title 12	No T&E Species identified. Additional plans may be required

24.20.2.1 Aquifer Protection Permit Amendment

The Arizona Department of Environmental Quality (ADEQ) grants and administers Aquifer Protection Permits (APPs). ADEQ adheres to licensing timeframes for the review and approval of permit applications.

An APP is required for facilities that have the potential to discharge and impact groundwater quality. APP-regulated surface activities related to open pit mining operations include, but are not limited to, heap leach pads, ponds, stockpiles, and tailing facilities. The Johnson Camp Mine is currently covered under permit P-100514. The currently permitted facilities include a waste rock stockpile, 3 leach pads, solution pond 1, intercept sump, solution pond 3, raffinate pond 1, ILS pond, 10 stormwater ponds, and 2 non-stormwater ponds.

EMJCM's significant APP amendment for a new facility, Leach Pad 5 and its associated impoundments was recently approved by ADEQ. This leach pad will be used for leaching mineralized material from JCM pits when mining resumes. A future amendment will be required for further expansion.

24.20.3 Water Management

Future actions include construction of a new, lined heap leach pad with associated ponds and pipelines. Other future actions may include the construction of additional ponds, either at the Gunnison Project or at JCM. These facilities will be designed to meet prescriptive Best Available Demonstrated Control Technologies (BADCT) which identifies design requirements for stability, liner specifications, capacities, freeboard, leak detection, operations, monitoring, and closure.

24.20.4 Closure and Reclamation Costs

Excelsior maintains surety bonds, posted with ADEQ for APP-regulated facilities, and Arizona State Mine Inspector (ASMI) for non-APP facilities. The closure (APP) and reclamation (ASMI) plans include cost estimates and financial assurance for implementing the plans. The closure/reclamation plans and surety bonds will be updated to reflect any changes in the regulated facilities.

APP-regulated facilities must be closed at the end of operations and post-closure monitoring must be conducted according to the permit. Closure of APP facilities will be conducted according to the most recently approved closure plan. The solution ponds will be emptied and cleaned. Liners will be inspected for signs of leakage. The soils beneath prospective defects will be investigated and remediated as necessary. After clearance, the liner materials will be folded into the bottom of the pond for burial in place. Perimeter berms above the natural land surface will be pushed into the pond to cover the liner, contoured, and revegetated to shed surface runoff and minimize infiltration. The APP for JCM does not require that closure costs be updated until 2026. The cost for closure of any new APP-regulated facilities will be added to the total closure costs and bonded.

Non-APP facilities, such as buildings and infrastructure, will be reclaimed in accordance with the approved Mined Land Reclamation Program overseen by ASMI. The Reclamation plan ensures safe and stable post-mining land use. Re-grading and resurfacing needs, if any, will be completed with good engineering practices minimizing unwanted surface disturbances.

24.20.5 Community Relations

Excelsior has worked extensively to build sustainable partnerships and bring value to the community. Excelsior's approach to community relations reinforces its core values and provides guidelines for making decisions on a variety of issues, ranging from charitable giving to resource development. To that end, Excelsior maintains a broad-based community relations and stakeholder outreach program. Various levels of activity and outreach occur as a function of the development of the Project from prefeasibility and feasibility studies, through Project construction and operations, to closure and rehabilitation. Elements of this program include:

- Targeted stakeholder outreach to government, community, business, non-profit and special interest groups, and leaders at the local, county and state level.
- Development of community relation and communication tools and resources (e.g., Project website, Project e-newsletter, and presentation materials);
- Public open houses, site tours and technical briefings when appropriate.

Crucial elements of Excelsior's community relations efforts will involve ensuring consistent and ongoing communication with stakeholders and providing opportunities for meaningful two-way dialogue and active public involvement. Excelsior will focus on ensuring the public benefits related to JCM and the Gunnison Project, such as employment opportunities, supplier services, infrastructure development and community investment are optimized for the local community.

24.21 CAPITAL AND OPERATING COSTS

The total capital cost requirements for the JCM heap leach operation including mining, Leach Pad 5 development, upgrades to the crushing and conveying equipment to handle material to Leach Pad 5 and pre-production costs totals $58.9 million of which $27.7 million (47%) is for the construction, piping, and electrical connection of the new leach pad, $21.4 million (36%) is to refurbish the crushing and conveying equipment. Most of the remainder $9.8 million (14%) is for pre-stripping/pre-production.

The total mine and plant operating cost exclusive of transportation, royalties, taxes, and reclamation costs is $960 million over the LoM equaling $1.95/lb Cu. Of that total, $1.02/lb Cu (52%) is attributed to mine operating cost, $0.46/lb Cu (24%) is attributed to heap leach cost, SX-EW cost is $0.27/lb Cu (14%), and G&A accounts for $0.20/lb Cu (10%).

The LoM contribution to operating cost from transportation, royalties, taxes, and reclamation is an additional $141 million ($0.29/lb Cu) of which 79% is attributed to royalties. The total operating cost of production for the entire JCM mining and heap leach is $2.24/lb Cu.

24.21.1 Mine Operating Costs

The mine operating costs assume contract mining and have been estimated using an average rate of $2.61 per ton mined for the contractor. An additional cost of $0.03 per ton is estimated for mine general services, which was provided by Excelsior and is included with the operation's G&A costs. The total cost per ton for mining is $2.64/ton. Mining costs for Year -1, both contractor costs and mine general services, are capitalized and included with initial CAPEX. In subsequent years, these costs are accrued as operating costs against the revenue produced by selling the copper product. The mining summary is shown in Table 24-51 below.

Table 24-51: Mine Operating Cost Summary

Mining Cost Summary	Units	Total
Mine General Services	US$000	$5,883
Mine Contracting Costs	US$000	$510,973
Total Mining Costs	US$000	$516,855
	$/ton mined	$2.64

24.21.1.1 Contractor Operating Costs

Table 24-52 shows the contractor estimated costs. These contractor costs are based on quotations provided by contract mining companies for recent studies. The costs were estimated using unit costs for both leach and waste material. The gallons of fuel per ton was also provided by the contractor which was used to adjust costs based on fuel price. The fuel price used for this study is $3.25/gallon. RESPEC believes that this diesel price is reasonable as a long-term price for diesel fuel.

Costs have been broken down into drill & blast, loading, auxiliary, and haulage. The unit costs used are broken down over time to reflect the benches that will be mined. Total contractor costs are estimated to be $510,973,000 or $2.61/ton mined after inclusion of fuel costs. Of this total, $8,833,000 is capitalized as pre-strip in Year -1.

Table 24-52: Contractor Mining Cost Estimate

Contract Mining Costs	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Total
Drilling & Blasting																						
Net Before Fuel	K USD	$1,140	$5,695	$2,537	$6,821	$4,292	$3,045	$1,990	$10,077	$6,652	$7,130	$5,076	$2,504	$2,435	$2,079	$1,742	$1,995	$4,903	$7,050	$6,490	$3,510	$87,163
Fuel	Gallons	159,582	797,367	355,112	954,910	600,914	426,351	278,575	1,410,847	931,254	998,157	710,645	350,514	340,899	291,113	243,913	279,307	686,411	987,049	908,531	491,433	12,202,884
	Gal/ton	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07
	K USD	$519	$2,591	$1,154	$3,103	$1,953	$1,386	$905	$4,585	$3,027	$3,244	$2,310	$1,139	$1,108	$946	$793	$908	$2,231	$3,208	$2,953	$1,597	$39,659
Total w/Fuel	K USD	$1,659	$8,287	$3,691	$9,924	$6,245	$4,431	$2,895	$14,663	$9,678	$10,374	$7,386	$3,643	$3,543	$3,025	$2,535	$2,903	$7,134	$10,258	$9,442	$5,107	$126,823
Loading																						
Net Before Fuel	K USD	$981	$2,850	$2,350	$4,217	$2,146	$1,523	$3,154	$5,475	$3,328	$3,565	$2,538	$1,252	$1,217	$1,040	$871	$998	$2,466	$3,580	$3,245	$1,755	$48,551
Fuel	Gallons	274,764	798,000	658,092	1,180,638	600,914	426,351	883,006	1,532,999	931,940	998,157	710,645	350,514	340,899	291,113	243,913	279,307	690,509	1,002,425	908,531	491,433	13,594,152
	Gal/ton	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07
	K USD	$893	$2,593	$2,139	$3,837	$1,953	$1,386	$2,870	$4,982	$3,029	$3,244	$2,310	$1,139	$1,108	$946	$793	$908	$2,244	$3,258	$2,953	$1,597	$44,181
Total w/Fuel	K USD	$1,874	$5,443	$4,489	$8,054	$4,099	$2,908	$6,023	$10,457	$6,357	$6,809	$4,848	$2,391	$2,325	$1,986	$1,664	$1,905	$4,710	$6,838	$6,197	$3,352	$92,732
Auxilliary																						
Net Before Fuel	K USD	$1,032	$3,761	$2,410	$5,050	$2,833	$2,010	$2,781	$6,948	$4,392	$4,706	$3,350	$1,652	$1,607	$1,372	$1,150	$1,317	$3,246	$4,691	$4,283	$2,317	$60,907
Fuel	Gallons	274,764	798,000	658,092	1,180,638	600,914	426,351	883,006	1,532,999	931,940	998,157	710,645	350,514	340,899	291,113	243,913	279,307	690,509	1,002,425	908,531	491,433	13,594,152
	Gal/ton	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07
	K USD	$893	$2,593	$2,139	$3,837	$1,953	$1,386	$2,870	$4,982	$3,029	$3,244	$2,310	$1,139	$1,108	$946	$793	$908	$2,244	$3,258	$2,953	$1,597	$44,181
Total w/Fuel	K USD	$1,925	$6,354	$4,549	$8,887	$4,786	$3,396	$5,651	$11,930	$7,421	$7,950	$5,660	$2,792	$2,715	$2,319	$1,943	$2,224	$5,490	$7,948	$7,236	$3,914	$105,088
Haulage																						
Net Before Fuel	K USD	$2,738	$8,094	$6,545	$11,878	$6,095	$4,507	$8,817	$16,101	$9,851	$10,552	$9,746	$4,807	$6,477	$5,531	$4,878	$3,830	$7,002	$10,161	$9,604	$5,195	$152,410
Fuel	Gallons	196,260	570,000	470,066	843,313	429,224	304,536	630,718	1,094,999	665,672	712,969	609,124	300,441	438,299	374,288	348,448	239,406	493,221	716,018	648,951	351,024	10,436,977
	Gal/ton	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.06	0.06	0.09	0.09	0.10	0.06	0.05	0.05	0.05	0.05	0.07
	K USD	$638	$1,852	$1,528	$2,741	$1,395	$990	$2,050	$3,559	$2,163	$2,317	$1,980	$976	$1,424	$1,216	$1,132	$778	$1,603	$2,327	$2,109	$1,141	$33,920
Total w/Fuel	K USD	$3,375	$9,946	$8,073	$14,619	$7,490	$5,497	$10,866	$19,660	$12,015	$12,869	$11,726	$5,783	$7,902	$6,748	$6,011	$4,609	$8,605	$12,488	$11,714	$6,336	$186,331
All Total w/Fuel	K USD	$8,833	$30,031	$20,801	$41,484	$22,620	$16,232	$25,436	$56,710	$35,471	$38,001	$29,619	$14,609	$16,485	$14,077	$12,152	$11,641	$25,939	$37,533	$34,589	$18,710	$510,973
Total w/Fuel	$/ton	$2.34	$2.50	$2.33	$2.43	$2.51	$2.54	$2.25	$2.50	$2.53	$2.54	$2.78	$2.78	$3.24	$3.24	$3.33	$2.78	$2.50	$2.50	$2.54	$2.54	$2.61

24.21.1.2 Mine General Costs

Mine general costs were estimated using a constant cost of $0.03 per ton mined based on previous studies. As the operation of Johnson camp is anticipated to be concurrent with other Excelsior mining, these general services will be shared between operations. For this reason, the cost is allocated to the tonnage being mined. This should be reflected in future studies for the area in which Excelsior operates. Total personal costs are about $5.88 million for the LoM or $0.03 per ton mined. The general services cost estimate is shown in Table 24-53.

Table 24-53: Mining General Services Cost Estimate

Mine General Services	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Yr 21	Total
Leach Mined	$000	$23	$149	$128	$130	$127	$123	$85	$117	$115	$165	$145	$102	$100	$107	$102	$113	$153	$172	$187	$162	$53	$-	$2,557
	$/ton	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$-	$0.03
Waste Mined	$000	$94	$193	$155	$376	$130	$60	$293	$540	$284	$262	$160	$48	$46	$17	$3	$7	$143	$258	$203	$49	$4	$-	$3,325
	$/ton	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$-	$0.03
Total Mine G&A Costs	$000	$118	$342	$282	$506	$258	$183	$378	$657	$399	$428	$305	$150	$146	$125	$105	$120	$296	$430	$389	$211	$57	$-	$5,883
	$/ton	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$0.03	$-	$0.03

24.21.2 Mine Capital Costs

Mine capital costs have been estimated by the QP based on assumed contract mining to achieve the production schedule presented in Section 24.16. The use of a contractor reduces the amount of capital required but does increase the operating cost. Table 24-54 shows the estimate for mining capital costs.

Contractor costs includes Mobilization and demobilization costs of $400,000 based on recent contractor quotations for similar projects. In addition, it is anticipated that the contractor will provide their own offices and line-out facilities. However, an estimated $25,000 has been included into the capital cost for concrete slabs and power to their facilities. The shop area will be outdoor and consist of a lined area covered with gravel where they work on equipment so that any contaminants are contained and do not escape into the environment. A total of $425,000 is estimated for contractor capital.

Owner mining capital is assumed to consist of mining software, survey equipment, light vehicles, computers, printers, plotters, and communications equipment. The total Owner capital is estimated to be $520,000.

Other owner mining capital is included for mining general services during Year -1. This totals $118,000 and is based on $0.03/ton mine as discussed in Section 24.16.

The largest component of mining capital is for pre-stripping during year -1. This is based on contractor operating cost rates which are discussed in the operating cost Section 24.21.4. Total pre-stripping costs were estimated to be $6,908,000 bringing the total mining capital cost to $7,971,000 as shown in Table 24-54.

Table 24-54: Mine Capital Costs Summary

	Units	Total
Contractor Capital		
Mob & DeMob	US$000	$400
Facilities	US$000	$25
Total Contractor Capital	*US$000*	*$425*
Owner Capital		
Mining Software	US$000	$175
Survey Equipment	US$000	$150
Light Vehicles	US$000	$150
Computers, Printers, Plotters	US$000	$25
Communications	US$000	$20
Total Owner's Capital	*US$000*	*$520*
Pre-stripping - Contractor	US$000	$6,908
Mining General Services	US$000	$118
Total Mining Capital	US$000	$7,971

24.21.3 Plant Capital Costs

The Johnson Camp SX-EW plant was upgraded in 2019 and 2020 in anticipation of the Stage 1 production of the Gunnison ISR Copper Project. These improvements were discussed in Section 24.17 of this report. There are no upgrades to the JCM SX-EW plant or ponds planned for the JCM heap leach operation.

To restart the JCM open pit mining, the improvements are related to upgrades to the existing crushing, agglomerating, and conveying system and the construction of the new Leach Pad 5.

In 2022, Excelsior received a vendor quote to replace the conveying and stacking system from the crushing agglomerating area to Pad 5 and make improvements to the mechanical and electrical equipment. Table 24-55 lists the main improvements by Area to refurbish, recondition, and replace equipment for crushing and material handling and the QP's estimate of the CAPEX associated with the improvements.

Table 24-55: JCM Crusher-Conveyor Improvements

Area	Discipline	Main Improvements	Cost
Primary Crusher	Mechanical	Replace concave liners, upgrade hydraulic system, modify apron feeder, replace air compressor,	$1,509,607
	Electrical	Rebuild substation, purchase 7.5 MVA transformer, add new vacuum breaker	
Secondary Crushers	Mechanical	Replace vibrating feeders and chutes, replace vibrating screens, rebuild Secondary Crusher #2, improve dust collectors, upgrade ASRi system	$2,824,195
	Electrical	Add 1500 kVA transformer, replace electrical conduit and cabling, refurbish MCC and switchgear	
Agglomerator	Mechanical	Install cross belt sampler, replace agglomerator liner, replace agglomerator feeder with belt feeder, add acid feed valve	$859,839
	Electrical	Install new VFDs for acid feed pump and 200 HP drive	
Conveying & Stacking	Mechanical	Install grasshopper conveyors from agglomerator the Pad 5, install new indexing conveyor and stacker, add conveyor from agglomerator to grasshoppers	$6,579,600
	Electrical		
Freight			$998,970
Direct Cost of Improvements			$12,772,211
Indirects			$3,046,500
Contingency (35%)			$5,536,600
Total Cost			$21,355,311

A number of studies were prepared for Nord Resources between 2007 and 2011 for the design, costing and construction of Pad 5 and other improvements to deliver mineralized material to the leach pad. These studies included the Bikerman feasibility study (Bikerman, 2010), the design package prepared by Glasgow Engineering (2010) and the cost estimate and scope definition prepared by Curtis Associates (2011).

Only half of the original footprint of Pad 5 is intended to be constructed as initial capital since the current anticipated ROM production is much less than the mine plan presented in 2011. Quantities for civil excavation, grading, overliner production and placement, and geomembrane were adjusted to account for 50% of the anticipated footprint of the original Pad 5. The largest cost adjustment came from the halving of geomembrane liner procurement and installation. Quantities have been adjusted to match the reduced leach pad footprint, however, the contingency is still held at 30% because updated material take-offs for civil and piping quantities on Pad 5 were not confirmed. The distribution piping on top of the new Pad 5 leach pad was not re-designed or taken off for this study.

An MTO's for the PLS and raffinate overland piping from the Pad 5 pump station to the JCM ponds as well as the pump station stainless steel piping were prepared. Some activities were estimated as whole installations: the emergency (event) pond below Pad 5, the containment trench, the mine shaft remediation, and the full pump station installation.

Electrical distribution costs for the Pad 5 facility from the previous cost estimates (Curtis Associates, 2011) included the installation of transformers and motor starters, a pole line to the pump station from the crusher substation, addition of duct banks, lighting, and reconfiguration of equipment at the crushing area which is now captured in the crusher-conveying upgrade costs. The electrical cost for solution pumping from the Curtis study was escalated to 2022 prices in whole because the price was quoted as a single line item. The electrical scope to electrify Pad 5 should be revisited in full.

Concrete scope was limited to installing pump pads at Pad 5. The former instrumentation costs were escalated to 2022 costs but appear to be on the low side. Control systems are already in place at Johnson Camp.

The cost for EPCM was calculated at 12.8% instead of the customary 16.8% for a greenfields project. It is assumed the construction management will largely be conducted by Excelsior staff on the JCM property. Engineering costs, project controls, and project services have been retained in full. The estimated capital costs for Pad 5 construction are shown in Table 24-56.

Table 24-56: Capital Costs Associated with Pad 5 Construction

Discipline	Items	Cost ($000)
Direct Construction		
Civil Earthworks	Excavation, grading, underliner, overliner, leachate piping, emergency pond, containment trench	$10,791
Concrete	Pump bases	$147
Plant Equipment	Pumps, VFDs	$348
Piping	HDPE pipe, 316SS, pipe, fittings & valves	$3,398
Electrical	Transformers, pole line, Control panels, lighting	$2,158
Instrumentation	Flowmeters, pressure gauges	$39
Freight	Mainly for piping & geomembrane	$648
Other	Mobilization, Temp facilities & power	$368
	Subtotal Directs	$17,897
Indirect Costs		
EPCM	Construction management provided by Owner	$2,103
EPCM facilities & Utility Set-up		$ 90
Commissioning		$ 13
Vendor Support		$ 33
Capital/Commissioning Spares		$ 33
	Subtotal Indirects	$2,270
	Contingency (30%)	$6,050
Total Plant Capex		$26,217

Oxidation and recovery of copper from sulfides requires aeration, as described in Section 24.17. The piping for aeration is considered sacrificial and is treated as an operating expense. The blowers that supply the air to the distribution piping for aeration are included in initial CAPEX at an estimated installed cost of $1.5 million, bringing the total leach pad CAPEX to $27.7 million.

24.21.4 Plant Operating Costs

24.21.4.1 Plant Operating Cost

The plant operating cost includes the management and irrigation of Pad 5, acid addition to raffinate sent back to the leach pad or added in agglomeration, and the operation of the JCM SX-EW plant. Components of the operating cost are operating and maintenance labor, power (mostly for electrowinning), reagents (mostly sulfuric acid) & consumables, spare & maintenance supplies, and services.

The heap leaching costs for Pad 5 are summarized in Table 24-57. The largest heap leach operating cost is purchased sulfuric acid for leaching the crushed and agglomerated material.

Table 24-57: JCM Heap Leaching Operating Costs

Cost Element	Operating Cost - LoM ($000)	$/st Mineralized Material Processed	$/lb Copper
Labor	$23,898	$0.28	$0.05
Power	$32,080	$0.38	$0.07
Reagents	$113,644	$1.33	$0.23
Maintenance	$46,051	$0.54	$0.09
Supplies & Services	$10,553	$0.12	$0.02
Total Plant Operating Costs	$226,226	$2.65	$0.46

Operating costs for the JCM plant established from recent operations of the Johnson Camp SX-EW plant during the operation of the Gunnison wellfield and from previous operations of the JCM SX-EW. Staffing for plant maintenance labor was provided by Excelsior with updated salaries and benefit rates. The JCM plant operating costs are summarized in Table 24-58.

Table 24-58: JCM Plant Operating Costs

Cost Element	Operating Cost - LoM ($000)	$/st Mineralized Material processed	$/lb Copper
Labor	$29,918	$0.35	$0.06
Power	$53,090	$0.62	$0.11
Reagents	$16,598	$0.19	$0.03
Maintenance	$26,345	$0.31	$0.05
Supplies & Services	$6,261	$0.07	$0.01
Total Plant Operating Costs	$132,211	$1.55	$0.27

24.21.4.2 General and Administrative Operating Costs

General and Administrative (G&A) costs include labor and fringe benefits for administration and support personnel and other support expenses are based on the 2023 JCM budget provided by Excelsior. Table 24-59 is the summary of Life-of-Mine G&A costs for the JCM Heap Leach operation. G&A costs are generally fixed costs and only G&A labor partially scales with increased or decreased production. Various services and expenses are estimated from recent studies of JCM for the Gunnison Project. The G&A cost for JCM averages $4.5 million annually of which labor is 38% and insurance is 22%. The G&A cost also includes the $0.03 per ton of material mined, which equals $0.07 per ton of material placed on the leach pad.

Table 24-59: JCM Heap Leach – LoM G&A Cost Summary

Cost Element	Operating Cost - LoM ($000)	$/st Mineralized Material processed	$/lb Copper
Labor & Fringes	$35,738	$0.42	$0.07
Accounting (excluding labor)	$513	$0.01	$0.00
Safety & Environmental (excluding labor)	$513	$0.01	$0.00
Human Resources (excluding labor)	$513	$0.01	$0.00
Security (excluding labor)	$513	$0.01	$0.00
Assay Lab (excluding labor)	$6,150	$0.07	$0.01
Office Operating Supplies and Postage	$820	$0.01	$0.00
Maintenance Supplies	$6,284	$0.07	$0.01
Propane, Power	$513	$0.01	$0.00
Communications	$1,435	$0.02	$0.00
Small Vehicles	$2,563	$0.03	$0.01
Claims Assessment	$205	$0.00	$0.00
Legal & Audit	$6,150	$0.07	$0.01
Consultants	$3,075	$0.04	$0.01
Janitorial Services	$1,025	$0.01	$0.00
Insurances	$20,500	$0.24	$0.04
Subs, Dues, PR, and Donations	$1,230	$0.01	$0.00
Travel, Lodging, and Meals	$3,075	$0.04	$0.01
Recruiting/Relocation	$2,563	$0.03	$0.01
Mine G&A	$5,883	$0.07	$0.01
Total General & Administrative Cost	**$99,257**	**$1.16**	**$0.20**

24.21.4.3 Reclamation and Closure Cost

Table 24-60 summarizes the reclamation and closure costs for JCM. The reclamation and closure costs for the Project include reclamation and closure activities at the JCM plant site and the JCM heaps and stockpiles. Reclamation and closure activities will be spread over three years after operations cease.

Table 24-60: Summary of Reclamation & Closure Costs

Item	Reclamation and Closure Cost $000			
	Total Cost LoM	Year 20	Year 21	Year 22
Solution Management	$1,840	$736	$736	$368
Leach Pad Closure	$12,240	$4,896	$4,896	$2,448
Total Direct Costs	**$14,080**	**$5,632**	**$5,632**	**$2,816**
Indirect costs	$282	$113	$113	$56
Contract Administration	$563	$225	$225	$113
Total Costs	**$14,925**	**$5,970**	**$5,970**	**$2,985**
Demolition Costs	$680	$442	$238	$0
Bonding Fees	$187	$77	$74	$36
Total Closure Costs	**$15,793**	**$6,489**	**$6,283**	**$3,021**

24.22 ECONOMIC ANALYSIS

The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the Project. Annual cash flow projections were estimated over the life of the operation based on the estimates of capital expenditures and production cost and sales revenue. The sales revenue is based on the production of a copper cathode.

Table 24-61 compares the financial indicators for JCM Heap Leach Project. **This economic assessment 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; there is no certainty that the financial indicators of this assessment will be realized.**

Table 24-61: Financial Indicators

Item	LoM
Years of Commercial Production	20
Total Copper Produced (klbs)	491,754
LoM Copper Price (avg $/lb)	$3.75
Initial Capital Cost ($M)	$58.9
Sustaining Capital Cost ($M)	$36.1
Payback of Capital (pre-tax / after-tax)	4.01 / 4.04
Internal Rate of Return (pre-tax / after-tax)	32.2% / 30.4%
LoM Direct Operating Cost ($/lb Copper recovered)	$1.95
LoM Total Production Cost ($/lb Copper recovered)	$2.24
Pre-Tax NPV at 7.5% discount rate ($M)	$212.5
After-Tax NPV at 7.5% discount rate ($M)	$180.0

Table 24-62 provides a sensitivity analysis for the Base Case project financial indicators with the financial indicators when other different variables are applied. The results indicate that Project economics are impacted the most by fluctuation in the copper price. Fluctuation in the initial capital cost has the least impact on Project economic indicators.

Table 24-62: Base Case After – Tax Sensitivities ($millions)

	Copper Price		
	NPV @ 7.5% ($M)	IRR%	Payback
	$180.0	30.4%	4.0
20%	$321	49.2%	2.1
10%	$251	39.9%	2.6
-10%	$107	20.9%	4.9
-20%	$32	11.5%	10.6
	Operating Cost		
	NPV @ 7.5% ($M)	IRR%	Payback
	$180.0	30.4%	4.0
20%	$141	24.7%	4.5
10%	$161	27.5%	4.3
-10%	$199	33.3%	3.4
-20%	$218	36.4%	2.9
	Initial Capital		
	NPV @ 7.5% ($M)	IRR%	Payback
	$180.0	30.4%	4.0
20%	$171	27.0%	4.3
10%	$176	28.6%	4.2
-10%	$184	32.4%	3.7
-20%	$189	34.9%	3.3

Figure 24-70 and Figure 24-71 are sensitivity plots of after tax IRR and NPV@7.5% for the Johnson Camp Heap Leach Project. Copper price has the biggest impact to financial results. Operating cost is the next most sensitive factor of which acid price is the biggest component.

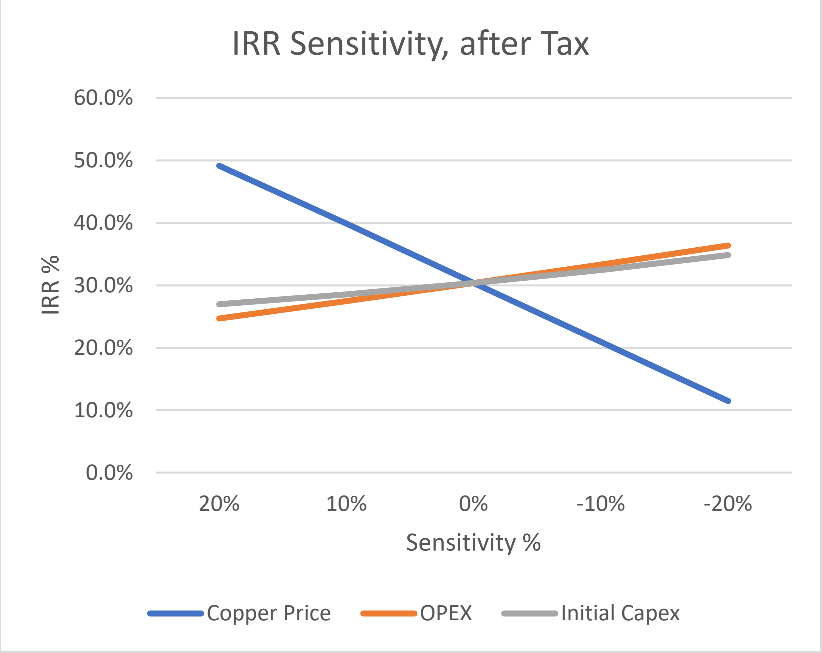


Figure 24-70: IRR Sensitivity for the Johnson Camp Heap Leach Project

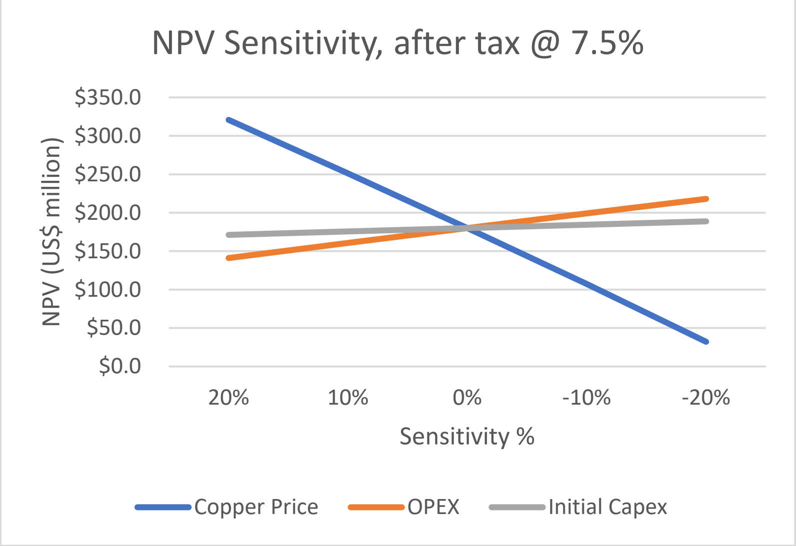


Figure 24-71: NPV Sensitivity for the Johnson Camp Heap Leach Project

24.23 ADJACENT PROPERTIES

Please refer to Section 23 of this report for information on adjacent properties.

24.24 OTHER RELEVANT DATA AND INFORMATION

There is currently no other relevant information.

24.25 INTERPRETATION AND CONCLUSIONS

Restarting the Burro and Copper Chief pits at Johnson Camp has been investigated by Nord Resources and by others since Nord closed the mine in 2012. Mineral resources were left unmined in the pits by Nord because it was unable to arrange financing to build Pad 5. Excelsior recognizes the opportunity to mine a subset of the remaining mineral resources at the currently higher copper prices.

The JCM plant has already been upgraded and JCM ponds are already operational. The crushing plant will be utilized and this capital upgrade has been included along with the construction of the new leach pad, Pad 5.

The full capital cost for restarting the JCM heap leaching operation between mining pre-production, first fills/Owners costs, leach pad construction, crusher and agglomerator refurbishment, new leach pad stackers and haul road construction is approximately $59 million. Significant staffing for the JCM Project is in place and some new hires will be needed to augment the staff that is already engaged by Excelsior.

Based on the current pit shell, mineral resources for the two pits total approximately 108 million tons of M&I and 51 million tons of Inferred at a cut-off grade of 0.1% CuT. The amount that is included in the conceptual mine plan over years of mining is 69.7 million tons of M&I and 15.6 million tons of Inferred. It is possible that the mine life for the JCM open pit operation could be extended for several more years if copper prices continue to be favorable.

The financial results for the Project indicate an NPV of $180.0 million after taxes at discount rate of 7.5% with an IRR of 30.4% and a payback period of 4.0 years. ***This economic assessment 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; and there is no certainty that the financial indicators of this assessment will be realized.***

24.25.1 JCM Opportunities

1. Additional infill and step-out drilling, including drilling focused on deeper sulfides, could yield increased tonnage and/or grade in some areas within the mineral resource.
2. Approximately 54% of the mineral resources for the two pits that total approximately 108 million tons of M&I and 51 million tons of Inferred at a cut-off grade of 0.1% CuT have been scheduled in the final pit design. Detailed mine planning and scheduling may result in higher production rates and reduced mining unit costs. Mine plan optimization could bring higher grade material closer to start for better initial cash flow and could reduce waste tons.
3. Planned metallurgical test work on sulfides and transitional mineralization in 2023 could generate higher recoveries or lower acid consumptions that presently estimated.
4. Testing may also demonstrate that less crushing is required to achieve the estimated copper recoveries resulting in reduced capital and operating costs.
5. More detailed mine design and planning focused on brining higher-grade mineralization forward in the mine schedule or delaying/reducing waste stripping, which could result in improved economics.
6. The estimated capital required for the JCM Project has been prepared to a PEA level. Additional detailed engineering work may reduce this cost.
7. Relocating the existing waste rock stockpile could be performed with smaller, cheaper equipment as well as timed later in the mine schedule to increase near-term revenue.

8. The copper price used for this study is $3.75/lb. Current copper spot copper prices are above this price level and may continue to increase, given the continued emphasis by the US and other governments towards renewable energy sources and electrification of transportation. Copper has a large role to play in these “green” initiatives.
9. Expansion of the current SX-EW facility to a production capacity of 50 mlbs per annum could be evaluated in a tradeoff study to evaluate whether it would improve the project economics by increasing the cash flow.
10. Demonstration of successful leaching of sulfide and transitional material could provide opportunities for mining additional satellite deposits that are known to exist in the Johnson Camp District, including the Strong and Harris and Gunnison deposits.
11. Integration of planning efforts for the Johnson Camp, Strong and Harris, and Gunnison deposits could reveal synergies or development strategies for improving financial returns and increasing the mine life.

24.25.2 JCM Risks

1. Metallurgical test work to be performed on sulfides and transitional mineralization in 2023 could generate lower recoveries or higher acid consumptions that presently estimated.
2. This testing could also indicate that finer grinding is required to achieve the copper recoveries estimated in this report resulting in increased capital and operating costs or reduced revenue.
3. Additionally, new column testwork may indicate that the overall sulfuric acid consumption is higher than reported in this PEA. Section 24.13 of this report discusses the history of column tests that used different sequential assay techniques, some using hot acid determinations for acid soluble copper. The current opinion is that cold acid assays for acid soluble copper are more appropriate. The acid consumption could increase raising the operating cost of heap leaching.
4. The cost of purchased acid is currently over $230/ton. This cost is considered to be an outlier since the recent historic acid price is between $95/ton to $125/ton over the previous six years. The acid price could remain high in the short term and could increase the operating cost of heap leaching.
5. Other reagent costs, principally diluent and extractant could increase materially, increasing SX-EW operating costs.
6. The cost of power in Cochise County has been affordable over the last several years. For this study, the cost of power has been approximately $0.088/kWh. The current increased price of natural gas which is used by the local generating company could impact the long-term cost of power needed for the Project.
7. The selected PEA copper price of $3.75/lb could be subject to volatility due to external factors. World events could result in a lessening of demand for a number of reasons. Central banks ramping up interest rates to fight inflation could also result in a lower long-term copper price.
8. Capital and operating costs increase during preparation of a Feasibility Study. More detailed engineering designs for Pad 5 could result in higher MTOs for bulk earthworks, geomembrane liner, crushed rock overliner, and piping than has currently been estimated. Unit prices for civil earthworks were updated for this study using recent rates for southern Arizona. Mining contractor costs for Pad 5 construction could materially increase beyond current estimates due to higher fuel prices when the leach pad is developed.
9. Increased lead times for construction could materially delay the start of leaching and generation of revenue.
10. The cost of financing capital for the JCM heap leach could become prohibitive.
11. Obtaining the necessary environmental permit amendments could take longer than anticipated.
12. Changes to the new Pad 5 permit may be required to optimize sulfide leaching which could delay mine start-up.

24.26 RECOMMENDATIONS

Excelsior management has launched a sampling and metallurgical testing program to evaluate the leaching strategy proposed in this study. The sampling and testwork program will assess the metallurgical zonation within the pits to more accurately estimate copper recoveries from each zone including testing the solubility of sulfide species. This program will help determine the long-term outlook for open pit mining and heap leaching at JCM.

The current plan includes crushing and agglomeration with conveying and stacking the agglomerated material on the leach pad. Excelsior should refine the cost to reactivate the crushing-agglomerating plant, design the conveyor system, and the stacking plan for the life of the mine.

Excelsior should consider conducting parallel large-diameter column (or equivalent) tests on a bulk sample. Large samples of at least 250 tons of ROM from the most important future mineralized zones should be collected, blended, and split as for these column tests. Column tests of this size are most easily carried out in cribbed structures built of railroad cross-ties and lined with plastic to minimize leaks.

Although augmentation of chalcopyrite oxidation and leaching with the aid of biological oxidation of pyrite, while supplying low-pressure aeration, now appears to be within reach after decades of effort by many metallurgists and chemists, the need for careful testwork on representative samples of the resource must be recognized. Arrangements are being made for column testing under the supervision of Rio Tinto/Nuton™. The evaluation will include mineralogical characterization of sulfides and gangue by QEMSCAN. The response of various minerals to leaching in dilute solutions of sodium cyanide and sulfuric acid was summarized by Frank Bazzanella (Metcon) in Copper Hydromet 2000, published by Randol International. For example, chalcocite, bornite, and native copper were reported to have solubilities of 90, 70, and 90 percent in dilute sodium cyanide solution with 3-5 percent solubility in dilute sulfuric acid. These data illustrate the risks of assuming that the CNCu and ASCu procedures are quantitatively selective.

In the event that heap leaching of the JCM sulfide resource does not prove to be feasible, a logical alternative is conventional grinding and flotation. The original Johnson Camp Mining District produced copper, silver, gold, lead, tungsten, and zinc. (Molybdenum was practically unknown at the time.) It is possible that silver, gold, and molybdenum are present in economically recoverable concentrations, so a limited amount of assaying, followed – if justified by the assays – by a modest laboratory flotation program, should be considered.

For example, potential increased revenues per ton milled with hypothetical resource grades of 1.0 oz/ton silver, 0.02 oz/ton gold, and 0.015% molybdenum, and respective recoveries of 90%, 90%, and 50%, would total approximately US$60/ton, compared with roughly US$20/ton for copper alone by heap leaching. Additionally, CNCu and SuCu recoveries would likely approach 90%, compared with 70% by heap leaching. Obviously, the added revenues would be offset by increased CAPEX and OPEX for fine grinding, flotation, concentrate dewatering, and tailings management.

Excelsior should commission the re-design and estimating of Pad 5 using a footprint that can accommodate all of the leaching material in the mine plan to improve the accuracy of the initial and sustaining capital cost estimates for the leach pad.

Excelsior should complete the current metallurgical program and if warranted proceed to a feasibility study and then detailed engineering for the leach pad and crusher refurbishment.

Table 24-63: Budget for Recommended JCM Heap Leach Investigations

Detail	Cost US$
Metallurgical Testwork	$250,000
Feasibility Study	$500,000
Detailed engineering for Leach pad and Crusher refurbishment	$500,000
Total	**$1,250,000**

24.27 REFERENCES

See Section 27 for a consolidated list of references.

25 INTERPRETATION AND CONCLUSIONS

The Gunnison Copper Project is an oxidized copper deposit in southeastern Arizona, USA that is amenable to in-situ recovery (ISR) technology for the leaching copper from oxidized mineralization below the water table and conventional solvent extraction and electrowinning (SX-EW) technology for making a saleable copper product. The mineral resource estimate in measured and indicated categories is estimated at 873 million short tons with total copper (TCu) grade of 0.29 percent at a cut-off TCu grade of 0.05%. RESPEC estimates probable mineral reserves at 782 million short tons at a TCu grade of 0.29 percent after applying engineering and operational design parameters.

25.1 CONCLUSIONS

A production schedule has been developed using input from independent consultants and existing Project data. The production schedule anticipates recovery of approximately 48 percent of the estimated contained copper in the reserves for production of 2,165 million pounds of cathode copper in a mine life of 24 years. Production from an ISR well field results in pregnant leach solution from which saleable cathode copper can be produced by a conventional SX-EW process. M3 designed an appropriately sized SX-EW copper recovery plant for construction on the Gunnison Project and provided capital cost estimates for its construction.

Excelsior plans to develop the full copper production capacity of 125 million pounds per annum (mppa) in three stages. Stage 1 capacity with a nameplate capacity of 25 mppa uses the existing SX-EW plant that was acquired in 2015 at the Johnson Camp Mine (JCM). Stage 1 was constructed in 2020 and acid was first injected in December 2020. The JCM plant presently currently has the capacity to produce the desired 25 mppa after minor modifications and upgrades were made to achieve the production goals.

The Stage 1 wellfield operation experienced some challenges in maintaining flow rates into and out of the subsurface, primarily due to the buildup of CO_2 in the fracture flow paths from the dissolution of calcite and other carbonates. The anticipated flow of 4,000 gpm was not reached and decreased with time as discussed in other sections of this report. The ongoing development for Stage 1 production is adding a new set of unit operations to restore flowrates. A water treatment plant is planned to provide neutralized solutions capable of dissolving CO_2 in the formation by alternating acidified solutions with neutralized solutions until the accessible carbonates are removed.

Stage 2 capacity is planned to be 75 mppa, which will require construction of a 50 mppa SX-EW plant that will be sited on the Gunnison property. Stage 3 capacity is planned to increase to 125 mppa through at 50 mppa addition to the Gunnison plant. The current schedule predicts Stage 2 operations beginning in Year 4 of operation and Stage 3 operations beginning in Year 7. Each stage of development is supported by a detailed capital cost estimate at the prefeasibility level.

M3 completed an economic analysis for this Prefeasibility Study using industry standard criteria for studies at this level. The results of this study indicate that ISR development of the Gunnison Copper Project offers the potential for positive economics based upon the information available at this time. Project economics are based on beginning production at approximately 25 mppa in Stage 1 with the JCM plant and Gunnison wellfield infrastructure, increasing production to approximately 75 mppa in Year 4 with the construction of the Stage 2 Gunnison plant, and increasing production to 125 mppa in Year 7 with construction of the Stage 3 Gunnison plant additions. Stage 3 includes construction of the sulfur-burning acid plant, cogeneration facilities, and rail spur for the delivery of molten sulfur, which are part of the base case.

The base-case economic analysis indicates an after-tax Net Present Value (NPV) of $1,167 million at a 7.5 percent discount rate with a projected Internal Rate of Return (IRR) at 37.3 percent. The base case includes a sulfuric acid plant constructed in Year 6 to supply the acid for ISR copper extraction. If the sulfuric acid plant and cogeneration facilities are not constructed and sulfuric acid continues to be supplied by truck or rail, the NPV at a 7.5 percent discount rate is $976 million with projected IRR of 38.1 percent. Payback is projected to occur in approximately 6.7 years with the acid plant and 6.0 years without it.

The economics are based on a copper price of $3.75. The maximum copper production rate is based on three years of 25 mppa for three years after pre-conditioning breakthrough, followed by three years of production at 75 mppa for three years, followed by full production at 125 mppa for the remaining years of the mine life. The direct operating costs are $0.95/lb Cu for the base case including the sulfuric acid plant and $1.35/lb for the purchased acid alternative case. The total cash costs including taxes, royalties, and reclamation & closure costs are $1.22/lb and $1.63/lb, respectively.

Initial capital costs are estimated at $47.6 million to cover the early Water Treatment Plant (WTP), thirteen drillholes and wellfield development, and the costs to operate the wellfield and WTP until stable copper production begins. Sustaining capital costs of $1,033 million are projected in the acid plant case and $880 million in the alternate case using purchased acid, of which approximately $527 million is attributable to ISR well drilling and wellfield infrastructure (the mine).

25.2 PROJECT RISKS

Certain risks and opportunities are associated with the Project, as is typical for mine development projects. These risks may include environmental permitting, title issues, taxation, public/political opposition, or legal impediments to operating this type of mining/processing operation at this location. The following Project-specific risks have been identified along with the measures that Excelsior envisages to mitigate the risk.

1. **Copper recovery.** The ISR process for recovering copper from oxidized mineralization in fractured bedrock has been tested on core in a variety of bench scale tests. Limited-scale production at the Gunnison site commencing in 2020 did not produce the flow rates and copper production as predicted. Flow rates have been negatively affected by CO_2 gas bubble formation, which block or restrict flow through mineralized structures. If flow cannot proceed through a mineralized structure, then copper cannot be recovered from that structure, and thus the effective sweep efficiency is heavily reduced. CO_2 gas bubble formation may not be the only mechanism for reduced flow rates and sweep efficiency. Metallurgical testing has established that mineralization is amenable to copper leaching and recovery. Laboratory testing results have been used to approximate results of ISR in bedrock, they may not reflect eventual performance. Potential deviations include:

    - Recovery rates (kinetics) that are slower than predicted
    - Hydrological conditions and hydrogeochemical reactions at depth resulting in reduced copper recoveries
    - Reduced acid strength due to neutralization by gangue (non-copper) minerals
    - Blind (non-connected) fractures, low fracture density and/or fracture widths resulting in poor contact of leach solution with copper oxides and hence lower sweep efficiency
    - Lower than expected sweep efficiency due to mineral precipitates or gas bubbles blocking or restricting flow along mineralized structures

    Mitigation. Many of these risks can be addressed by developing operational strategies during both the development and Stage 1 operation of the wellfield. This will include producing detailed local geological/structural and hydrological models while the wellfield is being emplaced to further aid in placement of final well locations. Operational strategies will involve predetermining flowrates and acid strengths based on these models for initiating the wellfield in order to maximize quick breakthrough and economic PLS grades. The average copper recovery estimate of 48 percent of total copper has been reduced from metallurgical testing maximums to address these uncertainties.

2. **Wellfield flow attenuation.** Introduction of acidified solutions to the Stage 1 wellfield resulted in significantly decreased flowrates that were shown to be reversible by the replacing the acidified solutions with well water, indicating that the flow-blocking agent (solid or gas) was removeable by dissolution or physical means. Blocking of fracture flow pathways by CO_2 and been postulated as the cause and flushing with neutralized raffinate as the solution to the problem. The following risks of pursuing this strategy have been identified.

- Fluid flow blocking or “blanketing” by gases has been studied in the hydrocarbon industry but has never been directly observed, therefore it’s importance at Gunnison is somewhat speculative.
- The potential blocking of flow paths by solids such as gypsum, drilling debris, or unfiltered particles in the injectate has not been ruled out as the source of flow attenuation.
- The buildup of gypsum in the formation could block off portions of the formation and make them unproductive.
- Flushing with neutralized solutions may not have the efficacy in restoring flow rates that has been observed using groundwater to flush the formation.
- Flushing with groundwater or neutralized solutions lowers the concentrations of free acid and dissolved copper in the formation which could cause copper precipitation.
- Fractures opened by flushing may not have leachable copper mineralization in which case copper recovery would not be restored.

Mitigation. The Project should continue to carefully observe the performance of the wellfield as the pre-production modifications described in this report are implemented with the intent of addressing these concerns. Laboratory experimentation should be conducted on the efficacy of neutralized raffinate in the dissolution of CO_2 and compare it with that of groundwater, which has been shown to be effective. Experiments should also be conducted to evaluate changes in the saturation of gypsum when acidified solutions are mixed with neutralized solutions in a porous medium.

3. **Short Circuiting.** Short circuiting describes a situation whereby the bulk of the injection or recovery flow is along one large structure or pathway rather than numerous smaller pathways. Short circuiting results in high flow rates but low sweep efficiency and copper recovery. Excelsior interprets several of their operating wells to be short circuiting and are thus less productive than expected. However, it is not clear if the low copper production (low sweep efficiency) and short circuiting behavior isn’t of a function of CO_2 blocking the smaller structures and giving the appearance of natural short circuiting. More investigation is needed (see recommendations).

   Mitigation. Several mitigation strategies are available for short circuiting wells including grouting of the short-circuiting structures, application of down-hole packers above and below the offending structures to redirect flows, or specialized valving.

4. **Reagent consumption/cost.** This Project relies on large volumes of sulfuric acid to accomplish the mobilization of copper from the subsurface and to produce a saleable product. In addition, the project requires substantial quantities of lime (CaO) to neutralize excess solutions during water treatment and for raffinate neutralization during CO_2 flushing operations. Increases in the price of reagents with respect to the price of copper would have a negative impact on the economics of the Project.

   Mitigation. The Project has two options for obtaining sulfuric acid: purchasing liquid acid and making acid from molten sulfur. Since sulfuric acid is used extensively in the production of copper worldwide, a significant increase in the price of sulfuric acid or sulfur would likely be accompanied by an increase the price of copper, partially compensating for higher reagent costs, mitigating the impact. Lime and/or limestone can be produced locally in Cochise County if the quantities are sufficient to make the economics worthwhile. Lhoist (formerly Chemical Lime) owns a lime plant in Cochise County near Douglas, AZ that, according to a company official, could be put back into production should the demand increase, meaning one or two significant new projects.

5. **Well design and spacing.** The well design consists of a borehole cased through the alluvial material into the mineralized bedrock with an open borehole through the productive portion of the mineralized material. Problems may arise in the construction of these wells due to caving that would increase drilling costs that are part of initial and sustaining capital costs. Borehole instability could require perforated casing to keep the borehole open, potentially resulting in a larger borehole and additional costs for the materials, labor, and drilling. The current well design is part of the ADEQ Aquifer Protection Permit and the EPA’s Underground Injection Control permit so changes to the borehole design could require amending these two permits.

Borehole spacing is presently on 100-foot centers with a 50-foot offset resulting in 71 feet between an injection well and its nearest recovery wells. Drilling costs per pound of copper produced would increase, if this spacing proves to be too wide.

Mitigation. The proposed well design can be further tested in during Stage 1 production to evaluate the adequacy of construction method and borehole stability to minimize potential problems during implementation and reduce uncertainty concerning well field construction costs. Aquifer testing in the preproduction stage should provide additional data with which to evaluate the optimum borehole spacing.

6. **Gypsum formation/rinsing.** Mineralized areas with significant carbonate content may reach saturation and cause precipitation of calcium sulfate (gypsum) in the formation. Precipitates forming in fractures could reduce flow rates in the formation, retarding the leaching of copper oxides with a consequent reduction in the rate of copper recovery. Gypsum precipitates in the formation might also reduce the rinsing rate, causing an increase in water treatment costs.

   Mitigation. The box tests or fracture simulation tests clearly indicated that the precipitation of gypsum did not alter flow rates however, noting the possible impact in flow reduction and rinsing volume requirements should provide greater confidence in the copper recovery and rinsing projections. Leaching schedules have already been lengthened, pumping rates and porosity reduced through time in this prefeasibility study to compensate for uncertainties associated with these types of issues.

7. **Permitting difficulties.** Permitting for mining projects in the western US and Arizona in particular has been an arduous and unpredictable task in the recent past. Public opposition can be mobilized from outside of the local community by groups that tend to obstruct mining projects. Permitting the sulfuric acid plant may be more difficult in the future due to its air quality implications when compared to the well field/plant issues that are already mitigated somewhat by the presence of SX-EW operations in the immediate vicinity.

   Mitigation. The project is fully permitted to operate in its current configuration. Permitting difficulties for additions and expansion can be mitigated by developing support within the local community, identifying, and fixing potential areas of contention before they arise, getting support from community leaders in advance of applying for permits. Another measure is developing realistic permitting schedules that incorporate time to deal with challenges which also helps minimize deleterious consequences.

25.3 PROJECT OPPORTUNITIES

Several opportunities have been identified which could enhance the viability and economic attractiveness of the Project. Many of these opportunities may be realized by removal of risk and uncertainty that are present at the prefeasibility level.

1. **Copper Recoveries.** The anticipated copper recovery of 48 percent of total copper is an estimate based on the best interpretation of existing test work. This copper recovery could be exceeded in practice. Recovery increases could improve the rate of recovery as well as increase total copper recovered. Improvements in the rate of recovery would mean lower flows from the wellfield for the same level of copper production, lowering operational costs, or that the increased grade could result in higher copper production (revenue) for the same operating cost. Improvements in total copper recovered have the obvious benefit of increasing total revenue during the life of the mine.

2. **Water Treatment.** The high-density solids neutralization considered necessary to rejuvenate flow rates in the wellfield is well established technology for acid mine drainage. Vendors with mobile equipment for rent are available to assist with situations such as those Gunnison faces. Investigation of the capital and operating costs of using rental equipment for the first 3 years of operation should be considered. Using rental equipment would have the benefits of reducing the time necessary to resume operation, provide data for input into the design parameters of a permanent system, demonstrate the efficacy of the technology in addressing the condition, and reducing the capital costs necessary to implement the proposed strategy. Rental equipment offers the flexibility of reducing rental and operating costs when the estimated flows are reduced in Years 1 and 2 of the mine plan.

3. **Increased Copper Price.** The current financial analysis is based on an average copper price of $3.75 per pound. Over the last year, the copper price has ranged from $4.00 to $4.50 for most of the year but is currently approximately $4.60 per pound, which is well above the three-year trailing average of $3.25. Global demand increases for copper have the potential to drive copper prices higher, thereby increasing the economic (revenue) outlook for the Project.

4. **Additional Resources.** Section 14 reports 187.2 million tons of inferred mineral resources at an average grade of 0.17% total copper. It is uncertain if further exploration will result in this mineralization being delineated as an indicated or measured mineral resource. However, if these inferred mineral resources can be converted to the measured or indicated categories, they have the potential to increase the mineral reserve and improve the economic outlook of the Project.

5. **Well Field Optimization.** No effort has been made to optimize well spacing or diameter for the Project. The spacing between wells determines the number of wells required to leach the Gunnison oxide deposit. Operator experience over the life of operation also has the potential to increase individual well flow rates that would result in increased well spacing distances reducing the number of required wells to leach the entire wellfield. A decrease in the diameter of wells installed would result in decreasing well installation costs.

6. **Limestone and Lime Resources.** The Project has high quality limestone resources that could be used to supplement imported lime in the water treatment process resulting in a reduction in lime consumption. These limestone resources could also be evaluated for use in making lime onsite thereby saving the cost of transportation to the site.

7. **Well Stimulation Techniques.** Well Stimulation has the potential to increase fracture connectiveness, fracture intensity, fracture aperture, porosity and sweep efficiency. Well stimulation is defined by the EPA as several processes used to clean the well bore, enlarge channels, and increase pore space in the interval to be injected thus making it possible for solutions to move more readily into the formation, and includes (1) surging, (2) jetting, (3) blasting, (4) acidizing, (5) hydraulic fracturing. Excelsior is investigating well stimulation techniques developed in the oil and gas and mining industries to enhance flows of fluids and solutions in the subsurface. These techniques include the pressure development of fracturing that could increase flows between injection wells and recovery wells. Well stimulation also has the potential to solve or alleviate the CO_2 blocking problem, or if used in combination with raffinate neutralization, reduce the amount or time needed during flushing of CO_2. Well stimulation requires EPA approval.

8. **Well Spacing Reduction.** Excelsior is exploring the possibility to reduce the well spacing in the well field to shorten the flow path between injection and recovery wells.

26 RECOMMENDATIONS

Based on the results of this Prefeasibility Study, it is recommended that Excelsior proceed with the Project through the engineering, procurement, and construction necessary to restart active production once financing is secured. The engineering for the water treatment infrastructure needs to be advanced in accord with the project development schedule. The drilling, mineral resource estimation, wellfield mine planning, wellfield drilling and infrastructure development and the staged SX-EW plant have all been adequately defined. The initial wellfield is drilled and solution is being pumped for processing, but the addition of raffinate neutralization capability is considered necessary to resolve the wellfield circulation and production difficulties. Other factors may be contributing to circulation (sweep efficiency) and production difficulties that are masked by the CO_2 flow restrictions. The following sections discuss areas for potential investigation and risk reduction.

26.1 METALLURGICAL TESTWORK RECOMMENDATIONS

Section 13 outlines four recommendations for investigating in-situ leaching with different lixiviants: sulfurous acid, ammonium carbonate, ammonium sulfate with oxygen, and glycine leaching. All of these techniques have received some attention as opportunities to leach metals without the formation of gypsum. A program of laboratory testwork should be undertaken to determine if any of these lixiviants are worth pursuing.

26.2 WELLFIELD RECOMMENDATIONS

Flow attenuation associated with the addition of acidified leach solutions to the wellfield has been attributed to the buildup of CO_2 in the formation due to the dissolution of calcite and other carbonates. Continued research into the causes of the flow rate attenuation and buildup in the formation should be continued. Changes in wellfield operational parameters should be considered to learn more about the conditions which lead to reductions in flow and the methods which can be used to enhance flow. Flow profiling down wells and in adjacent wells whilst under acid operations and during CO_2 flushing should be considered to help determine individual flow paths and the relative effect of CO_2 blocking on structure size, depth and connectiveness. Flow profiling can also assist with the identification of short-circuiting structures that can then be address through remedial actions.

Laboratory experimentation is recommended to ensure that neutralized raffinate is effective in dissolving CO_2 in the subsurface while the engineering, procurement, and construction is at an early stage to enhance the water treatment design criteria. Those experiments should also address the solubility of gypsum in mixed acidified and neutralized raffinate solutions to avoid conditions which might result in damage to the formation.

Well stimulation trials should be undertaken to determine if the technique(s) have the potential to alleviate or solve CO_2 blocking, improve connectiveness, and increase flow rates and sweep efficiency. Given that the results of well stimulation have the potential to reduce the need for raffinate neutralization or change the design criteria for the neutralization plant, it should be undertaken before or in parallel with design activities on the water treatment plant. Well stimulation is allowed under Class III Underground Injection Control permits but requires EPA approval of the stimulation programs.

26.3 WATER TREATMENT

A scope of work and bid package should be assembled to select a water treatment vendor to design the water treatment system. Vendors should be screened and selected to advance the engineering process to shrink the implementation schedule. Selection criteria should favor rapid, low-cost solutions to demonstrate that the technology is effective in solving the wellfield challenges.

26.4 BUDGET FOR ADDITIONAL WORKS

Excelsior has proposed a list and budget for additional work that will support the feasibility study. Table 26-1 below defines the cost for the technical activities.

Table 26-1: Prefeasibility Budget for the Gunnison Copper Project

Detail	Cost US$
Metallurgical Testwork	
Sulfurous acid leaching	$50,000
Ammonium carbonate leaching	$40,000
Ammonium sulfate leaching with oxygen	$40,000
Glycine leaching investigation	$65,000
Subtotal Metallurgical Testwork	$190,000
Wellfield Studies	
Flow attenuation	$150,000
CO_2 dissolution in neutralized raffinate testwork	$100,000
Well stimulation trials	$1,500,000
Flow profiling (mapping)	$500,000
Subtotal Wellfield Studies	$2,250,000
Water Treatment Testwork	
Raffinate neutralization testwork	$50,000
Solids management and densification testwork	$50,000
Solid liquid separation and filtration studies	$75,000
Subtotal Water Treatment Studies	$175,000
Total	$2,615,000

27 REFERENCES

A. J. Gerbino, 2021. “Simulation study results for Excelsior”, OLI systems, inc., April 12, 2021.

Arizona Department of Environmental Quality (ADEQ), 1996, revised 2008. A Screening Method to Determine Soil Concentrations Protective of Groundwater Quality, prepared by the Leachability Working Group of the Cleanup Standards/Policy Task Force, September.

Arizona Department of Environmental Quality (ADEQ), 2004. Arizona Mining BADCT Guidance Manual. Publication #TB 04-01.

ADEQ, 2004b, Interoffice Memorandum on Alert Level Calculations for Mining APPs, from Bill Kopp to Mining Unit Staff, April 1, 2002, revised October 19, 2004.

ADEQ, 2004. Arizona Mining BADCT Guidance Manual. Publication #TB 04-01.

ADEQ, 2007. Arizona Administrative Code (A.A.C.). Title 18 - Environmental Quality, Chapter 7 -Department of Environmental Quality Remedial Action, Article 2. Soil Remediation. http://www.azsos.gov/ public services/title_18/18-07.htm

Arizona Department of Water Resources (ADWR), 2008. Well Abandonment Handbook. September 2008.

Arizona Geological Survey, 2015. Natural Hazards in Arizona. On-Line Arizona Natural Hazards Viewer. http://data.usgin.org/hazard-viewer.

Baker, Arthur III, 1953. Localization of Pyrometasomatic Ore Deposits at Johnson Camp, Arizona, AIME Technical Paper 36841; Mining Engineering, V.5, no. 12, pp. 1272-1277. Cooper, J.R. and Silver, L.T. 1964. Geology and Ore Deposits of the Dragoon Quadrangle, Cochise Co., Arizona: USGS Professional Paper 416, 196 pp.

Barton, I. F., and Hiskey, J. B. 2019. “Kinetics of Chalcopyrite Leaching in Novel and Exotic Lixiviants”, CIM 2019.

Bikerman Engineering & Technology Associates Inc. 2007, Johnson Camp Mine Project Feasibility Study, Cochise County, Arizona – NI 43-101 Technical Report, prepared for Nord Resources, 234 p.

Burt, D.M., 1977. Mineralogy and petrology of skarn deposits: Rendiconti della Soc. Ital. di Mineral. e Petrol., v. 33, p. 859-873.

Cochise County, Arizona, 2014. Master Final Zoning Regulations, revised February 25, 2014. https://www.cochise.az.gov/sites/default/files/planning_and_zoning/MASTER FINAL ZONING REGULATIONS 2-25-14.pdf

Colburn, N.I., Perry, A.J., March 17, 1976: Perry, Knox, Kaufman, Inc., Perry, Knox, Kaufman, Inc. internal company report, 34 pages plus figures.

Cooper, J. R., and L. T. Silver, 1964. Geology and Ore Deposits of the Dragoon Quadrangle, Cochise County, Arizona. Geological Survey Professional Paper 416. United States Government Printing office, Washington, 196 pp.

Curtis & Associates, 2011: Johnson Camp Mine Pad 5 Expansion Project, Feasibility Study with Revisions prepared for Nord Resources, 49p.

Curtis & Associates, 2013: Johnson Camp Mine Facility and Operation, A Comprehensive Report prepared for Nord Resources, 141p

Darling Environmental and Surveying, Ltd., 2010. Gunnison Project Threatened and Endangered Species Analysis: Internal AzTech Minerals, Inc. report, 25 p.

Darling Environmental and Surveying, Ltd., 2014. Gunnison Project 2014 State Land Threatened and Endangered Species Analysis: Internal Excelsior Mining Corp. report, 41 p.

Darling Environmental and Surveying, Ltd., 2015. Gunnison Project 2014 State Land Threatened and Endangered Species Analysis: Internal Excelsior Mining Corp. report, 9 p.

Darling Geomatics. 2016. Gunnison Copper Project biological evaluation 2016 Threatened & Endangered species analysis, T. 15S., R. 22E, Sec. 36 and T. 15S., R. 23 E., Sec. 31, Cochise County, Arizona.

Davis, S.N., and R.J.M. DeWiest, 1966. *Hydrogeology*. New York, NY. John Wiley and Sons, 463 pp.

Dickens, Chuck M., 2003. Characterization of Hydrogeologic Conditions, Johnson Camp Mine. Prepared in Support of an Application for an Aquifer Protection Permit. July 31, 2003.

Drewes, Harald, Kelley, W.N. and Munts, S.R., 2001, Tectonic map of southeast Arizona: a digital database for the west part: U.S. Geological Survey Miscellaneous Investigations Series Map I-1109, digital database, version 1.0, 29 p., 1 digital sheet, scale 1:125,000.

Early D., 2016, Metallurgical Test Model (MTM) Calibration Results for the Gunnison Feasibility Project April 25, 2016, 17 p.

Einaudi, M.T., Meinert, L.D., Newberry, R.J., 1980. Skarn Deposits *in* Skinner, B.J., ed., Economic Geology 75th Anniversary Volume 1905-1980: El Paso, Texas, The Economic Geology Publishing Company, p. 317-391.

Ekenes, J. M., and Caro, C. A., "Improving leaching recovery of copper from low-grade chalcopyrite ores", Minerals and Metallurgical Processing, 2013, Vol.30., No. 3, pp.180-185.

Engineering Enterprises, Inc., 1985. Guidance Document for the Area of Review Method. ftp://ftp.consrv.ca.gov/pub/oil/EPA/Guidance_Document_for_Area_of_Review_Requirement.pdf

Federal Emergency Management Agency (FEMA), 2008. Flood Insurance Rate Map. Cochise County, Arizona and Incorporated Areas. Map Number 04003C0985F. Number 040012 Panel 0985 Suffix F. Effective Date August 28, 2008.

Fellows, L. D., 2000. 3. *Arizona Geology*, Vol. 30, No. 1, Spring 2000. Published by the Arizona Geological Survey.

Freeze, R. A., and J. A. Cherry, 1979. *Groundwater*. Prentice-Hall Inc., Englewood Cliffs, NJ. 604 pp.

Glasgow Engineering Group Inc, 2010, Technical Design Report for Johnson Camp Mine Leach Pad No. 5; APP Facility No. 28 (Nord Resources), 355 p.

Griffith, G.E., Omernik, J.M., Johnson, C.B., and Turner, D.S., 2014. Ecoregions of Arizona (poster): U.S. Geological Survey Open-File Report 2014-1141, with map, scale 1:1,325,000.

Haley and Aldrich, 2015. Hydrogeologic Well Completion Report, Gunnison Copper Project, Cochise County, Arizona. July.

Halpenny, L. C., 1973. Letter from Leonard C. Halpenny, Water Development Corporation, to William E. Seaegart, Quintana Minerals Corporation, dated February 14, 1973. 16pp.

Harshbarger & Associates, 1973. Hydrogeological Conditions and Potential Groundwater Supply, Johnson Camp Mine Area. December 6, 1973.

HATCH, 2022. Excelsior Mining Corp. Gunnison Project Mine Water Treatment Concept Design. Prepared for Excelsior Mining Corp. February 14, 2022.

Hazen Research, Inc. ("HRI") for BHP Copper, 1996. Project No. 8619, "Oxide Copper Column Leaching Experiments with Sulfurous Acid." September 9, 1996.

Hazen Research, Inc. ("HRI") for Excelsior Mining Arizona, Inc., 2011. Project No 11245, "Copper Ore Column Leach Experiments" Draft. September 2011.

Kantor, J. A., 1977. Structure, Alteration and Mineralization on the Flanks of the Texas Canyon Stock, Cochise County, Arizona. Society of Mining Engineers Preprint Number 77-I-321. For presentation at the 1977 SME Fall Meeting and Exhibit, St. Louis, Missouri – October 19-21, 1977.

Keith, S.B., Gest, D.E. DeWitt, E., Toll, N.W., Everson, B.A., 1983, Metallic Mineral Districts and Production in Arizona: Bulletin 194, Arizona Bureau of Geology and Mineral Technology, p. 22-23.

Kim, Eui-Jun; Shin, Dongbok; Shin, Seungwook; Nam, Hyeong-Tae; Park, Samgyu; 2015. Skarn zonation and rock physical properties of the Wondong Fe-Pb-Zn polymetallic deposit, Korea, Geosciences Journal: The Association of Korean Geoscience Societies, V 19, no 4, p 587-598.

King, A.M. 2014. A cultural resources inventory of 256 acres near Dragoon, Cochise County, Arizona. WestLand Resources, Inc.

King, P.B. and Beikman, H.M., 1974, Geological Map of the United States. US Geological Survey.

Kuhnel V., 2016, Distribution of hydraulic conductivities in the Gunnison Ore body.

Livingston, D.E., Damon, P.E., Mauger, R.L., Bennett, R., and Laughlin, A.W., 1967. Argon 40 in Cogenetic Feldspar-Mica Mineral Assemblages: Journal of Geophysical Research, v. 72, no. 4, pp. 1361-1375.

M3 Engineering and Technology Corporation, 2014. Gunnison Copper Project NI 43-101 Technical Report Prefeasibility Study, Cochise County, Arizona, USA. Revision 0, 14 February 2014. Prepared for Excelsior Mining Corp. 287pp.

M3 Engineering and Technology Corporation, 2016. Gunnison Copper Project Prefeasibility Update, Cochise County, Arizona, USA. Prepared for Excelsior Mining Corp.

M3 Engineering and Technology Corporation, 2017. Gunnison Copper Project Feasibility Study, Cochise County, Arizona, USA. Prepared for Excelsior Mining Corp.

Magma Copper Co. Metallurgical Laboratory ("Magma"), 1992. Report, No. ML-2093, "Bottle Roll Leach Test: Johnson Camp Samples," December 8, 1992.

Magma Copper Co. Metallurgical Laboratory ("Magma"), 1993. "Addendum to ML 2093," January 14, 1993.

Magma Copper Co. Metallurgical Laboratory ("Magma"), 1995. Report No. ML-2343, "Bottle Roll Leach Tests: Johnson Camp Samples – Composite #1 and #2, Crushed to Minus 10-mesh," February 27, 1995.

Magma Copper Co. Metallurgical Laboratory ("Magma"), 1996. Report No. ML-2494, "Mini-Column Leaching of I-10 Epoxy Coated Samples." January 4, 1996.

McDonald, M.G., and Harbaugh, A.W., 1988. A modular three-dimensional finite-difference ground-water flow model: Techniques of Water-Resources Investigations of the United States Geological Survey, Book 6, Chapter A1, 586 p.

Meinert, L.D., Dipple, G.M., and Nicolescu, S., 2005. World Skarn Deposits, *in* Hedenquist, J.W., Thompson, J.F.H., Goldfarb, R.J., and Richards, J.P., *eds.*, Economic Geology 100th Anniversary Volume 1905-2005: Littleton, Colorado, Society of Economic Geologists, Inc., p. 299-336.

Metcon Research for The Superior Oil Co., 1972. Project No. 50-92.10," Leach Testing on JO-1, JO-2, and JO-3", June 29, 1972.

Mountain States Research & Development, Inc. ("MSRDI") for Quintana Minerals Corp., 1973a. Project No. 2019, "Preliminary Metallurgical Tests – Johnson Camp Venture # 2, January 18, 1973.

Mountain States Research & Development, Inc. ("MSRDI") for Quintana Minerals Corp, 1973b. Project 2019, Progress Report No. 2, March 30, 1973.

NORAM Engineering and Construction Ltd, 2022. 1,650 stpd Sulfur Burning Sulfuric Acid Plant Study Report. Prepared for Excelsior Mining Corp and M3 Engineering. January 31, 2022.

PAST. 2010. Archaeological survey of The North Dragoon Basin Project near Dragoon, Cochise County, Arizona (A).

PAST. 2010. Archaeological survey of The Thing View Project near Dragoon, Cochise County, Arizona (B).

Pearthree, P. A., and A. Youberg, 2006. Recent Debris Flows and Floods in Southern Arizona. *Arizona Geology*, Vol. 36, No. 3. Fall 2006. Published by the Arizona Geological Survey.

Petersen, M.D., Moschetti, M.P., Powers, P.M., Mueller, C.S., Haller, K.M., Frankel, A.D., Zeng, Yuehua, Rezaeian, Sanaz, Harmsen, S.C., Boyd, O.S., Field, E.H., Chen, Rui, Luco, Nicolas, Wheeler, R.L., Williams, R.A., Olsen, A.H., and Rukstales, K.S., 2015, Seismic-hazard maps for the conterminous United States, 2014. USGS Scientific Investigations Map 3325, 6 sheets, scale 1: 7,000,000, http://dx.doi.org/10.3133/sim3325.

Phelps Dodge Mining Company's Process Technology Center, by E. A. Rood (Phelps Dodge), 1996. "I-10 Project, Metallurgical Testing Update." December 17, 1996.

Professional Archeological Services of Tucson (P.A.S.T.), 2010a. Archeological Survey of the Thing View Project near Dragoon, Cochise County, AZ. Cultural Resources Report No. 101981.

Professional Archeological Services of Tucson (P.A.S.T.), 2010b. Archeological Survey of the North Dragoon Basin Project near Dragoon, Cochise County, AZ. Cultural Resources Report No. 101974.

R. J. Roman, 2011. "Johnson Camp Column Leach Program (8-inch Columns)," Final Report, Revision 1, for Nord Resources Corporation, May 12, 2011.

R. Zimmerman, et al., 2017. NI 43-101 Technical Report – Feasibility Study, "Gunnison Copper Project", prepared by M3 for Excelsior Mining Corp, issued on January 16, 2017.

Ramsahoye, L. E., and S. M. Lang, 1961. A Simple Method for Determining Specific Yield from Pumping Tests. Ground-Water Hydraulics. Geological Survey Water-Supply Paper 1536-C. Prepared in cooperation with the New Jersey Department of Conservation and Economic Development. USGS, U. S. Government Printing Office, Washington, D.C. 46 pp.

Rebolledo, M., Zarate, G., and Mora, N. 2019. “Catalytic Heap Leaching of Chalcopyrite Ores Using Jetti’s Technology”, CIM 2019.

Regnault O., Lagneau V., Fiet N., Langlais V., 2013, Reactive transport simulation of Uranium ISL at the block scale a tool for testing designs and operation scenarios.

Reynolds, S.J., Florence, F.P., Welty, J.W., Roddy, M.S., Currier, D.A., Anderson, A.V., Keith, S.B., 1986. Arizona Bureau of Geology and Mines Technical Bulletin 197, Compilation of Radiometric Age Determinations in Arizona, 258 pp.

Richard, Stephen M., Todd C. Shipman, Lizbeth C. Greene, and Raymond C. Harris, 2007, Estimated Depth to Bedrock in Arizona, Arizona Geological Survey Digital Geologic Map Series DGM-52, Version 1.0.

S. Twyerould, President, Excelsior Mining Corp, private communication, February 8, 2022.

Shipman, T.C., 2007. Cochise County Earth Fissure Planning Map, Arizona: Arizona Geological Survey Open File Report 07-01, v1, Sheet 3, scale 1:250,000.

Sillitoe, R., 1989. Gold Deposits in the Western Pacific Island Arcs: The Magmatic Connection. Economic Geology Monograph 6, pp 266-283.

Sillitoe, R.H, 2010. Porphyry Copper Systems. Economic Geology, 105, 3-41

Stone, B.W.S. 2017. A cultural resources inventory of approximately 66.85 acres of Arizona state trust land and 265.88 acres of private land northeast of Dragoon, Cochise County, Arizona. WestLand Resources, Inc.

United States Historical Climatology Network (USHCN), 2015. Monthly Climate Records for Station 026353, PEARCE SUNSITES, Arizona.

Veolia Water Technologies, 2016. Excelsior Mining Gunnison Project Water Treatment Concept Design, October 31, 2016.

Weitz, T.J., 1976. Geology and Ore Deposits at the I-10 Prospect, Cochise County, Arizona, Unpublished M.S. Thesis, University of Arizona, 85 pp.

Westland Resources Inc., 2014. Cultural Resources Report 2014-52A, A Cultural Resource Inventory of 256 Acres near Dragoon, Cochise County, Arizona,: Internal Excelsior Mining Corp. report, 53p.

Young, J., and P. Pearthree, 2014. Duncan M5.3 Earthquake of June 2014 and Temporary Seismic Network Deployment. Seismic News, Arizona Geological Survey, October 17, 2014. http://azgeology.azgs.az.gov/article/seismic/2014/10/duncan-m53-earthquake-june-2014-and-temporary-seismic-network-deployment.

Yunge, H. 1965. Chemical compounds and radio-activity in atmosphere; Clarendon –press: Oxford, 1965.

APPENDIX A: PREFEASIBILITY STUDY AND PRELIMINARY ECONOMIC ASSESSMENT CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE OF QUALIFIED PERSON

Richard K Zimmerman, M.Sc., R.G., SME-RM

I, Richard K Zimmerman, M.Sc., R.G., SME-RM, do hereby certify that:

1. I am currently employed as Environmental Geologist by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Ste. 101
   Tucson, AZ 85704

2. I am a graduate of Carleton College and received a Bachelor of Arts degree in Geology in 1976. I am also a graduate of the University of Michigan and received a Master of Science degree in Geology in 1980.

3. I am a:

   - Registered Professional Geologist in the State of Arizona (No. 24064)
   - Registered Member in good standing of the Society for Mining, Metallurgy and Exploration, Inc. (No. 3612900RM)

4. I have practiced geology, mineral exploration, environmental remediation, and project management for 41 years. I have worked for mining and exploration companies for 9 years, engineering consulting firms for 22 years. The past 11 years have been spent with M3 Engineering & Technology Corporation managing, planning, and constructing processing plants for base and precious metals including over 20 technical reports.

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

6. I am responsible for Sections 1, 2, 3, 4, 5, 17, 18, 19, 21, 22, 23, 24.1, 24.2, 24.3, 24.4, 24.5, 24.17, 24.18, 24.19, 24.21 (except 24.21.1, 24.21.2), 24.22, 24.23, 24.24, 24.25, 24.26, 24.27, 25, 26, 27 of the technical report "Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment" (the "Technical Report") dated effective February 1, 2023, prepared for Excelsior Mining Corp.

7. I had prior involvement with the property that is the subject of the Technical Report. I was involved in the preparation of the "Gunnison Copper Project, NI 43-101 Technical Report, Feasibility Study" dated effective December 17, 2016 and in the preparation of the technical report "Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment" (the "Technical Report") dated effective March 11, 2022.

8. I visited the Gunnison site on September 28, 2021.

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

10. I am independent of the issuer applying all tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them of the Technical Report for regulatory purposes including electronic publication in the public company files on their websites accessible by the public.

Dated this 22nd day of February 2023.

(Signed and Sealed) *Richard K. Zimmerman*
Richard K. Zimmerman, M.Sc., R.G., SME-RM

CERTIFICATE OF QUALIFIED PERSON

Jeffrey Bickel, C.P.G.

I, Jeffrey Bickel, C.P.G., do hereby certify that:

1. I am employed as a Senior Geologist of:

   Mine Development Associates, a Division of RESPEC.
   210 South Rock Blvd., Reno, Nevada 89502

2. I graduated with a Bachelor of Science Degree from Arizona State University in 2010.
3. I am a Certified Professional Geologist (#12050) with the American Institute of Professional Geologists, a Registered Geologist in the state of Arizona (#60863), and a Registered Member of the Society of Mining, Metallurgy, and Exploration (4184632RM).
4. I have worked as a geologist in the mining industry for more than 12 years. I have previously explored, drilled, modeled, and evaluated copper skarn deposits in Arizona and Nevada, and have participated in mineral resource estimations in accordance with NI 43-101 guidelines for such deposits.
5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.
6. I am responsible for Sections 6, 7, 8, 9, 10, 11, 12, 14, 24.6, 24.7, 24.8, 24.9, 24.10, 24.11, 24.12, 24.14 of the technical report “Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment” (the “Technical Report”) dated effective February 1, 2023, prepared for Excelsior Mining Corp.
7. I have visited the project site on May 3, 2021.
8. I worked as a geologist for Excelsior from 2010-2020.
9. I was a co-author of the technical report entitled “*Estimated Mineral Resources and Preliminary Economic Analysis, Strong and Harris Copper-Zinc-Silver Project, Cochise County Arizona*”, prepared for Excelsior Mining Corporation, filed October 22, 2021. I was also co-author of the technical report “Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment” (the “Technical Report”) dated effective March 11, 2022.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 22nd day of February 2023.

“Jeffrey Bickel” (“signed”)
Jeffrey Bickel, C.P.G.

CERTIFICATE OF QUALIFIED PERSON

THOMAS L. DYER, P.E., SME-RM

I, Thomas L. Dyer, P. E., SME-RM do hereby certify that I am currently employed as Principal Engineer by RESPEC, Inc., 210 South Rock Blvd., Reno, Nevada 89502 and:

1. I graduated with a Bachelor of Science degree in Mine Engineering from South Dakota School of Mines & Technology in 1996. I have worked as a Mining Engineer for 26 years since graduation. During my Engineering career, I have held various positions of increasing responsibility at operating mines performing life of mine planning and cost estimates. During the last 15 years, I have been engaged in consulting on various lead, zinc, gold, silver, copper, and limestone deposits both for underground and open pit operations. This consulting work has primarily consisted of providing production schedules, mine cost estimates, and cash-flow analysis.
2. I am registered as a Professional Engineer – Mining in the State of Nevada (# 15729). I am also a Registered Member of SME (# 4029995RM) in good standing.
3. 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. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
4. I am one of the authors of the Technical Report titled *"NI 43-101 Technical Report Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment"* dated effective February 1, 2023 (the "Technical Report"). I am responsible for the preparation of sections 24.16.1 through 24.16.4 as well as mining costs in sections 24.21.1 and 24.21.2. I have most recently visited the property on March 18, 2021, to review current infrastructure and scope out future infrastructure and road requirements.
5. I have not had prior involvement with the Gunnison Mine or the Johnson Camp Mine project that is subject to this Report.
6. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this February 22, 2023.

(Signed and Sealed) "Thomas L. Dyer"

Thomas L. Dyer, P.E., SME-RM

CERTIFICATE of QUALIFIED PERSON

I, Neil Prenn, MMSA-QPM, do hereby certify that:

1. I am currently working with Respec/MDA as a consultant at: 210 S. Rock Blvd., Reno, Nevada, 89502

2. I am a graduate of the Colorado School of Mines in 1967 with an Engineer of Mines degree.

3. I am a: Registered Professional Engineer in the State of Nevada, and a registered Qualified Person with the Mining and Metallurgical Associates of America (MMSA).

4. I have worked as a Mining Engineer for more than 50 years, providing mine designs, reserve estimates and economic analyses for dozens of base- and precious-metals deposits and industrial minerals deposits in the United States and various countries of the world.

5. I have read the definition of "qualified person" set out in National instrument 43-101 ("NI 43- 10l ") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for Section 15 of the technical report titled “Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment” (“Technical Report”) dated effective February 1, 2023.

7. I have prior involvement with the property that is subject of the Technical Report. I was responsible for Section 15 of the technical report titled "Gunnison Copper Project, NI 43-101 Technical Report, Feasibility Study" (the "Technical Report") dated effective December 17, 2016 and for Section 15 of the technical report titled “Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment” (“Technical Report”) dated effective March 11, 2022.

8. I have visited the Gunnison Project Site, and inspected core from drilling on the Gunnison Project on November 10, 2021.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Signed and dated this 22nd day of February 2023.

(Signed) *Neil Prenn*
Neil Prenn, MMSA-QPM

srk consulting

SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
City and County of Cardiff
CF10 2HH, Wales
United Kingdom
E-mail: enquiries@srk.co.uk
URL: www.srk.com
Tel: + 44 (0) 2920 348 150
Fax: + 44 (0) 2920 348 199

CERTIFICATE OF AUTHOR

Robert John Bowell

Corporate Consultant (Geochemist)
SRK Consulting (UK) Ltd
Email: rbowell@srk.co.uk

I, Robert J Bowell, a Chartered Professional Chemist, Chartered Geologist, and a Certified Professional European Geologist, do hereby certify that:

1. I am responsible for the preparation of the technical section on the wellfield performance (Section 13.2.3.1) of the report titled, "Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment" (the "Technical Report") dated effective February 1, 2023, prepared for Excelsior Mining Corp.
2. I was previously involved in the preparation of the technical section on the wellfield performance (Section 13.2.3.1) of the report titled, "Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment" (the "Technical Report") dated effective March 11, 2022, prepared for Excelsior Mining Corp.
3. I visited the Project site on September 28, 2021.
4. I am currently employed as a consulting geochemist to the mining and mineral exploration industry, as a Corporate Consultant Geochemist with SRK Consulting (UK) Ltd, with an office address of 5th Floor Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, UK.
5. I graduated with a Bachelor of Science Degree, First Class Honours in Geochemistry from Owen's College, Manchester University, Manchester UK, June 1988.
6. I graduated with a Doctorate in Geochemistry from Southampton University, Southampton, UK in June 1991.
7. I am a Chartered Chemist of the Royal Society of Chemistry, London, UK and have been since 1997. Membership number 332782.
8. I am a Chartered Geologist and Certified Professional European Geologist through the Geological Society of London since 1997 and European Association of Professional Geologists since 2000. Registration number 1007245.
9. I am a Fellow of the Institute of Mining, Metallurgy and Materials and have been since 2010.
10. I have been employed as a geochemist in the mining and mineral exploration business and in applied academia, for the past 33 years since my graduation from university.

11. I have read the definition of “qualified person” set out in National Instrument 43-101 of the *Standards of Disclosure for Mineral Projects* (“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 fulfil the requirements to be a “qualified person” for the purposes of NI 43-101. The Technical Report is based upon my personal review of the information provided by the Issuer. My relevant experience for the purpose of the Technical Report is:
    - Geochemist, SRK Consulting from 1995 to date
    - Exploration Geochemist with BHP Minerals, Hammersmith, London., 1991-1994
    - Exploration Geologist, Ashanti Goldfields, Ghana, 1988
    - Exploration experience as a geochemist and geometallurgical consultant, from 1995-2003; 2005-2006; 2008-current on In Situ Projects.
    - Experience in the above positions working with and reviewing Copper mineralogy and geology, analysis, resource estimation methodologies, geometallurgical testwork leaching mechanisms, In Situ Recovery, SX-EW recovery of copper, geochemical data quality, assurance and quality control in concert with resource estimation geologists and engineers.
12. As a consultant, I have been involved in several previous competent person’s reports for Cu projects including NI 43-101 technical reports.
13. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
14. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, for which the omission to disclose would make the Technical Report misleading.
15. I am independent of Excelsior applying the test in section 1.5 of NI 43-101.
16. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public.

Dated in Cardiff, United Kingdom, February 22, 2023.

This signature has been scanned. The author has given permission to its use for this particular document. The original signature is held on file.

Bowell

SRK CONSULTING (UK) LIMITED Co. No: 01575403

(“signed”) (“sealed”)

***Eur.Geol.* Robert Bowell PhD C.Chem. C.Geol**
Corporate Consultant (Geochemist)

CERTIFICATE OF QUALIFIED PERSON

Dr. Terence P. McNulty, PE, DSc

I, Dr. Terence P. McNulty, PE, DSc, do hereby certify that:

1. I am President of:

   T, P, McNulty and Associates, Inc,
   4321 North Camino de Carrillo, Tucson, AZ 85750

2. I graduated with a BS in Chemical Engineering from Stanford University in 1960 and earned an MS in Metallurgical Engineering from Montana School of Mines in 1963 and a doctorate (DSc) from Colorado School of Mines in 1966.
3. I am a Registered Professional Engineer in Colorado with reciprocity in most states. My registration is current (No. 24789) and I am in good standing.
4. I have worked as a metallurgical engineer for a total of over 55 years since completion of post-graduate studies. My experience includes serving as a Research Engineer, Mill Superintendent, Supervisor of Process Engineering, and Director of Corporate R&D for The Anaconda Company, VP-Technical Operations for Kerr-McGee Chemical Corp., President of Hazen Research, Inc., and President of T. P. McNulty and Associates, Inc. for the last 33 years.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for Sections 13 (except 13.2.3.1) and 24.13 of the Technical Report "Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment" (the "Technical Report") dated effective February 1, 2023, prepared for Excelsior Mining Corp.
7. I last visited the Johnson Camp Site in the 1990s when it was owned by Cyprus Minerals.
8. I had prior involvement with the property that is the subject of the Technical Report. I was responsible for the preparation of Sections 13 (except 13.2.3.1) and 24.13 of the Technical Report "Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment" (the "Technical Report") dated effective March 11, 2022, prepared for Excelsior Mining Corp.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer by applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 22nd day of February 2023.

(Signed and Sealed) Terence P. McNulty

Dr. Terence P. McNulty, PE, DSc

CERTIFICATE of QUALIFIED PERSON

R. Douglas Bartlett, C.P.G.

I, R. Douglas Bartlett, do hereby certify that:

1. I am currently employed as a Hydrogeologist by:

   Clear Creek Associates
   8777 N. Gainey Center Dr., Suite 250
   Scottsdale, Arizona, 85258

2. I am a graduate of Colorado State University.

3. I am a:

   Registered Geologist in the States of Arizona, California, Oregon, Washington, and Alaska

4. I have practiced geology and hydrogeology since 1977 at: Dames & Moore in Denver and Phoenix; Anaconda Minerals in Denver, Colorado; and Clear Creek Associates in Scottsdale, Arizona. My expertise includes mining-related hydrogeologic investigations and groundwater modeling.

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

6. I am responsible for Sections 16, 20, 24.16.5, and 24.20 of the technical report titled “Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment” (“Technical Report”) dated effective February 1, 2023.

7. I had prior involvement with the property that is the subject of the Technical Report. I was responsible for the Sections 16, and 20 of the technical reports titled “Gunnison Copper Project, NI 43-101 Technical Report, Prefeasibility Study Update” (the "Technical Report") dated effective January 28, 2016, and “Gunnison Copper Project, NI 43-101 Technical Report Feasibility Study” (“Technical Report”) dated effective December 17, 2016 prepared for Excelsior Mining Corp. I was also responsible for Sections 16, 20, and 24.20 of the technical report titled “Gunnison Copper Project Prefeasibility Study Update and JCM Heap Leach Preliminary Economic Assessment” (“Technical Report”) dated effective March 11, 2022.

8. I first visited the Gunnison Site on May 15, 2019.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Signed and dated this 22nd day of February 2023.

(Signed) *R. Douglas Bartlett*
R. Douglas Bartlett, C.P.G.

APPENDIX B: MINERAL CLAIM DETAIL

Patented Mining Claims (Johnson Camp)

Parcel 1
Arizona, Blue Grass, Puzzle, Enough, and Carlton patented lode mining claims, Mineral Survey No. 4340
Parcel 2
Afterthought, Burro, Burro No. 3, Coronado, Coronado No. 2, and Mason No. 1 patented lode mining claims, Mineral Survey No. 4571
Parcel 3
St. George patented lode mining claim, Mineral Survey No. 1966
Parcel 4
Mayflower (aka May Flower) patented lode mining claim, Mineral Survey No. 2764
Parcel 5
Acorn, A-Number One, A-Number Two, Chicago, Cochise, Copper Thread, Johnson, Little Johnnie, Rough Rider, Tenderfoot, and United Fraction patented lode mining claims, Mineral Survey No. 4314
Parcel 6
Blue Lead, North Star, Little Bush, Copper Chief, Southern Cross, Blue Lead Extension, Dwarf, and Esmeralda patented lode mining claims, Mineral Survey No. 3242 Anaconda, Last Chance, Delta, and Sara patented lode mining claims, Mineral Survey No. 1525
Parcel 8
Southern patented lode mining claim, Lot 45, Mineral Survey No. 327
Parcel 9
Mi-an-te-no-mah patented lode mining claim, Lot 48, Mineral Survey No. 330
Parcel 10
Peabody patented lode mining claim, Lot 39, Mineral Survey No. 286
Parcel 11
Donna Anna patented lode mining claim, Lot 40, Mineral Survey No. 287
Parcel 12
Highland Mary patented lode mining claim, Lot 37, Mineral Survey No. 284
Parcel 13
Copper King patented lode mining claim Lot 38, Mineral Survey No. 285 382681 v2
Parcel 14
Golden Shield patented lode mining claim, Lot 43, Mineral Survey No. 325
Parcel 15
Republic patented lode mining claim, Lot 42, Mineral Survey No. 324
Parcel 16
Chicora patented lode mining claim, Lot 44, Mineral Survey No. 326
Parcel 17
Tycoon patented lode mining claim, Lot 47, Mineral Survey No. 329
Parcel 18
Mammoth patented lode mining claim, Lot 49, Mineral Survey No. 331
Parcel 19
Clondike, Blue Jacket, Keystone, Blue Bell, Copper Bell, Dewey, True Blue, and Ross patented lode mining claims, Mineral Survey No. 1717
Parcel 20
382681 v2 Hillside, Pittsburg, and Teaser patented lode mining claims, Mineral Survey No. 3306
Parcel 21
San Jacinto patented lode mining claim, Lot 46, Mineral Survey No. 328

BLM Claims

CLAIM NAME AND NUMBER	BLM Serial # (AMC #)	TOWNSHIP, RANGE, SECTION*	Maintenance Costs	Area
		Mr Twn Rng Sec		
ALPHA #1	21945	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #2	21946	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #3	21947	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #4	21948	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #5A	351064	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #6	21950	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #7	21951	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #8	21952	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #9	21953	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #10	21954	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #11	21955	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #12	21956	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #13	21957	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #15	21959	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #16	21960	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #17	21961	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #18	21962	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #19	21963	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #20	21964	14 0160S 0220E 024	$165.00	Gunnison
ALPHA #22	21966	14 0160S 0220E 026	$165.00	Gunnison
ALPHA #23	21967	14 0160S 0220E 026	$165.00	Gunnison
ALPHA #24	21968	14 0160S 0220E 023	$165.00	Gunnison
ALPHA #25	21969	14 0160S 0220E 023	$165.00	Gunnison
ALPHA #26	21970	14 0160S 0220E 023	$165.00	Gunnison
ALPHA #31	21975	14 0160S 0220E 023	$165.00	Gunnison
ALPHA #32	21976	14 0160S 0220E 023	$165.00	Gunnison
ALPHA #33	21977	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 34 A	324360	14 0160S 0220E 023	$165.00	Gunnison
ALPHA #36	21980	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #37	21981	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #38	21982	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #39	21983	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #40	21984	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #45	21989	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #46	21990	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #49	21991	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #50	21992	14 0160S 0220E 025	$165.00	Gunnison
ALPHA #51	21993	14 0160S 0220E 025	$165.00	Gunnison
ALPHA 52 A	324361	14 0160S 0220E 026	$165.00	Gunnison
ALPHA 118	326439	14 0160S 0220E 001	$165.00	Gunnison
ALPHA 119	326440	14 0160S 0220E 001	$165.00	Gunnison
ALPHA 120	326441	14 0160S 0220E 001	$165.00	Gunnison
ALPHA 121	326442	14 0160S 0220E 001	$165.00	Gunnison
ALPHA 122	326443	14 0160S 0220E 001	$165.00	Gunnison
ALPHA 123	326444	14 0160S 0220E 001	$165.00	Gunnison
ALPHA 124	326445	14 0160S 0220E 001	$165.00	Gunnison
ALPHA 125	326446	14 0160S 0220E 011	$165.00	Gunnison

CLAIM NAME AND NUMBER	BLM Serial # (AMC #)	TOWNSHIP, RANGE, SECTION*	Maintenance Costs	Area
ALPHA 126	326447	14 0160S 0220E 011	$165.00	Gunnison
ALPHA 127	326448	14 0160S 0220E 011	$165.00	Gunnison
ALPHA 128	326449	14 0160S 0220E 013	$165.00	Gunnison
ALPHA 129	326450	14 0160S 0220E 013	$165.00	Gunnison
ALPHA 130	326451	14 0160S 0220E 013	$165.00	Gunnison
ALPHA 131	326452	14 0160S 0220E 013	$165.00	Gunnison
ALPHA 27	340653	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 28	340654	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 29	340655	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 30	340656	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 35	340657	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 41	340658	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 42	340659	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 43	340660	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 44	340661	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 56	340662	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 57	340663	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 58	340664	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 59	340665	14 0160S 0220E 023	$165.00	Gunnison
ALPHA 60	340666	14 0160S 0220E 023	$165.00	Gunnison
TALLSHIP 5-A	341334	14 0160S 0220E 013	$165.00	Gunnison
TALLSHIP 7-A	341335	14 0160S 0220E 013	$165.00	Gunnison
TALLSHIP 8-A	341336	14 0160S 0220E 013	$165.00	Gunnison
TALLSHIP 9-A	341337	14 0160S 0220E 012	$165.00	Gunnison
TALLSHIP 10-A	341338	14 0160S 0220E 012	$165.00	Gunnison
TALLSHIP B-1	341339	14 0160S 0220E 012	$165.00	Gunnison
TALLSHIP B-2	341340	14 0160S 0220E 012	$165.00	Gunnison
TALLSHIP B-3	341341	14 0160S 0220E 012	$165.00	Gunnison
TALLSHIP B-4	341342	14 0160S 0220E 012	$165.00	Gunnison
TALLSHIP B-5	341343	14 0160S 0220E 012	$165.00	Gunnison
TALLSHIP B-6	341344	14 0160S 0220E 013	$165.00	Gunnison
TALLSHIP B-7	341345	14 0160S 0220E 013	$165.00	Gunnison
TALLSHIP B-8	351062	14 0160S 0220E 013	$165.00	Gunnison
TALLSHIP B-9	351063	14 0160S 0220E 013	$165.00	Gunnison
TALLSHIP B10	341968	14 0160S 0220E 013	$165.00	Gunnison
TALLSHIP #C-1	73414	14 0160S 0220E 013	$165.00	Gunnison
TALLSHIP #C-2	73415	14 0160S 0220E 024	$165.00	Gunnison
TALLSHIP #C-3	73416	14 0160S 0220E 013	$165.00	Gunnison
TALLSHIP #C-4	73417	14 0160S 0220E 024	$165.00	Gunnison
TALLSHIP #C-5	73418	14 0160S 0220E 013	$165.00	Gunnison
TALLSHIP #C-6	73419	14 0160S 0220E 024	$165.00	Gunnison
TALLSHIP #C-7	73420	14 0160S 0220E 013	$165.00	Gunnison
PROSPECT 1	341969	14 0150S 0220E 035	$165.00	Gunnison
PROSPECT 2	341970	14 0150S 0220E 035	$165.00	Gunnison
PROSPECT 3	341971	14 0150S 0220E 035	$165.00	Gunnison
PROSPECT 4	341972	14 0150S 0220E 035	$165.00	Gunnison
PROSPECT 5	341973	14 0150S 0220E 035	$165.00	Gunnison
PROSPECT 6	341974	14 0150S 0220E 035	$165.00	Gunnison
PROSPECT 7A	341975	14 0150S 0220E 035	$165.00	Gunnison
PROSPECT 8A	341976	14 0150S 0220E 035	$165.00	Gunnison

CLAIM NAME AND NUMBER	BLM Serial # (AMC #)	TOWNSHIP, RANGE, SECTION*	Maintenance Costs	Area
PROSPECT 9	341977	14 0150S 0220E 035	$165.00	Gunnison
TEX 1	341978	14 0150S 0230E 031	$165.00	Gunnison
TEX 2	341979	14 0160S 0220E 001	$165.00	Gunnison
TEX 3	341980	14 0150S 0230E 031	$165.00	Gunnison
TEX 4	341981	14 0160S 0230E 006	$165.00	Gunnison
TEX 5	341982	14 0150S 0230E 031	$165.00	Gunnison
TEX 6	341983	14 0160S 0230E 006	$165.00	Gunnison
TEX 7	341984	14 0150S 0230E 031	$165.00	Gunnison
TEX 8	341985	14 0160S 0230E 006	$165.00	Gunnison
TEX 9	341986	14 0150S 0230E 031	$165.00	Gunnison
TEX 10	341987	14 0160S 0230E 006	$165.00	Gunnison
TEX 11	341346	14 0150S 0230E 031	$165.00	Gunnison
TEX 12	341988	14 0160S 0230E 006	$165.00	Gunnison
TEX 13	341347	14 0160S 0230E 006	$165.00	Gunnison
TEX 14	341989	14 0160S 0230E 005	$165.00	Gunnison
TEX 15	341990	14 0160S 0220E 001	$165.00	Gunnison
TEX 16	341348	14 0160S 0220E 001	$165.00	Gunnison
TEX 17	341991	14 0160S 0230E 006	$165.00	Gunnison
TEX 18	341349	14 0160S 0230E 006	$165.00	Gunnison
TEX 19	341992	14 0160S 0230E 006	$165.00	Gunnison
TEX 20	341993	14 0160S 0230E 006	$165.00	Gunnison
TEX 21	341994	14 0160S 0230E 006	$165.00	Gunnison
TEX 22	341995	14 0160S 0230E 006	$165.00	Gunnison
TEX 23	341996	14 0160S 0230E 006	$165.00	Gunnison
TEX 24	341997	14 0160S 0230E 006	$165.00	Gunnison
TEX 25	341998	14 0160S 0230E 006	$165.00	Gunnison
TEX 26	341999	14 0160S 0230E 006	$165.00	Gunnison
TEX 27	342000	14 0160S 0230E 005	$165.00	Gunnison
TEX 28	342001	14 0160S 0230E 005	$165.00	Gunnison
TEX 29	341350	14 0160S 0230E 006	$165.00	Gunnison
TEX 30	341351	14 0150S 0230E 031	$165.00	Gunnison
NANA-1	AZ105264914	14 0160S 0220E 026	$165.00	Gunnison
NANA-2	AZ105264915	14 0160S 0220E 026	$165.00	Gunnison
NANA-3	AZ105264916	14 0160S 0220E 026	$165.00	Gunnison
NANA-4	AZ105264917	14 0160S 0220E 026	$165.00	Gunnison
NANA-5	AZ105264918	14 0160S 0220E 026	$165.00	Gunnison
NANA-6	AZ105264919	14 0160S 0220E 026	$165.00	Gunnison
NANA-7	AZ105264920	14 0160S 0220E 026	$165.00	Gunnison
NANA-8	AZ105264921	14 0160S 0220E 026	$165.00	Gunnison
NANA-9	AZ105264922	14 0160S 0220E 026	$165.00	Gunnison
NANA-10	AZ105264923	14 0160S 0220E 026	$165.00	Gunnison
NANA-11	AZ105264924	14 0160S 0220E 026	$165.00	Gunnison
NANA-12	AZ105264925	14 0160S 0220E 026	$165.00	Gunnison
NANA-13	AZ105264926	14 0160S 0220E 026	$165.00	Gunnison
NANA-14	AZ105264927	14 0160S 0220E 026	$165.00	Gunnison
NANA-15	AZ105264928	14 0160S 0220E 026	$165.00	Gunnison
NANA-16	AZ105264929	14 0160S 0220E 026	$165.00	Gunnison
NANA-17	AZ105264930	14 0160S 0220E 026	$165.00	Gunnison
NANA-18	AZ105264931	14 0160S 0220E 026	$165.00	Gunnison
NANA-19	AZ105264932	14 0160S 0220E 026	$165.00	Gunnison

CLAIM NAME AND NUMBER	BLM Serial # (AMC #)	TOWNSHIP, RANGE, SECTION*	Maintenance Costs	Area
NANA-20	AZ105264933	14 0160S 0220E 026	$165.00	Gunnison
Alpha Omega #64	AMC 429559	14 0160S 0230E 004	$165.00	Gunnison
Alpha Omega #65	AMC 429560	14 0160S 0230E 004	$165.00	Gunnison
Alpha Omega #76	AMC 429561	14 0160S 0230E 004	$165.00	Gunnison
Alpha Omega # 77	AMC 429562	14 0160S 0230E 004	$165.00	Gunnison
GUNNY 1	AZ105789226	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 2	AZ105789227	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 3	AZ105789228	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 4	AZ105789229	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 5	AZ105789230	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 6	AZ105789231	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 7	AZ105789232	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 8	AZ105789233	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 9	AZ105789234	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 10	AZ105789235	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 11	AZ105789236	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 12	AZ105789237	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 13	AZ105789238	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 14	AZ105789239	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 15	AZ105789240	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 16	AZ105789241	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 17	AZ105789242	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 18	AZ105789243	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 19	AZ105789244	14 0160S 0230E 005	$165.00	Gunnison
GUNNY 20	AZ105789244	14 0160S 0230E 005	$165.00	Gunnison
ADDIE R	403667	14 0150S 0220E 023	$165.00	Johnson Camp
ALAMOSA	403668	14 0150S 0220E 023	$165.00	Johnson Camp
BEE R2	403669	14 0150S 0220E 024	$165.00	Johnson Camp
BEE R1	403670	14 0150S 0220E 024	$165.00	Johnson Camp
BEE R3	403671	14 0150S 0220E 024	$165.00	Johnson Camp
BEE R4	403672	14 0150S 0220E 024	$165.00	Johnson Camp
BEE R5	403673	14 0150S 0220E 024	$165.00	Johnson Camp
BEE R11	403674	14 0150S 0220E 024	$165.00	Johnson Camp
BEE R12	403675	14 0150S 0220E 024	$165.00	Johnson Camp
BONANZA	403676	14 0150S 0220E 022	$165.00	Johnson Camp
BUMBLE BEE	403677	14 0150S 0220E 023	$165.00	Johnson Camp
BURRO L	403678	14 0150S 0220E 035	$165.00	Johnson Camp
BURRO 4	403679	14 0150S 0220E 035	$165.00	Johnson Camp
BURRO 5	403680	14 0150S 0220E 035	$165.00	Johnson Camp
BURRO 6	403681	14 0150S 0220E 035	$165.00	Johnson Camp
BURRO 7	403682	14 0150S 0220E 035	$165.00	Johnson Camp
BURRO 8	403683	14 0150S 0220E 035	$165.00	Johnson Camp
BURRO NO 9	403684	14 0150S 0220E 035	$165.00	Johnson Camp
BURRO 19	403685	14 0150S 0220E 027	$165.00	Johnson Camp
CALUMET	403686	14 0150S 0220E 036	$165.00	Johnson Camp
CHARLES	403687	14 0150S 0220E 036	$165.00	Johnson Camp
CHELSIE FRACTION	403688	14 0150S 0220E 022	$165.00	Johnson Camp
COLORADO	403689	14 0150S 0220E 022	$165.00	Johnson Camp
DEFENDER	403690	14 0150S 0220E 022	$165.00	Johnson Camp
DORA	403691	14 0150S 0220E 036	$165.00	Johnson Camp

CLAIM NAME AND NUMBER	BLM Serial # (AMC #)	TOWNSHIP, RANGE, SECTION*	Maintenance Costs	Area
E-5 FRACTION	403692	14 0150S 0220E 013	$165.00	Johnson Camp
ECHO NO 1	403693	14 0150S 0220E 024	$165.00	Johnson Camp
ECHO R2	403694	14 0150S 0220E 024	$165.00	Johnson Camp
ECHO R3	403695	14 0150S 0220E 024	$165.00	Johnson Camp
ELEPHANT	403696	14 0150S 0220E 023	$165.00	Johnson Camp
ELLA	403697	14 0150S 0220E 036	$165.00	Johnson Camp
ELLENOR	403698	14 0150S 0220E 027	$165.00	Johnson Camp
ERICKA	403699	14 0150S 0220E 036	$165.00	Johnson Camp
ERNEST	403700	14 0150S 0220E 036	$165.00	Johnson Camp
EULA BELLE	403701	14 0150S 0220E 027	$165.00	Johnson Camp
GLADYS R	403702	14 0150S 0220E 036	$165.00	Johnson Camp
GUSTAVE	403703	14 0150S 0220E 036	$165.00	Johnson Camp
HAGERMAN	403704	14 0150S 0220E 036	$165.00	Johnson Camp
IMOGENE	403705	14 0150S 0220E 027	$165.00	Johnson Camp
INA	403706	14 0150S 0220E 036	$165.00	Johnson Camp
INDICATOR	403707	14 0150S 0220E 022	$165.00	Johnson Camp
KATIE	403708	14 0150S 0220E 022	$165.00	Johnson Camp
KENTUCKY	403709	14 0150S 0220E 023	$165.00	Johnson Camp
LAST CHANCE	403710	14 0150S 0220E 027	$165.00	Johnson Camp
LAURA J	403711	14 0150S 0220E 024	$165.00	Johnson Camp
LIME NO 1	403712	14 0150S 0220E 022	$165.00	Johnson Camp
LIME NO 2	403713	14 0150S 0220E 022	$165.00	Johnson Camp
LIME NO 3	403714	14 0150S 0220E 022	$165.00	Johnson Camp
LIME NO 4	403715	14 0150S 0220E 022	$165.00	Johnson Camp
LINDA SUE	403716	14 0150S 0220E 027	$165.00	Johnson Camp
LOUIE	403717	14 0150S 0220E 036	$165.00	Johnson Camp
MARY	403718	14 0150S 0220E 036	$165.00	Johnson Camp
MARY EILENE	403719	14 0150S 0220E 027	$165.00	Johnson Camp
MASON	403720	14 0150S 0220E 027	$165.00	Johnson Camp
MESCAL NO 5	403721	14 0150S 0220E 027	$165.00	Johnson Camp
MILLINGTON	403722	14 0150S 0220E 023	$165.00	Johnson Camp
MIRIAM	403723	14 0150S 0220E 022	$165.00	Johnson Camp
MOORE #1	403724	14 0150S 0220E 022	$165.00	Johnson Camp
MOORE #2	403725	14 0150S 0220E 022	$165.00	Johnson Camp
MOORE #3	403726	14 0150S 0220E 022	$165.00	Johnson Camp
NELDA LANE	403727	14 0150S 0220E 027	$165.00	Johnson Camp
PORTLAND	403728	14 0150S 0220E 023	$165.00	Johnson Camp
PRIMROSE	403729	14 0150S 0220E 023	$165.00	Johnson Camp
PRIMROSE BEE	403730	14 0150S 0220E 023	$165.00	Johnson Camp
PUZZLE NO 2	403731	14 0150S 0220E 022	$165.00	Johnson Camp
S-10	403732	14 0150S 0220E 023	$165.00	Johnson Camp
S-12	403733	14 0150S 0220E 023	$165.00	Johnson Camp
S-14	403734	14 0150S 0220E 023	$165.00	Johnson Camp
S-16	403735	14 0150S 0220E 023	$165.00	Johnson Camp
S-18	403736	14 0150S 0220E 023	$165.00	Johnson Camp
S-26	403737	14 0150S 0220E 024	$165.00	Johnson Camp
S-28	403738	14 0150S 0220E 024	$165.00	Johnson Camp
S-30	403739	14 0150S 0220E 024	$165.00	Johnson Camp
S-32	403740	14 0150S 0220E 024	$165.00	Johnson Camp
S-34	403741	14 0150S 0220E 024	$165.00	Johnson Camp

CLAIM NAME AND NUMBER	BLM Serial # (AMC #)	TOWNSHIP, RANGE, SECTION*	Maintenance Costs	Area
SHARIE LYNN	403742	14 0150S 0220E 027	$165.00	Johnson Camp
SHIRLEY LOUISE	403743	14 0150S 0220E 027	$165.00	Johnson Camp
ULTIMO	403744	14 0150S 0220E 036	$165.00	Johnson Camp
WOLFRIME	403745	14 0150S 0220E 036	$165.00	Johnson Camp
BRENDA KAYE	405106	14 0150S 0220E 027	$165.00	Johnson Camp
BURRO A	405107	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO B	405108	14 0150S 0220E 027	$165.00	Johnson Camp
BURRO 17	405121	14 0150S 0220E 027	$165.00	Johnson Camp
BURRO 18	405122	14 0150S 0220E 027	$165.00	Johnson Camp
BURRO 20	405123	14 0150S 0220E 027	$165.00	Johnson Camp
CHARLENE	405124	14 0150S 0220E 027	$165.00	Johnson Camp
FRANCINE	405126	14 0150S 0220E 027	$165.00	Johnson Camp
JANE RAE	405127	14 0150S 0220E 027	$165.00	Johnson Camp
BURRO C	408182	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO D	408183	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO E	408184	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO G	408185	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO H	408186	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO I	408187	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO 11	408188	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO 12	408189	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO 13	408190	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO 14	408191	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO 15	408192	14 0150S 0220E 026	$165.00	Johnson Camp
BURRO 16	408193	14 0150S 0220E 026	$165.00	Johnson Camp
CORNADO NO 1	408194	14 0150S 0220E 026	$165.00	Johnson Camp
ROSIE R	408195	14 0150S 0220E 026	$165.00	Johnson Camp
J SULLY #6	408909	14 0150S 0220E 036	$165.00	Johnson Camp
J SULLY #8	408911	14 0150S 0220E 036	$165.00	Johnson Camp
J SULLY #11	408914	14 0150S 0220E 036	$165.00	Johnson Camp
J SULLY #12	408915	14 0150S 0220E 036	$165.00	Johnson Camp
J SULLY #13	408916	14 0150S 0220E 036	$165.00	Johnson Camp
J SULLY #14	408917	14 0150S 0220E 036	$165.00	Johnson Camp
J SULLY #15	408918	14 0150S 0220E 036	$165.00	Johnson Camp
SULLY #16	408919	14 0150S 0220E 036	$165.00	Johnson Camp
ASHLEY	416211	14 0150S 0220E 024	$165.00	Johnson Camp
J-TRAVASSOS	416212	14 0150S 0220E 024	$165.00	Johnson Camp
N-TRAVASSOS	416213	14 0150S 0220E 024	$165.00	Johnson Camp
SUMMERTIME	416214	14 0150S 0220E 023	$165.00	Johnson Camp
SUNSET	416215	14 0150S 0220E 023	$165.00	Johnson Camp
T-ACKEN	416216	14 0150S 0220E 024	$165.00	Johnson Camp
WILDFIRE	416217	14 0150S 0220E 023	$165.00	Johnson Camp
GUNNY 21	AZ105799835	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 22	AZ105799836	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 23	AZ105799837	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 24	AZ105799838	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 25	AZ105799839	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 26	AZ105799840	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 27	AZ105799841	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 28	AZ105799842	14 0150S 0220E 027	$165.00	Johnson Camp

CLAIM NAME AND NUMBER	BLM Serial # (AMC #)	TOWNSHIP, RANGE, SECTION*	Maintenance Costs	Area
GUNNY 29	AZ105799843	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 30	AZ105799844	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 31	AZ105799845	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 32	AZ105799846	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 33	AZ105799847	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 34	AZ105799848	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 35	AZ105799849	14 0150S 0220E 027	$165.00	Johnson Camp
GUNNY 36	AZ105799850	14 0150S 0220E 027	$165.00	Johnson Camp
*Some claims may extend into adjacent Townships, Ranges or Sections				
			ANNUAL COST	TOTAL # OF CLAIMS
TOAL GUNNISON CLAIMS			$28,380.00	172
TOTAL JOHNSON CAMP CLAIMS			$22,275.00	133
GRAND TOTAL EXHIBIT A			$50,655.00	305

State Permits

Permit No.	1st Year	2nd Year	3rd Year	4th Year	5th Year
08-121919 **Sec. 7**	Rent: 911.46 App Fee: $500 Exp: $4,557.30 Due: 2/22/2022	Rent: None App Fee: $500 Exp: $4,557.30 Due: 2/22/2023	Rent: $455.73 App Fee: $500 Exp: $9,114.60 Due: 2/22/2024	Rent: $455.73 App Fee: $500 Exp: $9,114.60 Due: 2/22/2025	Rent: $455.73 App Fee: $500 Exp: $9,114.60 Due: 2/22/2026
08-121961 **Sect. 18**	Rent: 558.02 App. Fee: $500 Exp: $2,790.10 Due: 4/4/2022	Rent: None App. Fee: $500 Exp: $2,790.10 Due: 4/4/2023	Rent: $279.01 App. Fee: $500 Exp: $5,580.20 Due: 4/4/2024	Rent: $279.01 App. Fee: $500 Exp: $5,580.20 Due: 4/4/2025	Rent: $279.01 App. Fee: $500 Exp: $5,580.20 Due: 4/4/2026
08-121966 **Sect. 5**	Rent: 638.78 App. Fee: $500 Exp: $3,193.90 Due: 4/4/2022	Rent: None App. Fee: $500 Exp: $3,193.90 Due: 4/4/2023	Rent: $319.39 App. Fee: $500 Exp: $6,387.80 Due: 4/4/2024	Rent: $319.39 App. Fee: $500 Exp: $6,387.80 Due: 4/4/2025	Rent: $319.39 App. Fee: $500 Exp: $6,387.80 Due: 4/4/2026
08-121965 **Sect. 25**	Rent: 80.00 App. Fee: $500 Exp: $400.00 Due: 4/4/2022	Rent: None App. Fee: $500 Exp: $400.00 Due: 4/4/2023	Rent: $40.00 App. Fee: $500 Exp: $800.00 Due: 4/4/2024	Rent: $40.00 App. Fee: $500 Exp: $800.00 Due: 4/4/2025	Rent: $40.00 App. Fee: $500 Exp: $800.00 Due: 4/4/2026
08-122663 **Sect. 29**	Rent: 1280.00 App. Fee: $500 Exp: $6,400.00 Due: 11/2/22	Rent: None App. Fee: $500 Exp: $6,400.00 Due: 11/3/23	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/24	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/25	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/26
08-122662 **Sect. 8**	Rent: 1280.00 App. Fee: $500 Exp: $6,400.00 Due: 11/2/22	Rent: None App. Fee: $500 Exp: $6,400.00 Due: 11/3/23	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/24	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/25	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/26
08-122661 **Sect. 17**	Rent: 1280.00 App. Fee: $500 Exp: $6,400.00 Due: 11/2/22	Rent: None App. Fee: $500 Exp: $6,400.00 Due: 11/3/23	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/24	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/25	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/26
08-122660 **Sect. 32**	Rent: 1280.00 App. Fee: $500 Exp: $6,400.00 Due: 11/2/22	Rent: None App. Fee: $500 Exp: $6,400.00 Due: 11/3/23	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/24	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/25	Rent: $640.00 App. Fee: $500 Exp: $12,800.00 Due: 11/3/26
08-121733 **Sec. 20**	Rent: 1080.00 App Fee: $500 Exp: $5,400.00 Due: 11-2-2021	Rent: None App Fee: $500 Exp: $5,400.00 Due: 11-2-2022	Rent: $540 App Fee: $500 Exp: $10,800.00 Due: 11-2-2023	Rent: $540 App Fee: $500 Exp: $10,800.00 Due: 11-2-2024	Rent: $540 App Fee: $500 Exp: $10,800.00 Due: 11-2-2025
08-122253 **Sec. 2**	Rent: 996.04 App Fee: $500 Exp: $4,980.20 Due: 6/27/2022	Rent: None App Fee: $500 Exp: $4,980.20 Due: 6/27/2023	Rent: $498.02 App Fee: $500 Exp: $9,960.40 Due: 6/27/2024	Rent: $498.02 App Fee: $500 Exp: $9,960.40 Due: 6/27/2025	Rent: $498.02 App Fee: $500 Exp: $9,960.40 Due: 6/27/2026
08-122443 **Sec. 26**	Rent: 560.00 App Fee: $500 Exp: $2,800 Due: 8/27/2022	Rent: None App Fee: $500 Exp: $2,800 Due: 8/27/2023	Rent: $280.00 App Fee: $500 Exp: $5,600 Due: 8/27/2024	Rent: $280.00 App Fee: $500 Exp: $5,600 Due: 8/28/25	Rent: $280.00 App Fee: $500 Exp: $5,600 Due: 8/28/26

State Mineral Lease

Permit Number 11-53946 Sec. 36 Rent: $11,964.75 Minimum Royalty: $6,381.20 Due: June 16 each year

Lease expires 6-15-2034

Connie Johnson Deed

All mines and minerals in and under Section 31, Township 15 South, Range 23 East, Gila and Salt River Base and Meridian, containing *615.52* acres, more or less; together with the power to take all usual, necessary or convenient means for working, getting, laying up, dressing, making merchantable, and taking away the said mines and minerals, and also for the above purposes, or for any other purposes whatsoever, to make and repair tunnels and sewers, and to lay and repair pipes for conveying water to and from any manufactory or other building as reserved in that certain Warranty Deed from Hetty Wilson Johnson (formerly Hetty G. Wilson) and Conner Johnson, her husband, to Tom Adams and Lizzie E, Adams, husband and wife, dated May 19, 1943, and recorded at Book 136 Deeds of Real Estate, pages 123, 124 in the Office of the Cochise County, Arizona Recorder.

Fee Simple Land

The mineral rights and other interests in the following parcels located in Cochise County, Arizona, as more specifically described in Exhibit A to the Option:

Parcel A: The mineral estate only in approximately 39.06 acres of land in Section 19, T. 16 S., R. 23 E. and Sections 24 and 25, T. 16 S., R 22 E.

Parcel D: The property in approximately 14.24 acres of land in Section 19, T. 16 S., R. 23 E. and Section 25, T. 16 S., R 22 E.

Parcel E: The property in approximately 4.28 acres of land in Section 19, T. 16 S., R. 23 E.

Parcel F: The property in approximately 15.29 acres of land in Section 25, T. 16 S., R. 22 E. (save and excluding a 15-foot easement along the northern boundary of Parcels D and E)

JOHNSON CAMP FEE LANDS

The following parcels of fee land are all situated in Township 15 South, Range 22 East, G&SRB&M, Cochise County, Arizona

Parcel 1

Section 26: Lots 8, 9, 10, and 11 EXCEPT all coal and other minerals as reserved in the patent from the United States of America, containing 139.00 acres, more or less.

Parcel 2

Section 26: Those portions of the King and Wolfrime Queen patented lode mining claims lying within the Southeast Quarter (SE1/4) as shown on Mineral Survey No. 1800, U.S. Patent No. 40087, recorded in the records of Cochise County at Book 26, Deeds of Mines, Page 251, containing 1.00 acres, more or less.

Parcel 3

Section 24: Lot 16

Section 25: Lots 11, 13, 14, 16, 17, 18, 20, and 21 382681 v2 EXCEPT any portion of Section 25 lying in the Southeast Quarter of the Northwest Quarter and the Northeast Quarter of the Southwest Quarter of Section 25, Township 15 South, Range 22 East, G&SRB&M, conveyed by Special Warranty Deed dated January 26, 1987 from Cyprus Mines Corporation, Grantor, to David A. Rae, Grantee, recorded in the Cochise County records as Document No. 870102364. EXCEPT a right-of-way for ditches and canals constructed by the authority of the United States as reserved in the patent from the United States of America.
Containing 53.444 acres, more or less.

Parcel 4

Section 23: Lots 11, 12, 13, 15, and 16

Section 24: Lots 11, 12, and 13 EXCEPT any portion lying within the South Half of the Southeast Quarter of the Northwest Quarter (S1/2SE1/4NW1/4) and the East Half of the Southwest Quarter (E1/2SW1/4) of Section 24, Township 15 South, Range 22 East, G&SRB&M conveyed by Special Warranty Deed dated January 26, 1987 from Cyprus Mines Corporation, Grantor, to David A. Rae, Grantee, recorded in the Cochise County records as Document No. 870102364.

Section 25: Lot 12 EXCEPT any portion lying within the Southeast: Quarter of the Northwest Quarter (SE1/4NW1/4) and the Northeast Quarter of the Southwest Quarter (NE1/4SW1/4) of Section 25, Township 15 South, Range 22 East, G&SRB&M, conveyed by Special Warranty Deed dated January 26, 1987, from Cyprus Mines Corporation, Grantor, to David A. Rae, Grantee, recorded in the Cochise County records as Document No. 870102364.

Section 26: Lots 4, 14, 15, 16, 17, 18, and 19; Southwest Quarter of the Northwest Quarter (SW1/4NW1/4) EXCEPT a right-of-way for ditches and canals constructed by the authority of the United States as reserved in the patent from the United States of America. Containing 307.47 acres, more or less.

Section 25: Lot 15 consisting of 37.53 acres, more or less; and Lot 16 consisting of 38.26 acres, more or less; and Lot 19 consisting of 40 acres, more or less, subject to ownership of those portions of unpatented claims Gladys R and Erika that lie North of the Southern boundary of Lot 19; and

Those portions of Lots 20 and 21 that lie East of the survey line dated April 23, 1989 completed by H.W. Smith, Registered Land Surveyor; and Those portions of the Cochise Lode Claim and the United Fraction Lode Claim that lie East of the survey line dated April 23, 1989 completed by H.W. Smith, Registered Land Surveyor; and That portion of the Highland Mary Lode Claim lying East of the survey line dated April 23, 1989 completed by 382681 v2 H.W. Smith, Registered Land Surveyor. All described lands, in sum, containing 116.267 acres, more or less.