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Mineral Resource Estimate and Technical Report

Revision 0

Arizona Sonoran Copper Company, Inc.
Parks/Salyer
NI 43-101-Compliant Mineral Resource Estimate and Technical Report
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Stantec

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1.0 SUMMARY

1.1 Cautionary Statements

Certain information and statements contained in this section are "forward looking" in nature. Forward-looking statements include, but are not limited to, statements with respect to the economic and scoping-level parameters of the Project; Mineral Resource estimates; the cost and timing of any development of the Project; the proposed mine plan and mining methods; dilution and mining recoveries; processing method and rates and production rates; projected metallurgical recovery rates; infrastructure requirements; capital, operating and sustaining cost estimates; the projected life of mine (LOM) and other expected attributes of the Project; the net present value (NPV); capital; future metal prices; the Project location; the timing of the environmental assessment process; changes to the Project configuration that may be requested as a result of stakeholder or government input to the environmental assessment process; government regulations and permitting timelines; estimates of reclamation obligations; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this Report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this Report, the forward-looking statements in this Report are subject to the following assumptions.

- There being no significant disruptions affecting the development and operation of the Project.
- Any exchange rate assumptions being approximately consistent with the assumptions in the Report.
- The availability of certain consumables and services and the prices for power and other key supplies being approximately consistent with assumptions in the Report.
- Labor and materials costs being approximately consistent with assumptions in the Report.

Assumptions made in Mineral Resource and Preliminary Economic Analysis (PEA) estimates, including, but not limited to, geological interpretation, grades, metal price assumptions, metallurgical and mining recovery rates, geotechnical and hydrogeological assumptions, capital and operating cost estimates, and general marketing, political, business, and economic conditions.

1.2 Introduction

Arizona Sonoran Copper Company (Arizona Sonoran) is a North American-based mining company engaged in the exploration and development of the Cactus Project and the Merrill Properties, comprising the Parks/Salyer Project (collectively, the Project) located near Casa Grande, Arizona, USA.

Stantec Consulting Services Inc. (Stantec), in conjunction with Samuel Engineering, Inc. (Samuel Engineering), has prepared a technical report for Arizona Sonoran at their request on the mineral resource estimate of the Parks/Salyer Project. A Preliminary Economic Assessment (PEA) was previously completed on the Cactus Project and filed on SEDAR in a Technical Report entitled Arizona Sonoran Copper Company, Inc. (Arizona Sonoran). Cactus Project, Arizona, USA Preliminary Economic Assessment effective August 31, 2021 (the 2021 Cactus PEA) covering the mining, process, infrastructure design, capital cost, and operating cost of the Cactus Project. This report was prepared in accordance with the Canadian National Instrument 43-101 (NI 43 101) standards for reporting mineral properties. For certain chapters in this report, text and figures have been taken directly from the 2021 Cactus PEA. The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The effective date of the Cactus Resource is 01 March 2021, the Stockpile Resource has an effective date of 04 April 2021 and the inputs and assumptions used for economic assessment are valid as of as of those dates.

The 2021 Cactus PEA is preliminary in nature and is partly based on 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 assessment based on these Mineral Resources will be realized.

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

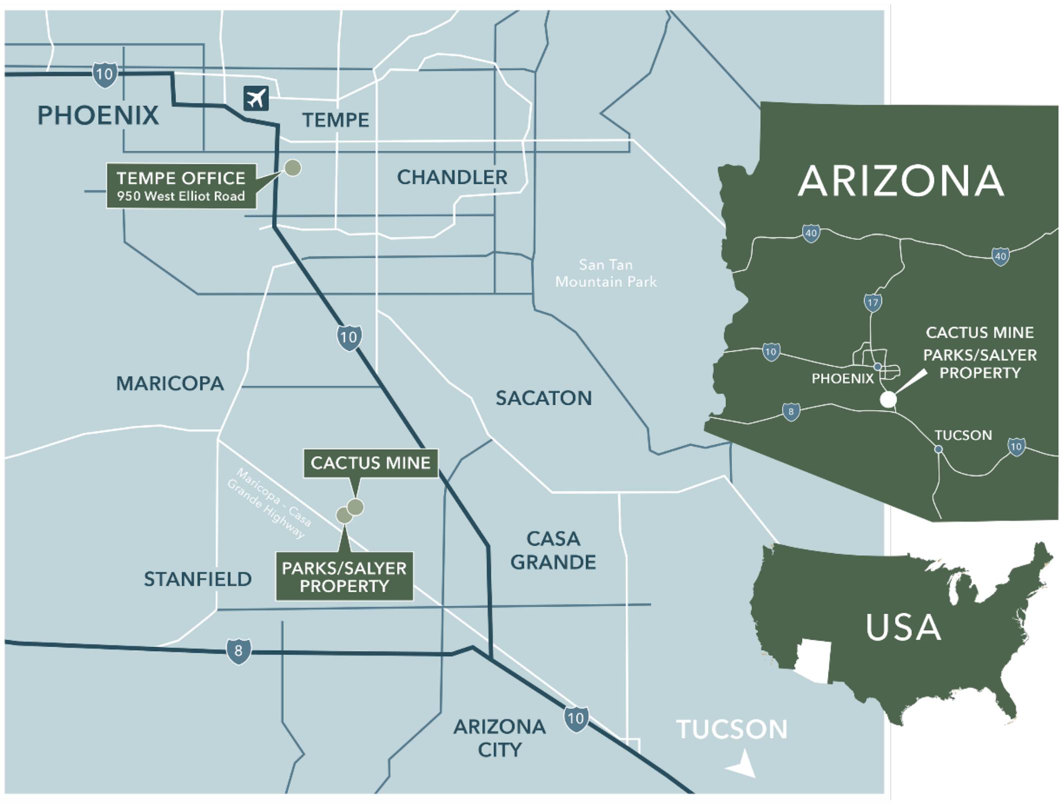
As required in NI 43-101, the effective date of the Parks/Salyer Resource is 26 September 2022; the date of this report is 10 November 2022.

1.3 Project Location and Description

The Project is located 40 road miles south southeast of the Greater Phoenix metropolitan area and approximately 3 miles northwest of the city of Casa Grande, Pinal County, Arizona.

The Project, located at the historic Sacaton Mine, is 10 miles due west of the Interstate 10 (I-10) freeway. Total site area is approximately 4,850 acres. Figure 1-1 shows the Cactus Project Location.

Figure 1-1: Cactus Project Location



In August 2019, Cactus110 LLC, a subsidiary of Arizona Sonoran, executed a purchase agreement (PA) and prospective purchaser's agreement (PPA) with a multi-state custodial trust and the Arizona Department of Environmental Quality (ADEQ), respectively, for the right to acquire all American Smelting and Refining Company (ASARCO) land parcels representing the Project, as well as all infrastructure therein, and all associated mineral rights.

In July 2020, Arizona Sonoran successfully closed on the property and acquired full title for the Project. In addition, Cactus 110 closed on the Merrill Properties, comprising the Parks/Salyer Project. Also in 2020, Arizona Sonoran acquired a prospecting permit for adjacent land owned by the Arizona State Lands Department.

In February 2021, Arizona Sonoran's wholly owned subsidiary Cactus 110 LLC executed an agreement with Arcus Copper Mountain Holdings LLC and several co-owners to purchase 750 acres of land also adjacent to the Project. Further, in May 2021, Arizona Sonoran's wholly owned subsidiary Cactus 110 LLC entered into an agreement with LKY / Copper Mountain Investments Limited Partnership LLP to purchase 1,000 acres of land adjacent to the Project referred to as the LKY Property. Additionally in February 2022, Arizona Sonoran entered into an agreement with Bronco Creek Exploration Inc. to transfer Bronco Creek Explorations Mineral Exploration Lease (MEP) with the Arizona State Lands Department to Arizona Sonoran. This MEP consists of 157.50 acres of State-owned surface and minerals. The Project comprises total landholdings of approximately 4,850 acres.

The privately-owned land assets represent, among other things, the mineral rights to the old Sacaton East, Sacaton West, and Parks/Salyer deposits. Arizona Sonoran Copper Company USA, Inc, is a subsidiary of Arizona Sonoran Copper Company, Inc, and intends to operate the mine under the name Cactus.

1.4 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The Project exists in relatively flat to slightly undulating ranching and mining locale. The Project is surrounded by other, current and past-producing, copper mines and processing facilities. The Greater Phoenix area is a major population center (approximately 4.5 million persons) with a major airport and transportation hub and well-developed infrastructure and services that support the mining industry. The cities of Casa Grande and Maricopa are nearby and, combined with Phoenix, can supply sufficient skilled labor for the Project.

Ecologically, the site is within the Sonoran Desert Section of the Basin and Range Lowlands Province of Arizona in the lower Santa Cruz Basin. The area is characterized by broad, level valley plains, gently sloping pediments, and widely separated mountain ranges. Elevations at the mine vary from approximately 1,360 ft amsl to 1,460 ft amsl. Soils have very low levels of available plant nutrients and vegetation on the property is typical of the Sonoran Desert and includes bunchgrasses, yucca, mesquite, and cacti.

Climate at the mine is typical of the Arizona Sonoran Desert, with temperatures ranging from 19 °F (-7 °C) to 117 °F (47 °C), and with average annual precipitation of 8.6-inch, falling primarily in high-intensity, short-duration events.

Electric power is available from Arizona Public Service's (APS) 115 kV transmission line which passes on the south side of the site and connects to an existing substation at the mine site and is owned by APS.

Arizona Sonoran, as part of the sale of the property, acquired the historic Type 2 Non-Irrigation grandfather rights (Certificate 58-100706.0005) for 136 acre-foot per year (afy). In addition to the grandfathered rights, Arizona Sonoran has obtained its permit from the Arizona Department of Water Resources (ADWR) (Permit 59-233782.0000) for an additional 3,600 afy under a Permit to Withdraw Groundwater for Mineral Extraction and Metallurgical Processing within an Active Management Area (A.R.S. § 45-514). This entitlement is expected to be sufficient for LOM as outlined in this PEA.

Situated within the Sonoran Desert the area is characterized by the Basin and Range Province of Intermontane Plateaus.

1.5 History

ASARCO geologists first discovered the Sacaton mineral deposit in the early 1960s while examining an outcrop of leached capping composed of granite cut by several thin monzonite porphyry dikes. The nature of this original find indicated the likely presence of porphyry copper-type mineralization. Following this lead, ASARCO initiated a drilling program which defined copper mineralization zones. The west zone contained the ore body which was ultimately accessed through the open pit. The deeper east zone was the target of potential mining by underground methods.

Project construction and mining of the west zone via open pit method commenced by 1972, and the mine operated continuously from 1974 until 1984. An underground copper deposit at Sacaton was under development until September 1981 when work was suspended because of high costs and a weak copper market. The Sacaton mine was permanently closed 31 March 1984 due to exhaustion of the open pit ore reserves.

The resultant Sacaton open pit mine is roughly circular, approximately 3,000 ft (914 m) in diameter and 1,040 ft (317 m) deep. The pit also has a visible internal lake with the surface at approximately 980 ft in depth from the pit rim. During operation, the Sacaton mine consisted of the pit, crushing facilities and coarse ore stockpile, a 9,000 tpd flotation mill, a tailings storage facility (TSF) that covered approximately 300 acres, a return water impoundment, an overburden dump, and a waste rock dump (WRD) that covered approximately 500 acres.

Production from the open pit was approximately 11,000 tpd. Copper flotation mill concentrate was sent by rail to the ASARCO smelter in El Paso, Texas. Over the mine's operating life 38.1 million tons of ore were mined and processed, recovering 400M lb of copper, 27,455 oz of gold, 759,000 oz of silver.

During mining of the open pit, a waste dump was created through dumping of defined waste material. All oxide copper mineralization, and sulfide copper mineralization below the working grade control cutoff of 0.3% copper (Cu), were deposited to the waste dump. The historic waste dump forms the basis of the Stockpile Project resource modelled in this PEA due to the level of mineralized material discarded.

1.6 Geologic Setting and Mineralization

The Cactus and Parks/Salyer Projects occur in the desert region of the Basin and Range province of Arizona. These combined deposits are part of a large porphyry copper system. Major host rocks are Precambrian Oracle Granite and Laramide monzonite porphyry and quartz monzonite porphyry. The porphyries intruded the older rocks and form mixed breccias; monolithic breccias and occur as large masses, poorly defined dike-like masses; and thin well-defined but discontinuous dikes. Structurally the deposit is complex with intense fracturing, faulting, and both pre-mineral and post-mineral brecciation. It is bounded on the east and west sides by normal faults.

Chalcocite and covellite are the only supergene sulfides recognized. The chalcocite blanket in the mineralized zone is irregular in thickness, grade, and continuity. The thickness of leached capping varies from less than 100 ft (30 m) to over 650 ft (198 m), with the thicker intercepts on the north side. Substantial quantities of oxidized copper minerals are found erratically distributed through the capping. Chrysocolla, brochantite, and malachite are the most common oxidized copper minerals. In upper portions of the capping, chrysocolla predominates, while brochantite and malachite predominate in the lower portions.

The dominant hypogene alteration assemblages in the deposit are phyllic and potassic. The major hypogene sulfide minerals in the deposit are pyrite, chalcopyrite, and molybdenite.

Hypogene sulfides occur as disseminated grains, veins, and vug fillings.

1.7 Deposit Types

The Cactus and Parks/Salyer deposits are portions of a large porphyry copper system that has been dismembered and displaced by Tertiary extensional faulting. Porphyry copper deposits form in areas of shallow magmatism within subduction-related tectonic environments (Berger et al., 2008). Cactus has typical characteristics of a porphyry copper deposit which Berger et al. (2008) define as follows.

- One wherein copper-bearing sulfides are localized in a network of fracture-controlled stockwork veinlets and as disseminated grains in the adjacent altered rock matrix.
- Alteration and mineralization at 1 km to 4 km depth are genetically related to magma reservoirs emplaced into the shallow crust (6 km to over 8 km), predominantly intermediate to silicic in composition, in magmatic arcs above subduction zones.
- Intrusive rock complexes that are emplaced immediately before porphyry deposit formation and that host the deposits are predominantly in the form of upright-vertical cylindrical stocks and/or complexes of dikes.
- Zones of phyllic-argillic and marginal propylitic alteration overlap or surround a potassic alteration assemblage.
- Copper may also be introduced during overprinting phyllic-argillic alteration events.

1.8 Exploration and Drilling

ASARCO geologists John Kinnison and Art Bloucher first identified the Sacaton mine area in early 1961 while doing regional mapping and sampling in and around the Sacaton Mountains. A lone outcrop of altered and weakly mineralized granite encompassed by alluvium was the only indicator of the potential for porphyry copper-type mineralization in the surrounding area. A six-hole drilling program was authorized and initiated in the fall of 1961. From 1962, through the first half of 1963, 82 additional holes were drilled. These 88 holes outlined a northeasterly trending alteration zone approximately 4 miles (6.4 km) long and 1.5 miles (2.4 km) wide dominated by what was recognized as two potential ore bodies, the Sacaton West and East deposits, as well as widespread intercepts of copper mineralization throughout. Low copper prices precluded any further exploration drilling at that time.

Improving market conditions prompted ASARCO to continue exploration drilling in 1968 and 1969 leading to 37 more holes being drilled. An additional 10 holes were drilled (1970 and 1971) to sterilize areas under planned facilities. After mining was initiated in 1972, development and definition drilling was conducted for the open pit (Cactus West deposit).

Through 1974 and 1976, eight additional holes were drilled in the Cactus East deposit for definition purposes.

In 2019, Arizona Sonoran drilled two vertical PQ core holes into the Cactus East mineralized zone for verification of grade and for metallurgical testing as part of the evaluation program prior to purchase. An additional vertical PQ core hole was drilled into Cactus East in 2020 for further metallurgical testing, for a total of 5,768 ft (1,758 m). Five angled HQ core holes totaling 9,252 ft (2,820 m) were drilled in late 2019 and 2020 around the northern and western edges of Cactus East to define and expand mineralization. Also, in 2020, 11 angled HQ core holes totaling 15,377 ft (4,687 m) were drilled around the perimeter of the West Pit to further define and expand Cactus West mineralization beyond the pit limits.

In 2019, 55 surface sonic drill holes totaling 5,120 ft (1,560 m) of 6-inch diameter holes were drilled across the Stockpile Project to support an initial resource based on approximately 750 ft (229 m) spaced drilling. Through late 2020 and early 2021, an infill surface sonic drill program was undertaken to reduce the spacing to 400 ft (122 m). The resource database for the Stockpile Project resource contains 210 holes. Drilling continues on the Project to ultimately reduce the spacing to 200 ft (61 m).

In late 2020, Arizona Sonoran successfully extended mineralization historically drilled at Parks/Salyer. Initially in 1996, two diamond drillholes totaling 3,753 ft (1,144 m) were drilled by ASARCO into the Parks/Salyer deposit, intercepting high grades of porphyry copper enrichment and primary sulphides. This drilling was a follow-up to previous drilling conducted to the south of Arizona Sonoran's property in which porphyry copper mineralization had been intersected and the characteristics indicated that the potential higher grades should be located to the north. In late 2020, Arizona Sonoran undertook two exploration holes totaling 4,573 ft (1,394 m) that continued to hit high grade mineralization 800 ft (244 m) further to the north. In late 2021, Arizona Sonoran began an exploration diamond drilling program over Parks/Salyer that through 2022 was expanded to cover the bulk of the interpreted deposit with 500 ft (152 m) spaced drilling. The total program covered 25 diamond drillholes for 56,303 ft (17,161 m).

1.9 Sample Preparation, Analysis, and Security

Arizona Sonoran has been exclusively using Skyline Assayers and Laboratories (Tucson, Arizona) for their sample prep and analysis. Upon arrival at the lab, totes were offloaded and stored. When the samples were ready to be processed, the bags were emptied into metal bins and the sample bags with tags placed on top. The bins and bags were placed in an oven at 220 °F (105 °C) for 24 hours to dry before moving into the lab for processing.

As a first pass each sample was assayed for CuT. To support potential heap leaching for metal recovery, a sequential acid leach assay procedure was conducted on each sample to return an acid soluble copper value (CuAS) and a cyanide soluble copper (CuCN) value. The remaining pulverized sample in the heavy paper envelope was returned to Arizona Sonoran together with the coarse reject.

Bagged samples with identification tags are placed in large 3 ft (1.0 m) square plastic totes which are stored at a core shed situated within the secured mine site away from any point of access until ready for transport. A transmittal sheet is prepared that lists all the samples in the shipment with an assay order sheet for the analysis to be done. A chain of custody sheet is signed by Arizona Sonoran upon dispatch, signed by Skyline Labs upon arrival, and returned to Arizona Sonoran to show secure delivery.

1.10 Mineral Processing and Metallurgical Testing

The metallurgical testing program for the Cactus mine resources is ongoing, with preliminary testing completed in some areas. Bottle roll and column testing data have been used along with typical recovery expectations for similar types of mineralization and run of mine (ROM) leaching operations and sequential assaying methodologies for mineralization variability. Recoveries employed are within typical values reported in industry.

Results obtained to June 2021 were used in the 2021 Cactus PEA as the basis for the metallurgical performance estimates. Averaged copper recovery and acid consumption estimates are presented in Table 1-1.

Table 1-1: Averaged Metallurgical Performance Criteria

Resource Component	Source Information	Net Copper Recovery (% - CuAS)	Net Copper Recovery (% - CuCN)	Gross Acid Consumption (lb/ton)	Net Acid Consumption (lb/ton)
Stockpile					
Oxide	Preliminary Column Tests	90%	40%	22	18
Open Pit & Underground					
Oxide	Preliminary Column Tests	90%	72%	22	18
Enriched	Preliminary Column Tests*	90%	72%	22	1

A 5% net recovery reduction adjustment has been included in the recovery estimates over the column extraction results to account for scale up and operational performance.

Oxide materials demonstrate a relatively rapid copper extraction potential, with copper extractions within two months achieved in column tests completed to date. A 3-month leach cycle has been considered for these materials. A one-year distribution of the recovery values used has been employed to account for heap inefficiencies, stacking planning and solution management activities. This will be refined with kinetic testing of the Stockpile Project and Cactus Project open pit materials.

Sulfide leaching completed to date indicates longer leaching cycles will be required. The materials will also be placed in a separate leach pad area that can be managed for bio-leaching kinetics and the longer cycle times required. A two-year distribution of the recovery values used has been employed to account for heap inefficiencies, stacking planning and solution management activities. This will be refined with kinetic testing of the Cactus Project open pit materials.

The initial sulfide columns are presently net acid producing due to the sulfide content and higher copper grades. This may be an advantageous feature once sulfide material is mined. For resource evaluations an experienced based long-term net acid consumption of approximately 1 lb per ton is considered as a conservative value for use in current economic evaluations until the column testing programs are completed.

1.11 Mineral Resource Estimate

The Cactus Project Resource Estimate including both the Cactus and Parks/Salyer deposits, was calculated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's (CIM's) Definitions Standards for Mineral Resources and Mineral Reserves. This represented the first Mineral Resource Estimate for the Cactus and Parks/Salyer Projects and updated Mineral Resource Estimate for the Cactus Stockpile Project. It includes the results of drilling programs undertaken by Arizona Sonoran between 2019 and 2022. The resource has been depleted of material mined in the Sacaton open pit in operation from 1974 through 1984. The estimate of the Mineral Resources supports both Indicated and Inferred Resources for Cactus, Inferred Resources from Parks/Salyer, and Inferred Resources for the Stockpile Project.

All data coordinates are presented in NAD83 ft. Zone 12 truncated to the last six whole digits for easting, and five whole digits for northing. All quantities are given in imperial units unless indicated otherwise. All copper values are presented in percent.

Cactus Project Mineral resources meeting the cutoff grades (COGs) for Cactus West and East, Parks/Salyer, and the stockpile are combined and reported in Table 1-2.

Table 1-2: Cactus Project Total Indicated and Inferred Resource

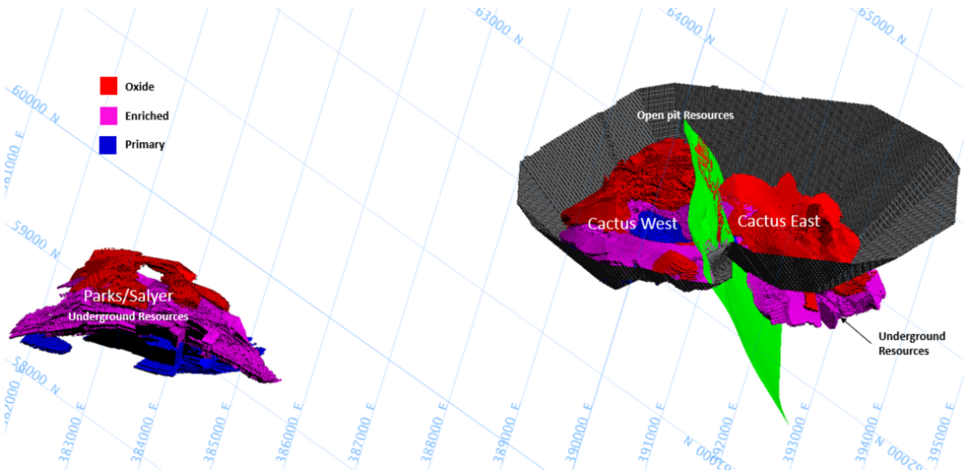
Material Type	Tons (kt)	CuT (%)	TSol (%)	Contained Copper (klb)
Indicated				
Total Leachable	73,900		0.723	1,065,200
Total Indicated	151,800	0.531		1,610,700
Inferred				
Total Leachable	310,400		0.590	3,663,700
Total Inferred	449,900	0.544		4,894,200

Notes:
1. CuT means total copper and TSolTSol means total soluble copper as the addition of sequential acid soluble and sequential cyanide soluble copper assays. Tons are reported as short tons.
2. Cactus East and West resources have an effective date of 01 March 2021, the Stockpile Resource have an effective date of 04 April 2021, and use a copper price of US$3.15/lb. The assumptions in respect of the Cactus and Stockpile Resource estimates are as stated in the PEA titled "Arizona Sonoran Copper Company, Inc. Cactus Project, Arizona, USA Preliminary Economic Assessment" with an effective date of August 31, 2021; Parks/Salyer Resource estimate has an effective date of 26 September 2022 and uses a copper price of US$3.75/lb.
3. Technical and economic parameters defining resource pit shell: mining cost US$2.45/t; G&A US$0.55/t, and 44°-46° pit slope angle.
4. Technical and economic parameters defining underground resource: mining cost US$28.93/t, and G&A representing 7% of direct costs.
5. Technical and economic parameters defining processing: Heap leach (HL) processing cost including selling US$1.77/t; HL recovery 83% of CuT; mill processing cost US$8.50/t.
6. For Cactus: Variable cutoff grades were reported depending on material type, potential mining method, and potential processing method. Oxide material within resource pit shell = 0.096% TSol; enriched material within resource pit shell = 0.098% TSol; primary material within resource pit shell = 0.205% CuT; oxide underground material outside resource pit shell = 0.56% TSol; enriched underground material outside resource pit shell = 0.70% TSol; primary underground material outside resource pit shell = 0.70% CuT.
7. For Parks/Salyer: Variable cut-off grades were reported depending on material type associated potential processing method. Oxide

underground material = 0.495% TSol; enriched underground material = 0.60% TSol; primary underground material = 0.586% CuT.
8.For the stockpile: There is a reasonable probability of eventual economic extraction of this resource using sulfuric acid leaching and SX/EW recover at a TSol cutoff of 0.095%
9. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, sociopolitical, marketing, or other relevant factors.
10. The quantity and grade of reported inferred mineral resources in this estimation are uncertain in nature and there is insufficient exploration to define these inferred mineral resources as an indicated or measured mineral resource; it is uncertain if further exploration will result in upgrading them to an indicated or measured classification.
11. Total may not add up due to rounding.

A graphical representation of the Oxide, Enriched and Primary material is shown Figure 1-2.

Figure 1-2: Total Material by Properties



1.11.1 Capping

Raw assay data was reviewed to determine if there were sufficient high grades in the various populations to require capping of the high grades during compositing. Histogram and log normal cumulative probability plots were reviewed for total copper (CuT) assays and total soluble copper (TSol) results in each of the mineral zones in the Cactus Project resource. A review of a log normal probability chart for CuT showed a good linear plot of values above the assay lab's detection levels. There is a visible minor break in linearity at 1.6 on the log normal scale, which transforms to 5% CuT. A review of a histogram plot of CuT values showed that 5% represents the high-end tail of the grades. A further review of a box plot of CuT grades, shows that 5% CuT does represent the high end of grades in the deposit. A capping grade of 5% CuT was chosen, with all grades above 5% set to 5% at time of compositing. This only affected two intervals in the dataset. The process was repeated for TSol, which identified 5% TSol as an appropriate capping grade. This affected 20 intervals in the diamond drill database.

For Parks/Salyer, top cutting was reviewed on a domain basis for both CuT and TSol and are presented in Table 14-7.

Table 1-3: Capping Levels for Parks/Salyer Estimation Domains

Domain	CuT Top Cut	Samples Cut	TSol Top Cut	Samples Cut
Leached	0.12	2	0.03	11
Oxide	1.71	11	1.67	9
Enriched	4.20	7	3.60	10
Primary	1.15	16	0.43	27

For the Stockpile Project, histogram and log normal cumulative probability plots were reviewed for CuT, CuAS, and CuCN assays. Cutoffs were defined within individual Stockpile Project lifts and ranged between 0.45% to 0.51% for CuT, 0.29% to 0.38% for CuAS, and 0.11% to 0.21% for CuCN.

1.11.2 Resource Cutoff Grades

To meet a reasonable expectation of eventual economic extraction (REEEE) requirement, as stated in CIM 2019 Best Practices, COGs were applied to both a potential open pit across the Cactus West deposit and a potential underground mine at depth in Cactus East.

Conceptually, copper from oxide and enriched material in the open pit would be recovered in a heap leach. Therefore, COGs in the amenable oxide and enriched zones were based on TSol assays. COGs for the sulfides in the primary material was based on CuT assays. High-level cost analysis for the open pit suggested COGs of 0.096% TSol for the oxides, and 0.098% TSol for the enriched material. A cutoff of 0.205% CuT was applied to primary material mined and therefore stockpiled for potential recovery in the future using a sulfide recovery process. Whittle open pit optimization software was ran using these parameters to define the ultimate pit shell for reporting of open pit resources.

Additional resources outside of the Whittle pit in Cactus East have the potential to be amenable to underground mining. High-level analysis of the material yielded cutoffs of 0.560% TSol for the oxides and 0.700% TSol for the enriched. The primary had a 0.700% cutoff applied to the CuT grade for potential recovery in a future sulfide recovery process.

Mineral resources for Parks/Salyer were also determined based on its amenability to underground mining. Due to the resources for Parks/Salyer having an effective date of 26 September 2022, a higher copper price of US$3.75/lb, was used in determining the cutoff grades for underground mining. High-level analysis of the material yielded cutoffs of 0.495% TSol for the oxides and 0.600% TSol for the enriched. The primary had a 0.586% cutoff applied to the CuT grade for potential recovery in a flotation mill.

The Stockpile Project resources were defined using a COG of 0.095% TSol.

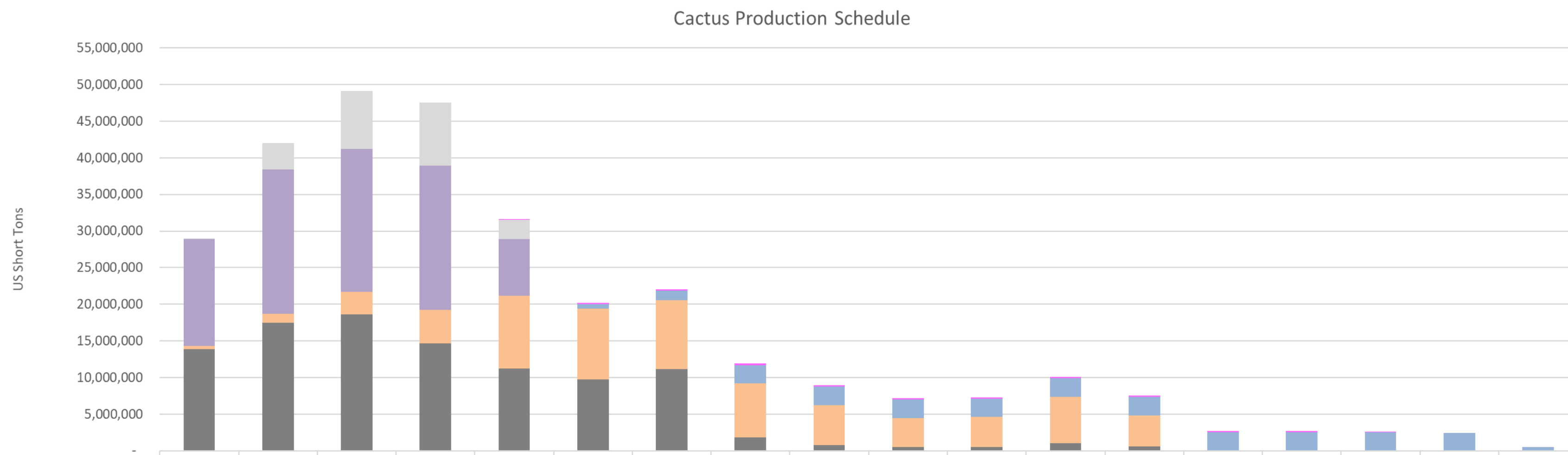
1.12 Mining Methods

The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as of 1 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report.

The Cactus Project considers mill feed originating from three sources: the existing surface Stockpile Project of previously mined material, an open pit operation – Cactus West, and an underground operation – Cactus East. To determine an appropriate mining approach, mine planning exercises were conducted consisting of combinations of processing and mining strategies. For the 2021 Cactus PEA, the outcome was to adopt a layered approach that considered initial Stockpile Project mining concurrent with Cactus West open pit stripping and early production for 1-4 years before Cactus West achieving steady state production by year 5. Once the pit reaches a suitable depth, development, and early production of Cactus East via a Transverse Longhole Stoping (TLS) method commences in year 6 and achieves steady-state production by year 8.

Complete extraction of the mineable resource is to take 17 years. The production profile for the LOM is provided in Figure 1-3.

Figure 1-3: Cactus Project Mine Plan



	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18
UG Waste	-	-	-	-	129,400	194,200	194,200	194,200	194,200	194,200	194,200	194,200	194,200	194,200	194,200	85,700		
UG Ore	-	-	-	-	-	630,005	1,266,316	2,520,043	2,519,981	2,519,962	2,519,997	2,519,997	2,519,979	2,519,996	2,520,027	2,519,956	2,418,515	529,226
External Stk - Waste	13,421	3,600,000	7,930,000	8,650,000	2,630,000	-	-	-	-	-	-	-	-	-	-	-	-	-
External Stk - Ore	14,560,000	19,730,000	19,530,000	19,690,000	7,731,940	-	-	-	-	-	-	-	-	-	-	-	-	-
Pit - Ore	442,000	1,189,000	3,113,000	4,540,000	9,982,000	9,662,000	9,455,000	7,353,000	5,464,000	3,987,000	4,094,000	6,305,000	4,276,000	-	-	-	-	-
Pit Waste	13,890,000	17,500,000	18,600,000	14,660,000	11,200,000	9,730,000	11,110,000	1,830,000	760,000	480,000	500,000	1,060,000	570,000	-	-	-	-	-

1.12.1 Stockpile Project and Open Pit

The Stockpile Project contains approximately 81.2 million tons of low-grade material and 22.8 million tons of waste. A similar but smaller sized equipment fleet will be used to move the low-grade material out of the Stockpile Project as that used for the open pit.

The open pit expansions will provide a total of approximately 71.8 million tons of mineralized material and 101.9 million tons of waste. Based on the planned production rate, the primary equipment fleet will consist of a fleet of rigid dump trucks in the 100-150 t range. Loading equipment will consist of at least two face digging units in the mine, assisted by a wheel loader. The sizing of these machines will be determined by the specifications of the haul truck fleet as well as the actual rock conditions.

The primary fleet will be complemented by a fleet of ancillary machines consisting of at least two track dozers, a road grader, a wheel dozer, a water truck and drill and blast equipment.

1.12.2 Underground

The remaining resource available in Cactus East was evaluated as an underground mine. The underground mining method used for this evaluation was a TLS mining method.

The top of the underground deposit, Cactus East, is roughly 800 ft (244 m) below the surface and extends an additional 1,000 ft (328 m) vertically. The deposit averages 800 ft (244 m) in thickness, from hanging wall to footwall.

To access the underground mine, twin declines will be developed from the wall of the new open pit. Due to the high daily production rate required, the declines will utilize one-way traffic to minimize traffic congestion. Preproduction development will excavate the twin declines down to the center of the deposit and split to opposite ends of the deposit.

A single ventilation raise will be driven at a central location off the footwall development. The mine will be split into two horizons with each horizon producing approximately 3,500 tpd for a total of 7,000 tpd at full production. Production of the first horizon will begin when the ventilation raise is established at surface.

1.13 Recovery Methods

The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on, or otherwise adversely affect, the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as of 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report.

A hydrometallurgical approach via a potential copper heap leaching and solvent extraction (SX) / electrowinning (EW) processing facility copper has been contemplated to process existing Stockpile Project oxidized copper resources and Cactus Project oxide and enriched sulfides (chalcocite / covellite dominant) material identified in the mineralized Cactus East and Cactus West extensions.

The integrated project has been designed to accommodate a 30,000 tons per day (tpd) permanent acid oxide heap leach, and permanent acid enriched heap leach. Material will be "as mined" from the new mining operations with no additional crushing or handling and stacked with mine trucks using an end dumping methodology. Table 1-4 shows the processing by source and material type.

Table 1-4: Processing by Material Type

Material	Source	Tons (Kt)
Oxidized	Stockpile Project	81,200
Cactus Project		
Oxide	Cactus West	46,800
	Cactus East	6,300
Enriched	Cactus West	23,100
	Cactus East	21,200
Totals		**178,600**

Table 1-5 shows the recovery assumptions achieved by bottle roll and column testwork by material type used for this PEA.

Table 1-5: Recovery Assumptions

Material	Source	Acid Soluble Recovery	Cyanide Soluble Recovery
Oxide	Stockpile Project	90%	40%
Oxide	Cactus West, East	90%	40%
Enriched	Cactus West, East	90%	72%

1.14 Environmental Studies, Permitting and Social Impact

Several documents were reviewed to provide an indication of the existing environmental conditions at the Cactus property near Casa Grande, Pinal County, Arizona.

Review of historical water quality data collected from 1972 through the present identified sulfates, nitrates, and fluoride exceedances over Arizona drinking water standards at various locations throughout the site.

No environmental fatal flaws that would materially impede the advancement of the project have been identified. Through due diligence research through the State of Arizona, the soil and groundwater at the site are contaminated with heavy metals. As such, the groundwater in the vicinity of the site is highly mineralized and contains elevated levels of arsenic, chromium, selenium, and zinc, and therefore is unfit for domestic, livestock, or agricultural use. The open pit from ASARCO's mining, contains water with high mineralization and a very low pH.

The Cactus property consists mostly of private surface and mineral rights, with the exception of two Arizona State Land Department Leases (ASLD) (Parcel No. 502-25-7020 Prospecting Permit No. 008-122116-00 and parcel number 503-26-7000 Prospecting Permit No. 008-121173-00-100). Permitting for an operation on private and ASLD lands will require the following major permits and certifications, already issued or in progress.

- Dust Permit Pinal Air Quality Control Permit (permit obtained).
- Arizona Pollutant Discharge Elimination System (AZPDES) permits (construction and Multi-Sector General Permit) (permit obtained for both the Mine Facility and the TruStone Facility). In Q2 of 2022, a new AZPDES was granted, this permit eliminated the TruStone Facility and incorporated that area into the new mine permit (LTF No. 95924, ID No. AZMS95924).
- ADWR Permit to Withdraw Groundwater for Mineral Extraction and Metallurgical Processing Permit No. 59-233782.0000. This permit allows Arizona Sonoran the rights to 3,600 afy for 50 years for Heap leach mining activities, dust control and processing at the Cactus Project site. The effective date of permit is 14 April 2021, and the expiration date of Permit is 14 April 2070.
- ADEQ Aquifer Protection Permit (APP) and Amended APP: Both APP applications have been accepted pending bond submittal.
- US Army Corp of Engineers (USACE) Approved Jurisdictional Determination (AJD) 404: On 11 February 2022 USACE issued the signed AJD for the site. US Army Corp of Engineers (USACE) Approved Jurisdictional Determination (AJD) 404: On 11 February 2022 USACE issued the signed AJD for the site.
- Pinal Air Quality Control Industrial Permit (applied for in October 2022).
- Arizona State Mine Inspector Mined Lands Reclamation Permit (applied for in October 2022).
- An estimate of $1.5 million will be required for the initial reclamation bond based on the initial construction plan and prior estimates for site closure for the Stockpile Project. An additional $3.5 million is estimated to be required to close the planned facilities and bonding will be adjusted as new facilities are added, particularly the Phase 2 leach pad. Closure funding is expected to be supplemented by resale of the modular SX/EW plant and other infrastructure and equipment, with an estimated salvage value of $5 million.
- Special Land Use Permit (SLUP) for use of State Surface to construct facilities for the mining operation: Application No. 023-123266-03-100 (Approved Contract signed and sent back to Arizona State Lands Department) Permit No. 23-123266-03.

In keeping with Arizona Sonoran's community engagement and partnership standards, the Cactus Project will be developed with a plan to establish and maintain the support of its host communities.

- Arizona Sonoran has commenced early-stage community outreach and is currently evaluating partnerships within the community. As the Project's permits will involve a public process and are based on the permit submission and review schedule, Arizona Sonoran plans to elevate outreach during the permitting process and throughout the life of the mine.

1.15 Economic Analysis, Key Operating, and Financial Parameters

The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on, or otherwise adversely affect, the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as of 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this Report.

There are no Mineral Reserves for the Project currently. The information reported in the 2021 Cactus PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Inferred Mineral Resources are based on limited geological evidence and sampling. The tonnage and grade of Inferred Mineral Resources have significant uncertainty as to their existence and as to whether they can be mined economically. There is no certainty that this PEA will be realized.

A discounted cash flow analysis was completed to evaluate the potential viability of the Mineral Resources at the Project.

The 2021 Cactus PEA highlights include the following.

- Life of Mine (LOM) average annual payable production of 28 ktpa LME Grade A copper cathode.
- An 18-year mine life based on current mine plan comprising leachable mineralized material only.
- Initial processing capacity of 22 ktpa with ramp up to 35 ktpa by Year 7 resulting in low initial construction CAPEX of $124 million.
- Low OPEX driven open pit mining in the initial phase from start of first production until commencement of underground mining anticipated in 6 years from first production.

- Average LOM cash cost (C1) (comprising mining, processing, applicable royalty, and direct general and administrative (G&A) costs of US$1.55/lb of copper produced. Average all-in sustaining costs (comprising of C1 costs and project sustaining capex) US$1.88/lb of copper produced.
- After tax, Project (NPV) (8%) and internal rate of return (IRR) of $312 million and 33% based on US$3.35/lb.
- Total inventory of 1.27 billion pounds of copper of a total leachable resource of 2 billion pounds providing significant upside opportunities for in-pit expansion.

Details of the assumptions and the outcome of the analysis are provided in Table 1-6.

Table 1-6: Financial Assumptions and Results

Assumption / Outcome	Value / Results
Copper Price	$3.35/lb
Total Mineralized Material Mined	179 Million Tons
Annual Average Processing Rate Over LOM	10 million tons per annum
Average Recovery Rates Over LOM	Stockpile Project: CuAS: 90%, CuCN: 40% OP/UG: CuAS: 90%, CuCN: 72%
Average Production Over LOM	28 ktpa
Operating Costs (per Ton Processed)	$9.06/ton
Average Cash Cost (C1) and All-In Sustaining Cost(C1 Cost+ Sustaining CAPEX)	C1: US$1.55/lb AISC: US$1.88/lb
Sustaining CAPEX Over LOM (OP and UG, SX-EW and Leach pad expansions)	$340 Million
LOM Free Cash Flow (FCF) (Post Tax Undiscounted)	$960 million
Post Tax NPV8	$312 million
Post Tax IRR	33%

The sensitivity of the economic outcome to copper price, operating costs and capital costs are provided in Table 1-7.

Table 1-7: Sensitivity Analysis

Total Project CAPEX	Project 8 NPV (Post Tax, $M) (US$3.35/lb Cu)	Post Tax IRR
15%	273.14	27%
10%	286.13	29%
0%	312.10	33%
-10%	338.07	37%
-15%	351.05	40%
LOM OPEX	**Project 8 NPV (Post Tax, $M) (US$3.35/lb Cu)**	**Post Tax IRR**
15%	239.40	28%
10%	263.64	30%
0%	312.10	33%
-10%	360.56	36%
-15%	384.79	37%

1.16 Key Opportunities and Risks

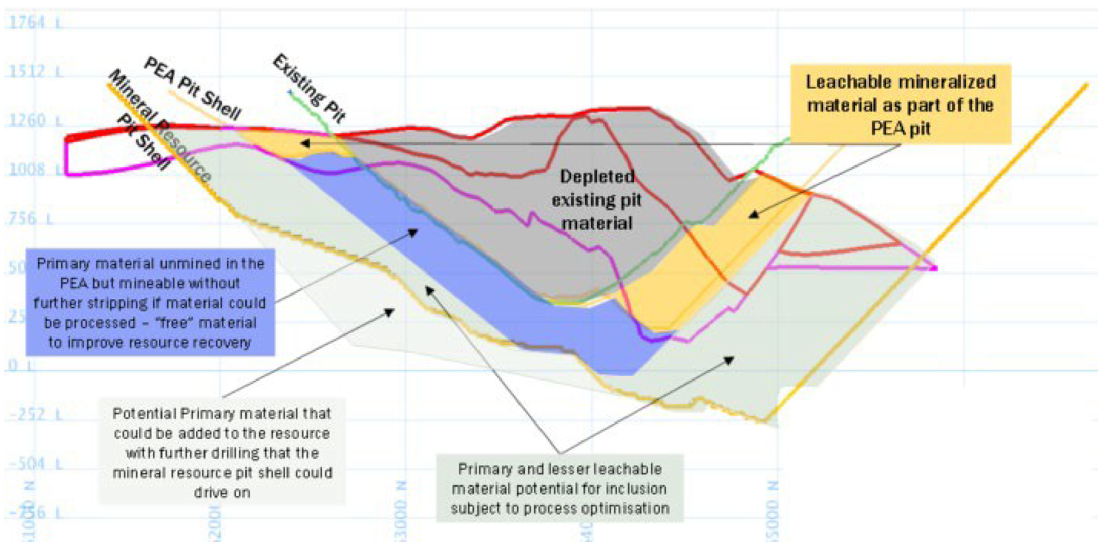
The following subsections outline key opportunities for the Project as previously outlined in the 2021 Cactus PEA and also reflect the Parks/Salyer mineral resource declared herein.

1.16.1 Resource Expansion

- In-Pit Potential: Based on the current level of exploration and planning, the Cactus West and East deposits comprise 2 billion pounds of leachable copper material.
- Only 1.27 billion pounds of leachable resource has been included within the 2021 Cactus PEA LOM, as the current pit mine plan has reached its natural limits for strip ratio due to increasing waste and decreasing grades on the periphery. Being able to process the primary material through sulfide leaching or flotation, which sits in the final pit floor, could add significant upside without additional waste stripping cost. This could result in depth expansion of the existing pit footprint, but also drive pit economics to support further pit expansions.
- Ex-Pit Potential: Based on the current level of exploration and planning, there is potential to add to the resource base through testing and conversion of material currently characterized as waste north of the Cactus West deposit.

- Parks/Salyer: There may be further potential to expand the Parks/Salyer mineral resource declared herein through further drilling. The mine trend between Parks/Salyer and Cactus West may hold potential for a down-dropped fault block of porphyry mineralization similar in nature to Cactus East.
- Figure 1-4 represents a cross-sectional view of the Cactus West pit. The green outline is the existing pit reflecting depletion. The 2021 Cactus PEA pit shell contains the leachable resource contemplated for that shell. The Mineral Resource pit shell captures all leachable and primary material as reflected in the Mineral Resource.

Figure 1-4: Cross Section Looking North Reflecting Depleted Material and Current Resource



1.16.2 Process Optimization

- Further metallurgical testing should be done to refine acid consumption and copper recoveries by source.
- Further metallurgical testing of sulfide recoveries could also demonstrate alternate process facilities thereby resulting substantial expansion of production rates.
- Upside from production of copper sulfate:
  - •Improved metallurgical performance (kinetics, acid consumption and copper recovery) and an alternative processing to an intermediate copper sulfate product are also potential opportunities to be pursued.
- A preliminary investigation was conducted regarding alternative processing routes. This included a site visit and discussion with the management team at a nearby processing facility in Arizona. The potential for producing copper sulfate and shipping 180 miles round trip was considered. Preliminary results indicate the following.
- Producing copper sulfate and sending it to existing unused Electro-Winning capacity at nearby facilities could provide savings in CAPEX of about $20 million to the Project by not building an EW circuit.

- A development schedule improvement of up to 3 months could be realized based on eliminating long lead items.
- Net Cactus site based OPEX savings, including shipping to nearby facilities, would be about $0.04/lb.
- Potential processing charges to recover external EW costs plus profit is assumed to be about $0.085/lb. No cost discussions have taken place.
- The likely net overall operating cost increase to Arizona Sonoran could be $0.05/lb ($1.32 million/y) against the $20 million in capital savings.
- Emerging technologies for improved leaching of sulfide copper ores are being developed. In particular, a proprietary catalytic bio-heap leaching technology may provide an alternative approach to improving the leach performance of primary sulfide content in the leach materials considered in this report and the primary sulfides presently not considered economically suitable for commercial heap leaching operation.
- The future potential for a copper concentrator for primary sulfide materials should also continue to be investigated.

1.16.3 Stockpile Project Sequencing

Significant opportunities exist to further enhance the Stockpile Project development in the areas of mining sequencing and heap leach feed grade distribution, Table 1-8 shows a preliminary mineral resource estimate for grade by lift in the Stockpile Project. Due to the uncertainty in possible low / no grade pockets within the lifts until more infill drilling is completed, an economic case has not been established at this time.

Table 1-8: Preliminary Estimate for Grade by Lift

Inferred Resources	Cu Sol Cutoff	Tons (million tons)	Cu Grade (%)				Pounds Cu (million pounds)			
			CuAS	CuCN	Cu Sol	TCu	CuAS	CuC N	Cu Sol	TCu
Lift 4	0.095	0.5	0.246	0.063	0.309	0.346	2.6	0.7	3.3	3.7
Lift 3	0.095	34.1	0.132	0.026	0.158	0.184	90.4	17.7	108.0	125.6
Lift 2	0.095	28.8	0.108	0.027	0.135	0.158	62.2	15.4	77.6	90.8
Lift 1	0.095	14.0	0.098	0.026	0.123	0.150	27.3	7.2	34.5	42.0
Total*	**0.095**	**77.4**	**0.118**	**0.026**	**0.144**	**0.169**	**182.5**	**40.9**	**223.5**	**262.2**

* Figures may not add up due to rounding

1.16.4 Project Schedule

Assuming permitting can be achieved as indicated, the overall project schedule could be brought forward 6-8 months by reducing the equipment delivery timeframes and commencing leach pad construction immediately upon receipt of permits. Any early execution or equipment purchase would be at the risk of project delays.

1.16.5 High Copper Commodity Environment

The Project development timeline driven by private land permitting is shorter, relative to other copper projects, which could see the Project developed in a higher copper price environment. For the purposes of LOM modelling, resources included in the open pit mine plan reflect an optimization run at a $2.27/lb copper price to present a robust initial mine plan, maximize grade inputs and consequently project value. There is significant room to expand the existing mineral inventory should US$+3.00/lb copper prices continue to prevail. There is potential room to expand the Integrated Cactus PEA inventory through improving strip ratios for certain areas adding approximately 10%-15% additional contained copper (resulting in +20 year mine life and increased production in the near term) and optimizing recovery methods for primary ore. Further trade-off studies in this context will also be pursued during the upcoming work programs.

Risks associated with the uncertainty of resource definition confidence in WRD, prevailing land issues, permitting processes and timing, and metallurgical testing are the most significant risks identified. The following is a description of the identified risks for this Project.

1.16.6 Stockpile Project Resource

- Unusual resource risks are associated with defining mineral content of waste rock facilities. Limited resource definition is available to be included in the estimates grade and tonnage made. Historic dump plans and information is not available for review and interpretation. Additional definition is required to ascertain a higher level of confidence in the resources included in this report. An average tons and grade approach have been used.
- As with resource definition, the ability to obtain truly representative samples from the Stockpile Project, or waste rock facility is somewhat compromised. An inherent risk exists as to representativeness of the samples tested to date or in future. Sequential assaying methodology provides a broader interpretation spatially with in the Stockpile Project related to recovery expectations.
- The potential for crushing larger materials may be required to achieve the recovery results projected and assessed against costs.
- Mitigation measures for the potential leach hydrodynamics may need to consider conveyor stacking to avoid surficial compaction and associated leach solution flow distribution and effectiveness.

1.16.7 Existing Litigation

- Ramm Power Group LLC (Ramm) had expressed interest in developing a pumped hydro renewable energy project at the site and had previously publicly announced that it would apply for a Federal Energy Regulatory Commission (FERC) license so that it could use FERC's eminent domain authority to acquire the property. The application was not contested and, consistent with its practice to issue preliminary permits to uncontested applications, by order of 19 July 2018, FERC granted the preliminary permit. The preliminary permit gives Ramm no rights in the site or rights to develop their project. The preliminary permit only initiates the longer permitting process. On 15 January 2020, Ramm began the formal licensing process by filing its Notice of Intent (NOI) and Pre-Application Document (PAD), together with a Letter Requesting Use of Traditional Licensing Process (TLP). The ASARCO Multi-State Environmental Trust, to which Arizona Sonoran is under contract to acquire the property from, Arizona Sonoran, and the Arizona Department of Environmental Quality all filed comments opposing Ramm's initiation of the licensing process. On 04 March 2020, FERC rejected Ramm's NOI and PAD as "patently deficient". FERC determined the pre-application document relied upon a single study conducted for the purpose of remediating a copper mine site, lacked agency or tribal consultation, and was therefore incomplete. FERC also cited the public comments received from ASCU that Ramm does not have rights to access the site to conduct the required studies.

- However, by 10 June 2020, ASCU was notified of a FERC application filed by REAggregators (REA) for a preliminary permit for Project No. 15010-000 to study the feasibility of developing an approximately 200 megawatt (MW) closed-loop, pumped-storage hydro project near Casa Grande in Pinal County, Arizona. Note that REA is a direct affiliation of Ramm. As portrayed in the Application, approximately 50-100 acres of the Project's site (Casa Grande Hydro Site) would overlap with land ASCU purchased in July 2020 from the ASARCO Multi-State Environmental Custodial Trust (the Trust). On 08 August 2020, ASCU filed their response with FERC, again outlining plans to develop a copper mine on the Mine Site (Cactus Project), further re-iterating that REA has no permission to access the property. The Casa Grande Hydro Site would encroach on the mine shaft of the Cactus Project materially impeding underground extraction activities. On 09 July 2021, Ramm requested a two-year extension of its preliminary permit. On 12 August 2021, FERC denied the request because Ramm filed the request after the deadline. FERC noted, however, that the rejection does not preclude Ramm from filing for an entirely new preliminary permit for the project. FERC typically only issues new preliminary permits to former permittees in extraordinary circumstances.

1.16.8 Permitting

Permitting for mining projects in the western US and Arizona 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. Although the Cactus Project is on private lands, these risks remain.

1.16.9 Geotechnical

Geotechnical risks associated with the Cactus Project, including the proposed heap leach pad locations, open pit and underground wall stability have not been fully assessed and will require extensive test work to confirm current work and assumptions.

1.16.10 Metallurgical Testing

The testing as outlined in this report is required to advance the level of confidence in leaching performance criteria such as recoveries, acid consumption, leach flow rates and hydrodynamic flow both for Cactus Project mineralization as well as the Stockpile Project.

As with resource definition, the ability to obtain truly representative samples from the waste rock facility is somewhat compromised. An inherent risk exists as to representativeness of the samples tested to date or in future. Sequential assaying methodology provides a broader interpretation spatially within the Stockpile Project related to recovery expectations.

Leach solution hydrodynamic performance risks in the heap leach pads due to excess fine materials, clays in intermixed alluvial materials and other factors are a risk to leaching metallurgical performance and heap stability. Testing and evaluation of these considerations should be conducted to confirm practical leaching parameters and reduce the potential risks.

The Park Salyer deposit requires metallurgical testing to verify heap leaching performance expectations for the materials defined. While the deposit is adjacent to the Cactus deposit resources, initial indications from geologic logging and physical observations indicate potentially significant mineralogical and geologic differences that could result in differing metallurgical performance from that of the Cactus deposit. Most notable is a more significant occurrence of both covellite and digenite copper minerals with a reduced percentage of oxide copper mineralization present in comparison to the Cactus deposit. These differences could manifest in leaching times to achieve expected copper recovery, net acid consumption requirements, and material handling properties.

A metallurgical drill core sampling program is in progress, twinning three known resource drill holes to provide sufficient materials for a comprehensive test work program for Park Salyer. Twinned holes target both the grade and mineralogical variability of the deposit as presently understood.

1.16.11 Tax Rates

The Project economics vary with the tax rate used in the evaluation. The all-in rate assumption of 24% is reasonable for this level of study, given that the depletion values have not been quantified. Should the full tax rate of 30.5% be applied to the project, the after-tax IRR reduces from 28% to 26%.

1.17 Conclusions and Recommendations

As set out in the 2021 Cactus PEA, the resource estimates established for both the Stockpile Project and Cactus Project combined with associated metallurgical testing appear adequate, with additional work warranted to continue to investigate the Project. The resource estimate for Parks/Salyer deposit further warrants additional drilling such that it can be included into an integrated technical study.

The primary goals of future work programs should be as follows:

- In-fill drill programs of the current resource volume to convert inferred material to indicated and measured resource categories.
- Continue to expand the current resource through additional, step-out drilling.
- Continue to explore the mineralized targets away from the deposit to evaluate the potential for additional deposits to add to the medium term expansion potential.
- Conduct additional metallurgical testing as outlined in this report.
- Complete an integrated technical report/prefeasibility study (PFS) of the project based on the positive outcome of the Cactus PEA and the Parks/Salyer Mineral Resource.

The QPs to this report recommend the completion of a PFS to advance the development of the Project. As set out in the 2021 Cactus PEA recommendations for further work study programs have been divided into two Phases to better define the goals and objectives and assist in planning and budgeting the work.

Table 1-9 captures all Phase 1 costs required to complete a PFS for the Cactus deposit, whereas Table 1-10 reflects the additional Phase 2 costs to bring the Project to a definitive feasibility study (DFS), including final detailed engineering and initial exploration drilling on Parks/Salyer and NE Extension. The budget has been estimated for project expenditures commencing in Q4 2021 for the next two phases of the work program. The results of the Lab Testing, particularly Metallurgical, will form the basis to proceed the study to a DFS. The results of additional drilling will be required prior to a scoping level evaluation of the economics for Parks/Salyer and are not included in the costs below.

Table 1-9: Phase 1 Prefeasibility Study Costs

Budget Category	Estimate Cost (US$ 000)	
	Q3 2021	Q4 2021
Drilling	2,782	1,232
Project Support	396	276
Technical Studies	750	750
Lab testing (Assaying and Metallurgical)	493	198
Permitting	59	80
Land Payments	7,000	
Exploration - Adjacent Properties		
Total	**11,479**	**2,535**

Table 1-10: Phase 2, Definitive Feasibility Study Costs

Budget Category	Estimate Cost (US$ 000)
Drilling	3,128
Project Support	750
Technical Studies	652
FEED Engineering	800
Lab Testing (Assaying and Metallurgical)	398
Permitting	124
Land Payments	7,900
Exploration - Adjacent Properties	2,916
Total	**16,669**

The following tasks should be undertaken as part of Phase 1, PFS work program as outlined in the 2021 Cactus PEA.

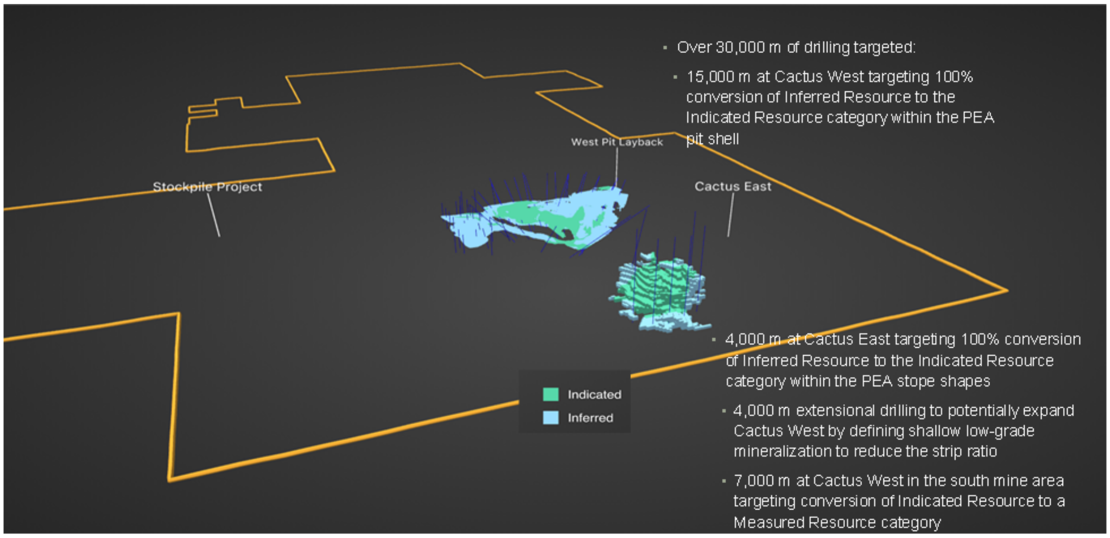
- Sustainability
  - Continue permitting activities and land acquisition as planned.
  - While adequate for this PEA, further hydrogeologic study is required to better quantify aquifer levels and impacts from mining.
- Geotechnical
  - Develop geotechnical information required for engineering design.
    - For example, the proposed pillar between open pit high wall and underground stopes is fairly represented in the PEA but needs geotechnical verification once additional data becomes available.

- Drilling
- The present Cactus West and East deposit outlines appear to be drill limited to the north and east. Continued step out drilling in these areas could very well extend the limits of known mineralization.
- Continue metallurgical sample drilling across the Project area.
- Condemnation/step-out drilling to be completed to confirm the placement of dumps, leach pads and plant facilities.
- If the decision is made to go underground at the Cactus East, plans should be made to have a close spaced definition drilling program to provide a more detailed understanding of mineralized material zone boundaries for stope design purposes.
- Lab Testing
  - Significant additional column testing, particularly large columns, recovery by size fraction to determine merits of crushing / agglomeration and importance of isolating oxides and sulfides from open pit, leaching characteristics of mixed oxides and sulfides will be required.
  - Reduce the number of calculated soluble grades in the model through assaying of historical pulps (currently 30% of composites use calculated CuAS and CuCN grades based on CuT grades and mineralization domains).
- Mine Design
  - Regarding the Cactus East underground
    - While current plans do not expect Cactus East to be operated as an in situ leach operation, this proposed leaching method should be considered further with the existing core and resource information. In Situ leach may be an alternative to underground mining in a low copper price environment, thereby still realizing high value material.
  - The proposed TLS mining method is suited for the deposit and the primary/secondary sequence with access from sublevels at 75 ft (23 m) spacing is logical. An economic trade-off study that envisions Avoca style TLS should be commissioned. With the relatively wide dimensions of the mineralized zones, additional opposite side access to set up Avoca mining (continuous mining and backfilling) may prove to add enough additional productivity gains to offset the additional development costs. If the timing of the open pit layback schedule is not conducive to commence portal excavation in a timely manner, then access from the surface, which lengthens the development declines, should be considered.
  - Additional detail related to pit layback design as well as the design of a bench portal platform is needed for the next stage of study. Sufficient working space for the portal and associated facilities must be designed such that underground / open pit equipment interactions are limited.
- Costs and Schedule
  - The mining costs seem reasonable and sufficient for a PEA-level evaluation but will need a higher level of detail and productivity analysis in the next stage. This will include a total buildup of equipment, personnel, and materials.

- A more detailed production and development schedule is required to verify the mines' ability to achieve the mining schedules presented for the Stockpile Project and Cactus Project.

A graphical representation of the drill plan is as provided in Figure 1-5.

Figure 1-5: Cactus Drill Plan



2.0 INTRODUCTION

2.1 Issuer and Purpose of Report

Arizona Sonoran is a North American-based mining company engaged in the exploration and development of the Cactus Project (the Project) located near Casa Grande, Arizona.

Stantec, in conjunction with Samuel Engineering, has prepared a technical report for Arizona Sonoran at their request on the mineral resource estimate of the Parks/Salyer Project. PEA was previously completed on the Cactus Project and filed on SEDAR in a Technical Report entitled Arizona Sonoran Copper Company, Inc. Cactus Project, Arizona, USA Preliminary Economic Assessment effective 31 August 2021 (the 2021 Cactus PEA) covering the mining, process, infrastructure design, capital cost, and operating cost of the Cactus Project. This report was prepared in accordance with the Canadian NI 43-101 standards for reporting mineral properties. For certain chapters in this report, text and figures have been taken directly from the 2021 Cactus PEA. The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021.

The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report.

2.2 Sources of Data

The following sources of information and data were used in preparing this report.

- Personal inspections of the Cactus project site and surrounding area.
- Technical information provided by Arizona Sonoran.
- Technical and cost information provided by Stantec, Samuel Engineering, Minerals Advisory Group (MAG), and Arizona Sonoran.
- Information provided by other experts with specific knowledge and expertise in their fields as described in Section 3.0 – Reliance on Other Experts.
- Additional information obtained from public domain sources.
- Additional reports relevant to the study are listed in Section 27.0 – References.

2.3 Qualified Persons

The qualified persons (QPs) responsible for this report and the dates of their visits to the Cactus project site and surrounding area are as presented in Table 2-1. Signed consent forms for these individuals can be found in Section 28.0.

Table 2-1: Qualified Persons Responsibilities

QP Name	Certification	Company	Dates of Site Visit	Section Responsibility
Allan Schappert	CPG, Reg. Member SME	Stantec	Cactus Project 13 Aug 19 03 Oct 19 03 Mar 20 25 Aug 20 20 Jan 21 21 Apr 22 19 Oct 22 Skyline Labs 27 Aug 19 03 Oct 19 03 Mar 20 02 Apr 21	Author 7,8, 9, 10, 11, 12, 14 Co-author 1, 25, 26
Jason Sexauer	P.Eng, PE	Stantec	03 Mar 20 21 Apr 22	Author 2, 3, 4, 5, 6, 15, 18, 19, 20, 21, 22, 23, 24 Co-author 1, 16, 25, 26
Dr. Martin Kuhn	PE, Reg. Member SME	Mineral Advisory Group	25 Feb 21 25 Mar 21	Author 13, 17 Co-author 25, 26
Wilhelm Greuer	PE	Stantec	03 Mar 20 07 Jun 22	Co-author 16

2.4 Units, Currencies, and Abbreviations

All currency amounts, costs, and commodity prices are stated in US dollars.

Quantities are stated in Imperial units. Where applicable, any System International (SI) units of measure have been converted to Imperial for reporting consistency, with the metric equivalent provided in parentheses following it.

Base metal grades are expressed as a percentage (%).

Table 2-2 provides a list of the units and abbreviations used throughout this report.

Table 2-2: List of Abbreviations

Acronym	Definition
% Cu	percent copper
AA	atomic absorption
AARL	American Analytical Research Laboratories
ADEQ	Arizona Department of Environmental Quality
ADWR	Arizona Department of Water Resources
afy	acre-foot per year
AMA	Active Management Area
APP	Aquifer Protection Permit
APS	Arizona Public Service
ASARCO	American Smelting and Refining Company
asp	alumino-phospho-sulfates
AZPDES	Arizona Pollutant Discharge Elimination System
BADCT	Best Available Demonstrated Current Technology
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
COG	Cutoff Grade
CPG	Certified Professional Geologist
CPVC	chlorinated polyvinyl chloride
CRF	cemented rockfill
Cu	copper
CuAS	acid soluble copper
CuCN	cyanide-soluble copper
CuCN-Seq	cyanide soluble copper grade
CuT	total copper assay
DFS	Definitive Feasibility Study
DGPS	differential global positioning system
EDA	exploratory data analysis
Arizona Sonoran	Arizona Sonoran Copper Company, Inc.
EW	electrowinning
FAA	Federal Aviation Administration
FAR	Federal Aviation Regulations
FCF	free cash flow
FERC	Federal Energy Regulatory Commission

Acronym	Definition
FRP	fiberglass reinforced plastic
FS	feasibility study
G&A	general and administrative
GRWS	Gila River Water Storage, LLC
GMS	groundwater modeling system
Grupo Mexico	Grupo Mexico S.A. De C.V.
GRWS	Gila River Water Storage
H2SO4	sulfuric acid
HCl	hydrochloric acid
$HClO_4$	percloric acid
HDPE	high density polyethylene
HL	high level
HNO_3	nitric acid
HR	hydraulic radii
I.D.	inner diameter
ID3	inverse distance
IP	induced polarization
IRR	internal rate of return
lb/t	pound per ton
LOM	life of mine
M&A	Montgomery and Associates
MAE	mean absolute error
masl	meter above sea level
METALEX	Metalex Technologies
MAG	Minerals Advisory Group
MRDS	mineral resource data system
NOI	notice of intent
NOITL	notice of intent to locate
NPV	net present value
OK	Ordinary Kriging
OP	open pit
PA	purchase agreement
PAD	pre-approved document

Acronym	Definition
PEA	preliminary economic assessment
PFS	prefeasibility study
PLC	Process Mineralogical Testing Ltd.
PLS	pregnant leach solution
PPA	prospective purchasers agreement
QA	Quality Assurance
QC	Quality Control
QP	qualified person
Ramm	Ramm Power Group LLC
REA	REAggregators
REEEE	reasonable expectation of eventual economic extraction
ROC	rapid ore characterization
ROM	run of mine
RQD	rock quality designation
SCSE	SAG Circuit Specific Energy
SecCuS	secondary copper sulfides
SIP	site improvement plan
SMCT	SMC Testing Pty Ltd
NaCN	Sodium Cyanide
SOP	standard operating procedure
Stantec	Stantec Consulting Services Inc.
SX	solvent extraction
t	ton
TECu	total economic copper
TDS	total dissolved solids
TLP	traditional licensing process
TLS	transverse longhole stoping
tpa	tons per annum
tpd	tons per day
TSF	tailings storage facility
TSol	total soluble
UCS	unconfined compressive strength
URF	unconsolidated rockfill

Acronym	Definition
UTM	Universal Transverse Mercator
WRD	waste rock dump
XTF	external tool force

3.0 RELIANCE ON OTHER EXPERTS

This technical report relies on information and conclusions from legal and technical experts who are not QPs as defined by NI 43-101. The QPs responsible for the preparation of this report have reviewed the information and conclusions provided and have determined that they conform to industry standards, are professionally sound, and are acceptable for use in this report.

The information, conclusions, opinions, and estimates contained herein are based on the following.

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

The QPs, while taking full responsibility for the contents of the report, recognize the support of the following.

- Arizona Sonoran's staff in Arizona, including Ian McMullan, COO, Doug Bowden, VP Exploration, and Travis Snider, VP Sustainability and External Relations for providing the data used throughout this report in all sections.
- Anthony Bottrill of InterGEO Resource Consulting Pty Ltd. for his contributions in section 14.0 providing input into the geological models for the Cactus Project and Stockpile Project resource.
- Stantec's team, including Christiaan Terblanche, David Vatterrodt, Diana Trieu, and Layne Kaufmann for inputs into section 16.0.
- Jim Sorensen and his associates at Samuel Engineering, for their work on the metallurgical testing and recovery calculations used for sections 13.0 and 17.0.
- None of the authors of this report has researched or verified property title or mineral or land access rights for the Arizona Sonoran property and the authors of this report express no opinion as to the legal status of property ownership and rights as disclosed in Section 4.0 of this report.

4.0 PROPERTY DESCRIPTION AND LOCATION

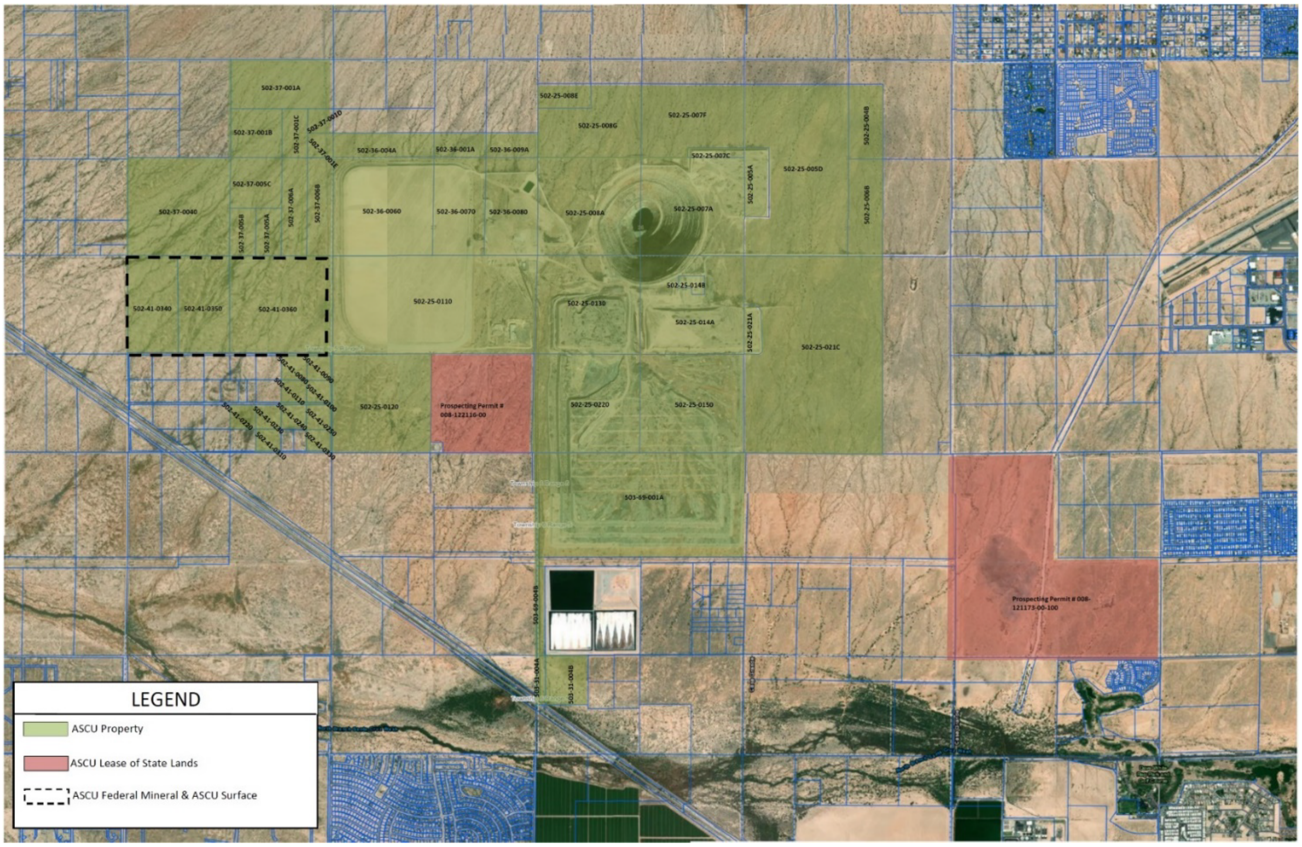
In 2019, Cactus110 LLC, a subsidiary of Arizona Sonoran executed both PA and PPA with a Multi-State Custodial Trust and the ADEQ, respectively, for the right to acquire all ASARCO land parcels representing the Project, as well as all infrastructure therein, and all associated mineral rights. In June of 2020, Arizona Sonoran successfully closed on the property and acquired full title for the Project. In addition, Cactus 110 LLC closed on the Merrill Properties comprising the Parks/Salyer Project. Also, in 2020, Arizona Sonoran acquired a prospecting permit for adjacent land owned by the Arizona State Lands Department. All lands are shown in Figure 4-1.

On 02 February 2021, Arizona Sonoran's wholly owned subsidiary Cactus 110 LLC executed an agreement with Arcus Copper Mountain Holdings LLC and several co-owners to purchase 750 acres of land also adjacent to the Project. Further, on 20 May 2021, Arizona Sonoran's wholly owned subsidiary Cactus 110 LLC entered into an agreement with LKY / Copper Mountain Investments Limited Partnership LLP to purchase 1,000 acres of land adjacent to the Project referred to as the LKY Property. Additionally in February 2022, Arizona Sonoran entered into an agreement to transfer Bronco Creek Explorations Mineral Exploration Lease (MEP) with the Arizona State Lands Department to Arizona Sonoran. This MEP consists of 157.50 acres of State-owned surface and minerals. The Project comprises total landholdings of approximately 4,850 acres. A summary of the current landholdings is as provided in Figure 4-1.

These private land assets represent, among other things, the mineral rights to the old Sacaton East, Sacaton West and Parks/Salyer deposits. Arizona Sonoran Copper Company USA, Inc, a subsidiary of Arizona Sonoran, intends to operate the mine under the name Cactus.

A 3.18% royalty is assumed to be applicable to the Cactus Project for the purposes of the Integrated Cactus PEA based on current contractual arrangements. In addition to the royalties granted by ASCU USA, the Cactus Project is also subject to existing 5% NSR on the SW1/4, W1/2SE1/4, and E1/2SE1/4 of Section 27 and the SW1/4, Township 5 South, Range 5 East, which are outside the areas contemplated by the mine plan in the Integrated Cactus PEA.

Figure 4-1: 2022 Land Ownership Position Including Federal Mineral Release Areas



4.1 Legal Description of Real Property

The property and rights owned by Arizona Sonoran are described in Table 4-1. These rights and title have not been independently verified and the title documents have been relied upon by the QP for this section of the report.

Table 4-1: Property and Rights

Owner	Parcel No.	Property Description	Township	Range	Section	Acres
Cactus 110 LLC						
CACTUS 110 LLC	503-31-004B	NWNW LESS WEST 215 FEET OF SEC 10, 6S-5E	6 South	5 East	10	33.5
CACTUS 110 LLC	502-36-004A	S1/2S1/2NW OF SEC 27, 5S-5E	5 South	5 East	27	40
CACTUS 110 LLC	502-36-001A	S1/2S1/2W1/2NE OF SEC 27, 5S-5E	5 South	5 East	27	20
CACTUS 110 LLC	502-36-009A	S1/2S1/2E1/2NE OF SEC 27, 5S-5E	5 South	5 East	27	20

Owner	Parcel No.	Property Description	Township	Range	Section	Acres
CACTUS 110 LLC	502-37-001E	SESENE OF SEC 28, 5S-5E	5 South	5 East	28	10
CACTUS 110 LLC	502-37-006B	E1/2E1/2SE OF SEC 28, 5S-5E	5 South	5 East	28	40
CACTUS 110 LLC	502-41-0080	LOT 7 OF SEC 33, 5S-5E	5 South	5 East	33	10
CACTUS 110 LLC	502-41-0090	LOT 8 OF SEC 33, 5S-5E	5 South	5 East	33	10
CACTUS 110 LLC	502-41-0100	LOT 9 OF SEC 33, 5S-5E	5 South	5 East	33	10
CACTUS 110 LLC	502-41-0110	LOT 10 OF SEC 33, 5S-5E	5 South	5 East	33	10
CACTUS 110 LLC	502-41-0220	LOT 21 OF SEC 33, 5S-5E	5 South	5 East	33	10
CACTUS 110 LLC	502-41-0230	LOT 22 OF SEC 33, 5S-5E	5 South	5 East	33	10
CACTUS 110 LLC	502-41-0240	LOT 23 OF SEC 33, 5S-5E	5 South	5 East	33	10
CACTUS 110 LLC	502-41-0250	LOT 24 OF SEC 33, 5S-5E	5 South	5 East	33	10
CACTUS 110 LLC	502-41-0310	LOT 30 OF SEC 33, 5S-5E	5 South	5 East	33	10
CACTUS 110 LLC	502-41-0330	LOT 32 OF SEC 33, 5S-5E	5 South	5 East	33	10
CACTUS 110 LLC	502-25-0120	SW OF SEC 34-5S-5E	5 South	5 East	34	160
CACTUS 110 LLC	503-69-004B	WEST 215 FET OF SW OF SEC 3-5S-5E	5 South	5 East	3	10
CACTUS 110 LLC	503-31-004A	WEST 215 FET OF NWNW OF SEC 10-5S-5E	5 South	5 East	10	6.5
CACTUS 110 LLC	502-36-0060	SW OF SEC 27-5S-5E	5 South	5 East	27	160
CACTUS 110 LLC	502-36-0070	W1/2SE OF SEC 27-5S-5E	5 South	5 East	27	80
CACTUS 110 LLC	502-36-0080	E1/2SE OF SEC 27-5S-5E	5 South	5 East	27	80
CACTUS 110 LLC	502-25-008A	SW OF SEC 26-5S-5E	5 South	5 East	26	160
CACTUS 110 LLC	502-25-007A	SE OF SEC 26-5S-5E	5 South	5 East	26	160

Owner	Parcel No.	Property Description	Township	Range	Section	Acres
CACTUS 110 LLC	502-25-007C	S-265.72 OF E-1450 OF NE OF SEC 26-5S-5E	5 South	5 East	26	8.85
CACTUS 110 LLC	502-25-005A	W-630 OF THE N-1855 OF THE S-2905 OF SEC 25-5S-5E	5 South	5 East	25	26
CACTUS 110 LLC	502-25-014A & 502-25-014B	NE OF SEC 35-5S-5E	5 South	5 East	35	160
CACTUS 110 LLC	502-25-0130	NW OF SEC 35-5S-5E	5 South	5 East	35	160
CACTUS 110 LLC	502-25-0110	N1/2 OF SEC 34-5S-5E AC E-CRETE IPR #502-25-800	5 South	5 East	34	320
CACTUS 110 LLC	502-25-0220	SW SEC 35-5S-5E (Surface only)	5 South	5 East	35	160
CACTUS 110 LLC	502-25-0150	SE OF SEC 35-5S-5E	5 South	5 East	35	160
CACTUS 110 LLC	502-25-021A	COMM @ NW COR OF SEC 36-5S-5E TH S-1316.64' TO POB TH S88D E- 227.58' TO POB THE POINT OF A TANG-CUR CONCAVE SW W/RAD OF 217.19' TH SWLY 325.21- TH S02D E-980.73' TO THE POINT OF A NON-TANG-CUR CONCAVE NW W/RAD OF 123.28' TH SWLY 192.7' TH W-360.55' TH N-1313.81' TO POB 13.50 AC (Surface only)	5 South	5 East	36	13.5
CACTUS 110 LLC	503-69-001A	LOTS 1-4 & S1/2N1/2 OF SEC 3-6S-5E	6 South	5 East	3	340.24
CACTUS 110 LLC	515-28-0020	SEC 28-5S-6E WATERWELL SITE #1 NWNENE AND PIPELINE RIGHT OF WAY EXTENDING IRREGULARLY FROM EAST EDGE OF NE TO N EDGE OF NE	5 South	6 East	28	15.46
CACTUS 110 LLC	515-28-0100	SEC 28-5S-6E WATERWELL SITE IN NENENESE AND PIPELINE RIGHT OF WAY ALONG EAST EDGE OF SE	5 South	6 East	28	15.12
CACTUS 110 LLC	502-37-006A	W1/2E1/2SE OF SEC 28-5S-5E	5 South	5 East	28	40
CACTUS 110 LLC	502-37-005C	NWSE OF SEC 28-5S-5E	5 South	5 East	28	40

Owner	Parcel No.	Property Description	Township	Range	Section	Acres
CACTUS 110 LLC	502-37-005A	E1/2SWSE OF SEC 28-5S-5E	5 South	5 East	28	20
CACTUS 110 LLC	502-37-005B	W1/2SWSE OF SEC 28-5S-5E	5 South	5 East	28	20
CACTUS 110 LLC	502-37-001A	N1/2NE OF SEC 28-5S-5E	5 South	5 East	28	80
CACTUS 110 LLC	502-37-001B	SWNE OF SEC 28-5S-5E	5 South	5 East	28	40
CACTUS 110 LLC	502-37-001C	W1/2SENE OF SEC 28-5S-5E	5 South	5 East	28	20
CACTUS 110 LLC	502-37-001D	NESENE OF SEC 28-5S-5E	5 South	5 East	28	10
CACTUS 110 LLC	502-37-0040	SW OF SEC 28-5S-5E	5 South	5 East	28	160
CACTUS 110 LLC	502-41-0360	NE OF SEC 33-5S-5E 160.00 AC (Surface only)	5 South	5 East	33	160
CACTUS 110 LLC	502-41-0340	W1/2NW OF SEC 33-5S-5E (Surface only)	5 South	5 East	33	80
CACTUS 110 LLC	502-41-0350	E1/2NW OF SEC 33-5S-5E (Surface only)	5 South	5 East	33	80
CACTUS 110 LLC	502-25-005D	THE ENTIRE WEST HALF OF SECTION 25, TOWNSHIP 05 SOUTH, RANGE 05 EAST; EXCEPT THE NORTH HALF OF THE NORTH HALF OF THE NORTHWEST QUARTER OF SAID SECTION 25; ALSO EXCEPT THE FOLLOWING DESCRIBED PARCEL: COMMENCING AT THE SOUTHWEST CORNER OF SAID SECTION 25, THENCE NORTH 1050.01 FEET TO THE POINT OF BEGINNING, THENCE CONTINUING NORTH 1589.24 FEET, THENCE CONTINUING NORTH 265.79 FEET, THENCE EAST 630.01 FEET, THENCE SOUTH 1855.03 FEET, THENCE WEST 630.01 FEET TO THE POINT OF BEGINNING, 11,108,062.86 SQUARE FEET, 255.01 ACRES	5 South	5 East	25	255.01

Owner	Parcel No.	Property Description	Township	Range	Section	Acres
CACTUS 110 LLC	502-25-004B	THE WEST 894.69 FEET OF THE SOUTH 1979.31 FEET OF THE NORTHEAST QUARTER OF SECTION 25, TOWNSHIP 05 SOUTH, RANGE 05 EAST, 1,770,868.86 SQUARE FEET, 40.65 ACRES	5 South	5 East	25	40.65
CACTUS 110 LLC	502-25-006B	THE WEST 894.69 FEET OF THE SOUTHEAST QUARTER OF SECTION 25, TOWNSHIP 05 SOUTH, RANGE 05 EAST, 2,360,406.95 SQUARE FEET, 54.19 ACRES	5 South	5 East	25	54.19
CACTUS 110 LLC	502-25-008E	THE SOUTH HALF OF THE NORTHWEST QUARTER OF THE NORTHWEST QUARTER OF SECTION 26, TOWNSHIP 05 SOUTH, RANGE 05 EAST, 874,495.13 SQUARE FEET, 20.08 ACRES	5 South	5 East	26	20.08
CACTUS 110 LLC	502-25-008G	THE ENTIRE NORTHWEST QUARTER OF SECTION 26, TOWNSHIP 05 SOUTH, RANGE 05 EAST; EXCEPT THE NORTH HALF OF THE NORTHEAST QUARTER OF SAID NORTHWEST QUARTER; ALSO EXCEPT THE NORTHWEST QUARTER OF THE NORTHWEST QUARTER OF SAID SECTION 26, 4,377,953.92 SQUARE FEET, 100.50 ACRES	5 South	5 East	26	100.5
CACTUS 110 LLC	502-25-007F	THE ENTIRE NORTHEAST QUARTER OF SECTION 26, TOWNSHIP 05 SOUTH, RANGE 05 EAST; EXCEPT THE NORTH HALF OF THE NORTH HALF OF SAID NORTHEAST QUARTER; ALSO EXCEPT THE FOLLOWING DESCRIBED PARCEL: COMMENCING AT THE SOUTHEAST CORNER OF SAID SECTION 26, THENCE NORTH 2639.25 FEET TO THE POINT OF BEGINNING, THENCE WEST 1450.01 FEET, THENCE NORTH 265.79 FEET, THENCE EAST 1450.01 FEET, THENCE SOUTH 265.79 FEET TO THE POINT OF	5 South	5 East	26	111.87

Owner	Parcel No.	Property Description	Township	Range	Section	Acres
		BEGINNING, 4,873,050.52 SQUARE FEET, 111.87 ACRES				
CACTUS 110 LLC	502-25-021C	THE ENTIRE WEST HALF OF SECTION 36, TOWNSHIP 05 SOUTH, RANGE 05 EAST AND THE WEST 894.69 FEET OF THE EAST HALF OF SAID SECTION 36; EXCEPT THE FOLLOWING DESCRIBED PARCEL: COMMENCING AT THE NORTHWEST CORNER OF SAID SECTION 36, THENCE SOUTH 1316.64 FEET TO THE POINT OF BEGINNING, THENCE SOUTH 88 DEGREES EAST 227.57 FEET TO A TANGENT CURVE TO THE RIGHT, HAVING A RADIUS 217.19 FEET, THENCE SOUTHEASTERLY ALONG THE CURVE WITH A CENTRAL ANGLE OF 85 DEGREES 47 MINUTES 34 SECONDS, AN ARC DISTANCE 325.21 FEET, THENCE SOUTH 02 DEGREES EAST 980.73 FEET TO A NON-TANGENT CURVE TO THE RIGHT, WITH A RADIAL BEARING OF SOUTH 89 DEGREES 36 MINUTES 12 SECONDS WEST, HAVING A RADIUS 123.28 FEET, THENCE SOUTHWESTERLY ALONG SAID CURVE WITH A CENTRAL ANGLE OF 89 DEGREES 33 MINUTES 32 SECONDS, AN ARC DISTANCE OF 192.70 FEET, THENCE WEST 360.55 FEET, THENCE NORTH 1313.81 FEET TO THE POINT OF BEGINNING, ALSO KNOWN AS PARCEL 2 OF SURVEY 2022-016495, 18,193,685.88 SQUARE FEET, 417.67 ACRES (Surface only)	5 South	5 East	36	417.67
Total for Cactus 110 LLC						**4,209.14**

Owner	Parcel No.	Property Description	Township	Range	Section	Acres
Arizona State Lands Department (Leased Lands)						
Arizona State Lands Department (Prospecting Permit # 008-121173-00-100)	503-26-7000	Lots 3 4 S2NW S2	6 South	5 East	1	489.12
Arizona State Lands Department (Prospecting Permit # 008-122116-00) (optioned from Bronco Creek Exploration)	502-25-7020	SE EX SWSWSWSE	5 South	5 East	34	157.5
					Total	**4,855.76**

Along with these properties, Arizona Sonoran filed a Notice of Intent to Locate (NOITL) with the Bureau of Land Management Arizona in October of 2019 (AZA 37933) and staked 18 lode claims on 17 January 2020 and acquired the rights to the Federal Minerals under the ARCUS surface. The claims are S-1 through S-18 (AMC459838 through AMC459855) detailed in Table 4-2. These claims are for lands in the north half of section 35, Township 5 South, Range 5 East, of which Arizona Sonoran purchased from ARCUS in 2021.

Table 4-2: Acquired Claims S-1 through S-18 (AMC459838 thru AMC459855) Property and Rights

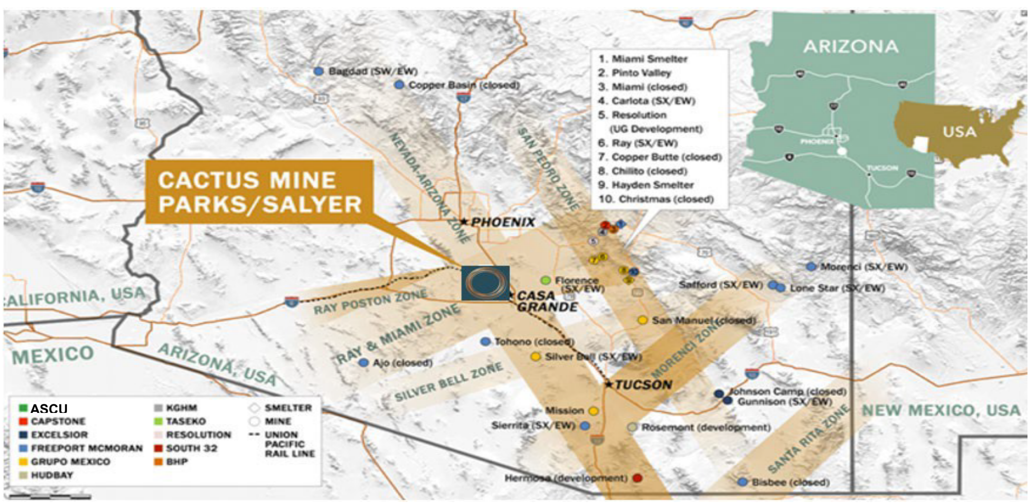
Serial Number	Lead Serial Number	Mer Twn Rng Sec	Quad	Claim Name	Claimant Name	Case Type	Status	Loc Date	Last Assessment Year
AMC459838	AMC459838	14 0050S 0050E 034	NW	S1	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459839	AMC459838	14 0050S 0050E 027	SW	S2	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459840	AMC459838	14 0050S 0050E 033	NE	S3	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459841	AMC459838	14 0050S 0050E 028	SE	S4	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459842	AMC459838	14 0050S 0050E 033	NE	S5	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459843	AMC459838	14 0050S 0050E 033	NE	S6	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459844	AMC459838	14 0050S 0050E 033	NE	S7	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459845	AMC459838	14 0050S 0050E 028	SE	S8	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459846	AMC459838	14 0050S 0050E 033	NE, NW	S9	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459847	AMC459838	14 0050S 0050E 028	SW, SE	S10	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459848	AMC459838	14 0050S 0050E 033	NW	S11	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459849	AMC459838	14 0050S 0050E 028	SW	S12	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459850	AMC459838	14 0050S 0050E 033	NW	S13	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459851	AMC459838	14 0050S 0050E 028	SW	S14	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459852	AMC459838	14 0050S 0050E 033	NW	S15	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459853	AMC459838	14 0050S 0050E 028	SW	S16	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459854	AMC459838	14 0050S 0050E 033	NW	S17	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022
AMC459855	AMC459838	14 0050S 0050E 028	SW	S18	CACTUS 110 LLC	LODE	ACTIVE	01/17/2020	2022

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Project is located 40 road miles south southwest of the Greater Phoenix metropolitan area, in a relatively flat to slightly undulating ranching and mining locale. Access to the Project is approximately 4.6 miles west of AZ-387 on North Bianco Road off of West Maricopa-Casa Grande Highway. The Project, as shown in Figure 5-1 is surrounded by other, current and past-producing, copper mines and processing facilities. As such, the Greater Phoenix area is a major population center (approximately 4.5 million persons) with a major airport and transportation hub and well-developed infrastructure and services that support the mining industry. The cities of Casa Grande and Maricopa are nearby and, combined with Phoenix, can supply sufficient skilled labor for the Project.

Figure 5-1: Regional Copper Mines and Processing Facilities



Copper is a key product in Arizona's economy. Within Pinal County there are currently two operating mines. These mines are the Florence Copper Mine, owned and operated by Taseko Mines Ltd. (approximately 25 miles ENE) and the Ray Mine, owned, and operated by ASARCO, a subsidiary to Grupo Mexico S.A. De C.V. (Grupo Mexico) (approximately 50 miles ENE) of the Cactus Mine.

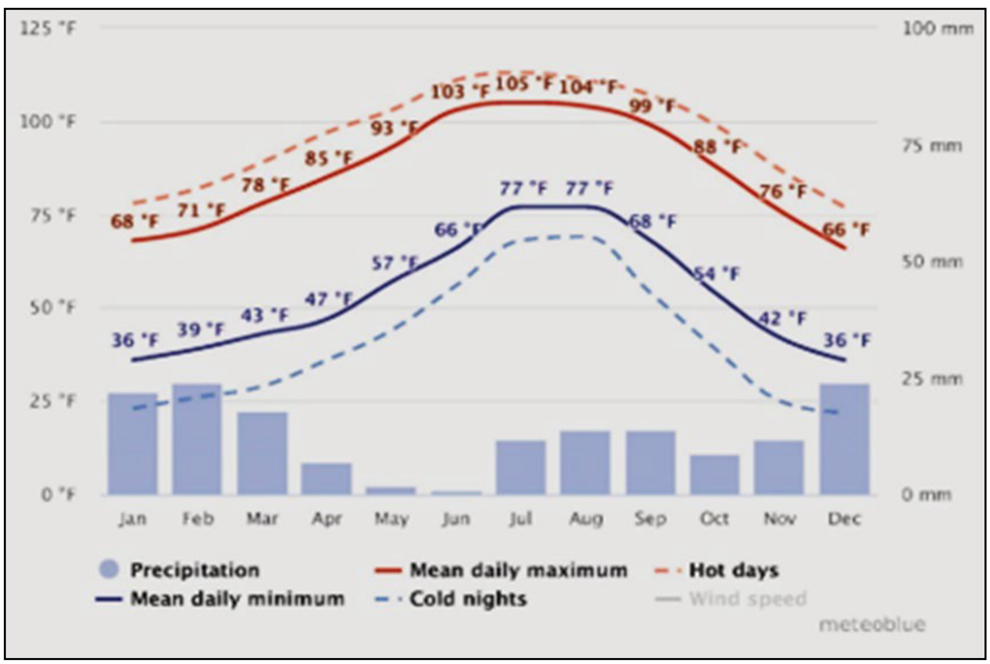
Ecologically, the site is within the Sonoran Desert Section of the Basin and Range Lowlands Province of Arizona in the lower Santa Cruz Basin. The area is characterized by broad, level valley plains, gently sloping pediments, and widely separated mountain ranges. Elevations at the mine vary from approximately 1,360 ft amsl to 1,460 ft amsl. Soils have very low levels of available plant nutrients and vegetation on the property is typical of the Sonoran Desert and includes bunchgrasses, yucca, mesquite, and cacti.

5.2 Climate

Climate at the mine is typical of the Arizona Sonoran Desert, with temperatures ranging from 19 °F (-7 °C) to 117 °F (47 °C), and with average annual precipitation of 8.6-inch, falling primarily in high-intensity, short duration events. The project will operate 365 days a year with no expected delays due to seasonal climate changes. The mine site contains no surface water resources. Storm runoff waters from the site are drained toward the Santa Cruz River by minor tributaries to the Santa Rosa and Brawley washes. Groundwater flows generally are to the south and southwest and towards the open pit, which acts as a terminal sink. A terminal sink occurs as the result of at least two factors. First, the pit lake is below the surrounding water table. Second, the area is arid, leading to significant evaporation from the pit lake. What storm and groundwater enters the pit lake evaporates before migrating into the surrounding groundwater.

Average relative humidity is approximately 25%. The least humid month is June (10.2% relative humidity), and the most humid month is December (39.3%). Figure 5-2 shows the temperature highs and lows.

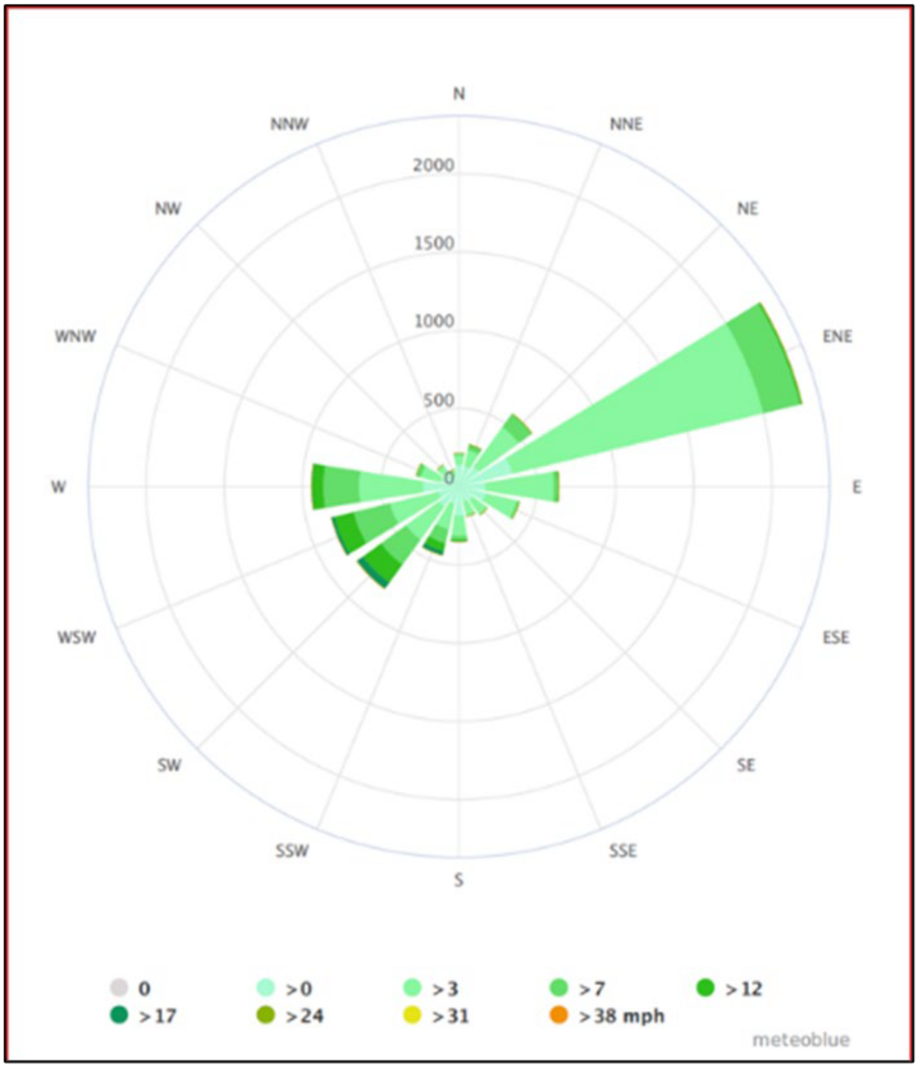
Figure 5-2: Climate (High/Low)



Wind is usually calm. The windiest month is May, followed by April and July. May's average wind speed of around 5.5 knots (6.4 mph or 10.3 km/h) is considered a light breeze.

Maximum sustained winds (the highest speed for the day lasting more than a few moments) are at their highest in mid-July where average top sustained speeds reach 17.3 knots, which is considered a fresh breeze. The wind rose for Casa Grande in Figure 5-3 shows how many hours per year the wind blows from the indicated direction. Example SW: Wind is blowing from southwest (SW) to northeast (NE). Arizona Sonoran will institute measures to reduce dust that could be produced at the mine site. For additional information see Chapter 20.0 Environmental Studies, Permitting and Social or Community Impact of this document.

Figure 5-3: Wind Speed and Direction



5.3 Local Resources and Infrastructure

Utilities and Infrastructure

Electric power is available from APS 115 kV transmission line which passes on the south side of the site and connects to an existing substation at the mine site and is owned by APS.

Arizona Sonoran, as part of the sale of the property, acquired the historic Type 2 Non-Irrigation grandfather rights (Certificate 58-100706.0005) for 136 afy. In addition to the grandfathered rights Arizona Sonoran has obtained its permit from the ADWR (Permit 59-233782.0000) for an additional 3,600 afy under a Permit to Withdraw Groundwater for Mineral Extraction and Metallurgical Processing within an Active Management Area (A.R.S. § 45-514). This entitlement is expected to be sufficient for LOM as outlined in this PEA.

5.4 Physiography

Table 5-1 shows the physiography.

Table 5-1: Physiography

General Physiographic Area:	Intermontane Plateaus	
Physiographic Province:	Basin and Range Province	
Physiographic Section:	Sonoran Desert	
Alteration Type:	L	
Alteration:	Phyllic and Argillic – More Intense in Mineralized Area; Sericitization	
Associated Rocks:	Breccia Dacite Porphyry Monzonite Porphyry Quartz Monzonite Porphyry Limestone Quartzite Schist Conglomerate Diabase Dike Diorite Granite	
Rock Unit Name:	Pinal Schist Oracle Granite Three Peaks Monzonite Porphyry Sacaton Granite	
Associated Minerals:	Alunite Anhydrite / Gypsum Azurite / Malachite Biotite Bornite Brochantite Chalcocite Chalcopyrite Chlorite Chrysocolla Copper	Covellite Cuprite Goethite / Limonite Hematite / Specularite Jarosite Kaolinite Magnetite Molybdenite Pyrite Sericite Sphalerite

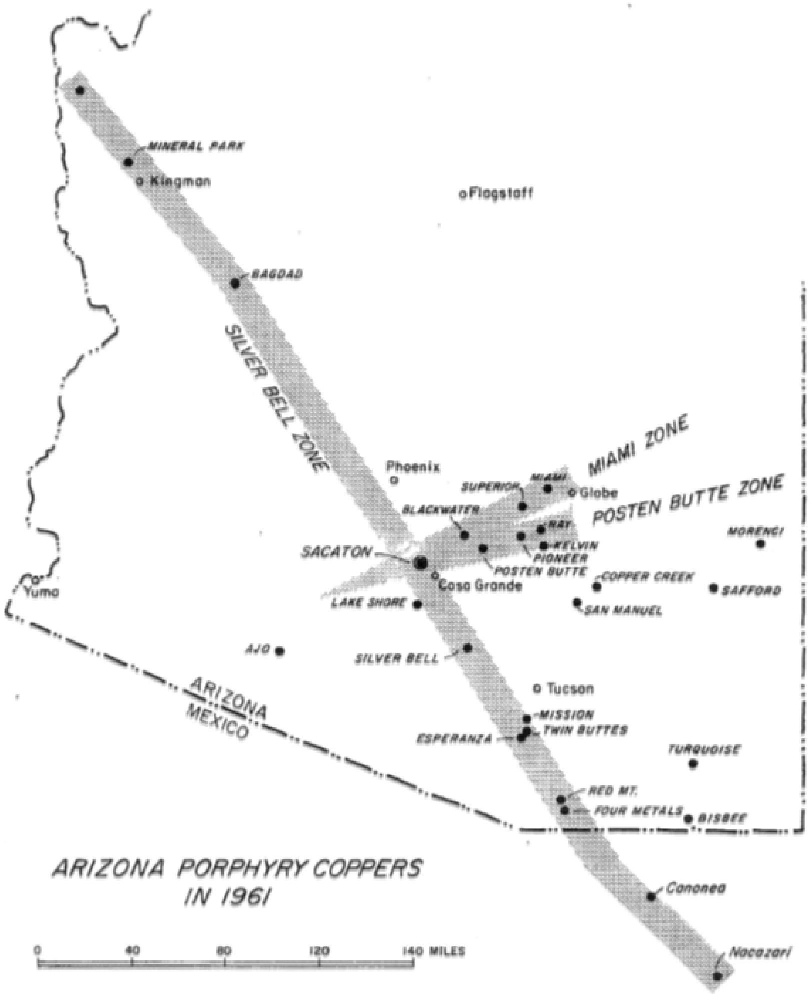
6.0 HISTORY

ASARCO geologists first discovered the Sacaton mineral deposit in the early 1960s while examining an outcrop of leached capping composed of granite cut by several thin monzonite porphyry dikes. The search was based on re-prospecting large areas of the US, including central Arizona, and used the exploration philosophies of Harold Courtright and Kenyon Richards. They had observed that many porphyry copper deposits did not contain large areas of copper oxide mineralization above the ore body. They used observations related to the oxidized products of the sulfide mineralization (leached capping interpretation) on the surface to evaluate the sulfide mineralization below.

In the 1960s, very few porphyry copper deposits were expected to be found outcropping in well prospected areas. The program was designed to search for unrecognized or partially covered altered rocks that could host porphyry copper deposits. Explorationists at the time had many ideas about regional structures that may have controlled the emplacement of copper deposits. According to Kenyon Richard (1983), ASARCO did not feel that this was a significant exploration tool, but they did see that alignment of altered zones and deposits could be useful.

Figure 6-1 approximates what was felt to be significant at the time.

Figure 6-1: Arizona Porphyry Coppers in 1961



Part of the exploration program was to understand the post mineral stratigraphy and examine areas on the edge of these cover rocks which may contain clues to underlying mineralization. Accordingly, ASARCO geologist John Kinnison was mapping the area SW of Superior in 1960 and discovered a small, altered outcrop at the base of Poston Butte just north of Florence. This led to the discovery of the Poston Butte deposit which is now known as the Florence deposit. Reconnaissance mapping continued to the SW and on 10 February 1961, Kinnison, along with ASARCO geologist Art Blucher, noticed an inconspicuous outcrop (Discovery Outcrop) east of Casa Grande. The exposure was about 300 ft in diameter and surrounded by alluvial cover. The nearest bedrock exposures were a mile and a half to the north. The hill, composed of granite and cut by a monzonite porphyry dike, contained pervasive sericite and argillic alteration. Both rock types exhibited limonite derived from the oxidation of pyrite and traces of live limonite derived from the oxidation and leaching of chalcocite. Photos of Discovery Outcrop are in Figure 6-2 and Figure 6-3.

Figure 6-2: View from Discovery Outcrop from ASARCO Exploration Site Looking Out Over What Would Become the Sacaton Pit

Figure 6-3: View from Discovery Outcrop Today Post-Mining of the Sacaton Pit

The nature of this original find indicated the likely presence of porphyry copper-type mineralization. Following this lead, ASARCO initiated a drilling program which defined copper mineralization zones. The west zone contained the ore body which was ultimately accessed through the open pit. The deeper east zone was the target of potential mining by underground methods.

During the life of the project ASARCO drilled an approximate 223,246.4 ft of both Core and Rotary exploration drilling. A detailed list of historic drilling is provided in Appendix A.

Project construction and mining of the west zone via open pit method commenced by 1972, and the mine operated continuously from 1974 until 1984. An underground copper deposit at Sacaton was under development until September 1981 when work was suspended because of high costs and a weak copper market. The Sacaton Mine was permanently closed 31 March 1984 due to exhaustion of the open pit ore reserves. Table 6-1 presents historic production rates.

Table 6-1: Sacaton Mine Historic Production (Fiscal Years Ended 31 December)

Year	Ore Milled Short Tons	Mill Grade Cu%	Mill Grade Ag Oz/T	Cu Short Tons	Au Troy Oz	Ag Troy Oz
1974	2,020,000	0.63	0.05	9,516	N/A	N/A
1975	3,630,000	0.74	0.06	21,918	3,153	N/A
1976	3,782,000	0.71	0.07	22,021	3,151	N/A
1977	3,471,000	0.70	0.06	19,872	3,103	N/A
1978	4,153,000	0.67	0.07	23,042	3,691	N/A
1979	4,006,000	0.65	0.07	21,367	3,558	142,000
1980	3,819,000	-	-	16,097	2,504	124,000
1981	4,103,000	-	-	21,015	3,334	172,000
1982	4,165,000	-	-	20,892	2,499	154,000
1983	4,003,000	-	-	18,794	1,983	134,000
1984	1,000,000	-	-	4,496	479	33,000
Total	**38,152,000**	**0.69**	**0.06**	**199,030**	**27,455**	**759,000**

Source: Sacaton Mining Operations Report Version 2005 By David F. Briggs 22 October 2004.

The resultant Sacaton open pit mine is roughly circular, approximately 3,000 ft (914 m) in diameter and 1,040 ft (317 m) deep (Figure 6-4). The pit has a visible internal lake with the surface at approximately 980 ft in depth from the pit rim. During operation, the Sacaton mine consisted of the pit, crushing facilities and coarse ore stockpile, a 9,000 tpd flotation mill, a (TSF) that covered approximately 300 acres, a return water impoundment, an overburden dump, and a WRD that covered approximately 500 acres. Production from the open pit was approximately 11,000 tpd. Copper flotation mill concentrate was sent by rail to the ASARCO smelter in El Paso, Texas.

Figure 6-4: Historic Overview of Prior Sacaton Mine Site

During mining of the open pit, a waste dump was created through dumping of defined waste material. All oxide copper mineralization, and sulfide copper mineralization below the working grade control cutoff of 0.3% Cu, were deposited to the waste dump. The historical waste dump forms the basis of the Stockpile Project resource modelled in this PEA due to the level of mineralized material discarded.

During the operating period, ASARCO sank a 2,000 ft shaft (Figure 6-5) just east of the pit to access the deeper east deposit. Development of the underground mine was suspended in 1981, and the site further suspended overall activity in 1984. Since then, intermittently and per a site improvement plan (SIP), fixed equipment and rolling stock have been removed from the site, and fixed plant locations and the tailings disposal facility were covered with previously salvaged and stockpiled desert alluvial soil material and revegetated.

Figure 6-5: Historic Overview of Sacaton Pit and Underground Shaft with Headframe

Parks/Salyer was first drill intercepted in January 1976 as part of a work commitment hole. S - 144 was ultimately located on the very eastern edge of the current Parks/Salyer resource. Later in 1976, three follow-up holes were drilled on the property immediately to the south of Arizona Sonoran's property and intercepted the southern side of the Parks/Salyer deposit as part of an ASARCO-Freeport joint venture. No immediate further exploration work was undertaken at Parks/Salyer. However, exploration targeting interpretations in 1978, 1981, and 1984 had interpreted the potential of higher-grade enrichment mineralization to the north in the area now known for the Parks/Salyer deposit. Four holes had been planned in 1984 but were undrilled at the time. In May 1996, two of those planned holes were drilled (S-200 and S-201) which were successful in intercepting higher grade and thicker enriched and primary mineralization however no further exploration was undertaken at Parks/Salyer until Arizona Sonoran acquired the property in 2020.

In 2005, ASARCO filed for reorganization under Chapter 11 of the Bankruptcy Code in the United States Bankruptcy Court for the Southern District of Texas, Corpus Christi Division. By 2008, the Bankruptcy Court for the Southern District of Texas, Corpus Christi Division approved the process by which ASARCO would pursue the selection of a plan sponsor and sale of its operating assets.

During that year, and after a bidding process for the purchase of ASARCO's assets, Sterlite (USA), Inc., a subsidiary of Vedanta Resources P (an Indian corporation), executed a purchase and sales agreement in the amount of $2.6 billion for ASARCO's assets. After the purchase and sales agreement was executed, copper prices began to decline, and by

October 2008, Sterlite representatives informed the United States Bankruptcy Court for the Southern District of Texas, Corpus Christi Division that the company could not honor the contract.

On 05 June 2009, the Bankruptcy Court for the Southern District of Texas, Corpus Christi Division approved a Custodial Trust Settlement Agreement that resolved claims pertaining to past and potential future cleanup costs associated with approximately 18 ASARCO owned sites in 11 states. The agreement required the establishment of a custodial trust to oversee cleanup of the sites and transfer of site property to the custodial trust.

The settlement agreement provided funding in the amount of $20 million to clean up the Sacaton site and to fund the administrative expenses associated with the custodial trust.

From 2009 up to 2018, attempts were made by other parties to purchase the Sacaton site and associated facilities. In 2018, Cactus110 LLC, a subsidiary of Arizona Sonoran Copper Company, Inc, executed both purchase and PPA with said Trust and the ADEQ respectively for the right to acquire all ASARCO land parcels representing the historic Sacaton Mine, as well as all infrastructure therein, and all associated mineral rights. Final purchase acquisition closed July 2020, following the completion and approval of SIP activities undertaken by the Trust and approved by the ADEQ. In addition, Cactus 110 holds title to the Merrill land parcels (as shown in Section 4.0. With associated royalties, these private land assets represent, among other things, the mineral rights to the old Sacaton East, Sacaton West and Parks/Salyer deposits. Further landholdings acquired by Arizona Sonoran or leased are also referred to above (as shown in Section 4.0. The Sacaton deposits since 2020 are now referred to as the Cactus deposits.

ASARCO had worked continuously on the project from the early 1960s to the mid-1980s. Significant records of the development of the geological understanding, mining operations, and processing results remained with the property. Arizona Sonoran is benefiting from the high quality of work and historical records remaining from the past operators.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Cactus project occurs in the desert region of the Basin and Range province of Arizona. The basal formation in the area is the Proterozoic Pinal Schist. At the close of Older Precambrian, the Oracle Granite batholith intruded the Pinal Schist. In Younger Precambrian time Apache Group sediments were deposited and igneous activity resulted in the emplacement of the Sacaton Granite northwest of the mine along with numerous diabase dikes. In the Paleozoic Era, an unknown thickness of sediments was deposited and later eroded along with most of the Apache Group rocks. During the Laramide Orogeny two granitic stocks, the Three Peaks Monzonite and the Sacaton Peak Granite () were emplaced in the vicinity of the Project. Figure 7-1 shows the major intrusive rocks in the Project area.

Figure 7-1: Generalized Geology Map Showing Major Intrusive Rocks in the Cactus Project Area

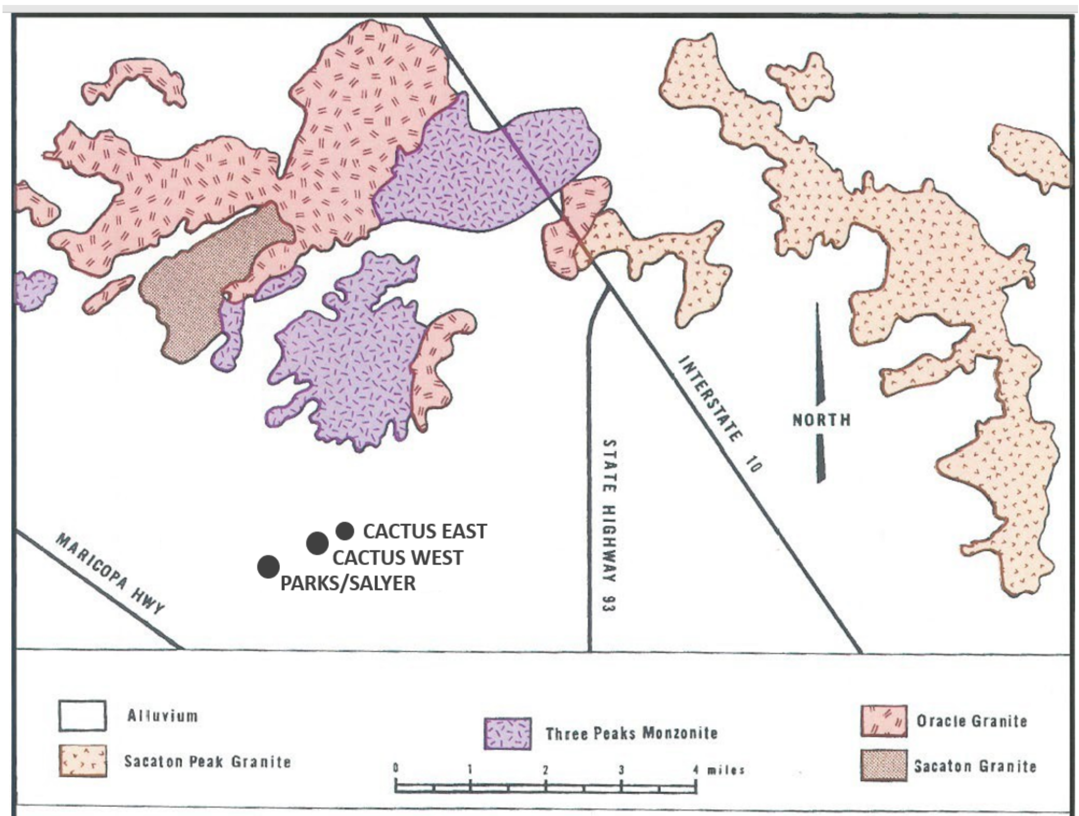


Figure 25.1. Generalized geologic map of the Sacaton Mountains. Modified after Balla (1972).

At a location removed from the current mine, Laramide porphyries of a similar composition intruded the Oracle Granite and introduced hydrothermal solutions which altered and mineralized a large area of the surrounding rocks. Subsequent Tertiary extension rotated and dismembered the mineralized rocks. A low angle listric fault (the Basement fault) moved the Sacaton deposits to their current location. Quaternary basin-fill deposits covered all evidence of mineralization except for the small Sacaton discovery outcrop. The Parks/Salyer project, also owned by Arizona Sonoran, is located 1.3 miles to the SW of Cactus and displays the same geological characteristics as Cactus. Located within a separate horst block to Cactus, it is a portion of the same larger porphyry system that shows lesser displacement from the in situ source.

With the exception of the Pinal Schist, found below the Basement fault, all pre-mineral rocks in the vicinity of the mineralized deposits are pervasively altered. In addition, two stages of brecciation are present, often resulting in an intimate mixture of rock types. These features have complicated the delineation and identification of the rocks. Major host rocks are Precambrian Oracle Granite, Laramide monzonite porphyry, and quartz monzonite porphyry.

The porphyries are similar in composition and texture but are distinguished by the presence of 10% clear quartz phenocrysts in the latter. They intrude the older rocks and occur as large masses, poorly defined dike-like masses, and thin well-defined but discontinuous dikes.

They also form monolithic breccias and mixed breccias containing varying percentages of granite. Discontinuous pre-mineral diabase and post-mineral dacite porphyry dikes intrude the older rocks in both deposits. Figure 7-2 through Figure 7-5 show the rock type distribution of the geological units in both deposits.

Figure 7-2: Plan View through the Cactus West Deposit on the 1,040 ft Elevation

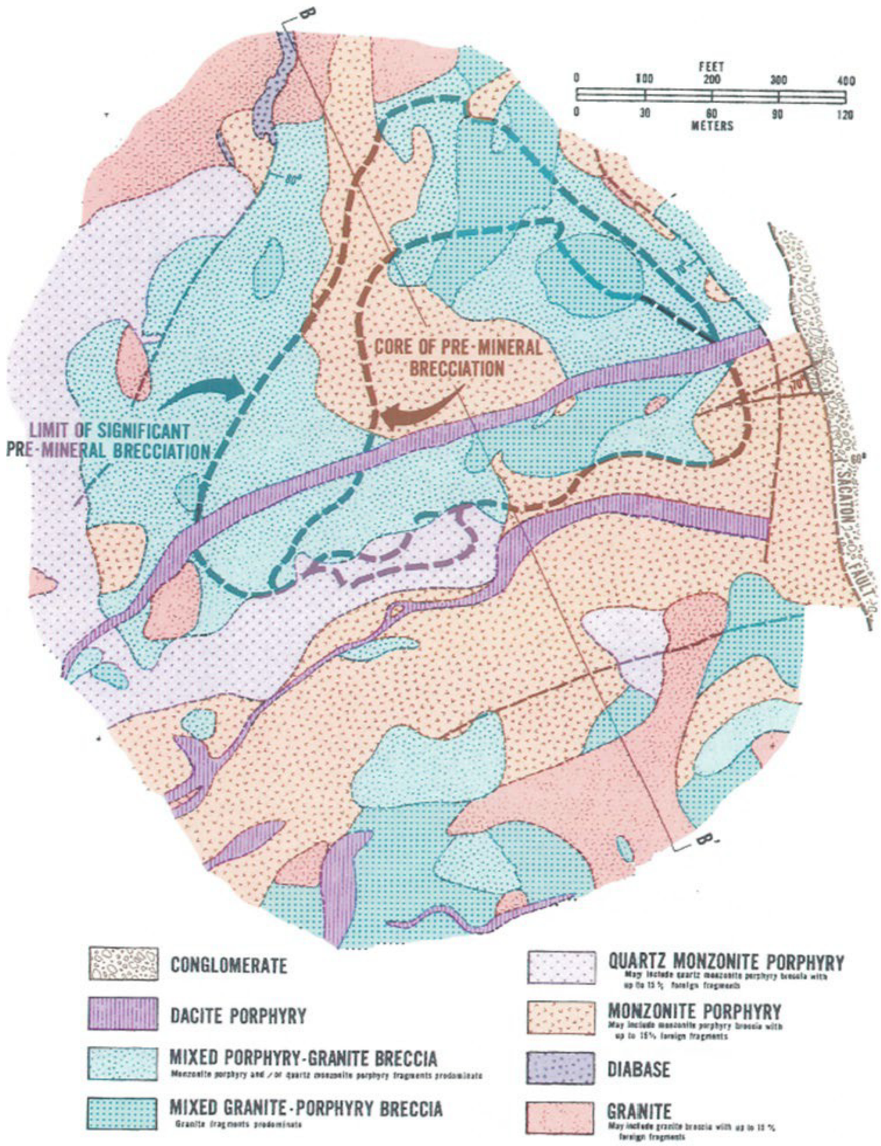


Figure 7-3: Location of Cross Sections B-B' and C-C' through the Cactus West and East Deposits

Figure 7-4: Cross Section B-B' Through the Cactus West Deposit

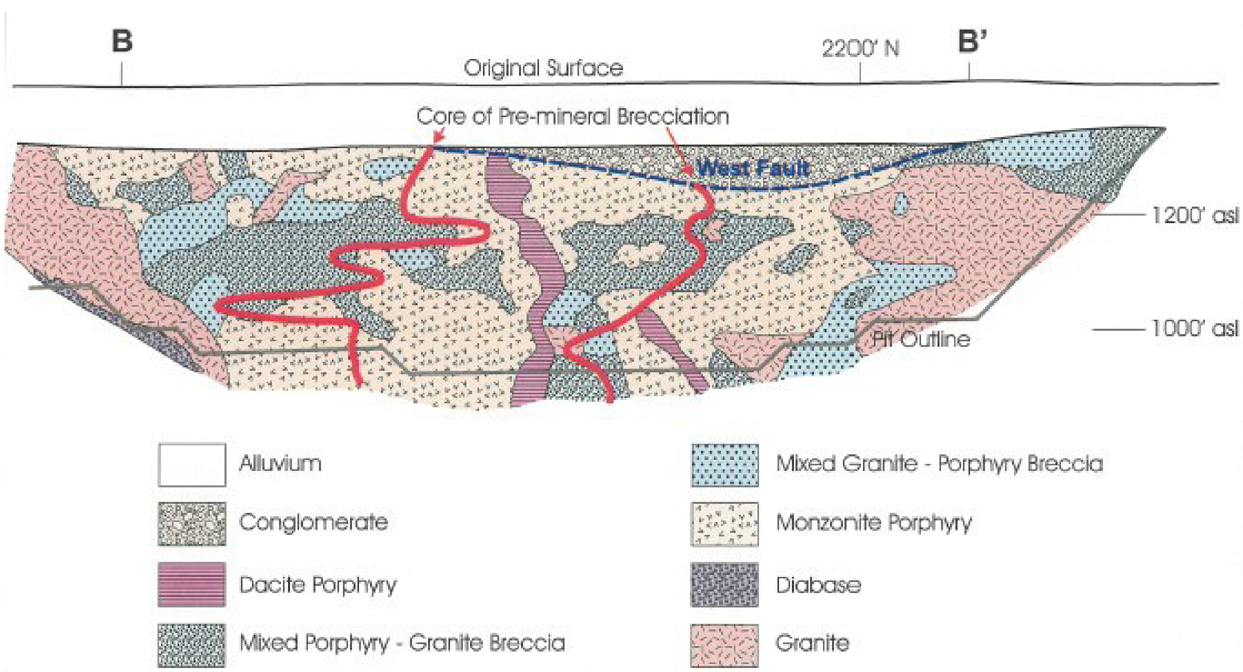
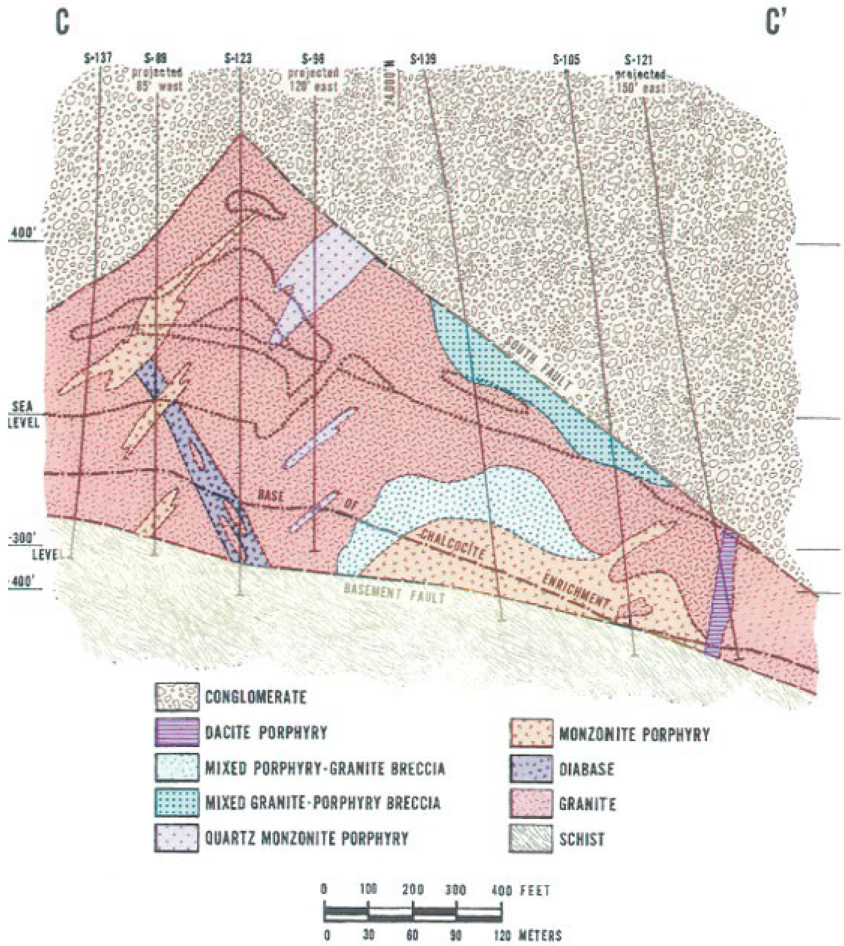


Figure 7-5: Cross Section C-C' through the Cactus East Deposit



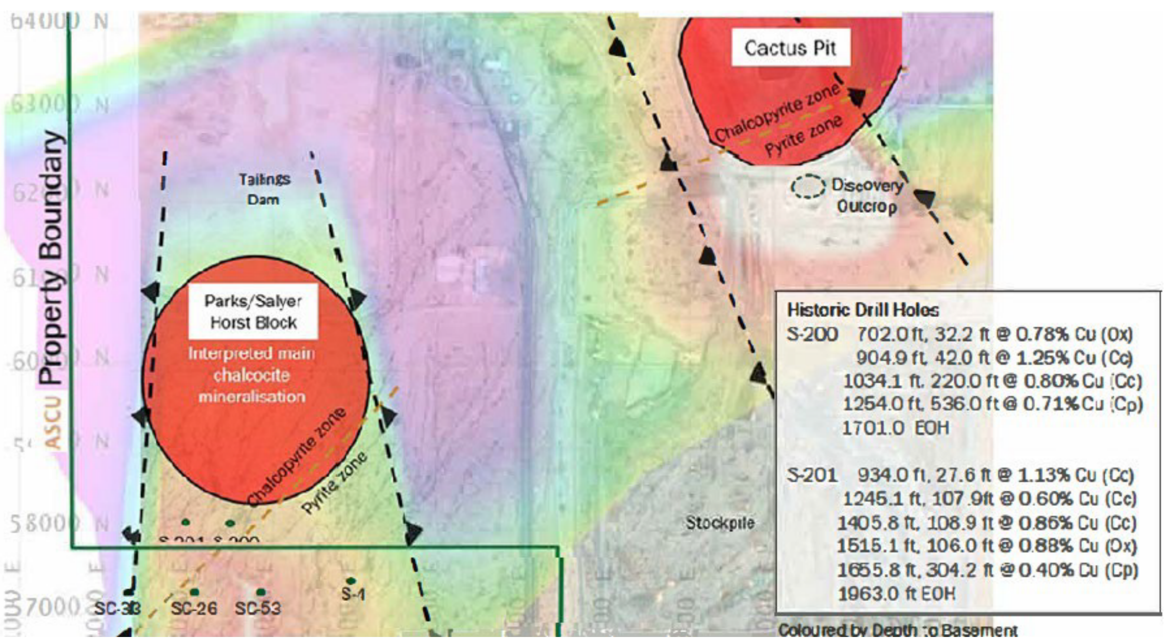
Structurally both deposits are complex with intense fracturing, faulting, and brecciation. Pre-mineral brecciation is related to the intrusion of the Laramide porphyries and occurs primarily in the west deposit which had a central core of pre-mineral brecciation that was a control for hypogene mineralization. Angular vugs are a diagnostic feature of the pre-mineral breccia. They occur between fragments in the breccia and vary in size from 0.2 inch to 2.0 inch. Post-mineral brecciation is ubiquitous in both deposits and has affected the rocks in a number of ways, depending on rock composition, degree and type of alteration, and relative location in the mineralized deposits. Manifestations of this period of brecciation include shattering, crushing and granulation, mixing of rock types, and the presence of linear breccia structures containing crushed sulfides. Mineralized fractures in the west deposit generally strike E-NE while post-mineral fractures strike N-NW.

A great number of minor faults have been mapped in the West mineralized deposit. The faults are often variable in strike and dip and are usually difficult to trace along strike. The prevailing strike direction is N60°E to E-W. Slickensides on some of the faults indicate that horizontal components of displacement are relatively common. Generally, the lack of predictable lithologic contacts to act as markers makes the direction and magnitude of displacement difficult to estimate. Total displacement on most of the faults is thought to be less than 100 ft. Both pre-mineral and post-mineral movement is often present.

Besides being terminated at depth by the Basement fault, both deposits are bounded by normal faults that drop post-mineral conglomerate into contact with the mineralized rocks. The west deposit is in a horst block formed with the Sacaton fault forming the east side which strikes N20°W and the West fault trending N45°W on the west side. The Sacaton Fault dips 60° to the east and has a displacement of up to 1,500 ft (457 m). The east deposit is the displaced portion of the west deposit in the hanging wall of the Sacaton fault.

The Parks/Salyer project, also owned by Arizona Sonoran, is located 1.3 miles to the SW of Cactus and displays the same geological characteristics as Cactus. Located within a repeat horst block similar to Cactus (Figure 7-6), it is a portion of the same larger porphyry system that shows lesser displacement from the In Situ source. Similar northwest trending normal faults are interpreted to bound the Parks/Salyer mineralization.

Figure 7-6: Plan View of Parks/Salyer Project with Respect to the Cactus West Pit



Note: Parks/Salyer is a separate displaced portion of the same larger porphyry copper system contained within a repeat horst block formation similar to the Cactus deposits.

7.2 Alteration and Mineralization

The dominant hypogene alteration assemblages in the deposit are phyllic and potassic. Phyllic alteration is characterized by quartz, sericite, and clay, but quartz and sericite predominate. Secondary silica in the porphyries occurs as a fine-grained replacement of the groundmass (intergrown with sericite and clay). Minor amounts of quartz are also found, with sericite and clay replacing plagioclase phenocrysts in the porphyries and granite. Quartz-sulfide veinlets are associated with the phyllic assemblage and comprise up to 1% of the rock by volume. Alteration minerals occurring in rocks of the potassic assemblage include varying quantities of biotite, chlorite, quartz, sericite, and clay, with traces of secondary K-feldspar, calcite, and anhydrite. Secondary biotite and chlorite characterize the potassic assemblage. Since phyllic and supergene alteration are superimposed upon, and largely destroy, potassic alteration, it is uncertain how much of the quartz, sericite, and clay are part of the original potassic suite. Supergene alteration associated with the process of secondary enrichment of sulfides has modified the suite of hypogene alteration minerals. In Cactus West, effects of this supergene overprint are not always assessable due to post-enrichment oxidation and leaching penetrating the chalcocite blanket into the primary sulfide zone.

Similar if not identical alteration assemblages can be found in Parks/Salyer. Both assemblages include hypogene and supergene alteration overprint. Hypogene alteration assemblages include both potassic and phyllic. Alteration minerals occurring in the potassically altered rock include secondary K-feldspar, magnetite, biotite, chlorite, quartz, sericite, and clay. Such zones are typically low grade. Secondary biotite, magnetite and chlorite characterize the potassic assemblage. Phyllic assemblages are noted to include strong secondary silicification, bleaching, quartz, sericite, pyrite, and clays. The secondary silica replacement appears as fine-grained replacement of the groundmass, intergrowing between the sericite and other clays. Alteration halos surrounding quartz-sericite and sulfide veins are common within these phyllic alteration zones. These phyllic zones are typically higher in grade compared to the potassic zones. It should be noted that much of the potassic alteration is found to the north of the section and above the Basement fault.

The major hypogene sulfide minerals at Cactus are pyrite, chalcopyrite, and molybdenite. Traces of bornite and sphalerite have been observed in concentrate samples. Hypogene sulfides occur as disseminated grains, veins, and vug fillings. Disseminated sulfides are more abundant in the granite and strongly brecciated rocks than in the porphyries and weakly brecciated rocks. In the West mineralized zone, disseminated grains usually comprise less than 50% of the hypogene sulfides, but in the East mineralized zone, where granite breccia is the main rock type, disseminated grains account for over 50% of the sulfides.

The major hypogene sulfide minerals at Parks/Salyer are pyrite, chalcopyrite, and molybdenite. Trace amounts of bornite and sphalerite have been observed within the upper sections of the hypogene and lower edges of the supergene mineralization. Hypogene sulfides occur as disseminated grains, veins/veinlets, and patchy blebs. Disseminated sulfides are abundant in the brecciated rocks, Monzonite porphyry, and in the granite. Sulfides are also present within quartz veins and veinlets throughout the deposit. Disseminated sulfides account for roughly 50% of the hypogene sulfides within the site, but in zones of intensely brecciated porphyries, disseminated grains appear comprise of less than 50% of the sulfides, instead favoring veinlets and patches.

The total sulfide content for both mineralized zones is variable, ranging from approximately 1.0% to 4.0% by volume. Rock type and pre-mineral brecciation cannot be directly correlated to variations in total sulfide content. North and south of the mineralized zones the total sulfide content decreases similarly to the overall alteration intensity. Drilling and pit mapping have defined a core zone within which the grade of hypogene mineralization is at least 0.40% copper as chalcopyrite. Outside the zone the copper grade gradually drops off to less than 0.10% copper. The pyrite: chalcopyrite ratio varies from 1:1 to 3:1 within the core zone and increases to 10:1 or more outside of it. Molybdenite occurs in quartz veins and as smears on fractures. The molybdenum content averages approximately 0.010% for the West mineralized zone and 0.025% for the East mineralized zone.

Similarly, within the Parks/Salyer, molybdenite occurs in quartz veins, as smears on fractures, as well as in disseminated crystals in the groundmass. Molybdenite content averages between 0.010%-0.025%. We see similar ratios of pyrite: chalcopyrite within this deposit as in Cactus. The major supergene sulfide mineral at Cactus is chalcocite. Covellite and digenite are also present in much smaller quantities. The intensity of secondary enrichment is greatest at the top of the enriched zone and decreases gradually toward the base. In the upper portions of the enriched zone chalcocite completely replaces chalcopyrite and partially replaces pyrite. Toward the base of the zone chalcopyrite is partially replaced and pyrite is rimmed by thin coatings of chalcocite. The enrichment factor (the ratio of supergene copper grade to hypogene copper grade) varies from 3:1 to 5:1 for both mineralized zones. The most important control for supergene enrichment is the grade of primary mineralization. The bulk of economic supergene mineralization is underlain by primary sulfides averaging at least 0.40% copper.

The major supergene sulfide minerals at Parks/Salyer are chalcocite, covellite, and pyrite. Digenite is also present in smaller quantities. The intensity of the secondary enrichment is greatest at the upper portion of the enriched zone, decreasing gradually towards the base. In the upper portions chalcocite and covellite completely replace chalcopyrite and partially replace pyrite. Near the base of the zone, chalcopyrite is partially replaced, and pyrite is rimmed by chalcocite. Covellite is discontinuous and often is seen as replacing blebs and grains of pyrite. The enrichment factor varies from 3:1 to 5:1 for both mineralized zones. The most important control for supergene enrichment is the grade of primary mineralization.

The Cactus deposits have undergone two periods of oxidation and leaching. The first period resulted in the formation of what was probably a uniform high grade chalcocite blanket that was continuous through the East and West deposits. Some, and probably all, of the original blanket formed prior to movement on the Sacaton and West faults. Substantial quantities of oxidized copper minerals are found erratically distributed through the capping of both deposits. In the East deposit, the oxide minerals usually occur just above chalcocite mineralization and are thought to have resulted from in-place oxidation of chalcocite along zones of deep oxidation. Copper grades over 1.0% are common. In-place oxidation is also found in the West deposit, but generally the oxides occur over a greater horizontal and vertical range, and the copper has likely been transported from the point of oxidation.

Chrysocolla, brochantite, and malachite are the most common oxidized copper minerals. In upper portions of the capping chrysocolla predominates, while brochantite and malachite predominate in the lower portions.

The Parks/Salyer deposit has undergone at least two periods of oxidation and leaching. A large suite of transported iron oxide is present, along with remnant copper oxide minerals left behind after the initial leaching and oxidizing events. Oxidized copper occurs erratically within the leach capping; most commonly observed near the lower contact between the leached zone and the enrichment. Minerals observed include hematite, limonite, goethite, jarosite, manganese oxides, chrysocolla, malachite, brochantite, azurite, atacamite, native copper, tenorite, and cuprite. Native copper is often observed at the contact. Chrysocolla, malachite, azurite and brochantite are the most common oxidized copper minerals, with a few zones of cuprite appearing erratically with the native copper and chrysocolla.

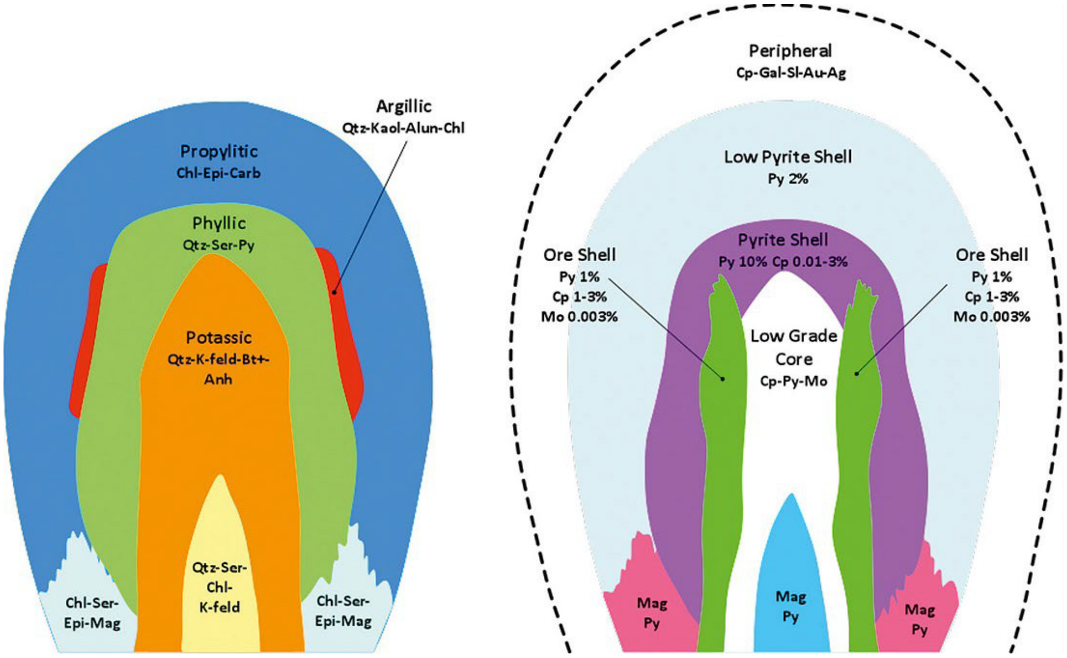
8.0 DEPOSIT TYPES

The Cactus and Parks/Salyer deposits are a portion of a large porphyry copper system that has been dismembered and displaced by Tertiary extensional faulting. Porphyry copper deposits form in areas of shallow magmatism within subduction-related tectonic environments (Berger et al., 2008). Both Cactus and Parks/Salyer have typical characteristics of a porphyry copper deposit which Berger et al. (2008) define as follows.

- A deposit wherein copper-bearing sulfides are localized in a network of fracture-controlled stockwork veinlets and as disseminated grains in the adjacent altered rock matrix.
- Alteration and mineralization at 1 km to 4 km depth are genetically related to magma reservoirs emplaced into the shallow crust (6 km to over 8 km), predominantly intermediate to silicic in composition, in magmatic arcs above subduction zones.
- Intrusive rock complexes that are emplaced immediately before porphyry deposit formation and that host the deposits are predominantly in the form of upright-vertical cylindrical stocks and/or complexes of dikes.
- Zones of phyllic-argillic and marginal propylitic alteration overlap or surround a potassic alteration assemblage.
- Copper may also be introduced during overprinting phyllic-argillic alteration events.

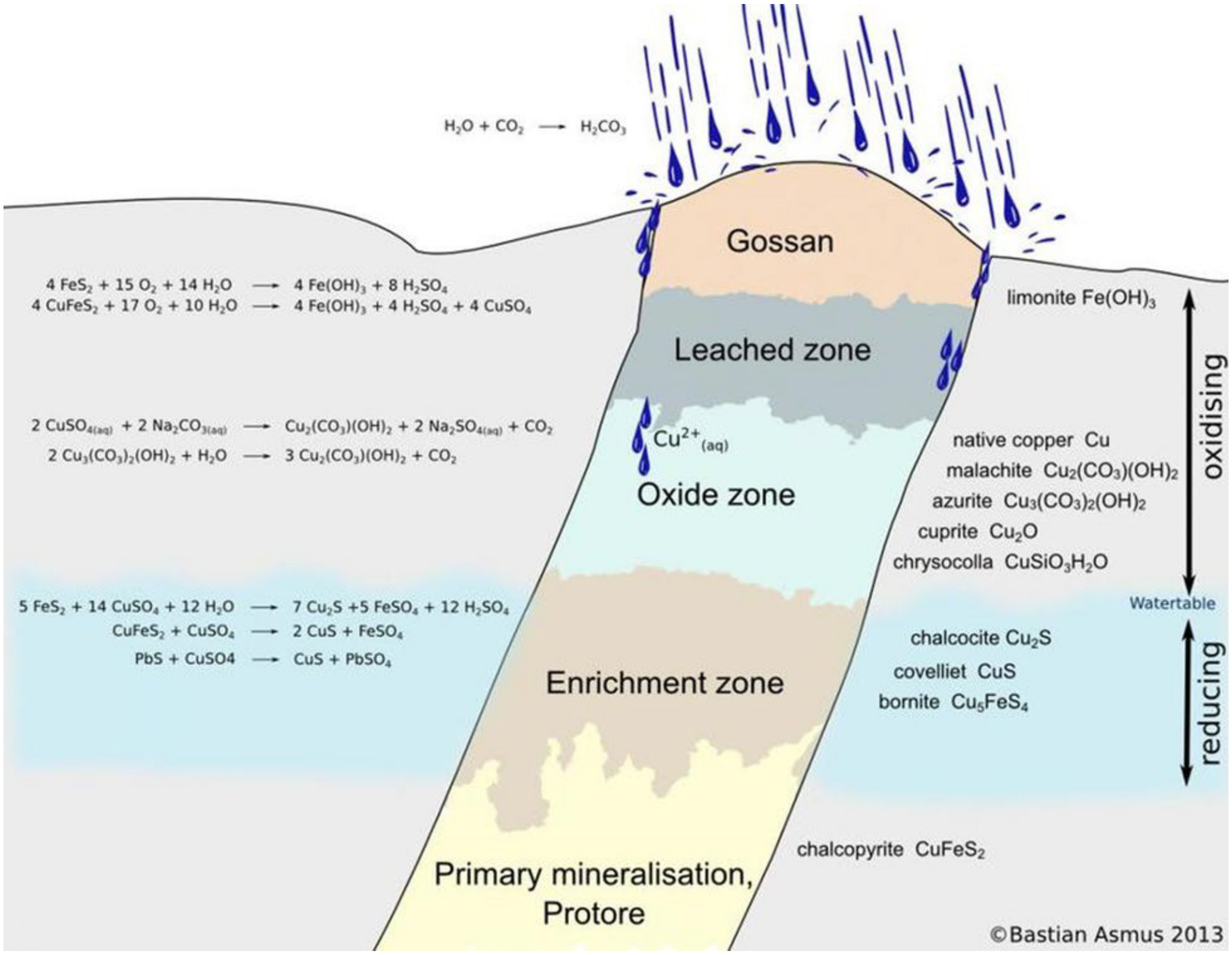
Hypogene (or primary) mineralization occurs as disseminations and in stockworks of veins, in hydrothermally altered, shallow intrusive complexes, and their adjacent country rocks (Berger et al 2008). Sulfides of the hypogene zone are dominantly chalcopyrite and pyrite. The hydrothermal alteration zones of porphyry copper deposits are well known and provide an excellent tool for advancing exploration. Schematic cross sections of the typical alteration zonations and associated minerals are presented in Figure 8-1 which were originally presented by Lowell and Guilbert in 1970. Left is a schematic cross section of the hydrothermal alteration minerals and types associated. Right is the sulfide minerals and typical percentages.

Figure 8-1: Deposit Model of a Porphyry Copper Deposit (Modified from Lowell and Guilbert, 1970)



Uplift of the porphyry system to shallow depths can result in secondary enrichment processes where copper is leached from the weathering of hypogene mineralization and redeposited below the water table as supergene copper sulfides such as chalcocite and covellite. Above the water table, copper oxide minerals typically form. Figure 8-2 represents a schematic section through a secondary enriched porphyry copper deposit identifying the main mineral zones formed as an overprint from weathering of the hypogene system. Both the Cactus and Parks/Salyer deposits have a history of oxidation and leaching which resulted in the formation of an enriched chalcocite blanket. A later stage of oxidation and leaching modified the blanket by oxidizing portions of it in place and mobilized some of the chalcocite to a greater depth.

Figure 8-2: Schematic Cross Section of the Secondary Enrichment Zonation of a Porphyry Copper Deposit and Typical Copper Minerals Present (After Asmus, 2013)



9.0 EXPLORATION

ASARCO geologists John Kinnison and Art Bloucher first identified the Sacaton mine area in early 1961 while performing regional mapping and sampling in and around the Sacaton Mountains. A lone outcrop of altered and weakly mineralized granite in a sea of alluvium was the only indicator of the potential for porphyry copper-type mineralization in the surrounding area. Following acquisition of mineral rights ASARCO conducted several geophysical surveys, including magnetics and induced polarization (IP). The IP survey identified a large area just south of the outcrop with a chargeability response indicative of sulfide mineralization. A modest drilling program was authorized and initiated in the fall of 1961.

The first drill hole was located just north of the discovery outcrop; intersecting approximately 50 ft (15 m) averaging close to 0.5% Cu. The next four holes were drilled south, east, and west of the first hole in the geophysical target area but did not hit significant results. The sixth and final budgeted drill hole (located to the northwest of the IP anomaly and the Discovery Outcrop) did intercept high grade mineralization—the discovery of the Sacaton West deposit. No further ground geophysics work was done at Sacaton by ASARCO. In 1962 through the first half of 1963 eighty-two more holes were drilled. These 88 holes outlined a northeasterly trending alteration zone approximately 4 miles (6.4 km) long and 1.5 miles (2.4 km) wide dominated by what was recognized as two potential ore bodies, the Sacaton West and East deposits, as well as widespread intercepts of copper mineralization throughout. Low copper prices precluded any further exploration drilling at that time.

Improving market conditions prompted ASARCO to continue exploration drilling in 1968 and 1969 leading to thirty-seven more holes being drilled. The additional information led to the decision to plan and develop the mine. An additional 10 holes were drilled (1970 and 1971) to sterilize areas under planned facilities. After mining was initiated in 1972, development and definition drilling was conducted for the open pit (Sacaton West deposit). Through 1974 and 1976, eight additional holes were drilled in the Sacaton East deposit for definition purposes.

The adjacent Parks/Salyer property has been variably explored from the 1970s through the late 1990s. Parks/Salyer is also a displaced portion of the larger porphyry copper system. A number of diamond holes drilled to the south of the then current resource area identified mineralization and geological characteristics consistent with the Cactus deposits in a similar horst block environment. Two exploration diamond drill holes were undertaken in 1996 by ASARCO at the southern edge of the current resource area (S-200 and S-201). As interpreted, they intersected well mineralized zones of oxide, enriched, and primary material that indicated grades were increasing to the north.

Arizona Sonoran conducted an ionic leach soil geochemistry program over the Parks/Salyer property in 2019 on 325 ft (100 m) spacing. This confirmed anomalous soil geochemistry across the property for copper, molybdenum, silver, and gold and a general NE trend of the higher anomalous values. Arizona Sonoran followed this work up with two diamond drill holes in 2020 (ECP-018 and ECP-019). This extended mineralization a further 900-1,000 ft (275 305 m) to the NE of previously drilled mineralization. Drilling resumed in late 2021 with hole ECP-042, continued throughout 2021 and into 2022 with the completion of ECP-096, resulting in 26 holes totaling 58,481 feet of HQ core.

Figure 9-1 plots the location and scale of the potential Parks/Salyer deposit with respect to the Cactus Mine deposits and highlights the significant intercepts defined by the four exploration holes drilled into the deposit on the property to date.

Figure 9-1: Location and Scale of the Potential Parks/Salyer Deposit with Respect to the Cactus Mine Deposits

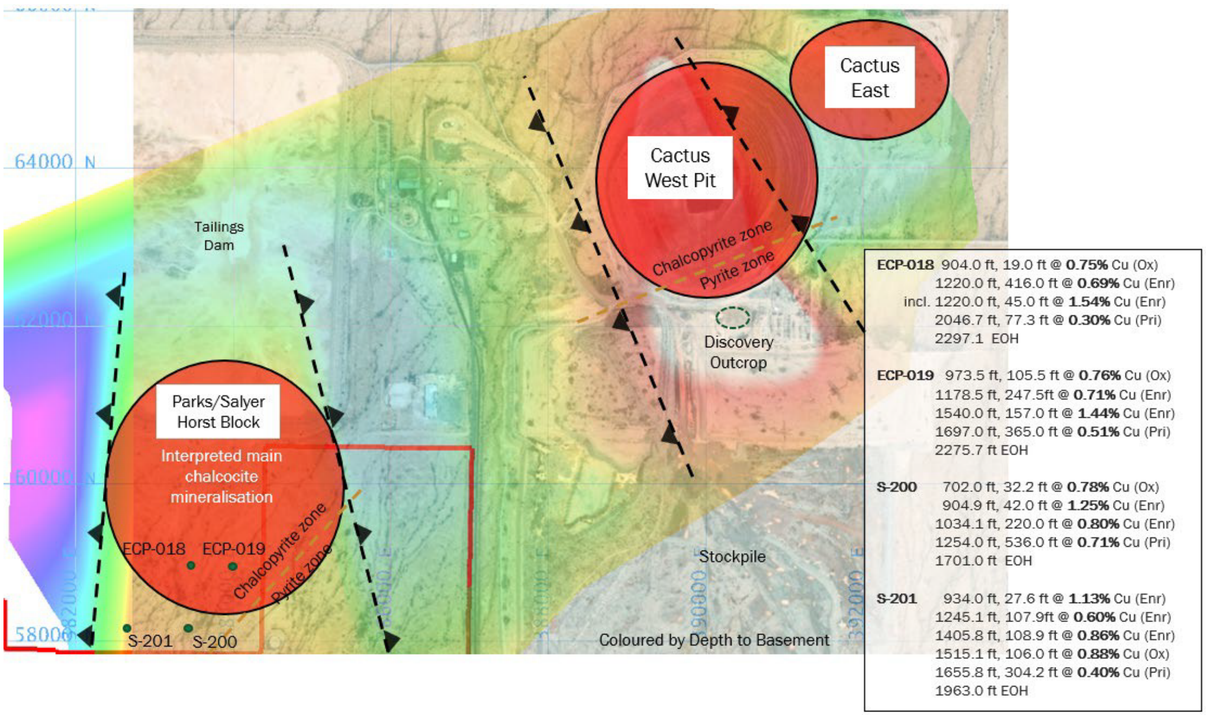
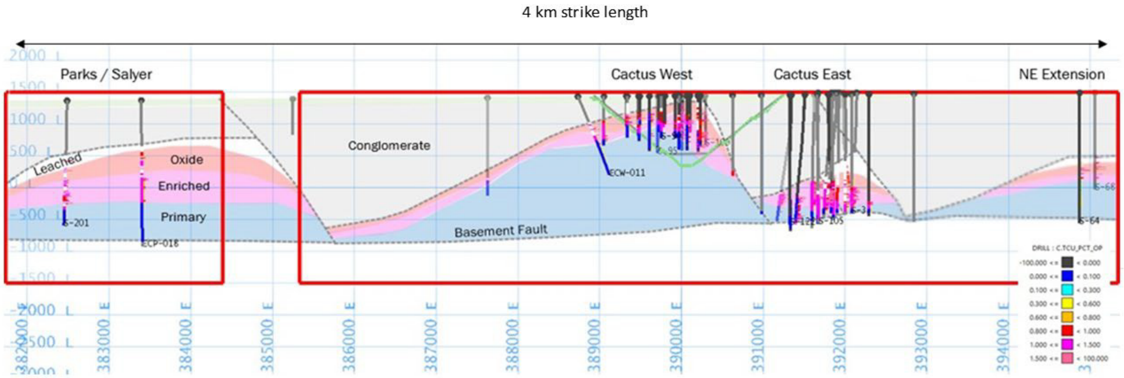


Figure 9-2 is a NE oriented long section displaying the horst and graben block fault and mineralization interpretation from the Parks/Salyer deposit in the SW through to the NE Extension mineralization in NE. NE movement along the basement fault was accommodated by block rotation and the formation of NW trending normal faults. The red boxes indicate Arizona Sonoran controlled property boundaries. The existing Cactus West pit is displayed on the long section.

Figure 9-2: NE Oriented Long Section Displaying Mineralization Interpretation and Property Boundaries



The NE Extension is located 3,000 ft (915 m) to the NE of Cactus East. ASARCO defined the mineralized zone with wide spaced exploration drilling (> 1,000 ft, 305 m) in 1962 and 1963 as part of the initial property wide exploration program. Table 9-1 reports the significant intercepts of the main holes drilled into the NE Extension mineralization. Arizona Sonoran has not performed any exploration programs on the NE Extension area to date.

Table 9-1: Significant Intercepts for the Two Main Holes Drilled into the NE Extension Mineralization

Holeid	From (ft)	To (ft)	Length (ft)	TCu (%)	Mineral Zone
S-68	1,016.5	1,044.5	28.0	1.27	oxide
	1,078.5	1,125.8	47.3	0.95	oxide
	1,161.0	1,208.8	47.8	3.05	oxide
	1,275.0	1,290.1	15.1	1.96	enriched
	1,322.4	1,354.1	31.7	0.97	enriched
	1,354.1	1,526.0	171.9	0.38	primary
S-64	1,093.9	1,104.2	10.3	1.01	oxide
	1,163.0	1,227.3	64.3	1.37	enriched
	1,333.7	1,350.9	17.2	0.89	enriched
	1,350.9	1,776.0	425.1	0.34	primary

Arizona Sonoran has focused their exploration by way of definition and expansion core drilling around the two known mineralized zones (now known as Cactus East and West). In 2019, two vertical PQ core holes were drilled into the East mineralized zone for verification of grade and for metallurgical testing as part of the evaluation program prior to purchase. One additional vertical PQ core hole was drilled into Cactus East in 2020 for further metallurgical testing, for a total of 5,768 ft (1,758 m). Five angled HQ core holes totaling 9,252 ft (2,820 m) were drilled in late 2019 and 2020 around the northern and western edges of Cactus East to define and expand mineralization. Also in 2020, 11 angled HQ core holes totaling 15,377 ft (4,687 m) were drilled around the perimeter of the West Pit to further define and expand Cactus West mineralization beyond the pit limits.

In 2019, 55 surface sonic drill holes totaling 5,120 ft of 6-inch diameter holes were drilled across the Cactus Stockpile Project to support an initial resource based on approximately 750 ft (229 m) spaced drilling. Through late 2020 and 2021, an infill surface sonic drill program was undertaken to reduce the spacing to 400 ft (122 m). The resource database for the Stockpile Project resource contains 210 holes. Drilling continues on the Project to reduce the spacing to 200 ft (61 m).

10.0 DRILLING

The Cactus (Sacaton) deposits are covered with post mineral alluvium and conglomerate, which may be up to 1,500 ft (457 m) thick. ASARCO rotary drilled through the cover alluvium and conglomerate and completed the remainder of the holes with NX/HX core tails. All ASARCO's drill holes, exploratory, and production holes, within the developing pit were drilled vertically and only a very few were down hole surveyed. Arizona Sonoran started a similar program in 2019 on the first two (PQ) metallurgy holes but converted to coring the full hole after unsatisfactory results. Core recovery, on average, was greater than 95%.

As detailed in Table 10-1, Arizona Sonoran completed a total of 20 core holes in the Cactus resource area in 2019 and 2020 for a total of 30,397 ft (9,265 m) of drilling. Table 10-2 details the 27 drillholes undertaken by Arizona Sonoran in the Parks/Salyer resource area in 2021 and 2022 for a total of 60,876 ft (18,555 m) of drilling. Figure 10-1 shows the location of the drilling relative to the Cactus and Parks/Salyer deposits with green and red circles locating the collars of Arizona Sonoran's recent holes, and white circles locating the collars of ASARCO's historical holes.

Of the 20 diamond drill holes completed in the Cactus area, 19 were used for the Cactus Mineral Resource estimates. All 27 holes completed in the Parks/Salyer areas were used for the Parks/Salyer Mineral Resource estimates.

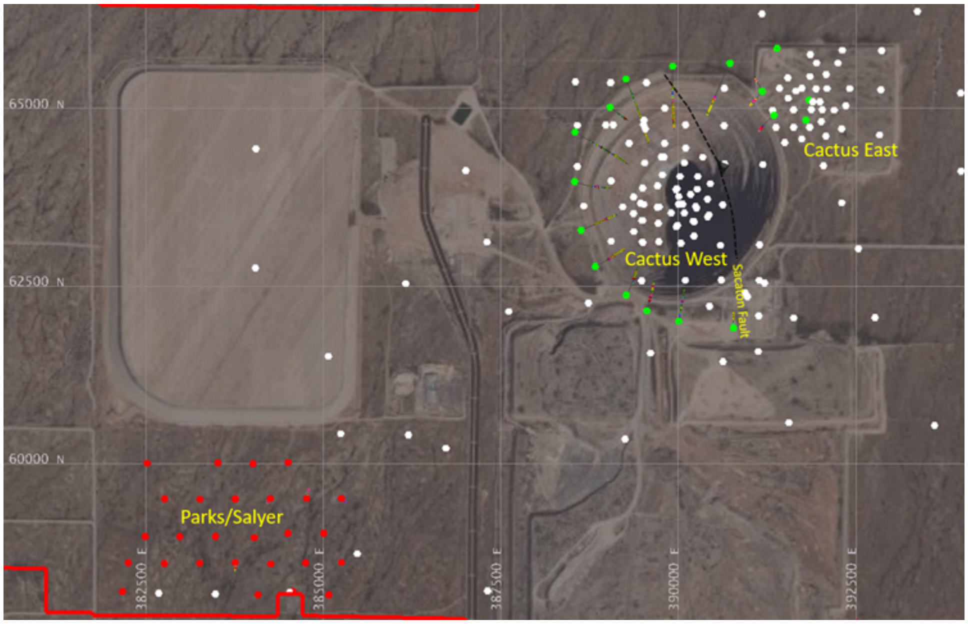
Table 10-1: 2019–2020 Drilling Completed by Arizona Sonoran

Drill Hole	Core	Total Depth (ft)	Total Depth (m)	Azimuth	Dip	Deposit
SE-01	PQ	2,058	627.3	0	-90	CE
SE-02	PQ	2,013	613.6	0	-90	CE
SE-03	PQ	1,697	517.2	0	-90	CE
ECE-001	HQ	1,896	577.9	220	-80	CE
ECE-002	HQ	2,013	613.6	230	-80	CE
ECW-003	HQ	1,936	590.0	180	-60	CW
ECW-004	HQ	500	152.4	0	-60	CW
ECW-005	HQ	664	202.4	129	-60	CW
ECW-006	HQ	1,000	304.8	10	-60	CW
ECW-007	HQ	1,811	552.0	123	-55	CW
ECW-008	HQ	1,000	304.8	20	-65	CW
ECW-009	HQ	906	276.1	30	-60	CW
ECW-010	HQ	1,469	447.8	110	-65	CW
ECW-011	HQ	1,329	414.2	60	-65	CW
ECW-012	HQ	1,459	444.7	65	-65	CW
ECW-013	HQ	1,616	492.6	205	-55	CW
ECW-014	HQ	1,687	514.2	160	-50	CW
ECE-015	HQ	1,723	525.2	0	-90	CE
ECE-016	HQ	1,783	543.5	0	-90	CE
ECE-017	HQ	1,837	559.9	0	-90	CE
Totals		**30,397**	**9,265.0**			

Table 10-2: 2021–2022 Drilling Completed by Arizona Sonoran

Drill Hole	Core	Total Depth (ft)	Total Depth (m)	Azimuth	Dip	Deposit
ECP-018	HQ	2,297.1	700.2	0	-90	PS
ECP-019	HQ	2,275.7	693.6	0	-90	PS
ECP-042	HQ	2,151.5	655.8	0	-90	PS
ECP-045	HQ	2,127.0	648.3	0	-90	PS
ECP-057	HQ	2,345.3	714.8	0	-90	PS
ECP-061	HQ	2,317.0	706.2	0	-90	PS
ECP-065	HQ	2,379.2	725.2	0	-90	PS
ECP-068	HQ	2,051.0	625.1	0	-90	PS
ECP-071	HQ	2,436.0	742.5	0	-90	PS
ECP-074	HQ	2,441.5	744.2	0	-90	PS
ECP-075	HQ	2,452.0	747.4	0	-90	PS
ECP-077	HQ	2,691.0	820.2	0	-90	PS
ECP-079	HQ	2,071.5	631.4	0	-90	PS
ECP-080	HQ	2,373.8	723.5	0	-90	PS
ECP-081	HQ	2,455.8	748.5	0	-90	PS
ECP-083	HQ	2,354.4	717.6	0	-90	PS
ECP-084	HQ	2,167.5	660.7	0	-90	PS
ECP-086	HQ	1,973.6	601.6	0	-90	PS
ECP-087	HQ	2,412.3	735.3	0	-90	PS
ECP-088	HQ	2,068.9	630.6	0	-90	PS
ECP-089	HQ	2,192.6	668.3	0	-90	PS
ECP-090	HQ	1,900.0	579.1	0	-90	PS
ECP-091	HQ	1,627.3	496.0	0	-90	PS
ECP-092	HQ	1,807.0	550.8	0	-90	PS
ECP-093	HQ	2,463.3	750.8	0	-90	PS
ECP-094	HQ	2,498.0	761.4	0	-90	PS
ECP-095	HQ	2,545.5	775.9	0	-90	PS
Totals		**60,875.8**	**18,554.9**			

Figure 10-1: Map of The Cactus and Parks/Salyer Deposit Areas Locating the Collar Locations of Arizona Sonoran's Recent Core Drill Holes (green and red) and ASARCO's Historical Drill Holes (White)



The Stockpile Project has been infilled drilled by Arizona Sonoran to 400 ft (122 m) spacing by sonic surface drilling since the initial 750 ft (229 m) spacing completed in 2019. This accounts for 206 holes in addition to 4 historical sterilization holes drilled into the barren alluvium dumps to the immediate north of the Stockpile Project.

10.1 Collar Surveying

The coordinates for the drill hole collars were determined using a Trimble R8 Model 2 Base and Rover GNSS GPS System, surveyed in Real Time Kinematic. Accuracy for this system is rated to be sub-centimeter. Post processing of baseline vectors are not required on Real Time Kinematic, however the data processing and preparation for delivery to the client was completed by Harvey Surveying using Trimble Business Center software. The report was delivered in Universal Transverse Mercator (UTM) Zone 12-grid projection. Units were measured in metric. The collar coordinates for the Parks/Salyer drilling used the same equipment and methodology that was used and Cactus East and West.

10.2 Downhole Surveying

All Arizona Sonoran's diamond drill holes for the Cactus Project, including vertical drill holes, have downhole surveys completed by the drill contractor using either a Reflex EZTRAC XTF magnetic survey instrument or a Reflex EZGYRO MEMS gyroscopic survey instrument.

Surveys were taken nominally every 100 ft (30.5 m) while the hole was being drilled. The downhole surveys completed for each of the holes at Parks/Salyer used the same Reflex Gyroscopic instrumentation and methodology.

All drill holes for the Cactus Stockpile Project were drilled vertically. Due to the depth of holes averaging approximately 80 ft (24.4 m), downhole surveys were not deemed necessary.

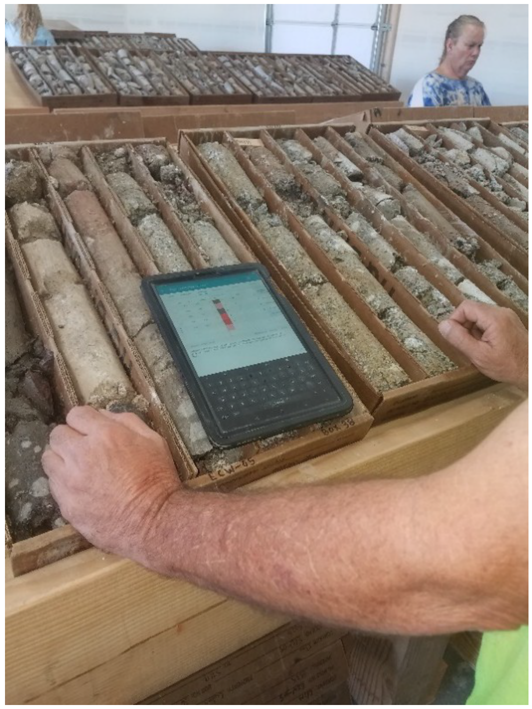
10.3 Core Logging and Photography

Core logging was performed in Arizona Sonoran's core shed at the Project site. Drill core was delivered to the core shed by the drillers at the end of each drill shift. The following preparation and logging processes were performed on the core.

- The core was given a final cleaning.
- Core boxes were marked for identification / verification of footages.
- Core boxes were photographed.
- Point-load testing was performed.
- Geological characteristics of the core such as lithology, copper mineralogy, brecciation, alteration, and oxidation were logged.
- Geotechnical characteristics of the core such as core recovery, rock quality designation (RQD), fracture frequency, and joint types were logged.
- Two holes (one in Cactus West and one in Cactus East) were drilled with oriented core. For these holes, structures were measured for orientation data and the information was logged into the database.

Data logging of all core characteristics is performed digitally on Galaxy S5e tablets that write directly into the cloud-based MX Deposit drill hole database when internet connection is available. When internet connection is not available, holes are locked by the logging geologist who can then log the hole offline. Locking out of the hole ensures two geologists cannot edit the same hole at the same time. Once internet is available, the logging information is uploaded to the database. In addition to the digital table view of the database for logging, a visual strip log view is used to review logging. Figure 10-2 is a photograph of Cactus core and the tablet used for logging. Note, the visible strip log as data is entered along the hole.

Figure 10-2: Cactus Drill Core with Logging Tablet



Core sample intervals are determined by the logging geologist based on logging characteristics. Sample interval breaks are determined by geological parameters, but within core containing the same geological characteristics, samples are undertaken on a regular 10 ft (3.28 m) sample length. Each sample interval is defined as follows.

- Sample interval is marked at its beginning in the core box with the interval and a unique sample identification number.
- The sample number is taken from a tag book of sequential sample cards to ensure duplicate samples cannot be produced. The sample tag is stapled into the box at the sample start location.
- A twin sample tag is stapled to a clean sample bag to collect the sample when it is split and then will be sent to the lab.
- Interval information for the hole Identification, and from/to depths is written in the tag book.
- The logging geologist enters the same from/to intervals directly into the sample logging table of MX-Deposit for the drillhole being logged.

All cores sampled were split into two equal portions along the long axis of the core, using either a diamond saw or a hydraulic blade splitter. One half of the split core is placed into the sample bag marked with that sample's unique sample number. The bagged samples are placed in a shipping tote for transport to the analytical lab in Tucson. The other half of the split core is placed back in the core box and is archived in Arizona Sonoran's secure core storage room located at the Cactus site. Figure 10-3 is a photograph of the rock saw used to split core at the Cactus Project core shed. Figure 10-4 is a photograph of a box of sawn core to be stored at the Cactus Project core shed.

Figure 10-3: Cactus Project's Rock Saw and Hydraulic Splitter

Figure 10-4: Sawn and Split Core to be Stored

For the Cactus Stockpile Project, sonic drill holes are logged for main material type, lithologies, color, iron oxide minerals, copper minerals, and clast size distribution. Data logging of all characteristics is performed digitally on Galaxy S5e tablets that write directly into the cloud-based MX Deposit drill hole database and use the same lockout version control features as the Cactus Project Deposit logging. Cactus Stockpile Project drill holes are managed in a separate database activity to the Cactus Project deposit drill holes.

All Stockpile Project samples are collected at the drill in plastic tubing at regular 2.5 ft (0.7 m) intervals. After logging, each sample interval is placed into a new sample bag with a unique sample number unrelated to drill hole number or drill interval in a manner similar to that described for core samples.

10.4 Qualified Person Opinion

The QP reviewed the survey methodology and results of the drill hole location and down hole data for historical and recent drilling on the Cactus Project. The QP also reviewed abnormal grades within the mineralized zone to ensure they were based on visible mineralization.

Individual high grades were dealt with in the capping grades as explained in Section 14.1.5.

The drill recovery has been consistently above 95%, with good control of sample location with the downhole survey program. The QP feels that the drilling results of the In Situ mineralized zones and the stockpile resource meets the expected standards and best practices as defined in CIM's Best Practices Guidelines 2019. The drill hole spacing and sample location data meets the level of accuracy expected for this PEA report.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sample Preparation

Arizona Sonoran has been exclusively using Skyline Assayers and Laboratories (Tucson, Arizona) for their sample prep and analysis. This laboratory is accredited in accordance with the recognized International Standard ISO/IEC 17025:2017, Certificate #2953.01. This accreditation demonstrates technical competence for a defined scope and the operation of a laboratory quality management system. The QP has visited this lab to review the procedures used for sample preparation, analysis, and the lab's internal quality assurance (QA) / quality control (QC) system.

The lab dispatches drivers to pick up samples at the mine site when they are informed there is a full shipment ready. Upon arrival at the lab, totes were offloaded and stored. When the samples were ready to be processed, the bags were emptied into metal bins and the sample bags with tags placed on top. The bins and bags were placed in an oven at 220 ºF (105 ºC) for 24 hours to dry before moving into the lab for processing.

Each sample was crushed in a TM Engineering – Terminator roll crusher to 95% passing ¼ inch. This material was passed through a riffle splitter and mixed three times to ensure homogeneity of the sample. Three-quarters of the sample was then bagged, labelled, and returned to Arizona Sonoran as coarse reject. The remaining material was returned to the roll crushers and crushed to 95% passing -10 mesh. A 280-gram sample of this material was put in a Labtech LM2-P puck pulverizer and run to 95% passing -150 mesh. This sample was placed into labelled heavy paper envelopes and sent to the lab for assay.

11.2 Sample Security

Bagged samples with identification tags are placed in large 3 ft (1.0 m) square plastic totes which are stored at the core shed which is within the secured mine site away from any point of access. Arizona Sonoran uses a private contractor to transport the samples totes to the lab. When 8 to 10 totes are filled, the contractor is called to make a pickup. A transmittal sheet is prepared that lists all the samples in the shipment with an assay order sheet for the analysis to be done. A chain of custody sheet is signed by Arizona Sonoran upon dispatch, signed by Skyline Labs upon arrival, and returned to Arizona Sonoran to show secure delivery.

11.3 Sample Analysis

As a first pass each sample was assayed for CuT. The pulverized samples were received from sample prep and a measured portion of the sample was digested in a mix of hydrochloric acid (HCl), nitric acid (HNO_3), and perchloric acid ($HClO_4$) on a hot plate for 15 minutes to 20 minutes. The sample was left to cool, rinsed with distilled water, and then digested in HCl for an additional 15 minutes on a hot plate. The sample was then cooled and sent to atomic absorption (AA) analysis to return a CuT value.

To support potential heap leaching for metal recovery, a sequential acid leach assay procedure was conducted on each sample. These samples were first run using a digestion in 5% sulfuric acid (H2SO4) for 1 hour on a shaker table, then 15 minutes in a centrifuge before the liquid was transferred to a 250 mL flask. The residue was rinsed, and that liquid was used to top up the flask. The flask was sent to the assay lab for AA analysis to return an CuAS value.

The residue from the centrifuge was then digested in 10% sodium cyanide (NaCN) for 30 minutes on a shaker table. After 15 minutes in the centrifuge, the liquid portion was transferred to a flask and the residue was rinsed and that liquid used to top off the flask. That sample was sent to the assay lab for AA analysis to return a CuCN value. The remaining pulverized sample in the heavy paper envelope was returned to Arizona Sonoran together with the coarse reject.

11.4 Lab Quality Assurance / Quality Control

Skyline Assayers and Laboratories is accredited in accordance with the recognized International Standard ISO/IEC 17025:2005. Their quality management system has been certified as conforming to the requirements defined in the International Standard ISO 9001:2015. The standard operating procedure (SOP) used while processing the Arizona Sonoran samples was to process samples in groups of 20. Each tray consisted of 18 samples with samples No. 1 and No. 10 repeated as duplicates. The results from each tray were analyzed and any variance in the duplicates of more than 3% would result in the entire tray being re-assayed.

The results of these analyses, including the QA / QC checks, were transmitted to a select set of individuals at Arizona Sonoran and Stantec.

11.5 Qualified Person Opinion

The QP for Section 11 has reviewed the assay lab’s procedures and QA/QC results in detail and finds that it meets all of the expected standards and best practices as defined in CIM’s Best Practices Guidelines 2019. The assay results and associated data meets the level of accuracy expected for this PEA report.

12.0 DATA VERIFICATION

The bulk of the Cactus drilling database was rebuilt from historical drilling logs and assay certificates from exploration work undertaken by ASARCO. Since 2019, Arizona Sonoran has drilled 20 new holes at the Project to support verification, metallurgical testing, and resource extension for the new Cactus mineral resource estimate. The Parks/Salyer resource database holes are composed primarily of 27 new holes drilled by Arizona Sonoran between 2021 and 2022. There were only four historical holes supporting the Parks/Salyer resource estimate.

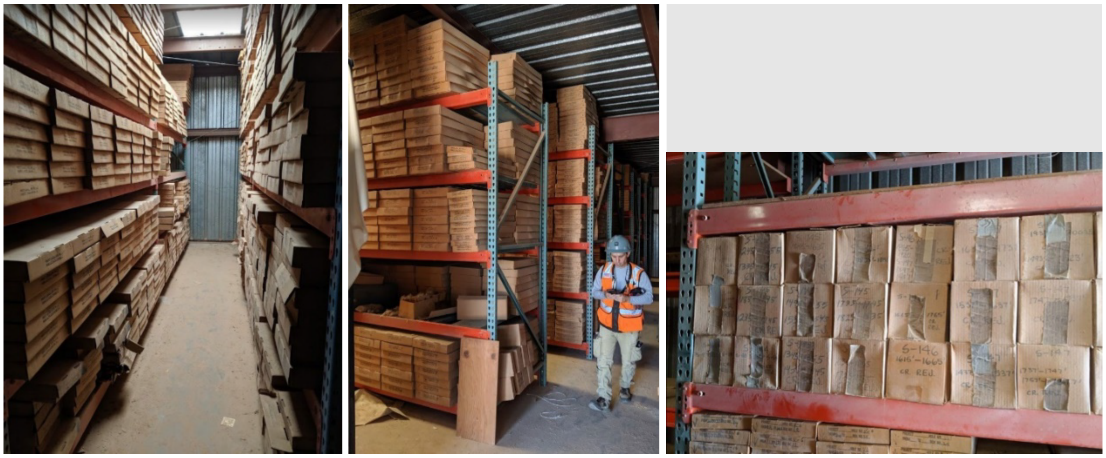
12.1 Historical Asarco Exploration Data

Two core sheds (Figure 12-1) were located at the Project that stored the historical drill core and sample pulps from ASARCOs exploration programs. This physical data verified the historical data quality and its use in the new mineral resource statement. While modern assay QA/QC procedures have evolved significantly, there is evidence in the historical records that ASARCO was using best practices of the day. In addition to these procedures, ASARCO ran a series of pulp duplicate checks against their regular laboratories to test assay quality.

Specific data verification work undertaken by Arizona Sonoran for the historical drill holes included the following.

- Verification of the collar locations.
- Reinstatement of downhole survey data drilled into the Cactus East deposit.
- Verification of drill hole locations and geological interpretations against historical cross sections and pit maps.
- Relogging of historical drill hole lithology, copper mineral zones, and alteration.
- Re-assaying of historical pulp samples to compare CuT grades and establish soluble copper contents confirming expected copper mineral zones and leachable copper mineralogies.

Figure 12-1: Onsite Core Shed with Historical Core and Pulps



12.2 Historical Collar Locations

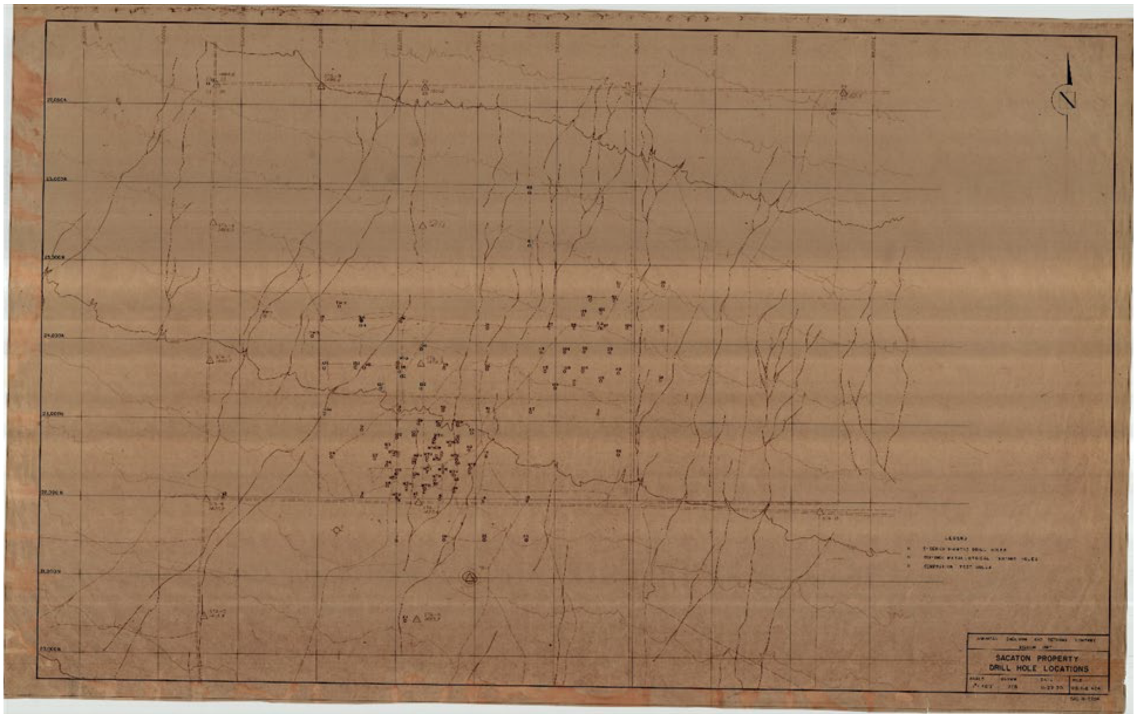
Historical collar locations were verified through the identification of historical survey control and field survey pickup. A final ASARCO control document entitled Sacaton – Drill Hole Files and Information produced in 1998 by Bret S. Canale was located. A page from this document detailed the final collar survey coordinates for all Sacaton drill holes and the aerial control survey points for the property (Figure 12-2). The coordinates were specified in two local grids: the Santa Cruz coordinate system and the Sacaton coordinate system. The Sacaton coordinate system was used for all drilling and mapping information related to the Cactus deposits. In addition to this document, a survey control map (Figure 12-3) was located at site that detailed the location of the historical drill holes and survey control points spatially and in conjunction with site locations such as land sections and the discovery outcrop. From this information, new survey control could be established from the known historical locations in the field to tie the historical local grid coordinates to a modern grid system.

Figure 12-2: Survey Control Points Reported in the Sacaton Coordinate System

SACATON AERIAL CONTROL SURVEY POINTS
(SANTA CRUZ and SACATON COORDINATE SYSTEMS)

POINT	SC COORDINATES		SAC COORD'S		ELEV
	NORTH	EAST	NORTH	EAST	
EPNG	74531.48	75292.07	13854.48	27634.07	1375.70
NW26	87927.37	67342.53	27250.37	19684.53	1494.60
NW27	87939.34	62020.02	27262.34	14362.02	1458.20
S1/4,23-26	87914.55	69986.19	27237.55	22328.19	1501.60
S1/4,24-25	87896.00	75289.69	27219.00	27631.69	1521.50
ST-1	86153.25	69970.84	25476.25	22312.84	1477.20
ST-2	84394.09	69955.02	23717.09	22297.02	1458.90
ST-3	82636.49	69938.42	21959.49	22280.42	1437.40
ST-4	81137.84	69925.26	20460.84	22267.26	1423.30
ST-5	81151.31	67225.64	20474.31	19567.64	1413.50
ST-6	82648.26	67249.65	21971.26	19591.65	1427.90
ST-7	84408.50	67283.70	23731.50	19625.70	1446.70
ST-8	86168.19	67313.65	25491.19	19655.65	1468.00
ST-9	87915.70	68664.83	27238.70	21006.83	1499.60
ST-10	82680.06	62049.26	22003.06	14391.26	1408.50
ST-11	74556.21	61913.04	13879.21	14255.04	1340.00
ST-12	74553.97	68554.80	13876.97	20896.80	1361.70
ST-13	82555.18	75016.98	21878.18	27358.98	-
TRI-1	81669.50	70588.64	20992.50	22930.64	-

Figure 12-3: 1970 Survey Control Map

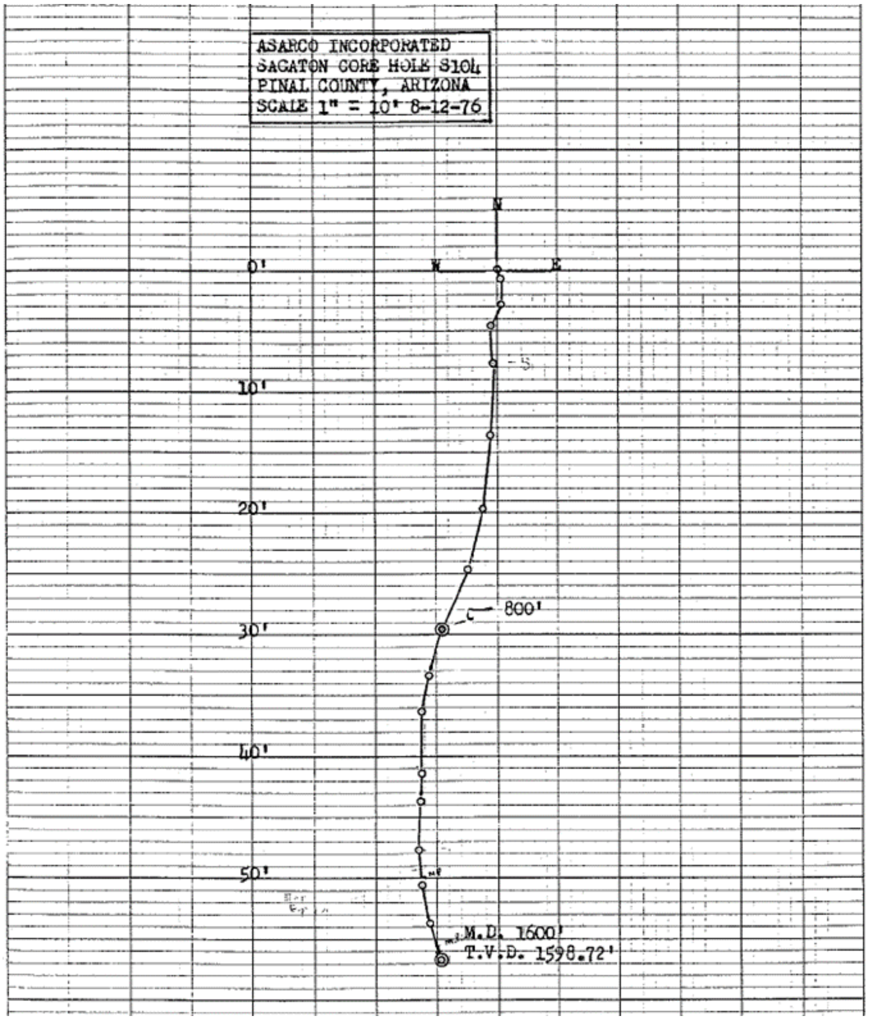


As a cross validation of this work, historical drill hole collars were located in the field and their collars were surveyed by differential GPS (DGPS). There were holes which could not have their collar surveys checked due to their location being within the mined pit extents or under alluvium dumps. The consistency of the field collar locations and historical collar coordinates for those that could be located, and consistency of historical drill hole locations against historical cross sections and pit maps, confirmed that collars that could not be verified in the field, are correctly spatially located.

12.2.1 Historical Downhole Survey Data

In the Cactus East deposit, deep vertical holes were drilled. In some cases, the holes deviated significantly as a function of depth and local drilling conditions. The downhole survey data was plotted on downhole survey plots (Figure 12-4). Using Vulcan software, the plots could be remapped into 3D and the downhole survey data reinstated. From these strings, downhole surveys were created so that the drill holes were plotted correctly in three dimensions. Holes were then compared against historical cross sections to confirm downhole survey data had been reinstated correctly. The following holes from CE contained historical downhole surveys – S-49, S-98, S-99, S-104, S-108, S-113, S-118, S-121, S-123, S-137, S- 138, S-139, S-140, S-142, S-145, S-146, S-147, and S-149. All other historical holes within CE and all historical holes within CW were drilled vertically and contain no downhole surveys.

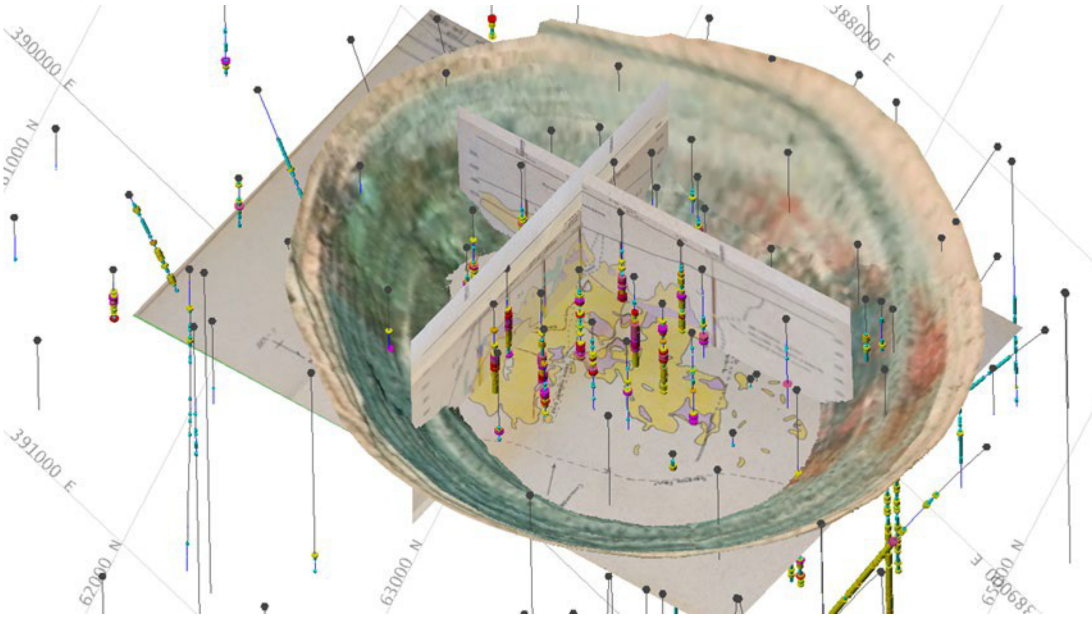
Figure 12-4: Example Downhole Survey Plot for Hole S-104



12.2.2 Comparison Against Historical Maps

ASARCO compiled a dataset of maps and cross sections to interpret the geology of the Cactus deposits. This information provided support to the verification of historical drilling information, fault interpretation, and copper mineral zonation modelling (Figure 12-5). The consistency of independent datasets to correlate with one another and the in-pit geology that can be observed in the field, provided verification that data were well located spatially and supported the deposit style and characteristics. The addition of Arizona Sonoran's 22 modern drill holes provided further confirmation that the geological model, historical data, and modern data were consistent with one another.

Figure 12-5: Three-Dimensional View of the Cactus West Pit, Facing Southwest



12.2.3 Relogging of Historical Core

Arizona Sonoran used the MX-Deposit drill hole database software to relog historical drill holes within the Cactus West, Cactus East, and Parks/Salyer deposits. Holes were logged digitally on a tablet, directly into the drill hole database, or where internet connection was not available, onto the tablet for later uploading to the drill hole database. Holes being logged are locked when offline, so two people cannot log the same drill hole at the same time. There were two objectives to the relogging effort of historical drill holes.

- To re-instate logging of drill holes where historical drill core exists, but no historical log was present.
- To re-log historical holes to ensure consistency of the logging process.

The logging processes used by ASARCO historically were very similar to the logging processes used by Arizona Sonoran. Areas of focus in the geological logging were lithology, copper mineral zone, alteration, and oxidation. Where historical and modern logs were undertaken, there was consistency between the two sets of logs, particularly for the critical areas with respect to resource modelling and metallurgy such as the copper mineral zones.

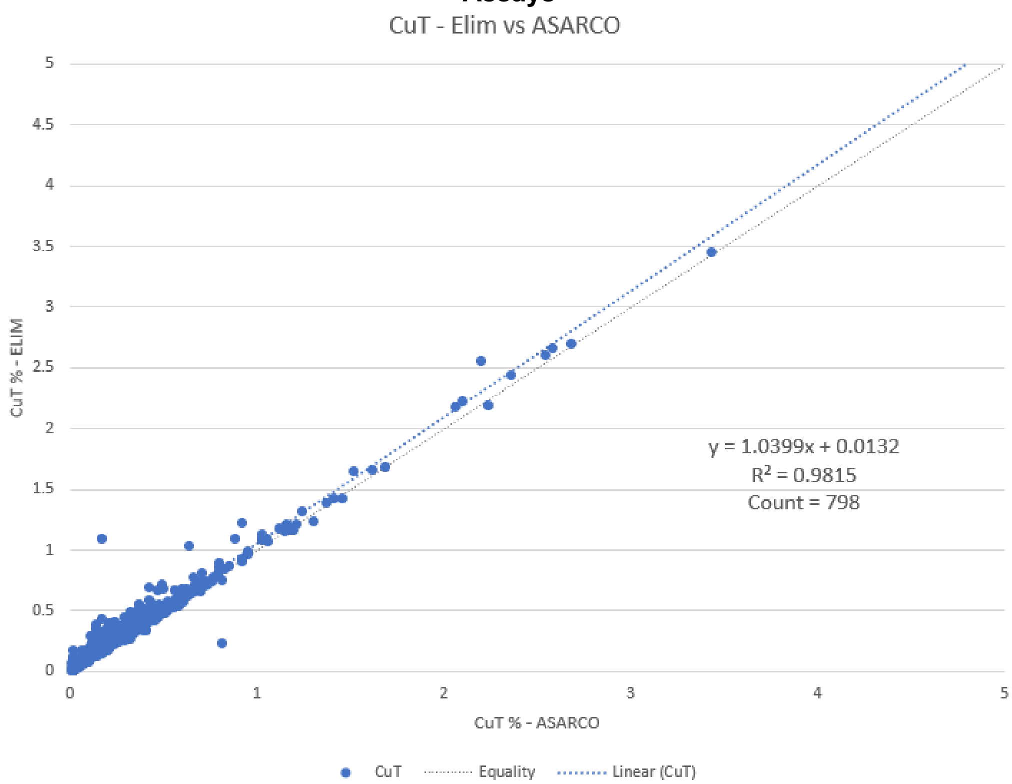
12.3 Re-Assaying of Historical Pulps

The historical core and pulp samples provided the opportunity to verify the historical assay results as follows.

- Historical pulps were re-assayed to enable comparison of the CuT assays against the historical CuT assay results. In some cases, where historical assays did not exist, the re-assays provided the opportunity to reinstate this data.
- Historical pulps were re-assayed with sequential copper analyses to measure the TSol copper present representing oxides and supergene sulfides. In addition to TSol copper, sequential assays for acid soluble and cyanide soluble results supported the determination of the copper mineral zones into oxide, enriched, and primary.
- Historical core was re-assayed where historical pulps were not present, or where core had not been historically sampled. This occurred rarely but did occur in oxide zones due to ASARCO's focus on sulfide zones to support mill flotation.

There were 798 re-sampled pulps to compare against the historical ASARCO assay results for CuT. The scatter plot in Figure 12-6 shows this comparison and confirms a strong correlation between historical CuT assays and modern re-assays of the pulps (correlation coefficient = 0.98). This supports the use of historical assays in the new mineral resource estimate.

Figure 12-6: Historical ASARCO Total Copper Grades against Modern Arizona Sonoran Pulp Re-Assays

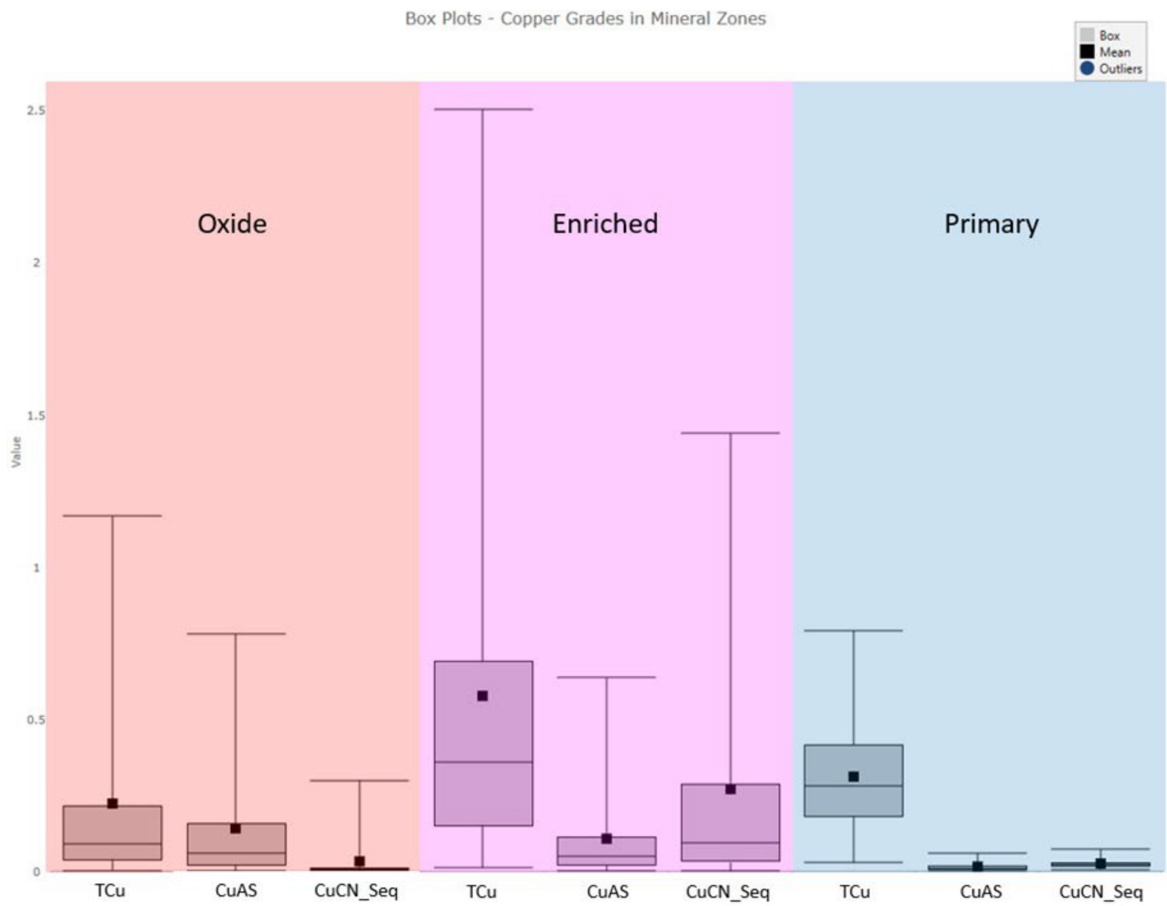


ASARCO did not undertake the same level of QA/QC with blanks, standards, and duplicates compared to current industry best practices. However, there is evidence in the historical records of significant pulp duplicate analysis and comparison work being undertaken.

ASARCO's procedures and assaying methodologies would have been considered industry best practice for that time in that deposit style.

The addition of the re-assay dataset, inclusive of sequential copper acid soluble and cyanide soluble assay results, provided a check against the modelling of the copper mineral zones to ensure mixing of mineral types or the presence of significant transition zones of mixed mineral types was understood. Figure 12-7 shows box plots of the main copper mineral zones and the makeup of the soluble copper distributions within them. The results support the logging, that mineral zones mostly transition rapidly and that there are not considerable zones of transitional or mixed mineralogies.

Figure 12-7: Box Plots for the Copper Mineral Zones



In the oxide zones, the CuT is mostly made up of CuAS grade as expected due to the presence of highly soluble oxide minerals. In the enriched copper zones, the CuT is mostly made up of cyanide soluble copper grade (CuCN-Seq) as expected due to the presence of chalcocite and covellite supergene enriched sulfides. In the primary zones, the CuT is not made up of either of the soluble copper grades as expected due to the presence of low solubility chalcopyrite. This provides verification of the logging and modelling of the copper mineral zones with the historical and modern drill holes.

12.4 Recent Drilling

For the 20 new Cactus drill holes, 27 new Parks/Salyer drillholes, and 206 new Stockpile Project drill holes undertaken by Arizona Sonoran since 2019, physical checks on collar, downhole survey, and logging have been completed by the QP.

12.4.1 Collar Location Checks

Collar locations were picked up in the field by DGPS and the coordinates imported into the MX-Deposit drill hole database by CSV file. Collar coordinates were independently field checked by the QP on a site visit at the end of the drilling program to ensure surveyed collar coordinates matched their field locations. Visual inspection by the QP confirmed that the drill holes were located as shown in the drilling database. This was also confirmed with a handheld GPS.

12.4.2 Downhole Surveys

All modern drill holes, regardless of the drill angle or depth, are surveyed with a Reflex EZTrac XTR instrument for their downhole deviation. Downhole surveys were reviewed by the QP against the designed survey and in the field for the collar survey orientation. A review of the downhole survey data for a few of the early holes drilled in Arizona Sonoran's 2019 / 2020 drill campaign revealed that magnetic declination had been improperly applied. This was fixed in the affected holes. The entire database was reviewed to ensure that the error did not occur elsewhere. The database was found to be correct.

12.4.3 Core Logging

All modern drill holes are logged for lithology, copper minerals and mineralization, alteration, oxidation, brecciation, and geotechnical attributes. Logging is viewed in three-dimensional software to confirm consistency with surrounding drilling and the geological interpretation.

Once assays are attained, results are compared back against the logged copper mineral zones to ensure consistency and as continuous improvement of the logging process.

The QP reviewed specifically requested drill holes to confirm logging and assays against the physical core. Three pseudo-random drill holes were selected, as each had intervals that were inconsistent in comparison to intervals on either side.

The first reviewed drill hole contained an interval with a comparatively high CuAS assay. It was explained by a zone of near massive malachite and other copper oxides.

The second reviewed drill hole contained high grades in a dacite dyke. Visual inspection revealed the presence of significant covellite mineralization.

The third drill hole reviewed contained high grades over a narrow zone. This occurred on the contact between the oxide and the enriched zone which typically contains the highest grades intercepted within the enriched zone.

All the pseudo-random checks of drilling showed compliance with logging.

12.4.4 Drill Hole Database Checks

In addition to validation checks performed in the MX Deposit drill hole database, specific drill hole database checks are undertaken on the Vulcan ISIS drill hole database to be used for the resource estimate. Checks that were undertaken and passed were as follows.

- All drill hole collars had a unique collar location.
- No collar end of hole depth was less than individual intercept depths logged within the hole.
- There were no overlapping from/to intervals in any table.
- All fields (including depths) that should increase between records were increasing.
- All hole IDs and sample IDs were unique.
- All assay grades were within expected tolerance ranges.
- All mandatory critical fields were populated in the database (e.g., easting, northing, elevation, total depth, from, to, azimuth, dip, and assay values).

12.5 Sample Quality Assurance / Quality Control

For the 20 new Cactus drill holes, 27 new Parks/Salyer drillholes, and 206 new Cactus Stockpile Project drill holes undertaken by Arizona Sonoran since 2019, and the re-assay program undertaken on historical pulps, a modern QA/QC program was undertaken composed of blanks and standards. Pulp duplicates were discussed earlier with respect to historical pulp samples and will feature in future programs on modern pulp samples.

12.5.1 Standards

Site-specific standards were created from onsite samples. The following standards were created, with the specific purpose of characterizing the mineral and grade characteristics of the Cactus deposits. Figure 12-8 shows the standards in use and the certified results attained from independent round robin testing for CuT grade. The main standards created are as follows.

- R-Blank – unmineralized rhyolite blank acting as a waste standard.
- OX-1 – oxide standard.
- EN-H, EN-M, EN-L – enriched standards of high, medium, and low grades.
- PR-H, PR-M, PR-L – primary standards of high, medium, and low grades.

Figure 12-8: Arizona Sonoran Drilling Program Standards and Certified Values

CRM Code	Sample Decomposition	Analytical Method	Element	Unit	Certified Values	Standard Deviation	95% Confidence	Minimum Value	Maximum Value
R-Blank	AD	ICP	CuT	%				0	0.015
OX-1	AD	ICP	CuT	%	0.725	0.043	0.173	0.683	0.818
EN-H	AD	ICP	CuT	%	1.958	0.074	0.295	1.72	2.109
EN-M	AD	ICP	CuT	%	0.978	0.021	0.082	0.913	1.02
EN-L	AD	ICP	CuT	%	0.417	0.018	0.073	0.388	0.465
PR-H	AD	ICP	CuT	%	0.787	0.055	0.221	0.675	0.911
PR-M	AD	ICP	CuT	%	0.52	0.025	0.099	0.475	0.579
PR-L	AD	ICP	CuT	%	0.336	0.016	0.066	0.304	0.384

Standards were inserted into the sample stream to test for precision of the lab to replicate an expected assay value. Standards were inserted in the sample stream at a rate of 1 per 20 samples or 5%.

Figure 12-9 through Figure 12-24 show the performance of the standards across both the drilling and re-assay programs supporting the new mineral resource estimates at Cactus West and Cactus East, Parks/Salyer, and the Stockpile Project. All figures plot the expected mean grade (black line), and dashed lines representing values two standard deviations above (max.) and below (min.) the mean. Orange squares represent the CuT grade to which the mean and two standard deviation values relate. Blue squares represent the CuAS grade and red squares represent the cyanide soluble sequential copper grade as a measure of the consistency of the measurement of the TSol copper grade contents. In all cases, the assayed CuT grades were within expectations.

Figure 12-9: Oxide Standard (OX-1) – Cactus

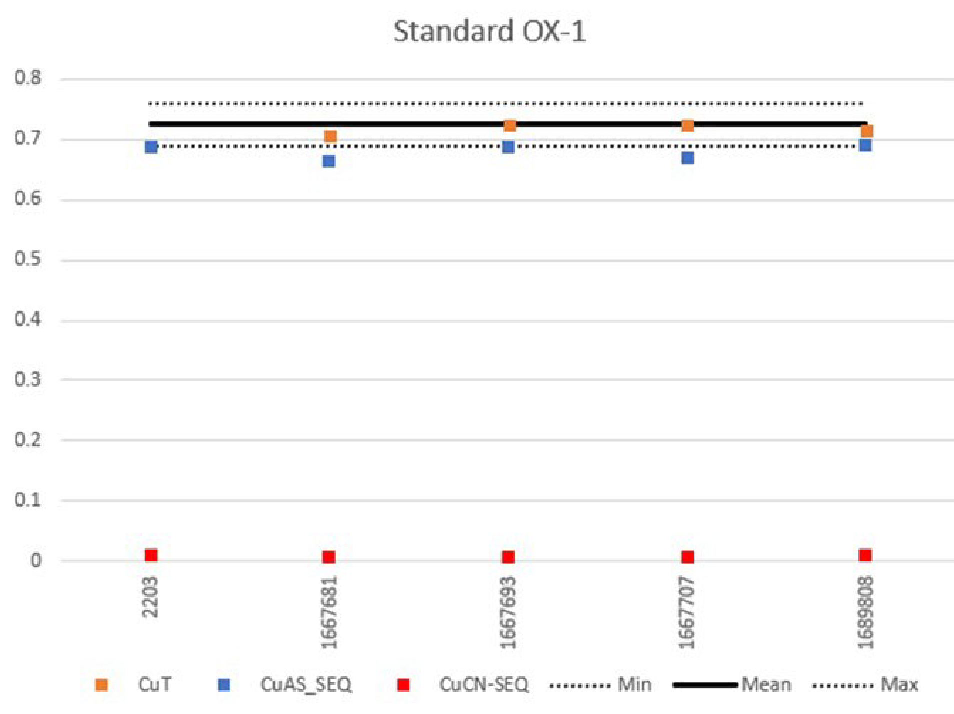


Figure 12-10: Oxide Standard (OX-1) – Parks/Salyer

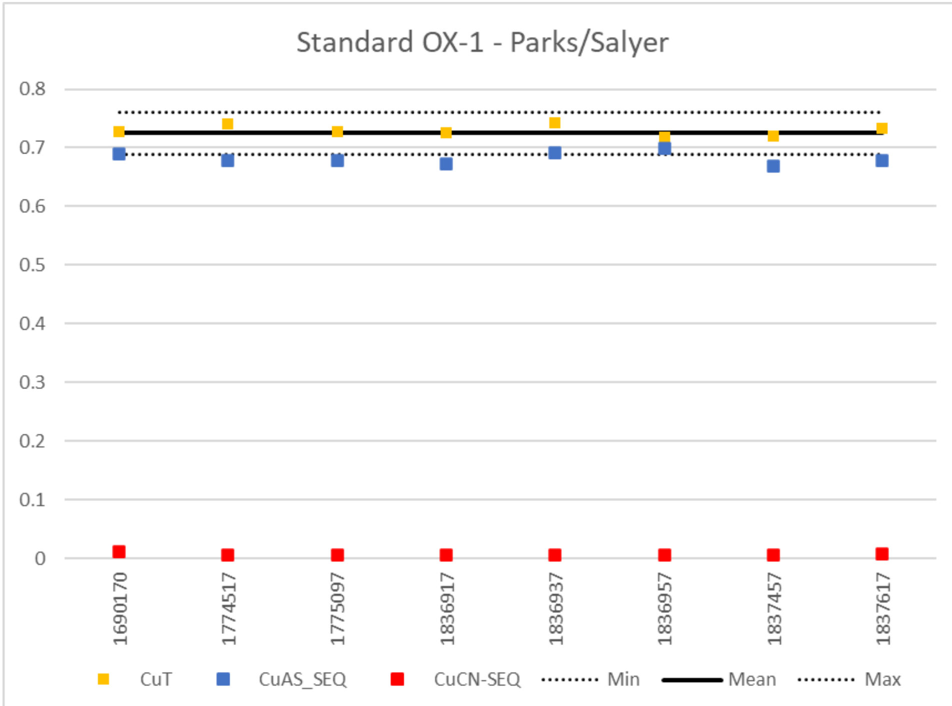


Figure 12-11: Oxide Standard (OX-1) – Cactus Stockpile Project

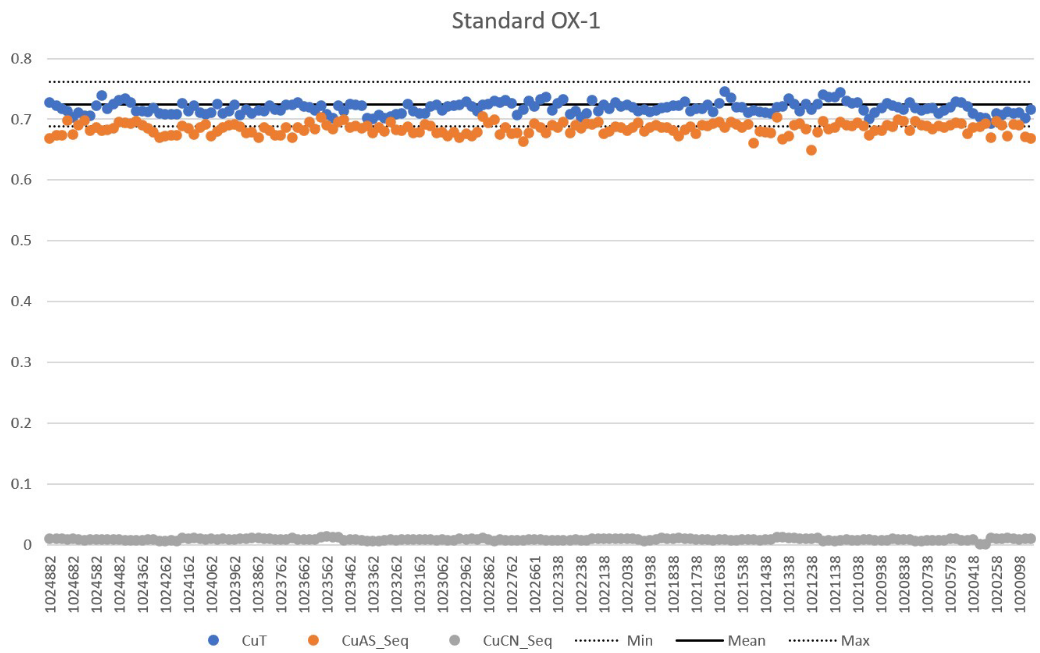


Figure 12-12: High Grade Enriched Standard (EN-H) – Cactus

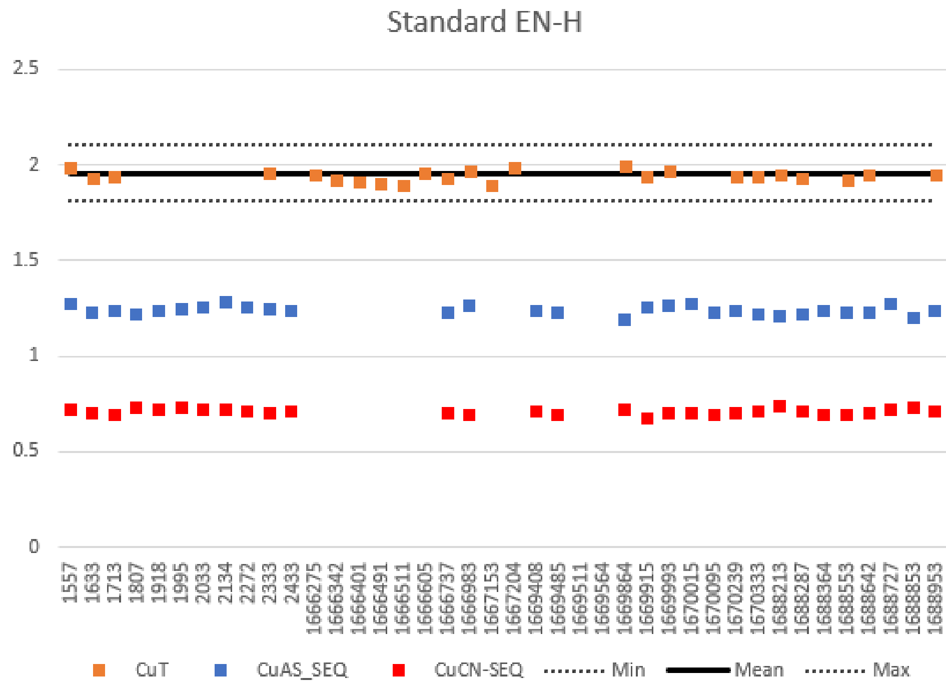


Figure 12-13: High Grade Enriched Standard (EN-H) – Parks/Salyer

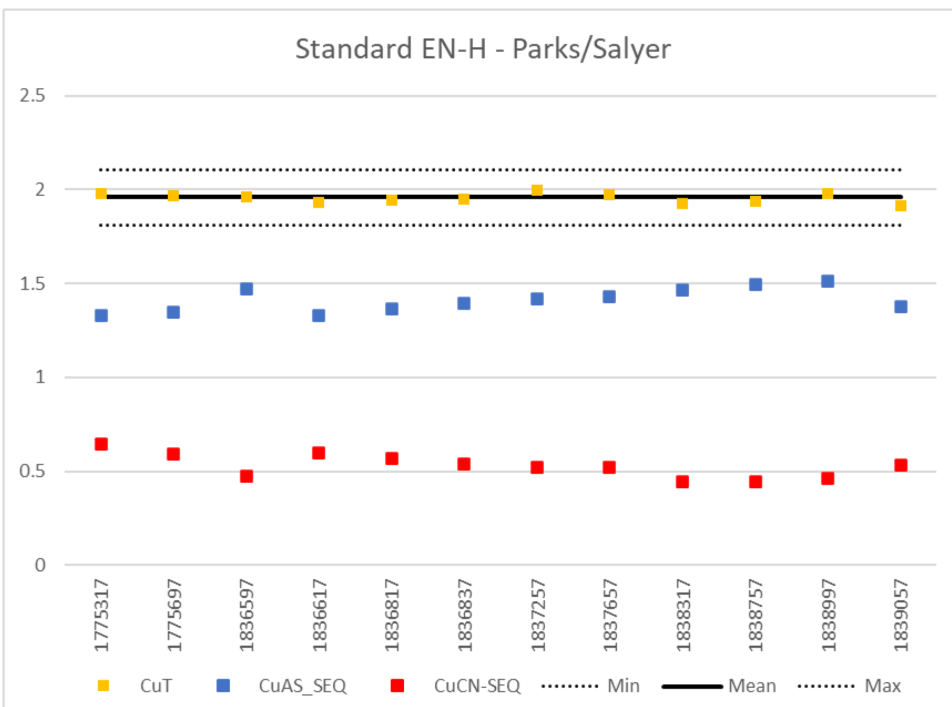


Figure 12-14: Medium Grade Enriched Standard (EN-M) – Cactus

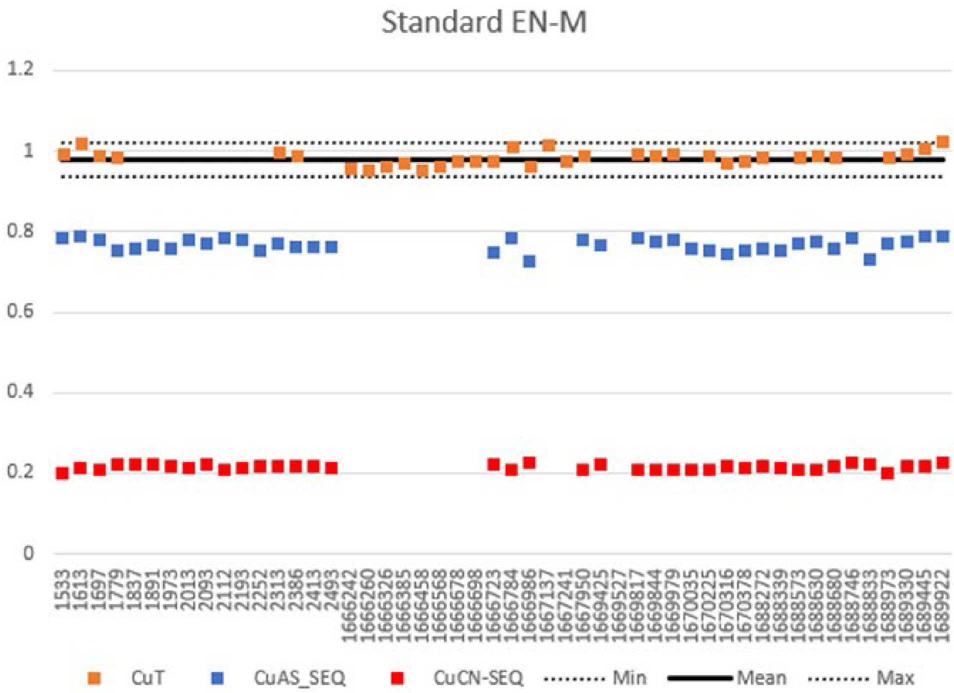


Figure 12-15: Medium Grade Enriched Standard (EN-M) – Parks/Salyer

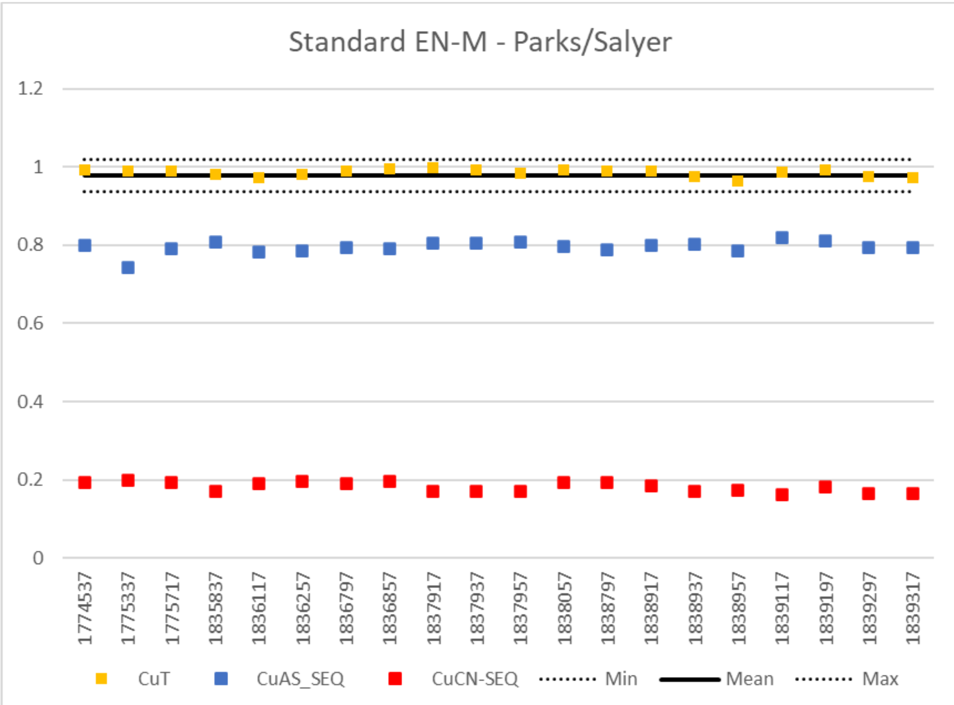


Figure 12-16: Low Grade Enriched Standard (EN-L) – Cactus

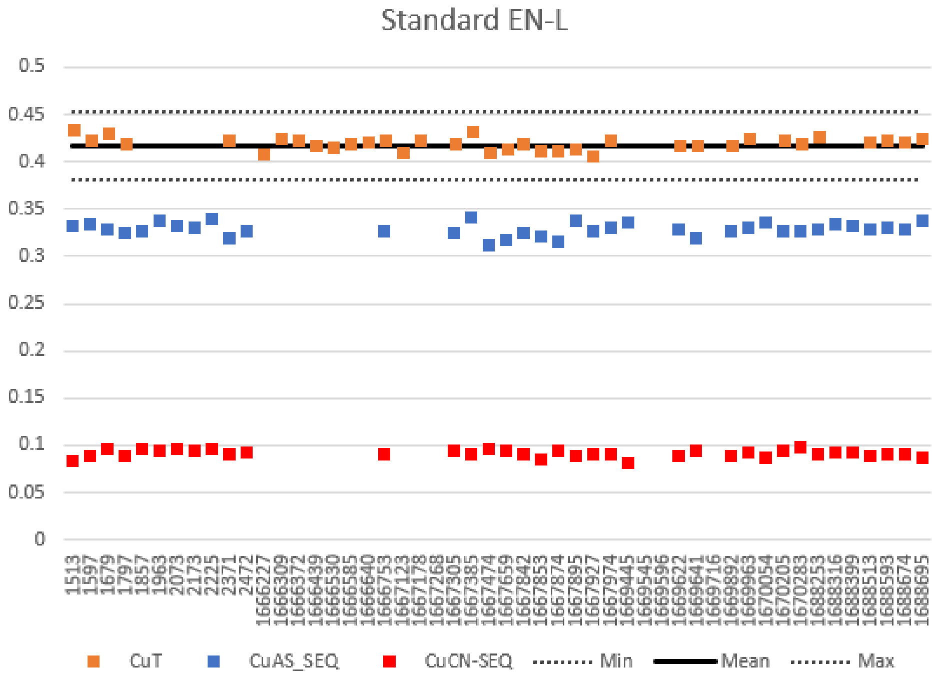


Figure 12-17: Low Grade Enriched Standard (EN-L) – Parks/Salyer

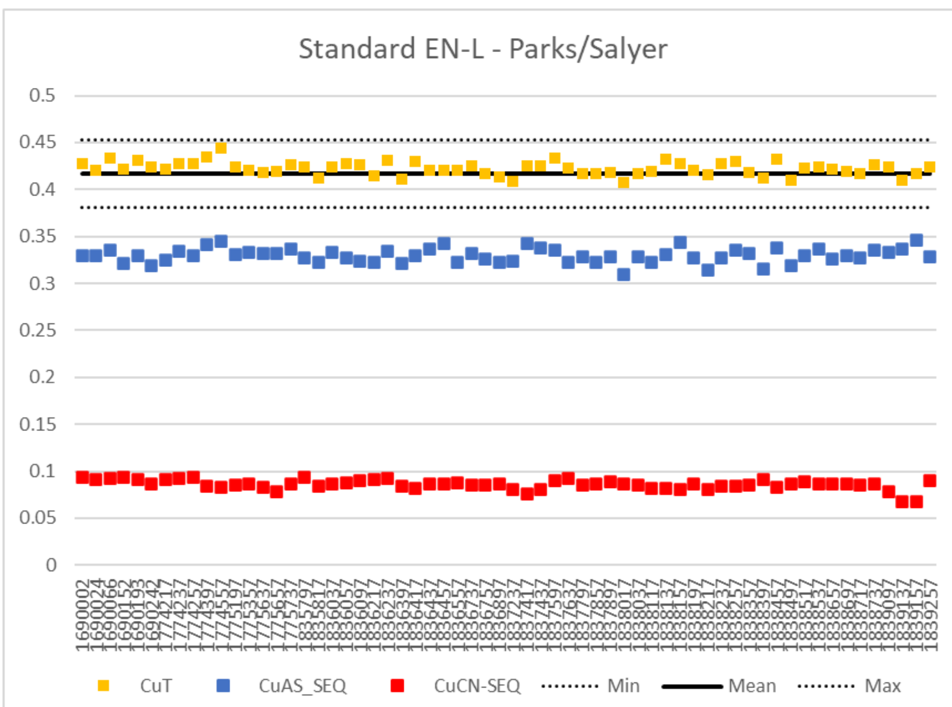


Figure 12-18: Low Grade Enriched Standard (EN-L) – Cactus Stockpile Project

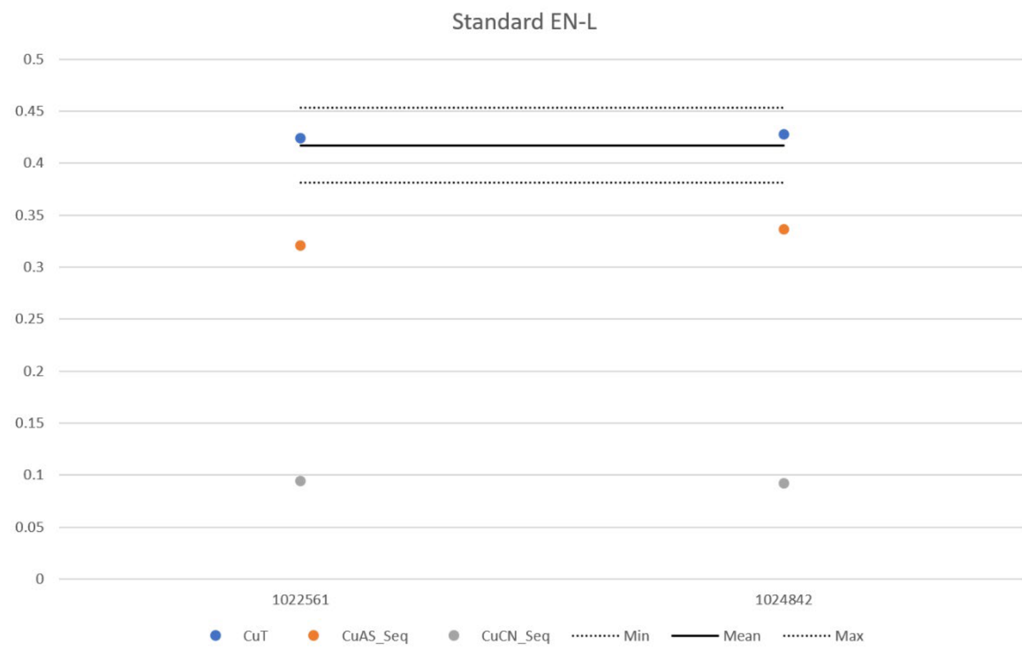


Figure 12-19: High Grade Primary Standard (PR-H) – Cactus

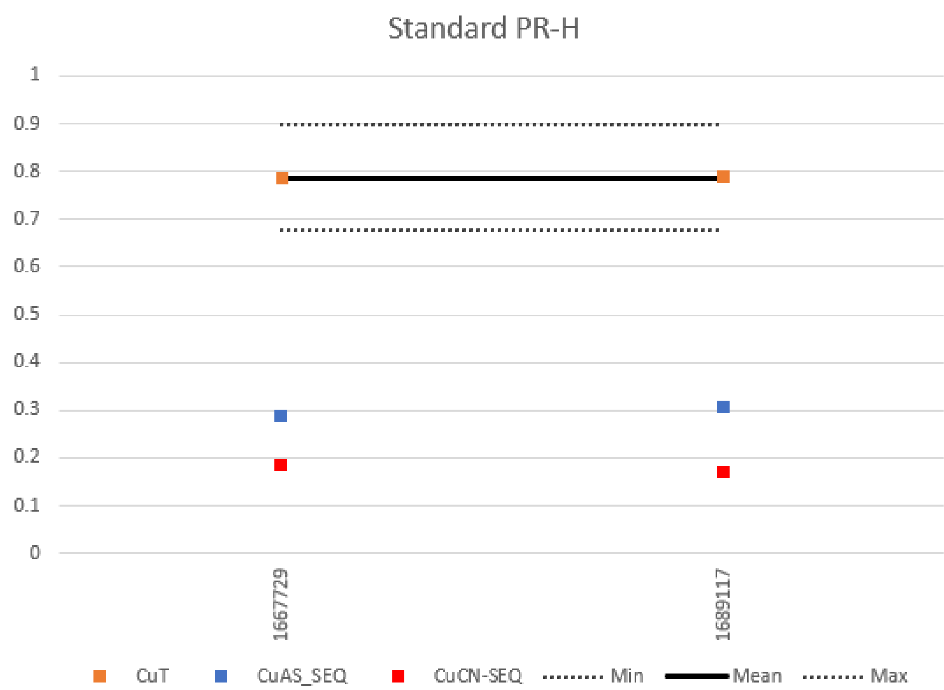


Figure 12-20: High Grade Primary Standard (PR-H) – Parks/Salyer

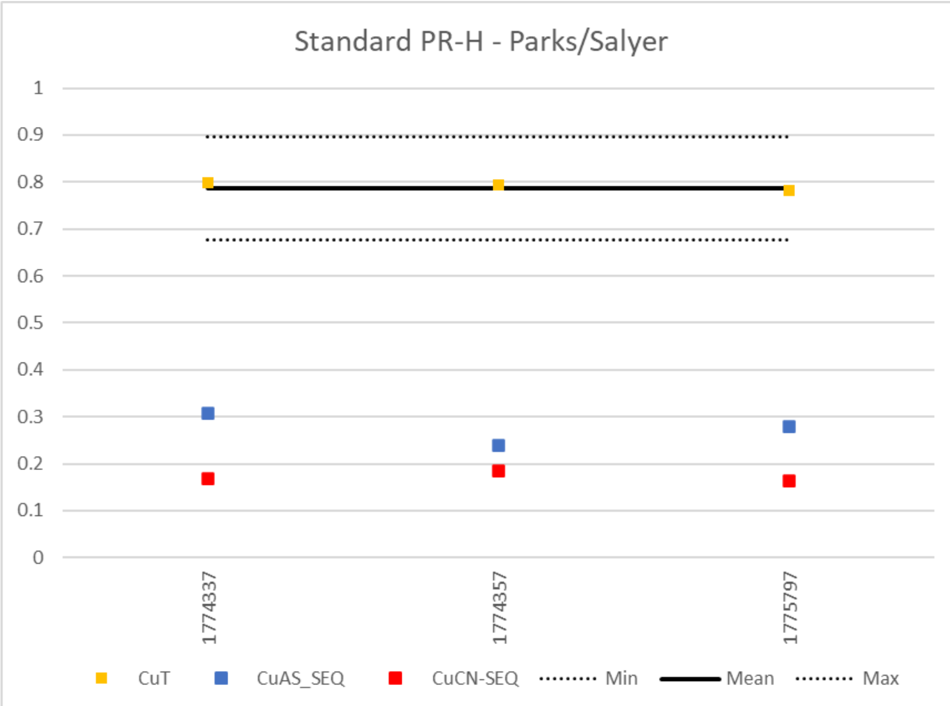


Figure 12-21: Medium Grade Primary Standard (PR-M) – Cactus

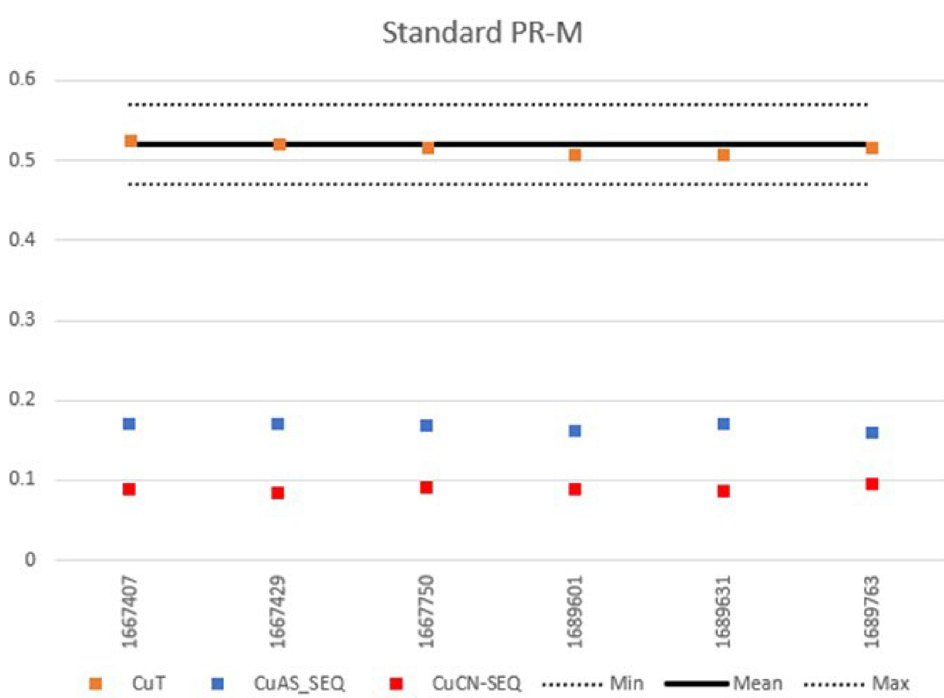


Figure 12-22: Medium Grade Primary Standard (PR-M) – Parks/Salyer

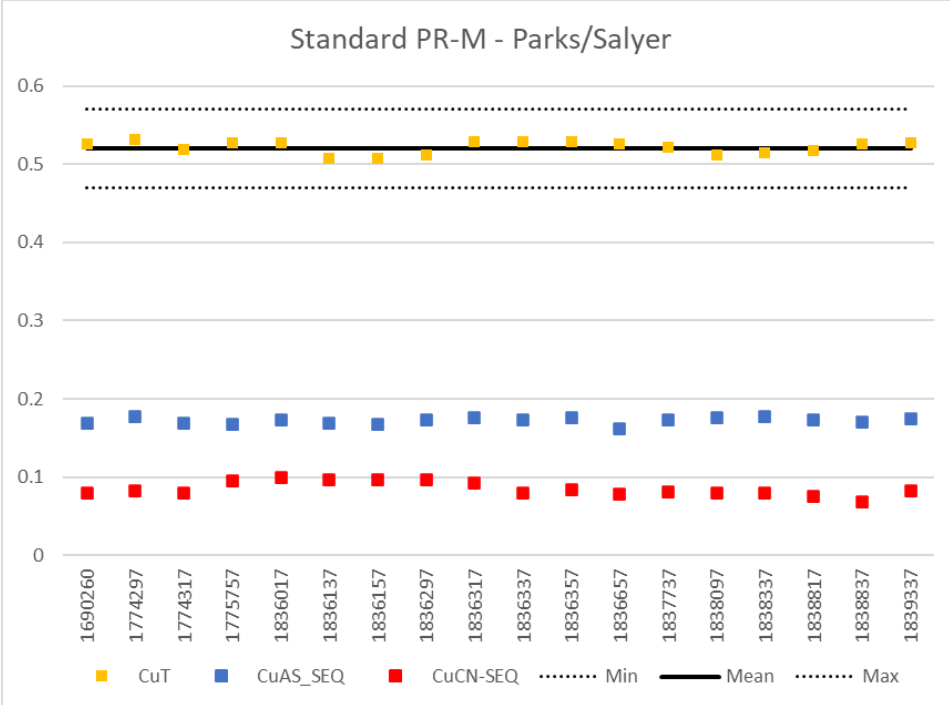


Figure 12-23: Low Grade Primary Standard (PR-L) – Cactus

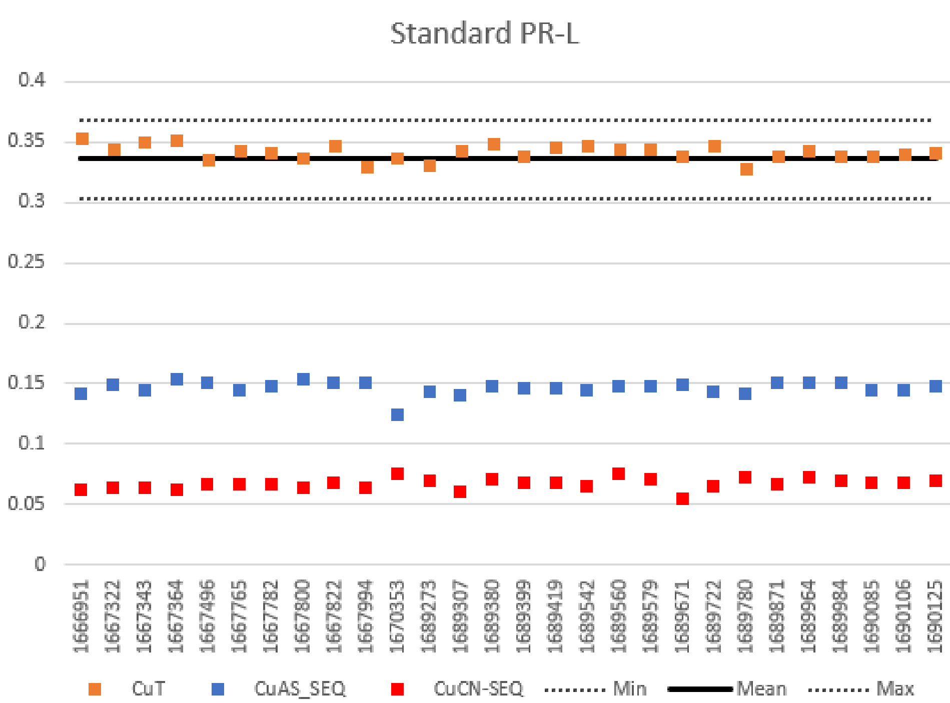
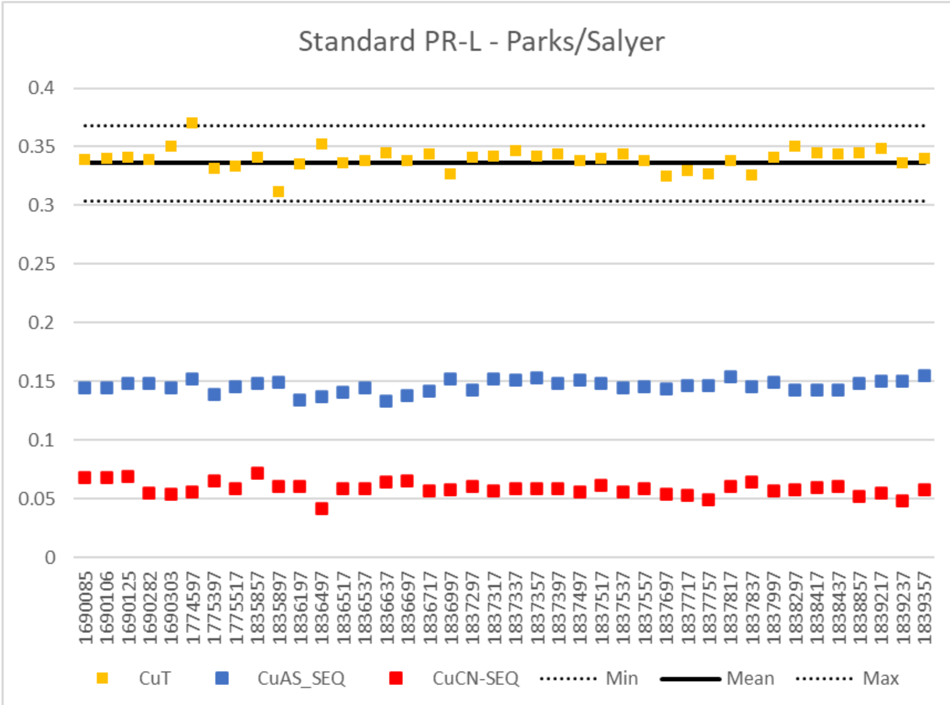


Figure 12-24: Low-Grade Primary Standard (PR-L) – Parks/Salyer



12.5.2 Blanks

Blanks were inserted into the sample stream at a rate of 1 per 20 samples or 5%, to test for contamination in the sample preparation process. Two blanks were used.

- R-Blank – an unmineralized rhyolite blank.
- MEG-Blank – an unmineralized blank.

Figure 12-25 through Figure 12-30 show the performance of the blanks across both the drilling and re-assay programs supporting the new mineral resource estimate. All figures plot the maximum expected total copper grade as a dashed line (0.015% CuT). Orange squares represent the total copper grade, blue squares represent the CuAS grade, and red squares represent the cyanide soluble sequential copper grade. In all cases, the assayed total copper grades were below the maximum value and indicate no evidence of contamination in the sample preparation process.

Figure 12-25: R-Blank – Cactus

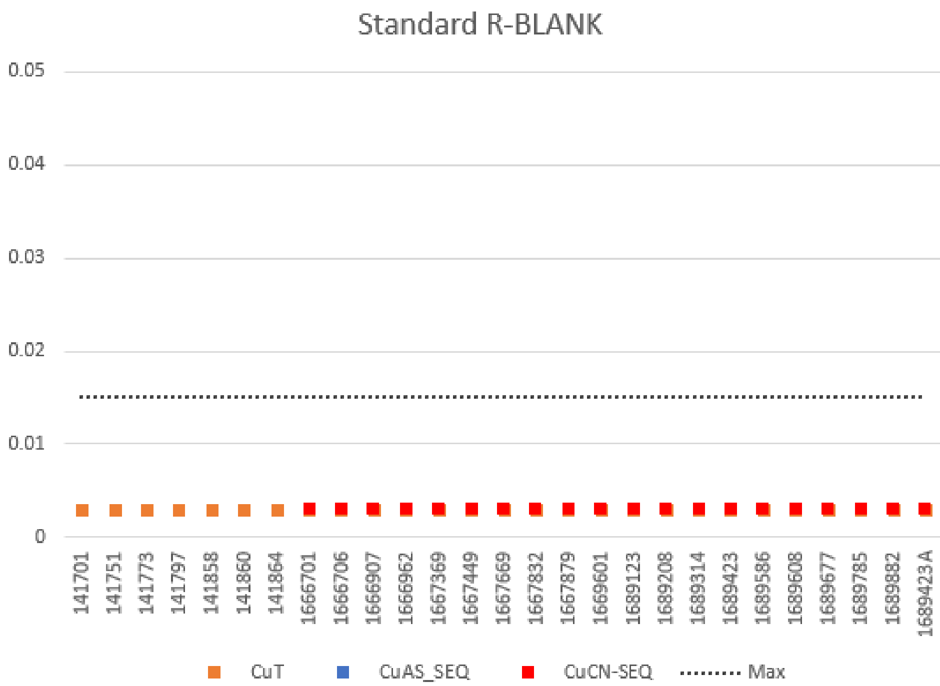


Figure 12-26: R-Blank – Parks/Salyer

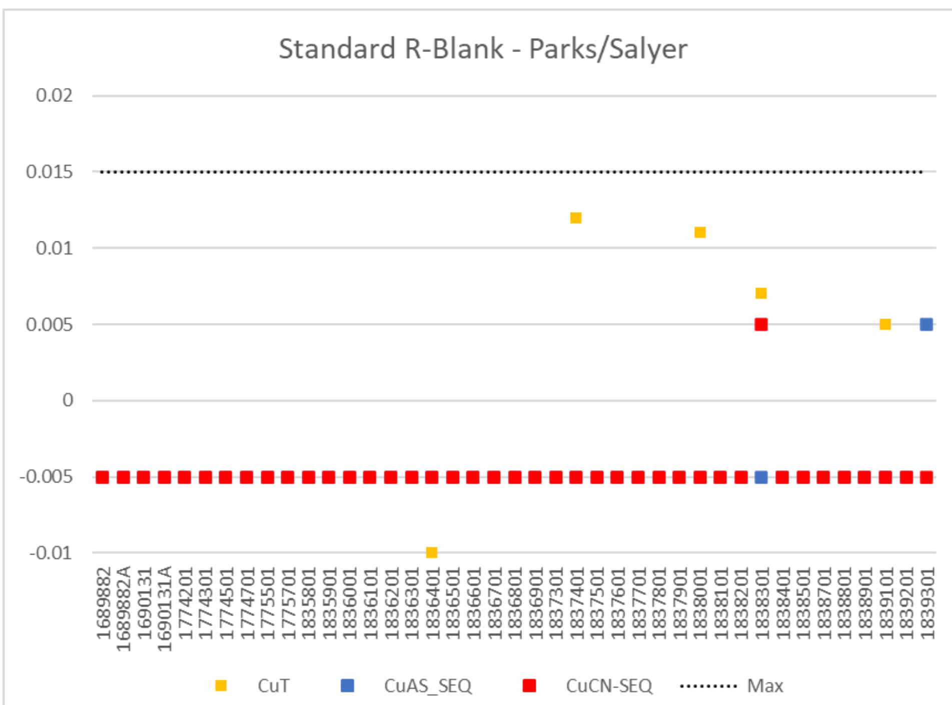


Figure 12-27: R-Blank – Cactus Stockpile Project

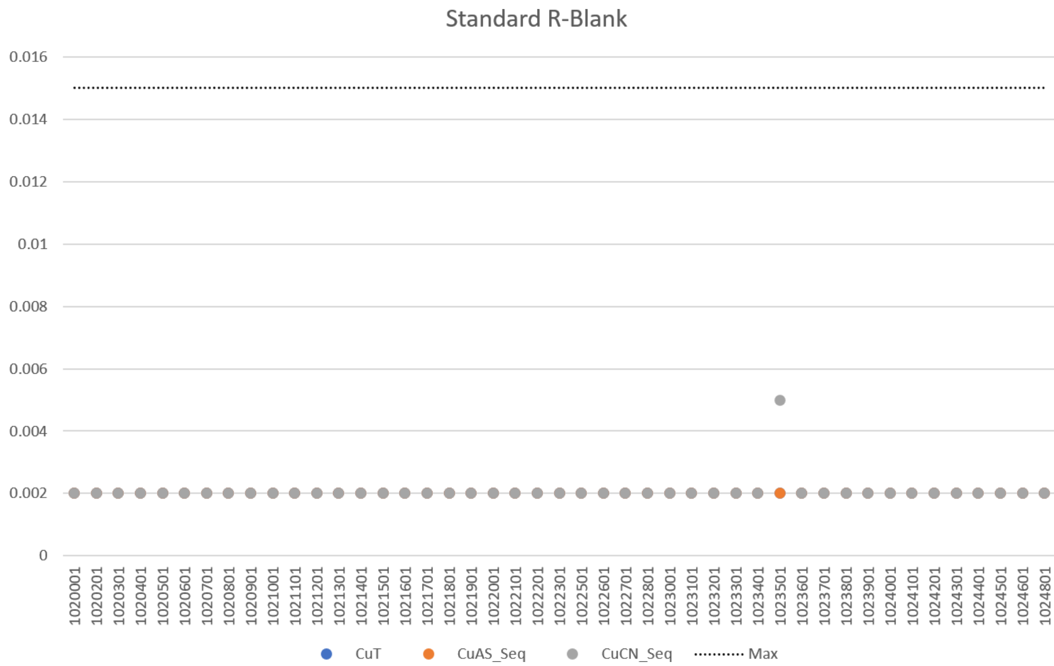


Figure 12-28: MEG-Blank – Cactus

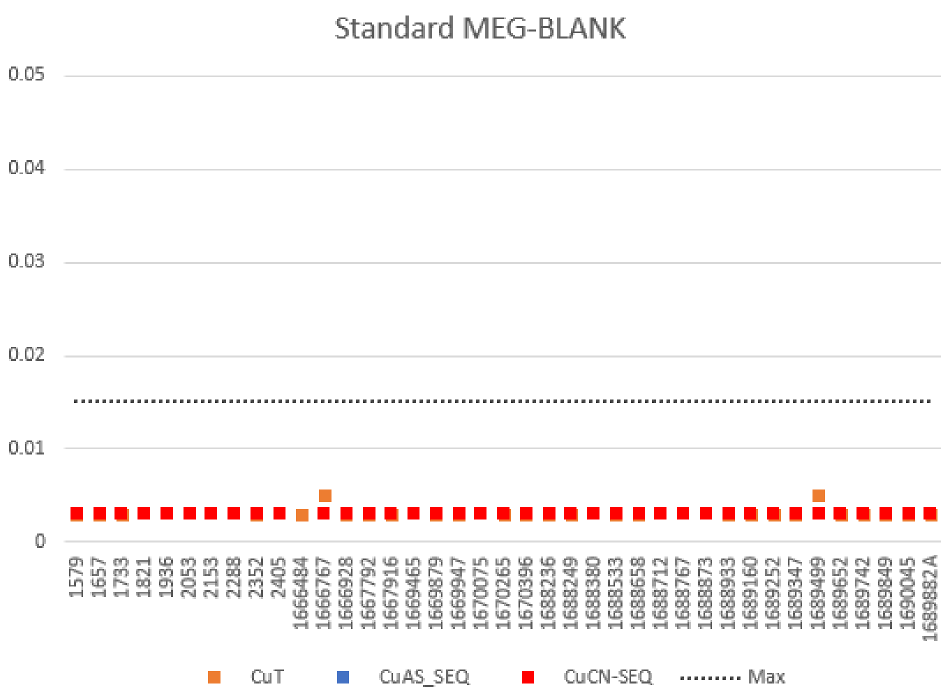


Figure 12-29: MEG-Blank – Parks/Salyer

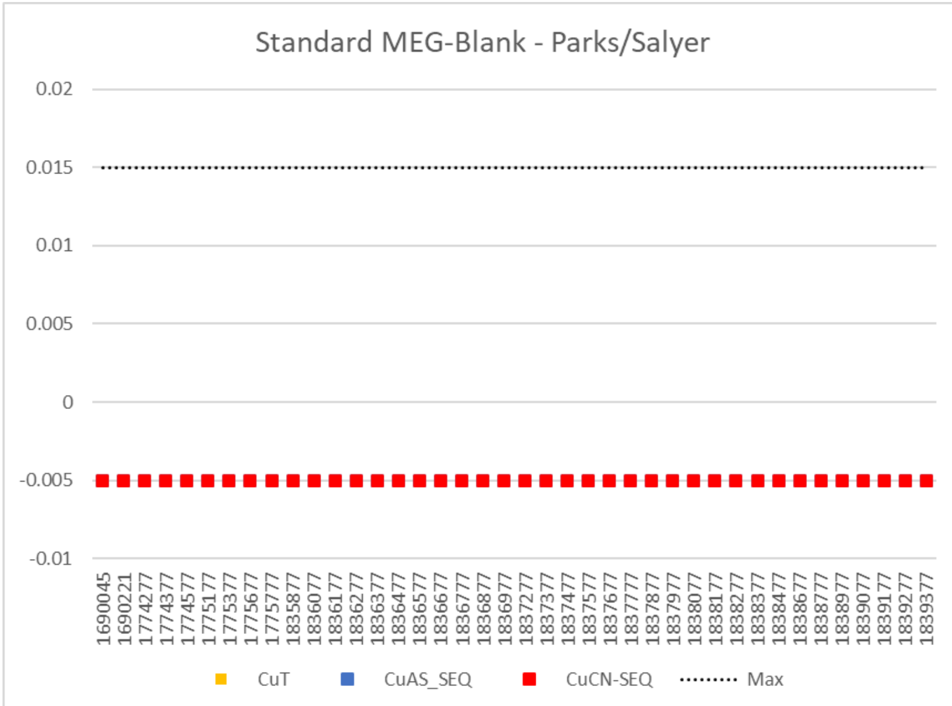
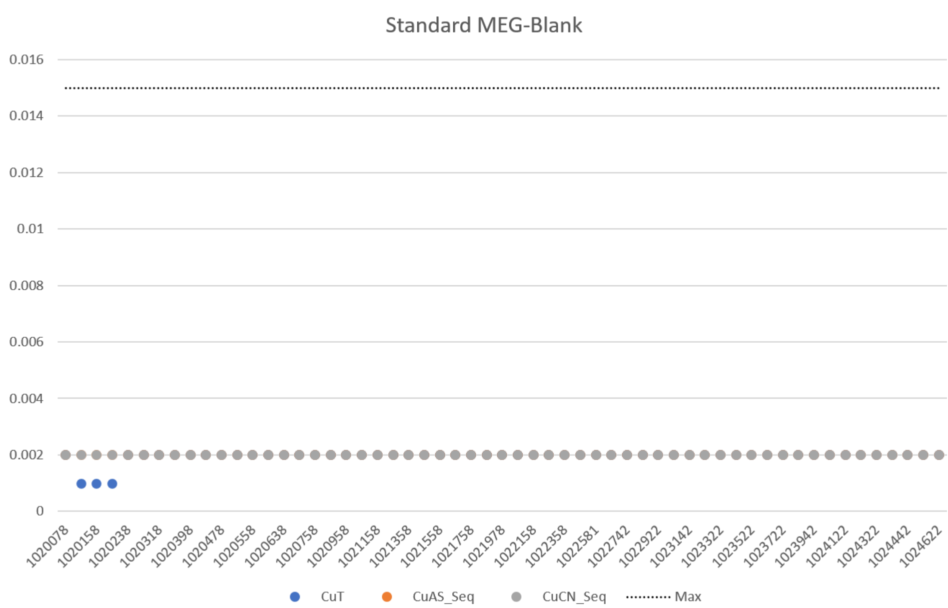


Figure 12-30: MEG-Blank – Cactus Stockpile Project



12.6 Qualified Person Opinion

During early visits to the mine site and core sheds, the QP worked with the geologists to select a number of pulps from historical core and requested that they be sent to Skyline labs to compare results with historical assay records and certificates. These data were analyzed and verified by the QP as an independent check of the assaying controls and procedures used by the assay lab and core samplers. Particular attention was paid to the QA/QC records for this group of samples both internal to the lab and the blanks, duplicates, and standards submitted by Arizona Sonoran.

The QP for Section 12 has reviewed all the associated data in detail and finds that it meets all the expected standards and best practices as defined in CIM's Best Practices Guidelines 2019. The drill results and associated data meets the level of accuracy expected for this PEA report.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as at 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report. The material to be processed as part of the Cactus open pit expansion project is an extension of the open pit mining operations by ASARCO that took place in the 1970s and early 1980s. The prior operations considered traditional copper milling and flotation concentration operations to produce copper sulfide concentrates for processing at local smelters.

In consideration of a potential copper heap leaching and SX/EW processing facility at Cactus based on processing existing Stockpile Project oxidized copper resources, a hydrometallurgical approach was also contemplated to process the oxide and enriched sulfides (chalcocite / covellite dominant) material identified in the mineralized Cactus East and Cactus West extensions to the existing open pit reported in this Mineral Resource Estimate.

Arizona Sonoran geologists are working with metallurgical engineers to quantify the recovery of copper from samples obtained in a large drilling campaign. The drill core samples are safely recovered and placed in bags to be studied by geologists and shipped to a well-established Mineral Processing research and development firm in Reno, Nevada (McClelland Analytical Service Laboratory, an ISO 17025 accredited facility). The metallurgical test program has been developed by and supervised by Mr. James L. Sorensen.

Metallurgical characterization testing has been completed as part of this study in the form of sequential assay (H2SO4 and cyanide steps) for the resources considered and bottle roll testing. Three samples from newly drilled core were selected to reflect copper grades close to the presumed average of the economically processable material in the open pit resource for column testing to be completed in the next phase of work. Assay data and bottle roll testing was completed for this study on head samples from the three column test samples currently under acidic and bioleach conditions.

Based on typical recovery estimates for CuAS and CuCN as provided by a standard sequential copper assay methodology developed at the Skyline Laboratory facility in Tucson, Arizona, projected copper recovery estimates have been derived based on leachable copper content that will be validated in the ongoing column testing program.

Based on the current understanding or the potential Stockpile Project resources to be processed, the leachable materials are characterized as oxide having an CuAS content of greater than or equal to 80% and a CuCN content for the balance to a COG of 0.095% CuAS + CuCN content, or soluble copper (TSol) that is potentially recoverable.

The COG considered at 0.095% is estimated from preliminary operating costs and is not based on a mining evaluation or detailed analysis and was therefore used to establish a potential economically viable component of the resources estimated. There is a reasonable probability of eventual economic extraction of this resource using H2SO4 leaching and SX/EW recovery at a cutoff of 0.095% TSol.

Materials with a TSol grade above the cutoff of 0.095% TSol but having a CuAS content of less than 80% is classified as sulfide or enriched materials for leaching purposes. Primary mineralization that is not acid or cyanide copper soluble (e.g., chalcopyrite) that reports in the CuT assays is not considered as recoverable metal in the current analysis.

In the current mine plans, the distribution of leachable oxide and enriched material types is provided in Table 13-1.

Table 13-1: Potential Leach Materials Distribution

Mining Source	Material Type	Tons of Leach Material (tons)	Grade % TSol (% Cu)	Leachable Cu (tons)	Distribution Percent	
					Material	Cu
Stockpile Project	Oxide	82,331,000	0.141%	116,279	100%	100%
Open Pit	Oxide	46,810,000	0.190%	88,939	67%	48%
	Enriched	23,131,000	0.420%	97,150	33%	52%
Underground	Oxide	6,317,000	1.100%	74,271	23%	21%
	Enriched	21,208,000	1.330%	274,597	77%	79%
Total	**Oxide**	**135,458,000**	**0.203%**	**279,489**	**75%**	**43%**
	Enriched	**44,339,000**	**0.822%**	**371,747**	**25%**	**57%**
	Total	**179,797,000**	**0.355%**	**651,236**		

In parallel, copper flotation testing is also being conducted on higher grade sulfide material to consider the possible future incorporation of a traditional copper milling and flotation operation to treat higher grade enriched and primary mineralization (chalcocite/chalcopyrite dominant) material identified. Resources containing a maximum of 20% oxidized copper content are considered potential mill feed based on ASARCO historical performance.

13.1 Historical Processing and Mineralogical Information

Information has been obtained from the Arizona Geological Survey archives related to the Sacaton deposit, now renamed Cactus Project. The information consists of a set of internal operating reports and memorandum identified as coming from the James Doyle Sell Mining Collection. The records are collected in a single file and date from 1961 to 1972. Included in these are reports and memorandum from 1963 that discuss acid leaching investigations of drill core samples from the Sacaton deposit.

ASARCO mined material for the Sacaton West ore body and milled ore containing primarily primary sulfide mineralization consisting mainly of pyrite and chalcopyrite. In the better part of the primary zone these sulfides occur in a volume proportion of about 1.5 parts pyrite to 1 part chalcopyrite. The total sulfide content (by volume) averaged between 1.5% and 3.0%. The primary sulfides occur both as thin veinlets and as discrete grains in roughly equal proportions. Chalcocite and minor covellite occurred as supergene replacements of both pyrite and chalcopyrite. Chalcocite predominates in the upper portion of the ore zones and chalcopyrite in the lower parts. In addition to copper, the ore contains minor amounts of molybdenum and traces of gold and silver.

The material placed in the waste rock facility were low grade and oxidized resources not suitable for processing in an existing 11,000 tpd (9,000 tpd initially) copper milling and concentrator operation to produce copper concentrates for processing in ASARCO owned smelters between 1974 and 1984. A significant amount of the prior exploration data was recovered and available for review, much of the operational data is not available and only a few production reports and files that were abandoned in offices provide some context for the prior materials processed. While there is some evidence collected that ASARCO considered processing the more oxidized components of the deposit through sulfidation and flotation techniques it does not appear this was pursued or if so on a very limited basis.

Material not sent to the mill was characterized in ASARCO reports recovered or available through the US Geological Survey Mineral Resource Data System (MRDS) and Arizona Geological Survey records archives. A synopsis of the relevant information contained in these available reports and memos is presented in the following.

Leached capping varied in thickness from 100 ft to 500 ft overlies both deposits (east and west ore bodies). The capping is characterized by the presence of "live" limonite’s derived from the oxidation and leaching of chalcocite. Copper values in the capping average less than 0.1% copper, except where appreciable amounts of perched sulfides or oxidized copper minerals are present. Deep post-enrichment oxidation and leaching has destroyed portions of the chalcocite blanket. Oxidized copper minerals including antlerite, brochantite, azurite, malachite, and chrysocolla are found in varying quantities in the capping and below where second stage oxidation has penetrated the sulfides.

As reported by Briggs in a Mining Operations Report Version 2005 dated 22 October 2004 (Copyright 1990-2006 David F. Briggs), the historic Sacaton process operation performance summary was described. A synopsis of the information was presented by Briggs and included in Figure 13-1.

Figure 13-1: Summary Historic Mill Performance (D.F. Briggs 22 October 2004)

Design Capacity: **Mill**

9,000 short tons/day (1974-1975)	11,000 short tons/day (1976-1984)

Actual Processing Rate: **Mill**

8,279 short tons/day (1974)	10,434 short tons/day (1980)
9,945 short tons/day (1975)	11,241 short tons/day (1981)
10,333 short tons/day (1976)	11,411 short tons/day (1982)
10,984 short tons/day (1977)	10,967 short tons/day (1983)
11,378 short tons/day (1978)	10,989 short tons/day (1984)
10,975 short tons/day (1979)	

Note: Estimated mill rates calculated from reported annual production data, assuming a 7 day/week (365 day/year) work schedule. The 1977 rates were affected by a seven week shut-down for economic reasons.

Reagent Consumption - Lime - 2.0 lbs./ton (1975)
A-238 - 0.021 lbs./ton (1975)
Z-6 - 0.020 lbs./ton (1975)
Frother - 0.06 lbs./ton (1975)

Reagent Consumption - Oxide Ore - NaHS - 0.9 lbs./ton (1975)
A-404 - 0.015 lbs./ton (1975)
Frother - 0.6 lbs./ton (1975)

Metallurgical Recovery: **Mill**

74.8% Cu (1974)	82.0% Cu (1976)	82.8% Cu (1978)
81.6% Cu (1975)	81.8% Cu (1977)	82.1% Cu (1979)

Note: Calculated from reported production data. Silver recoveries for 1979 were 51%.

Concentrate Content: Copper Concentrate - 25% Cu

Oxide Copper – Metallurgical Tests

During examination of diamond drill core, it was observed that little or no chrysocolla occurred in the oxide mineralization capping the sulfide zone. The bulk of the mineralization proved to be composed of copper carbonates and sulfates (azurite, malachite, brochantite, and antlerite) and it was thought that this material might be amenable to flotation, thereby adding to the ore reserve. Arrangements were made to run flotation tests on this material at the Sacaton Unit. (Note, flotation results for some tests indicated a recovery of about 70% in ASARCO information recovered).

In addition, a sizeable tonnage of leachable copper mineralization composed of chalcocite and copper oxides was delineated by ASARCO in the East Sacaton ore body. This material would be available for leaching by solutions introduced from above. With this in mind, the entire East Sacaton ore reserve was optionally considered by ASARCO to be an in-place leaching operation utilizing the underground development crosscuts as collection basins for pregnant solutions and pumping them out through the shaft, a potential recovery of approximately 400 million pounds of copper is theoretically possible assuming a 50% recovery.

While current plans do not expect the East Sacaton ore body to be operated as an in situ leach operation, this is nonetheless an option to be considered with the existing core and resource information should the underground mining plans be further delayed or be proven uneconomical by escalating mining costs.

The following information was taken from an internal memorandum 27 August 1963 and provides some insight into the copper leaching test work ASARCO conducted. It is believed that these samples come from what was to be the Sacaton West deposit mined by ASARCO.

- The oxidized sample, No. 6206, contained no water-soluble copper. However, it leached readily with a one-hour H2SO4 leach at pH 2.0, followed by a five-hour acid sulfate leach also at pH 2.0. The acid ferric sulfate is necessary because of some chalcocite mineral in the sample. Overall copper extraction was 91.4%. A straight acid leach with no ferric sulfate over a 24 hour period dissolved 86.9% of the copper.
- An extraction of 49.4% of the copper was obtained from the chalcocite sample, No. 6207, in a 48 hour leach with acid ferric sulfate at pH 1.5 with, no additional oxidation. Ferric sulfate dosage was calculated according to the amount of copper present, presumably more might have been added since most ores contain organic material that is capable of reducing ferric sulfate to ferrous sulfate under beaker leach conditions. Also, microscopic examination of the chalcocite mineral at -65 mesh grind showed the panned chalcocite to be, granular rather than sooty and this condition of the mineral may call for greater leach retention time. Mineral counts on panned sulfides, at the ~65 mesh grind, were made by Messrs. Graybeal and Aliaga. In both cases, the estimate was that about 10% of the total copper was present as chalcopyrite rather than chalcocite.
- A 44 hour leach of the chalcopyrite sample, No. 6208 rendered soluble only 5.3% of the copper, as might be expected.
- Lime content of all three samples is low enough to be suited to acid leach, with moderate acid consumption.

Table 13-2: Acid-Acid Ferric Sulfate Leaches

Sample	Best Extraction
No. 6206, Antlerite-Brochantite	91.4%
No. 6207, Chalcocite	49.4%
No. 6208, Chalcopyrite	5.3%

A synopsis of H2SO4 consumption data available from ASARCO testing in 1968 is presented in Table 13-3. The first two samples appear to come from the Sacaton West deposit and are most relevant to the current work. The American Analytical Research Laboratories (AARL) methodology is not disclosed but is provided here for completeness. Acid consumption is taken as gross acid consumption related to pound or ton of ore, not copper. No copper recovery information was found for these tests.

Table 13-3: Historic Acid Consumption Information (ASARCO 1968)

Sample Description	Drill Core Hole	Intercepts		AARL		ASARCO Method minus $^3/_8$ inch material	
Acid Consumption		From	To	lb H_2SO_4/lb	lb H_2SO_4/t	lb H_2SO_4/lb	lb H_2SO_4/t
Sample A-8312	S-98	1146	1151	0.0154	30.8	0.0105	20.972
Sample A-8313	S-99	549	554	0.0184	36.8	0.0105	20.972
Sample A-8314	S-100	713	718	0.0162	32.4	0.0106	21.168

13.2 Stockpile Project Material Testing

Samuel Engineering, in conjunction with Stantec, prepared a Technical Report PEA for the Stockpile Project, effective date 01 March 2020, for Arizona Sonoran. Materials to be processed as part of the Stockpile Project included an existing waste rock facility placed by ASARCO because of mining the Sacaton West deposit. The material placed in the existing waste rock facility was of low grade and highly oxidized copper resources not considered suitable for processing in the ASARCO mill and concentrator operation.

Arizona Sonoran plans to process the Stockpile Project materials in a heap leach and SX/EW processing facility to be constructed as part of that project. Metallurgical testing and analysis focused on the heap leach materials in the Stockpile Project facility. The objectives of the metallurgical tests were to develop technical parameters and inputs for the process plant scoping level investigation including potential copper extraction, approximate timeframes for extractions, and relative H2SO4 consumption.

Metallurgical characterization testing has been completed as part of this study in the form of sequential assay (H_2SO_4 and cyanide steps) for the resources considered and bottle roll testing. Three sample locations were selected to reflect copper grades close to the presumed average of the economically processable material in the waste rock facility for column testing to be completed in the next phase of work. Assay data and bottle roll testing was completed for this study on head samples from the three column test samples.

A site visit was made to the Cactus Project on 06 December 2019 for the purposes of selecting sampling sites based on the sonic drilling of the waste dump conducted previously that could be used to help characterize the metallurgical performance of the materials currently in the Stockpile Project. Samples were selected based on total copper assays (CuTs), since sequential assays were not available from the lab at the time of the visit.

Sonic drill hole CuTs and drill hole locations were provided by Arizona Sonoran. Drill collar locations were identified and confirmed based on field staking.

A total of three bulk samples were collected from three locations on the waste dump. Each bulk sample consisted of two 55-gal drums of material to be used in a single column test requiring at least 880 lb (400 kg) of material per the protocols outlined by McClelland Laboratories, Inc. (McClelland). The two drums representing one test sample were arranged on a single pallet for shipment to McClelland in Reno, Nevada. McClelland Analytical Services Laboratory is an ISO 17025 accredited facility.

The samples taken were selected on the following criteria.

- Samples focused on Lift 3, which contains most of the potential copper resources and early mineable material.
- Locations from three distinct physical locations and away from the dump edges that met the other criteria and provided unique samples based on the surface of the waste dump area.
- Total copper grades of approximately 0.17% Cu, the expected average grade of the potentially economic materials that could be mined.
- Materials of at least 8-12 ft below the top surface of the waste dump to minimize surficial impacts and biases.

A 200 gram split of the assay head sample for each bottle roll test was collected and submitted to Skyline Laboratories in Tucson, Arizona, for sequential assay analysis. Skyline was chosen for this step as they are the lab performing all the assaying for geologic samples and the same assay methodology would be used. Sequential assaying methodology digests an assay sample first with H_2SO_4, the resultant solution is analyzed and then the remaining solids digested with NaCN.

The sequential assay method serves as a proxy for identifying copper included in minerals that are typically leachable in H2SO4 leaching operations. Copper minerals such as oxides, carbonates, silicates / chrysocolla, and sulfates are dissolved in the first acid digestion. A portion of any chalcocite mineralization is also digested. In the second digestion with cyanide, secondary copper minerals like chalcocite and covellite are dissolved. The secondary sulfides also leach in acid heap leach commercially, though it takes longer due to the oxidation step required. Results for the sequential assays in terms of grade in percent copper and analysis is provided in Table 13-4.

Table 13-4: Sequential Assays on Bottle Roll Test Head Samples

	Sequential Assay Grade (% Cu)		
	CuAS-SEQ	**CuCN-SEQ**	**TSol**
4517 WD-22	0.098	0.020	0.118
4517 WD-24	0.164	0.007	0.171
4517 WD-50	0.099	0.008	0.107

Column testing results were not available at the time of the 2020 Cactus Stockpile Project PEA disclosure. Results for the Stockpile Project testing have been reported by McClelland in a Revised Report on Heap Leach Testing Cactus Bulk Samples MLI Job Number 4715 dated 11 February 2021.

Tests were conducted in 12-inch inner diameter (I.D.) by 10 ft (3 m) tall columns containing approximately 880 lb (400 kg) of material. Column leach testing in closed circuit with SX was incorporated once sufficient solutions were developed. A summary of these results is presented in Table 13-5.

Table 13-5: Summary of Column Test Results – Stockpile Project Composite Samples. Summary Metallurgical Results, Oxide Acid Column Leach Tests, Stockpile Project Bulk Samples, -3-inch Feed Size

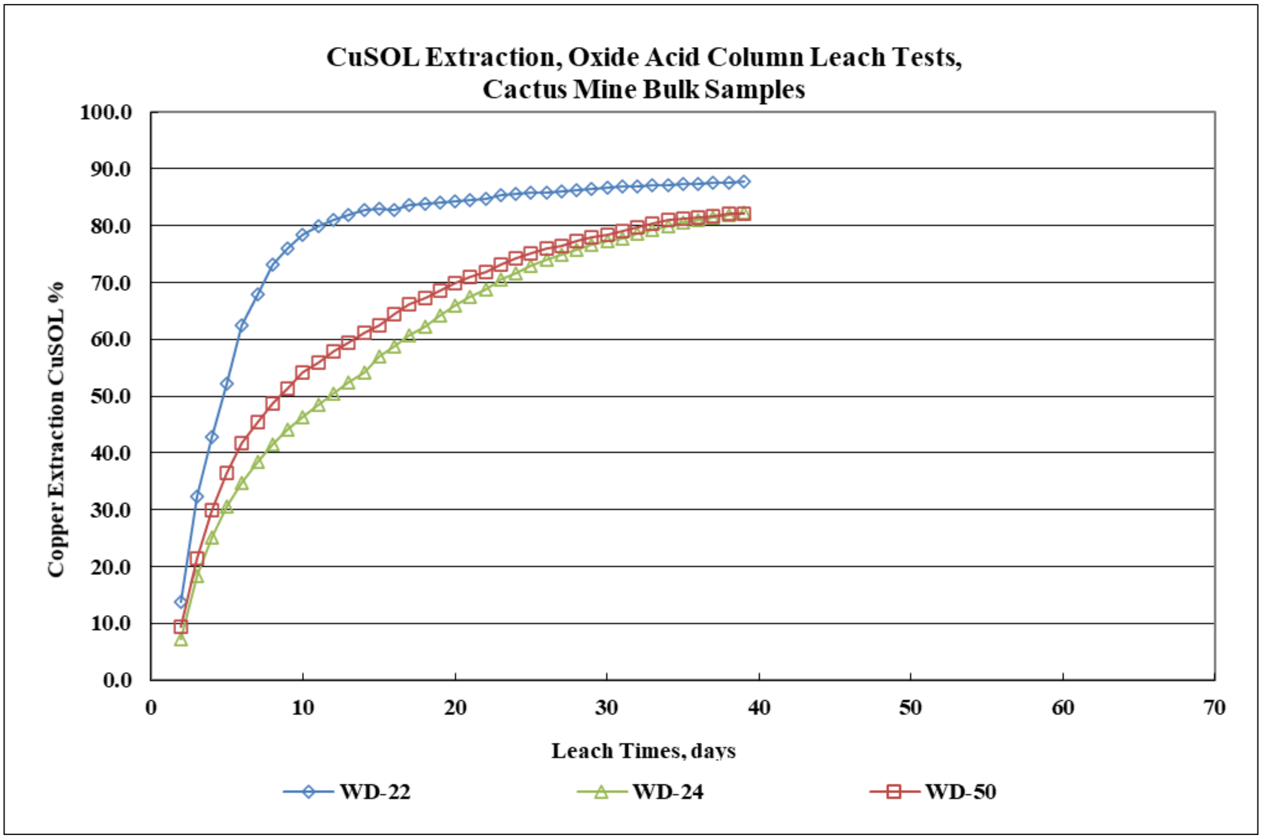
MLI Test No.	Sample	Leach / Rinse Time (days)	CuT Recovery (%)	Extracted	Tail	Calc/d Head	Avg. Head	Gross (lb/t ore)	Gangue (lb/t ore)	Specific (Gangue) lb/lb ore
AC-1	WD-22	39	75.9	0.082	0.026	0.108	0.119	31.0	28.5	17.4
AC-2	WD-24	39	74.8	0.193	0.065	0.258	0.215	21.7	15.7	4.1
AC-3	WD-50	39	70.4	0.112	0.047	0.159	0.151	25.7	22.3	9.9

13.2.1 Stockpile Project Column Test Copper Recovery

Column testing of the initial Stockpile Project scoping samples has yielded significant copper extractions in a relatively short leaching timeframe.

Copper extraction results for the contained TSol (acid soluble and cyanide soluble copper mineralization) is presented in Figure 13-2 based on calculated head grades from leach tails and solution assays.

Figure 13-2: Soluble Copper Extraction versus Time (McClelland, 2020)



Leaching was stopped after 39 days to allow for an assessment of the copper extraction based on a calculated head content due to the very high extractions indicated from the assay head basis.

A breakdown of copper recovery for CuAS, CuCN, TSol, and the CuT is presented in Table 13-6. Extraction estimates are based on head and tail assay data for the column tests. Based on experience, the initial concept was at least two leach cycles of 90 days over a 2-year period to achieve the bottle roll copper leaching extractions predicted for initial economic assessment in the 2020 Cactus Stockpile Project PEA.

Table 13-6: Copper Extraction by Copper Assay Method Copper Recovery (%) at 39 Days Leach/4 Days Drain and 7 Days Rinse

	WD-22		WD-24		WD-50		AVG %	PEA Predicted/Modeled		
	Assay %	Recv %	Assay %	Recv %	Assay %	Recv %		180 days	Yr 1	Yr 2
CuAS	80.6	97.0	63.6	89.0	69.8	94.0	94.0	85.0	75.0	10.0
CuCN	17.6	64.0	5.8	39.0	10.7	30.0	44.0	75.0	35.0	40.0
TSol	98.1	88.2	69.4	82.3	80.5	82.2	84.0	83.3	68.2	15.1
TSol Pred		81.7		58.4		67.4	69.0			
CuT		76.0		75.0		70.0	74.0	71.9	59.0	12.9

CuAS copper content extraction averaged 94% for the three columns in 39 days of active leaching. CuCN recovery, representing enriched copper mineralization content (chalcocite and covellite) was also significant and averaged 44%, with a high of 64% in column WD-22. The combined TSol extraction averaged 84% for the material tested, slightly improved over the expected 83.3% estimated in the 2020 PEA. It was initially expected that a longer leach cycle time would be required to achieve the extraction levels for copper. Of note is that the PEA extraction was anticipated to occur over a 180-day leach cycle timeframe and the columns tested achieved that in less than 40 days of active leaching.

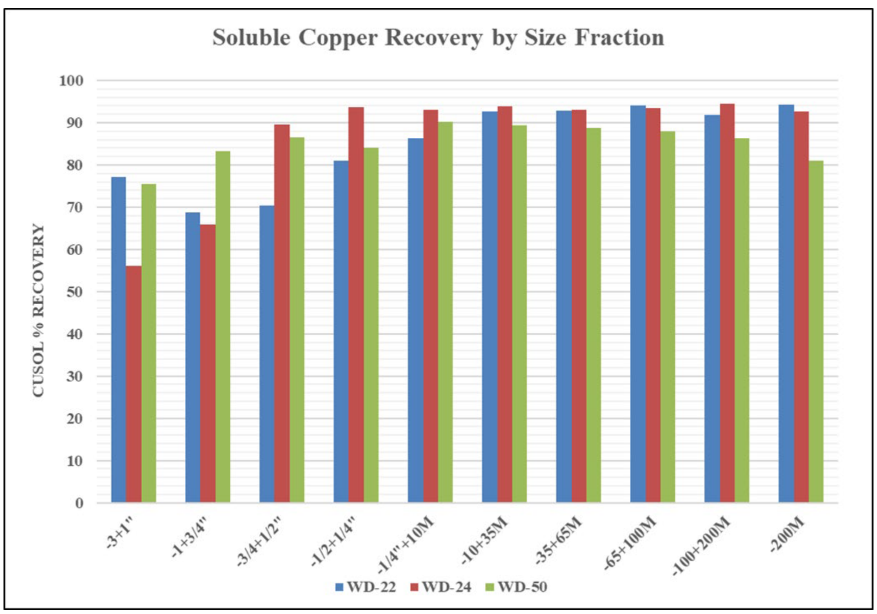
The Stockpile Project is visually composed of fine materials with little evidence of large (over 12-inch) particles during sample collection and site traverse. Samples were collected by backhoe with inclusion of all large rock encountered. The screen analysis for the samples as loaded into the columns is presented in Table 13-7. The effective P80 size distribution was 1.5-inch for the three columns.

Table 13-7: Column Screen Size Analysis (McClelland, 2020)

Size Fraction	WD-22 Weight, %		WD-24 Weight, %		WD-50 Weight, %	
	Head	Tail	Head	Tail	Head	Tail
-3" + 1"	28.4	21.9	29.8	27.0	31.6	22.6
-1 + ¾"	6.6	7.0	5.7	5.5	5.9	4.6
-¾ + ½"	11.8	10.0	7.9	7.2	9.4	6.8
-½ + ¼"	9.7	10.3	11.0	13.0	10.1	12.0
-¼" + 10 M	14.0	13.9	16.0	16.4	14.2	14.5
-10 + 35 M	12.2	14.5	12.1	12.7	11.2	14.6
-35 + 65 M	4.5	6.3	4.7	4.6	4.1	6.1
-65 + 100 M	2.2	3.0	1.7	1.6	2.2	2.6
-100 + 200 M	3.2	2.8	3.0	2.8	2.8	3.7
-200 M	7.4	10.3	8.1	9.2	8.5	12.5
Composite	100.0	100.0	100.0	100.0	100.0	100.0
P80	~1.5"		~1.5"		~1.5"	

Copper extraction was also considered by size fractions to assess the impact of particle size. Figure 13-3 shows the relative coper extraction by size fraction.

Figure 13-3: Soluble Copper Recovery by Size Fraction (McClelland, 2020)



Copper extraction appears to be impacted in particles sizes over 3/4 inch in these tests. Future testing will need to consider a larger distribution of sizes above 1-inch to fully evaluate the relative significance of these results to the run-of-Stockpile Project resources.

13.2.2 Stockpile Project Column Test Leaching Acid Consumption

Column testing of the initial Stockpile Project scoping samples has yielded significantly higher net acid consumption results in the leaching timeframe versus the bottle roll predictions.

Bottle roll acid consumption typically provides a higher-than-expected commercial performance result due to the fine particle size (-10 mesh) tested and excess acid added. Acid consumption results for the column tests are presented in Table 13-8.

Table 13-8: Stockpile Project Acid Consumption (Net of Copper Produced)

Net Acid Consumption (Pounds per Ton)				
	WD-22	**WD-24**	**WD-50**	**AVG**
Bottle Roll	16.7	17.9	7.1	13.9
Column	28.5	15.7	22.3	22.2
Difference	11.8	-2.2	15.2	8.3
Column lb acid/lb Cu	17.4	4.1	9.9	10.5

Gross acid consumption ranges from 25 lb/t to 30 lb/t of material leached. Net acid consumption is a function of copper recovery in an SX/EW facility which returns a stochiometric 1.54 lb of acid for each pound of copper recovered as cathode. As a result, net acid consumption will also be a function of the copper grades and extraction achieved.

13.3 Metallurgical Sample Selection – Open Pit Leach Resources

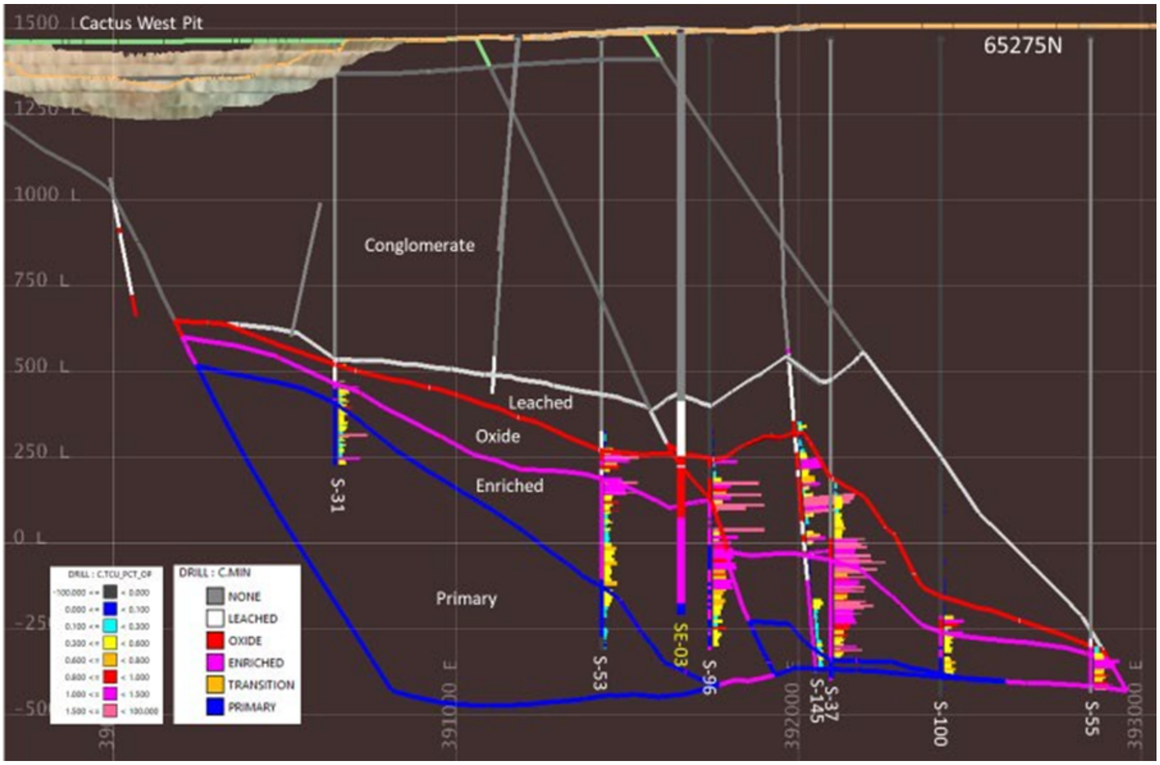
A site visit was made to the Cactus Project on 05 December 2019 for the purposes of selecting sampling sites based on the metallurgical drilling conducted that could be used to help characterize the metallurgical performance of the materials currently in the resource outlines. Samples were selected based on CuTs, since sequential assays were not available from the lab at the time of the visit.

Metallurgical core SE-03 (PQ size) diamond drill core hole was drilled in the Cactus East deposit area, near historic hole S-96 used as a reference for potential copper content and grades. Figure 13-4 describes the location of Drill Hole SE-03.

Figure 13-4: Metallurgical Sample Drill Hole SE-03 Location (Arizona Sonoran, 2020)

SE-03 met core drilling ended at approximately 1,700 ft (1,697 ft) and three intervals selected for the met samples. A cross section of the SE-03 hole and nearby drilling is presented in Figure 13-5.

Figure 13-5: Metallurgical Hole SE-03 Section (Arizona Sonoran, 2020)



A total of three bulk samples were collected from three locations from SE-03. Each bulk sample consisted of two 55-gal drums of material to be used in a single column test requiring at least 880 lb (400 kg) of material per the protocols outlined by McClelland Laboratories, Inc. (McClelland). The two drums representing one test sample were arranged on a single pallet for shipment to McClelland in Reno, Nevada. McClelland Analytical Services Laboratory is an ISO 17025 accredited facility.

Table 13-9 provides the sample intervals selected from SE-03 for column testing.

Table 13-9: Open Pit SE-03 PQ Core Sample Intervals

Name	Drill Hole	From, ft	To, ft	Feed Size, inch	Weight,lb (kg)	Notes
BAR 1	SE-03	1,268.0	1,340.0	$-3^{1}/_{2}$	461.2 (209.2)	oxide
BAR 2	SE-03	1,340.0	1,415.0	$-3^{1}/_{2}$	519.9 (235.8)	oxide
BAR 3	SE-03	1,415.0	1.460.1	$-3^{1}/_{2}$	567.5 (257.4)	sulfide
BAR 4	SE-03	1,460.1	1,521.0	$-3^{1}/_{2}$	546.7 (248.0)	sulfide
BAR 5	SE-03	1,521.0	1,574.8	$-3^{1}/_{2}$	537.5 (243.8)	sulfide
BAR 6	SE-03	1,574.8	1,627.5	$-3^{1}/_{2}$	533.5 (242.0)	sulfide

Barrels 1 and 2 were composited to form Sample 4600-001 (oxide); barrels 3 and 4 were composited to form Sample 4600-002 (higher grade enriched sulfide); and barrels 5 and 6 were composited to form Sample 4600-003 (lower grade enriched sulfide).

13.4 Hydro-Metallurgical Testwork – Open Pit

13.4.1 Sample Characterization

The drill core samples collected were shipped to McClellan Laboratories in Reno, Nevada, for preparation and analysis. McClelland has demonstrated prior experience in copper leach testing and associated protocols. A summary of the samples head assay information is provided in Table 13-10.

Table 13-10: Composite Head Assay (McClelland, 2020)

Determination	% Cu		
Sample	**4600-001**	**4600-002**	**4600-003**
Direct Assay, Init	0.844	2.510	0.613
Direct Assay, Dup	0.844	2.350	0.613
Direct Assay, Trip	0.827	2.440	0.627
Direct Assay, (Seq. Assay)	0.810	2.350	0.597
Calc'd., BRT, -10M	0.778	2.313	0.584
Calc'd., Head Screen	0.835	2.310	0.687
Calculated, Column Test, -3 inch			
Average	**0.823**	**2.379**	**0.620**
Std. Deviation	0.025	0.080	0.036
Relative Std. Deviation, %	3.0	3.4	5.8

Preliminary bottle testing has been completed on splits from each sample composite. Material for a column test on each of the three composite samples has been crushed to -3 inch, screened and loaded into columns for kinetic testing.

13.4.2 Sample Mineralogy

Mineralogy work by Process Mineralogical Testing Ltd. using a rapid mineral characterization testing method was also conducted on the sulfide sample composites to better understand the sulfide mineralization present and other factors that could influence bioleaching success. PLC's Rapid Ore Characterization (ROC) Report DEC 2020-03 dated 18 January 2021 is summarized as follows.

- The examined samples show that the copper deportment is dominated by secondary copper minerals of chalcocite/digenite in both samples. Moderate amounts are present as covellite with minor amounts pre-sent as Cu-bearing goethite (Cuprous goethite) and to a lesser extent as primary sulfides of chalcopyrite / bornite.
- Pyrite (py) is present in moderate amounts (approximately 10%) in sample 4600-002 and present in minor amounts (approximately 1%) in 4600-003. The 4600-003 sample also contains significant amounts of K-feldspar which sets it apart from 4600-002.
- Sample 4600-002 contains minor amounts of secondary copper minerals overall (approximately 6%) whereas 4600-003 contains lesser amounts (approximately 2%) of overall Cu-bearing minerals. Cu-minerals in both samples are fairly coarse, both demonstrating an 80% passing size of approximately 100 µm.
- Covellite is present in greater amounts in 4600-002 comprising 20% of the Cu in this sample, where it is still a significant contributor to the Cu content in 4600-003 but in lesser amounts.
- The Cuprous Fe-oxy hydroxide phase is essentially Cu-bearing goethite and would be a source of loss in a flotation circuit. Leaching of Cu from this phase may be limited.
- Porosity of the coarse particles is approximately 2% volume overall in both samples.
- Clay minerals are present in both samples in minor amounts.

Table 13-11 provides the PLC mineralogical determinations for the samples provided.

Table 13-11: Sulfide Composite Mineralogy (PLC, 2021)

Mineral Abundance	4600-002	4600-003
Mass %	100.00	100.00
Pyrite / Pyrrhotite	9.77	0.97
Chalcocite / Digenite	4.57	1.20
Covellite	1.22	0.17
Chalcopyrite / Bornite	0.09	0.04
Other Sulphides	0.15	0.03
Cuprous FE-Oxy Hydro	0.90	0.54
Alumino-Phospho-Sulphate	0.57	0.01
Quartz	51.30	54.60
Plagioclase	1.45	0.74
K-Feldspar	7.93	26.60
Muscovite / Sericite	19.10	12.90
Biotite	1.34	0.94
Clays	1.52	1.05
Other Minerals	0.10	0.17
Total	**100.00**	**100.00**
Porosity (Volume %)	1.93	2.44
Copper Deportment		
Chalcocite / Digenite	74.60	82.9
Covellite	20.60	12.9
Chalcopyrite / Bornite	1.05	0.88
Other Sulphides	0.03	0.01
Cuprous Goethite	3.35	2.73
Alumino-Phospho-Sulphate	0.44	0.63
Total	**100.00**	**100.00**

Copper deportment for the sulfide composite samples by selected size fraction is presented in Table 13-12.

Table 13-12: Sulfide Composite Copper Deportment by Size Fraction (PLC, 2021)

Copper Deportment	**4600-002**			
Fraction	**+1 mm**	**+150 um**	**-150 um**	**Head**
Mass %	20.00	41.00	39.00	100.00
Pyrite-Chalcocite Transition	28.50	30.30	21.10	26.30
Chalcocite	62.20	66.00	74.20	68.40
Bornite	1.77	0.64	1.13	1.05
Other Sulphides	0.11	0.01	0.03	0.04
Biotite	0.78	0.25	0.18	0.33
Others	6.64	2.84	3.34	3.79
Total	**100.00**	**100.00**	**100.00**	**100.00**
Copper Deportment	**4600-003**			
Fraction	**+1 mm**	**+150 Um**	**-150 um**	**Head**
Mass %	24.00	40.00	36.00	100.00
Pyrite-Chalcocite Transition	21.10	11.00	24.80	18.40
Chalcocite	74.20	85.80	70.90	77.60
Bornite	1.13	0.67	0.49	0.71
Other Sulphides	0.03	0.00	0.01	0.01
Biotite	0.18	0.23	0.09	0.17
Alumino-Phospho-Sulphate	3.34	2.32	3.72	3.07
Total	**100.00**	**100.00**	**100.00**	**100.00**

Photomicrographs of polished sections were also completed for the samples evaluated. Figure 13-6 is from the 4600-002 composite and presents an SEM-BSE image showing a particle with a breccia texture consisting of quartz (qtz, with minor potassic feldspar) with Fe- oxy / hydroxide infilling.

The Fe-oxy / hydroxide contains minor Cu (hence cuprous) and fine-grained inclusions of secondary Cu sulfides (white, SecCuS) such as covellite / chalcocite. Coarser pyrite grains show rims of secondary Cu sulfides.

Inclusion of copper mineral may represent the need for longer leaching timeframes as pyrite must first be leached to expose copper mineralization. The higher (approximately 10% py) than typical (1%-2% py) pyrite content is evident.

Figure 13-6: Sample 4600-002 Column Composite Material

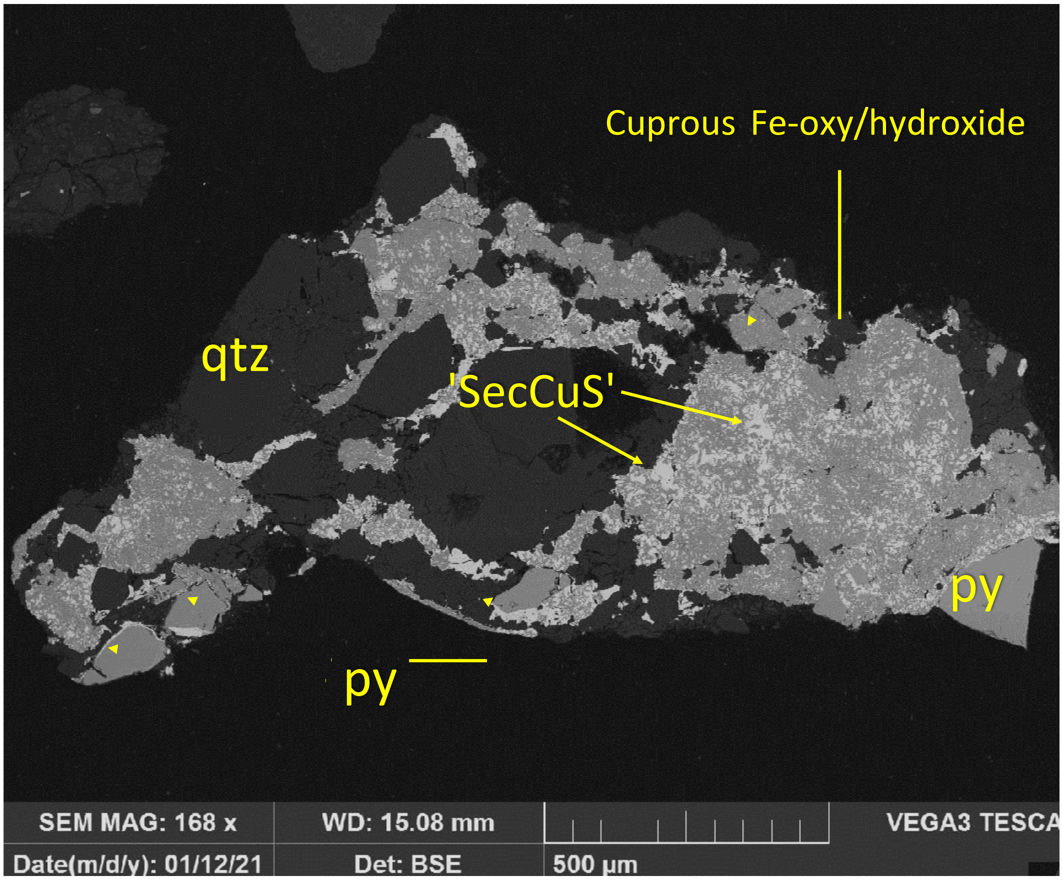
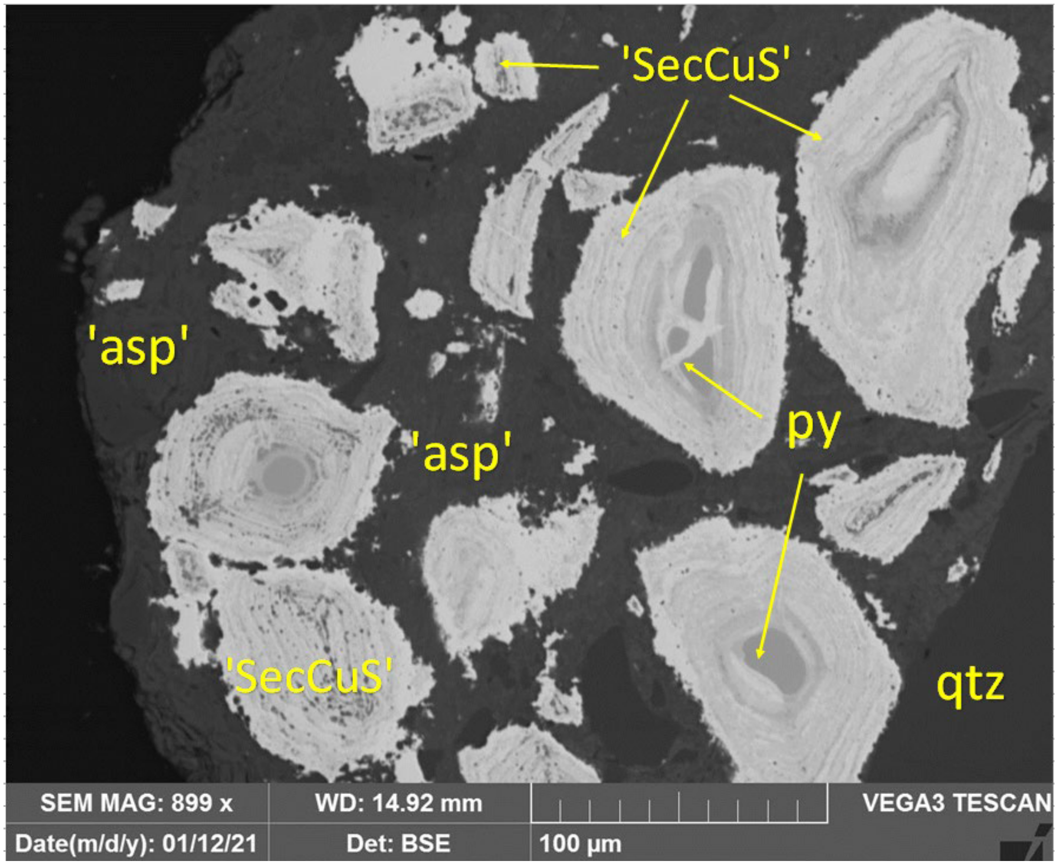


Figure 13-7 is also from the 4600-002 composite and presents an SEM-BSE image showing SecCuS developed from replacement of pyrite (preserved as cores; py).

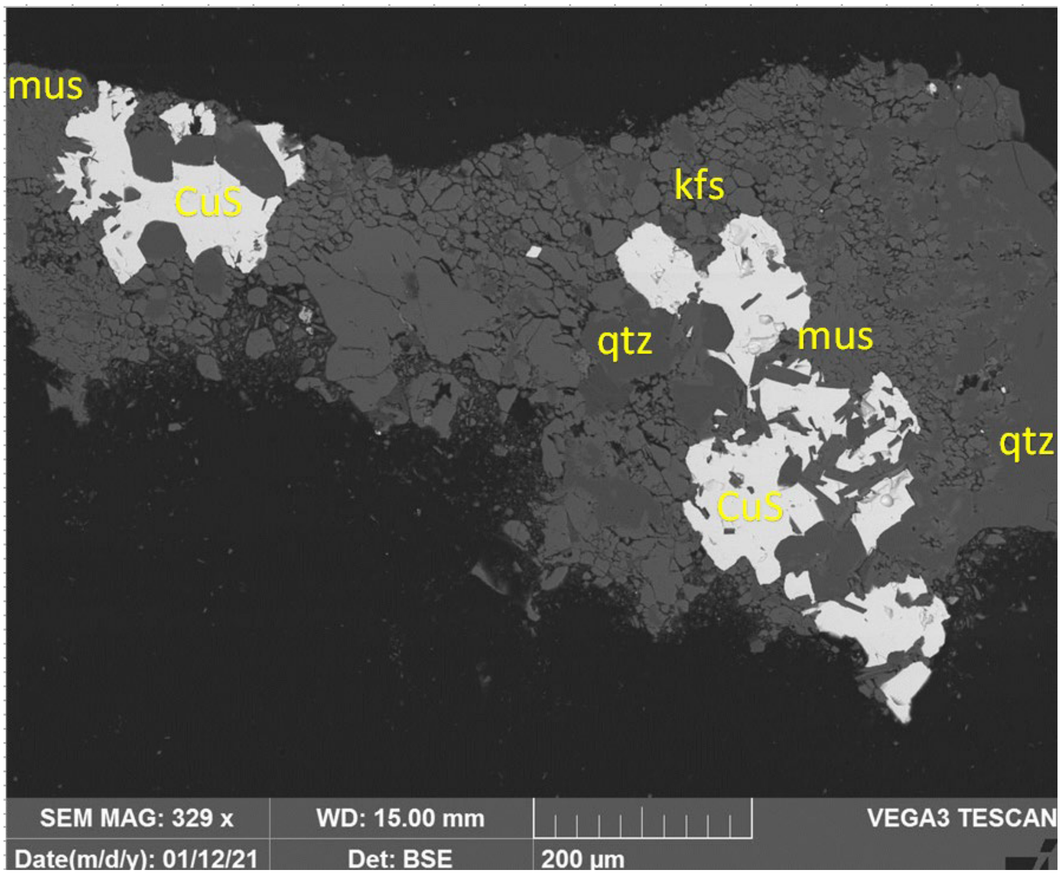
Figure 13-7: Sample 4600-002 Column Composite Material



The secondary Cu-sulfides are likely zoned with respect to copper and have a composition between covellite and chalcocite with minor iron. The sulfides are surrounded by a microcrystalline matrix of (tentatively identified) alumino-phospho-sulfates (asp) with trace amounts of copper. Fragments of quartz are also observed in this unit.

Figure 13-8 is also from the 4600-003 composite and presents an SEM-BSE image of quartz-feldspar-muscovite particle with infilling, massive secondary sulfide mineralization (covellite / chalcocite). Sulfide encapsulation may indicate size sensitivity (leach or flotation process options).

Figure 13-8: Sample 4600-003 Column Composite Materials



13.4.3 Bottle Roll Testing

A split of each sample was taken, prepped to 100% -10 mesh and subjected to bottle roll acid testing for 24 hours. Two main parameters were to be demonstrated from the work: a maximum acid consumption; and a maximum CuAS recovery. The summary of the results is presented in Table 13-13.

Table 13-13: Summary Metallurgical Bottle Roll Test Results (McClelland, 2020)

Sample	Cu Recovery, % of Total	%Cu				H2SO4 Added, lb/ton Ore	H2SO4 Consumption		
		Extracted	Tail	Calculated Head	Head Assay		Gross lb/t Ore	Gangue lb/t Ore	Specific (Gangue) lb/lb Cu
4600-001 (Oxide)	90.7	0.706	0.073	0.778	0.838	57.6	14.2	6.7	0.5
4600-002 (Sulfide)	8.8	0.204	2.130	2.313	2.433	32.8	6.5	6.7	1.7
4600-003 (Sulfide)	11.3	0.066	0.523	0.584	0.618	38.7	4.7	7.3	5.5

Results from the bottle roll testing suggest oxide copper recovery can be expected to be high with significantly lower acid consumption versus the prior Stockpile Project leach testing.

Sulfide results indicate the potential for low copper recovery in an acid only test with the sulfide mineralization composites.

13.4.4 Open Pit Copper Recovery

The open pit columns are currently in progress and results to date are considered preliminary. Tests are conducted in 12 inch I.D. by 10 ft tall columns containing approximately 880 lb (400 kg) of material. Column leach testing in closed circuit with SX was incorporated with saved solution from the Stockpile Project testing as a starting solution.

Until tails assays are confirmed, copper recovery based on head assays are only indicative and significant variations may be present. Soluble copper-based extraction for column.

4600-001 (oxide) is shown in Figure 13-9. This column has been completed, and assay and screen size data are available to use for soluble copper components; however, a final CuT was not available to confirm total copper metrics.

Results for the oxide columns are consolidated in Figure 13-9 and Table 13-14.

Figure 13-9: Oxide Copper Column Copper Extraction

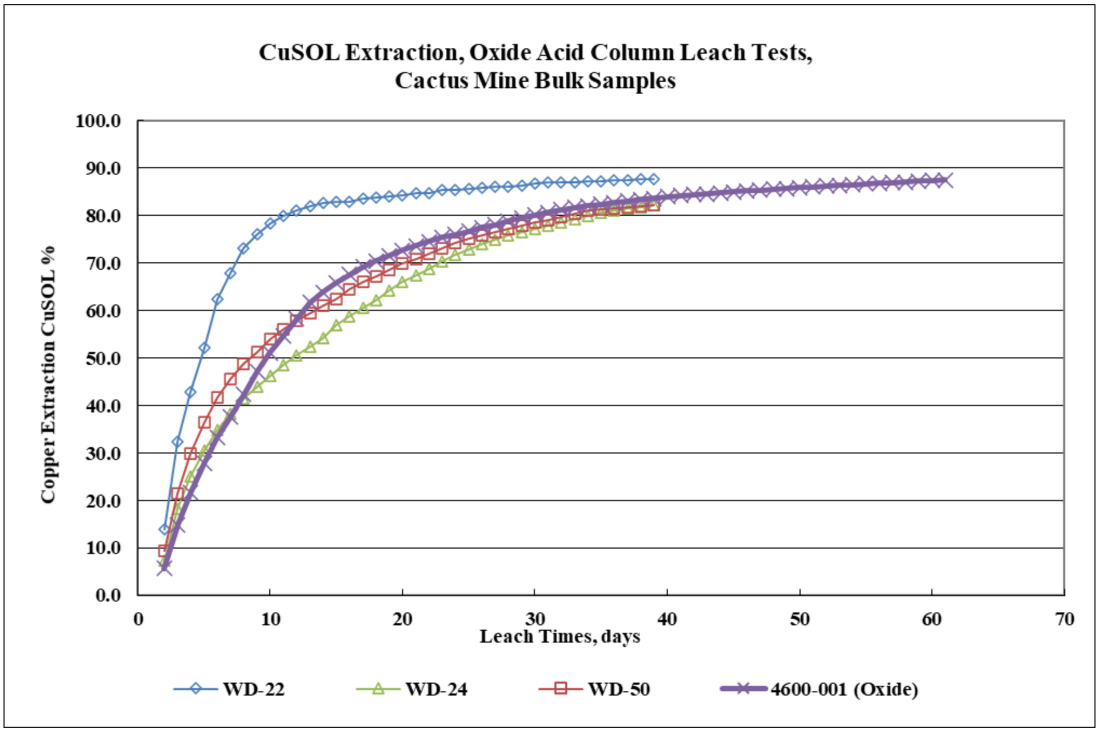


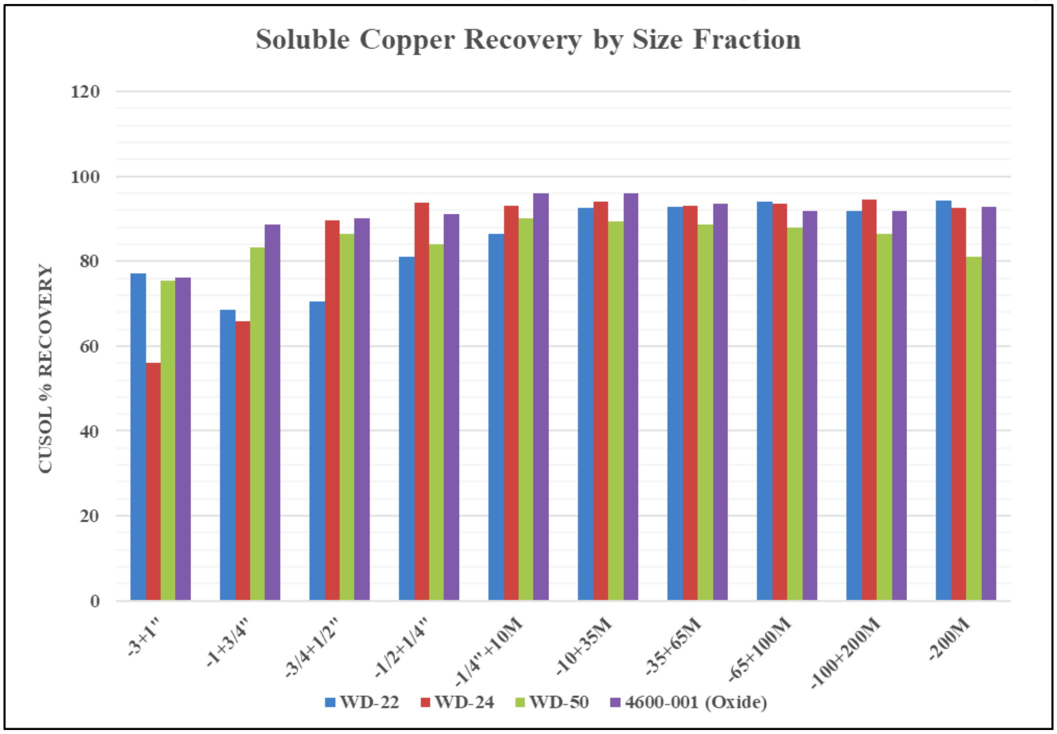
Table 13-14: Consolidated Oxide Column Results to Date

Summary Metallurgical Results, Oxide Acid Column Leach Tests, Cactus Mine Bulk Samples, -3-inch" Feed Size											
MLI Test No.	Sample	Leach / Rinse Time, days	TCu Recovery %	Extracted	% Cu Calc'd. Tail	Head	Avg. Head	H2SO4 Added lb/ton ore	H2SO4 Consumption Gross lb/ton ore	Gangue lb/ton ore	Specific (Gauge), lb/lb Cu
AC-1	WD-22	50	75.9	0.082	0.026	0.108	0.119	72.8	31.0	28.5	17.4
AC-2	WD-24	50	74.8	0.193	0.065	0.258	0.215	72.9	21.7	15.7	4.1
AC-3	WD-50	50	70.4	0.112	0.047	0.159	0.151	71.7	25.7	22.3	9.9
AC-1	4600-001 (Oxide)	61	TBD	0.685	TBD	TBD	0.832	120.4	26.8	5.7	0.4

The completed Stockpile Project oxide materials columns are included for comparison. Preliminary results indicate that open pit oxide material should perform in a similar manner to the Stockpile Project oxide material.

Soluble copper based extraction for all three open pit columns 4600-001 (AC-1 oxide), 4600-002 (AC-2 sulfide) and 4600-003 (AC-3 sulfide) is shown in Figure 13-10. Results for these columns are based on head assay information and should be considered indicative.

Figure 13-10: Soluble Copper Recovery by Size Fraction (McClelland, 2020, 2021)



A breakdown of copper recovery for CuAS (acid soluble), CuCN (cyanide soluble), TSol (combined CuAS and CuCN) and the CuT is presented in Table 13-15. Extraction estimates are based on head and tail assay data for the column tests.

Table 13-15: Copper Extraction by Copper Assay Method

Copper Recovery (%) @ 39 Days Leach/4 Days Drain and 7 Days Rinse												
	WD-22		WD-24		WD-50		4600-01		AVG	Predicted/Modeled		
	Assay %	Recv %	Assay %	Recv %	Assay %	Recv %	Assay %	Recv %		180 days	YR 1	YR 2
CuAS	80.6%	97%	63.6%	89%	69.8%	94%	90.9%	94%	94%	85%	75%	10%
CuCN	17.6%	64%	5.8%	39%	10.7%	30%	5.9%	60%	48%	75%	35%	40%
CuSOL	98.1%	88.2%	69.4%	82.3%	80.5%	82.2%	96.8%	92.4%	86%	83.3%	68.2%	15.1%
CuSOL Pred		81.7%		58.4%		67.4%		81.7%	72%			
CuT		76%		75%		70%		TBD	74%	71.9%	59.0%	12.9%

CuAS copper content extraction averaged 94% for the four columns completed. CuCN recovery, representing enriched copper mineralization content (chalcocite and covellite) was also significant and averaged 48%, with a high of 64% in column WD-22. The combined TSol extraction averaged 86% for the material tested.

The screen analysis for the samples as loaded into the columns is presented in Table 13-16. The effective P80 size distribution was 1.5 inch for the three Stockpile Project columns and approximately 1 inch for the open pit sample.

Table 13-16: Column Screen Size Analysis (McClelland, 2020, 2021)

Size Fraction	WD-22 Weight, %		WD-24 Weight, %		WD-50 Weight, %		4600-01 Weight, %	
	Head	Tail	Head	Tail	Head	Tail	Head	Tail
-3"+1"	28.4	21.9	29.8	27.0	31.6	22.6	16.3	15.6
-1+3/4"	6.6	7	5.7	5.5	5.9	4.6	22.5	10.8
-3/4+1/2"	11.8	10	7.9	7.2	9.4	6.8	13.7	10.1
-1/2+1/4"	9.7	10.3	11.0	13.0	10.1	12.0	13.8	12.3
-1/4"+10 M	14	13.9	16.0	16.4	14.2	14.5	15.2	18.2
-10+35 M	12.2	14.5	12.1	12.7	11.2	14.6	7.3	12.8
-35+65 M	4.5	6.3	4.7	4.6	4.1	6.1	2.5	4.1
-65+100 M	2.2	3	1.7	1.6	2.2	2.6	1.0	1.4
-100+200 M	3.2	2.8	3.0	2.8	2.8	3.7	1.3	2.5
-200 M	7.4	10.3	8.1	9.2	8.5	12.5	6.4	12.2
Composite	100	100	100	100	100	100	100	100
P80	~1.5"		~1.5"		~1.5"		~1"	

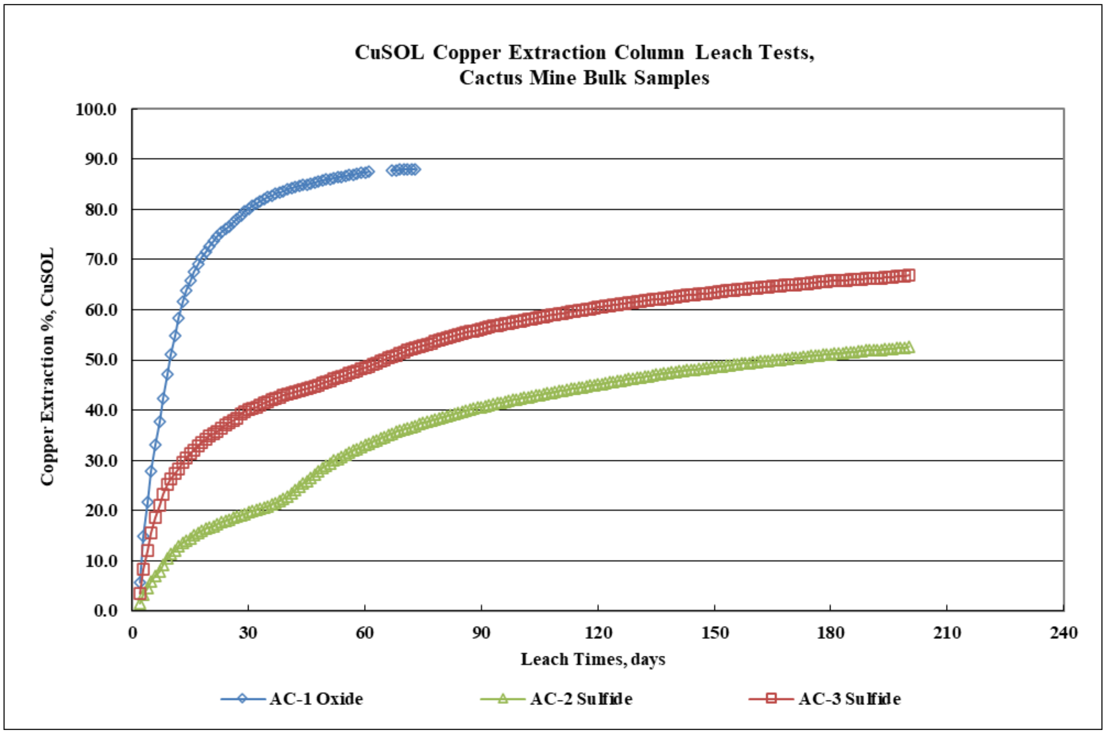
Copper extraction was also considered by size fractions to assess the impact of particle size. Figure 13-10 shows the relative copper extraction by size fraction.

Copper extraction appears to be impacted in particles sizes over 1 inch in these tests. Future testing will need to consider a larger distribution of sizes above 1 inch to fully evaluate the relative significance of these results to the run-of-Stockpile Project resources.

Columns AC-2 and AC-3 were started with biomass produced from mineralized samples from Cactus drill core added to the leaching solutions to simulate a mature bioleaching system.

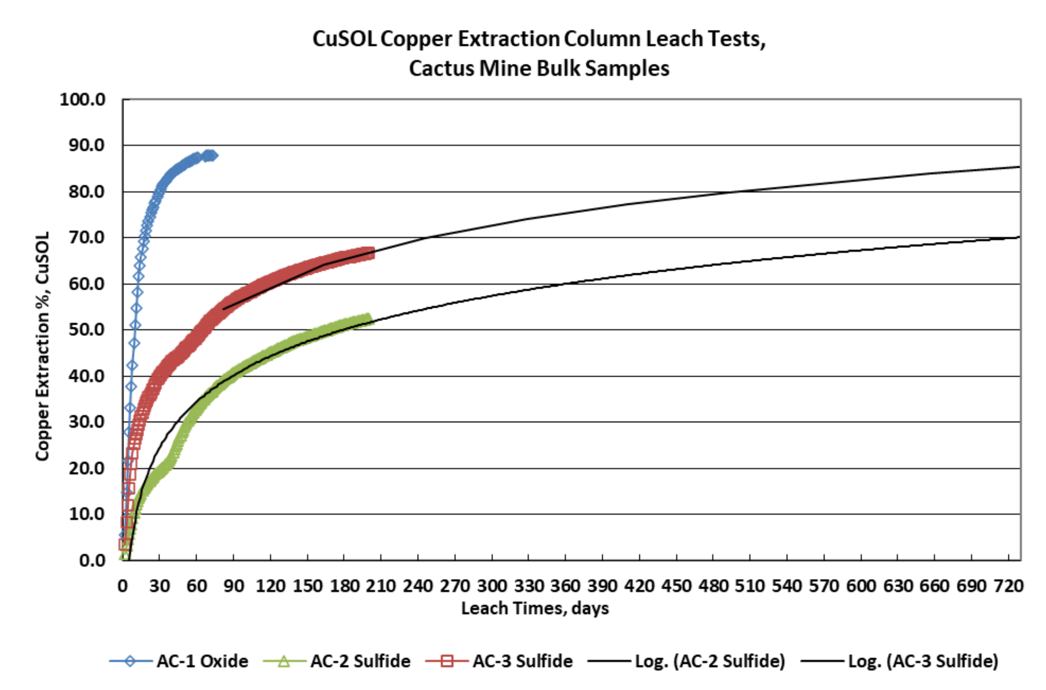
Copper extraction for the columns is shown in Figure 13-11. Column tests AC-2 4600-02 and AC-3 4600-03 are in progress and results are indicative only.

Figure 13-11: Soluble Copper Extraction for Open Pit Columns



Based on the indicative results for sulfide materials at 200 days of leaching to June 2021, a longer leaching time will be required to achieve copper extraction of 70% to 75% for the soluble copper components as demonstrated in Figure 13-12. Column results indicate a minimum of two years will be required to achieve extraction. Mineralogical and grade variability between the two sulfide columns will require further testing and understanding to access applicability to the overall enriched resource at the Cactus Project.

Figure 13-12: Extrapolated Long-Term Copper Extraction



Mineralogy also suggests that gangue encapsulation and pyrite inclusion is present, which also indicating a longer leaching time requirement should be expected.

13.4.5 Open Pit Sulfuric Acid Consumption

Historically, ASARCO testing in 1968 suggested a gross acid consumption of approximately 20.8 lb/t for the Sacaton West fresh core material. Table 13-17 shows the bottle roll acid consumption information for the open pit core composites.

Table 13-17: Open Pit Column Material Bottle Roll Results Bottle Roll Tests, Cactus Project, 100%-10 M Feed Size, 24 Hour

		H_2SO_4 Consumption	
Sample	Gross lb/t ore	Gangue lb/t ore	Specific (Gangue) lb/lb Cu
4600-001 (Oxide)	14.2	6.7	0.5
4600-002 (Sulfide)	6.5	6.7	1.7
4600-003 (Sulfide)	4.7	7.3	5.5

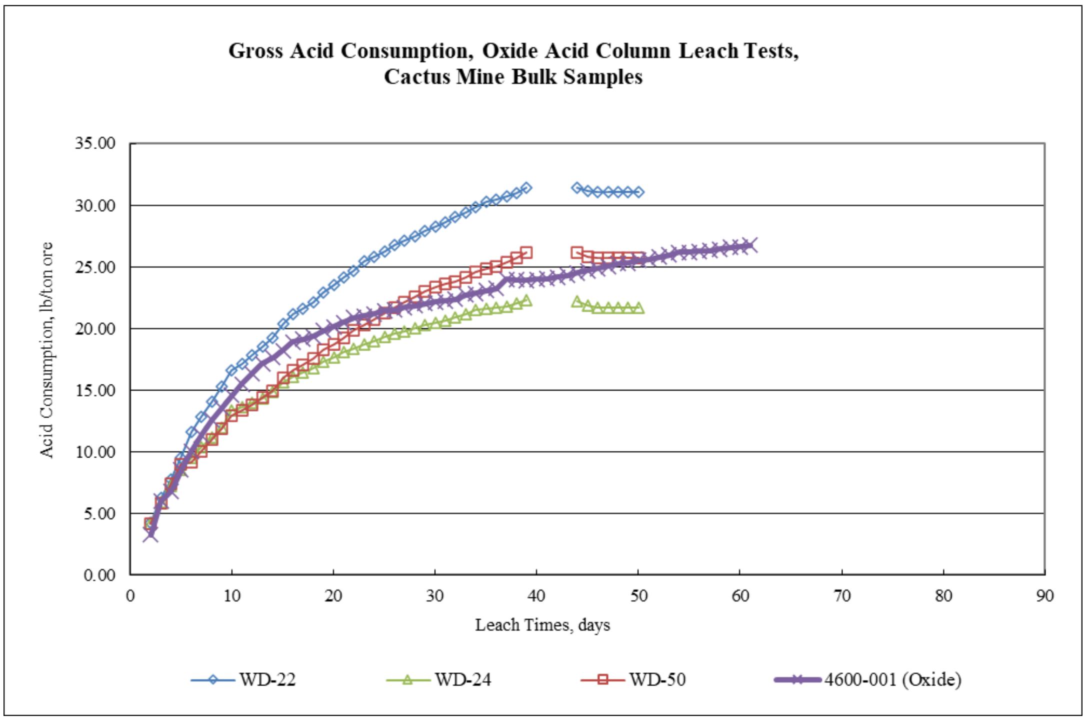
In general, the acid consumption indicated is significantly lower than the Stockpile Project materials tested so far. For the oxide materials, the comparisons are shown in Table 13-18.

Table 13-18: Column Testing Acid Consumption Results Net Acid Consumption (lb/t)

		WD-22	WD-24	WD-50	4600-01	AVG
Bottle Roll	Net	16.7	17.9	7.1	6.7	12.1
Column	Net	28.5	15.7	22.3	5.6	18.0
	Gross	31.0	21.7	25.7	26.8	26.3
Column lb acid / lb Cu		17.4	4.1	9.9	0.4	7.9

The higher copper grade in 4600-01 contributes to the lower net acid consumption and unit consumption results. Gross acid consumption for the materials ranged from 21 lb/t to 31 lb/t as shown in Figure 13-13. WD-22 does show that there is the presence of higher acid consuming gangue material in the material expected to be leached. Although known gangue acid consumers, such as calcite, have not been generally described in the geology of the deposit to date there is the potential for localized variations.

Figure 13-13: Gross Acid Consumption Column Test Results



Sulfide content of the enriched materials will also contribute acid from the oxidation of the contained sulfur as leaching progresses. Net acid consumption will be both a function of copper grades and sulfide content leached. Figure 13-14 and Figure 13-15 shows the indicative gross and net results to date from the ongoing sulfide column testing, respectively.

Figure 13-14: Gross Acid Consumption Column Test Results

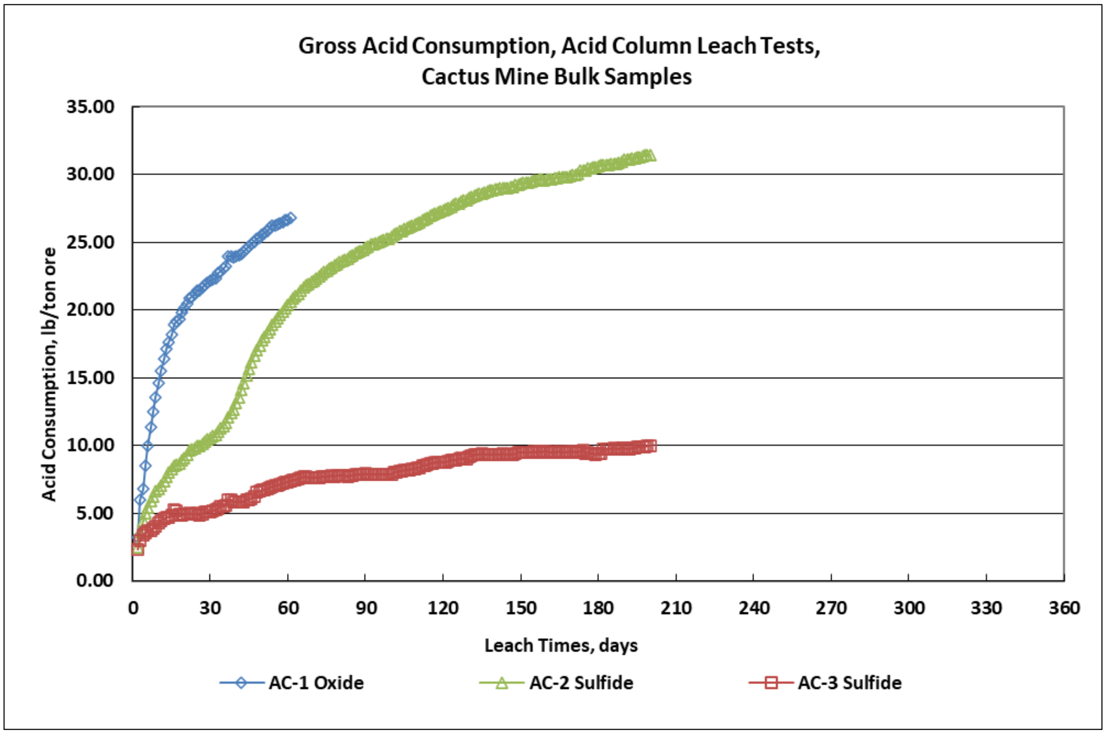
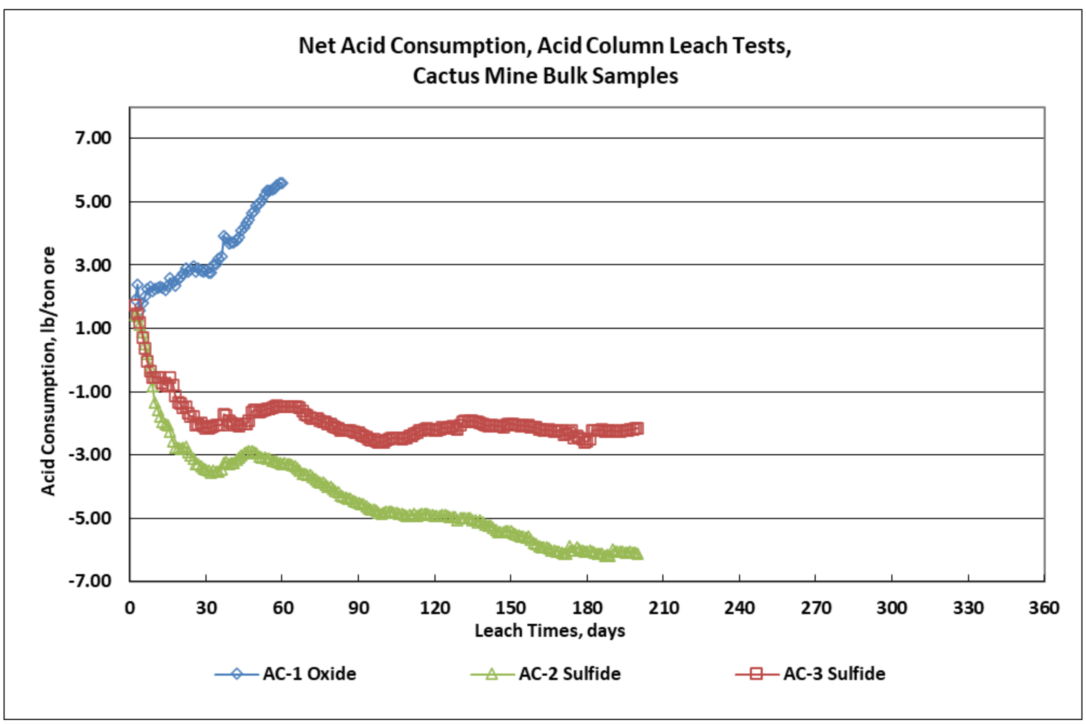
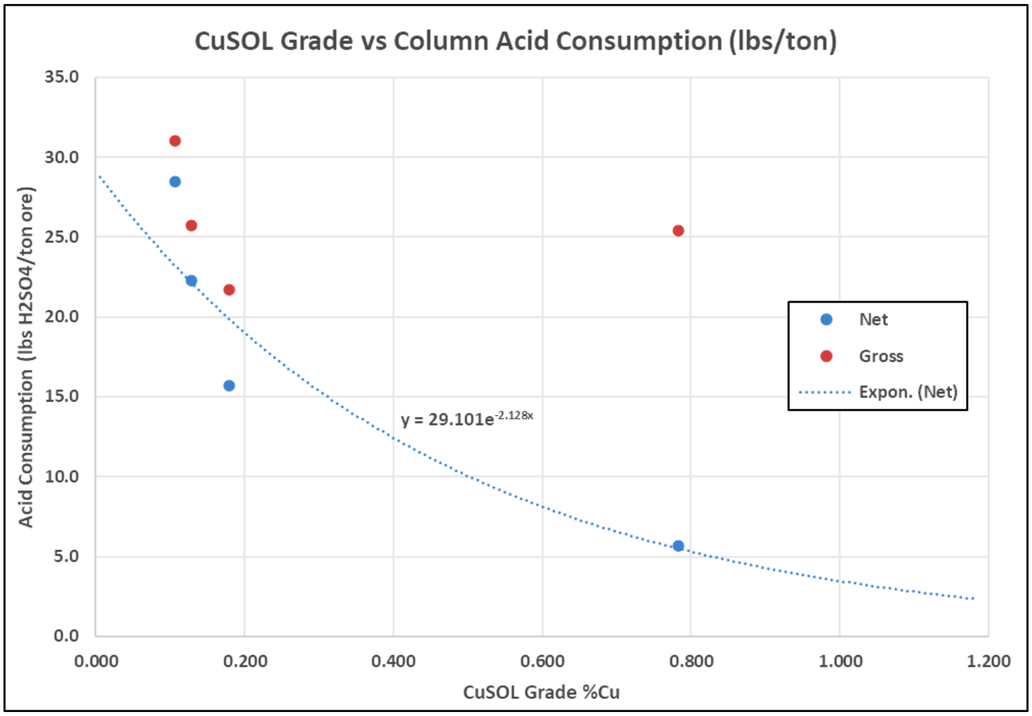


Figure 13-15: Net Acid Consumption Column Testing Results



Due to the higher copper content and sulfide mineralization oxidation, the sulfide columns are presently net acid producing. This may be an advantageous feature once sulfide material is mined. The relationship between copper grade and net acid consumption is still under development and based on several factors, including recovery of copper. However, a preliminary analysis is presented in Figure 13-16 that demonstrates the potential relationship at the Cactus Project.

Figure 13-16: Copper Grade Versus Net Acid Consumption



Continued column leach testing in closed circuit with SX for the sulfide materials and will provide a more considered value for acid consumption in future and any correlations to the bottle roll results.

13.5 Concentrator Opportunity Scoping

13.5.1 Introduction

ASARCO mined material for the Sacaton West ore body and milled ore containing primarily primary sulfide mineralization consisting mainly of pyrite and chalcopyrite with chalcocite. Ore processing was conducted in a 11,000 t/d (9,000 t/d initially designed) copper mill and concentrator operation to produce copper concentrates for processing in ASARCO owned smelters between 1974 and 1984. In addition to copper, the ore contains minor amounts of molybdenum and traces of gold and silver.

Very limited detail data from the historic ASARCO Sacaton concentrator operations has been recovered for review and consideration.

The potential to process higher grade enriched and the primary sulfide copper containing materials not considered economically suitable for heap leaching techniques was investigated on a preliminary basis as part of the current test work. The preliminary testing competed was aimed at the potential for employing modern comminution methods and flotation reagents to the Cactus (previously Sacaton) materials to improve historic recovery performance and concentrate grades.

Four composites were taken from two metallurgical recent drill holes into the proposed open pit/underground primary copper ore resource for scoping level comminution / flotation testing at McClelland Labs. With limited historic information, these samples were tested using the historic reagent and grinding scheme and compared to a more modern flotation scheme of developed for chalcocite dominant ore types. No optimization was conducted.

13.5.2 ASARCO Historic Process Plant

The processing facilities operated by ASARCO between 1974 and 1984 are described by Briggs (2004) as follows.

Run-of-mine ore was dumped in a 165-t coarse ore bin from which it passed through a vibrating grizzly feeder. The grizzly oversize reported to a primary 48-inch by 60-inch Allis-Chalmers jaw crusher. The primary crusher discharge and grizzly undersize (6-inches) were combined and conveyed to an intermediate stockpile, which had a live capacity of 4,750 ft.

This ore was recovered by four vibrating pan feeders and passed through a vibrating screen. The screen oversize reported to a secondary 7 ft Symons standard cone crusher, while the undersize (1.5 inch) reported to a 200-t capacity surge bin.

Two 72-inch belt feeders recovered the ore from the surge bin and it was conveyed to one of two vibrating screens. The screen oversize reported to a tertiary 7 ft Symons shorthead cone crusher, while the screen undersize (0.5-inch) reported to one of two fine ore bins, which each had a live capacity of 3,250 t. The tertiary crusher discharge was combined with the secondary crusher discharge and returned to the surge bin.

The fine ore bins fed a single-stage grinding circuit, consisting of two 15.5 ft diameter by 18 ft Allis-Chalmers ball mills, which were each operated in closed circuit with a cluster of six, 20 -inch Krebs cyclones. The grinding cyclone overflow (50%-55% minus 200-mesh) was split by a rougher feed distributor and fed to two banks of twelve conventional 300 ft3 rougher flotation cells. The rougher concentrates were directed to a cluster of three middling cyclones. The middling cyclone overflow (minus 325-mesh) reported to the 100-foot diameter middling thickener, while the underflow was classified by a cluster of four regrind cyclones.

The regrind cyclone underflow reported to a 9.5 ft diameter Allis-Chalmers regrind ball mill, which was operated in closed circuit with the regrind cyclones. The regrind cyclone overflow was combined with the middling thickener underflow and was treated by three banks of first cleaners (each bank containing five 100 ft3 cells), scavengers (each bank containing five 100 ft3 cells) and second cleaners (each bank containing three 100 ft3 cells) to produce a final copper concentrate product that reported to a 75 ft diameter concentrate thickener.

The concentrate thickener underflow was dewatered by two 10 ft diameter by 12 ft Eimco drum filters to produce the final copper concentrate product that was ship via rail to a local smelter.

Oxide dominant material was typically sent to the stockpile. Treatment of oxide material contained within the mill feed required special handling. When encountered, the final four cells within each of the rougher banks produced an oxide concentrate product. By-passing the regrind circuit, this material underwent a single stage of cleaning prior to reporting to the concentrate thickener.

The tails from the roughers and scavengers were combined and reported directly to a 275 ft diameter tailings thickener. The tailings thickener overflow was returned to the reclaim water reservoir, while the underflow was pumped the tailings impoundment.

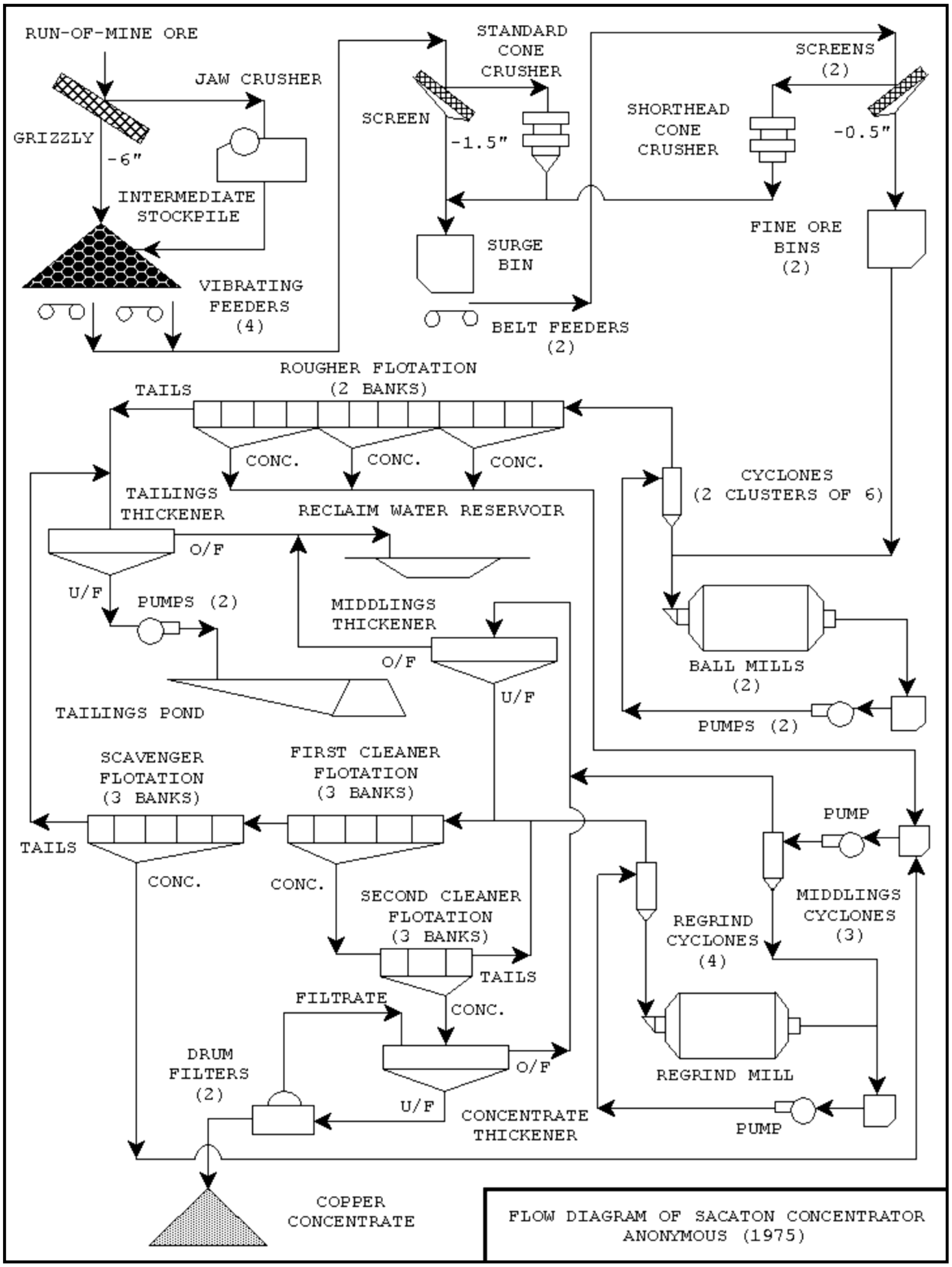
The flotation plant used the following scheme, typical for copper flotation circuits in the 1970s and 1980s. As reported by Briggs, the initial reagent scheme employed was as follows.

- Sulfide Ore Reagent Scheme:
- Lime: 2.0 lb/t (1975)
- A-238: 0.021 lb/t (1975)
- Z-6: 0.020 lb/t (1975)
- Frother: 0.06 lb/t (1975)
- Oxide Reagent Scheme:
- NaHS: 0.9 lb/t (1975)
- A-404: 0.015 lb/t (1975)
- Frother: 0.6 lb/t (1975)

Overall copper recovery reportedly ranged from 75% to 83% during the operations life. Gold and silver were recovered in the copper concentrate product. While molybdenum was known to be present in the ores processed, no record of molybdenum production was reported or found.

The process flow diagram for the prior Sacaton concentrator is shown in Figure 13-17 as included in the Briggs 2004 information.

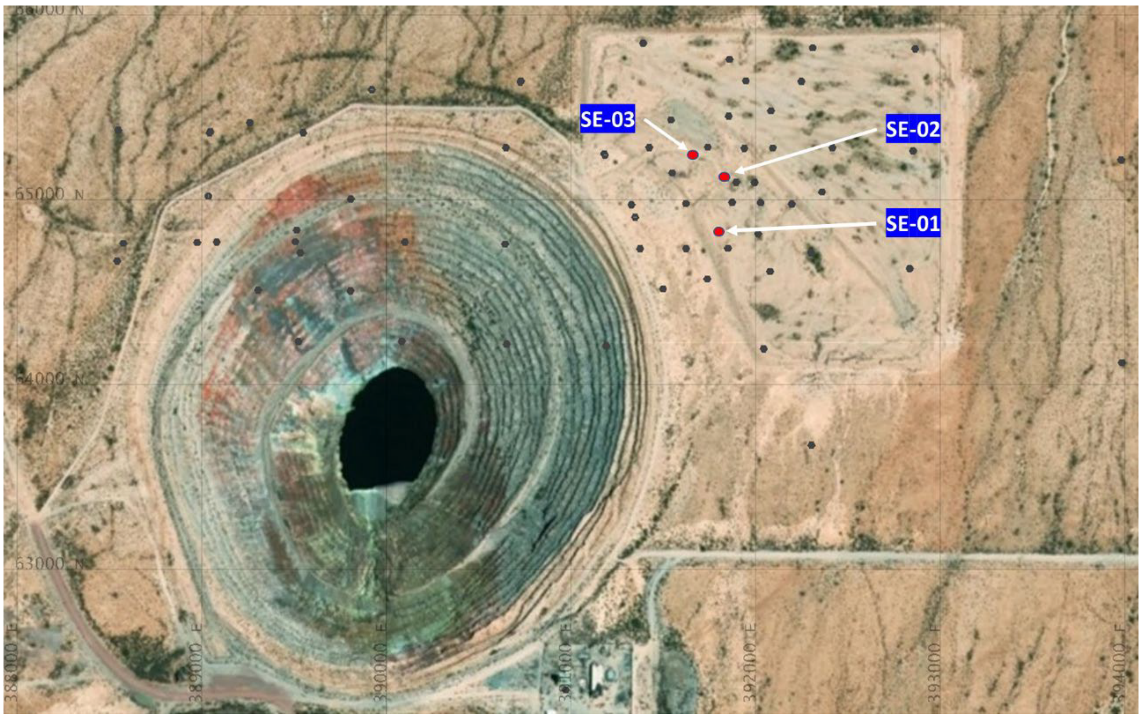
Figure 13-17: Historic Sacaton Concentrator Flow Diagram (Briggs)



13.5.3 Scoping Sample Descriptions

Four composites were taken from two metallurgical recent drill holes into the proposed open pit/underground primary copper ore resource at Cactus East for scoping level comminution/flotation testing at McClelland Labs. The metallurgical holes are designated as SE-01 and SE-02 with their locations as shown in Figure 13-18.

Figure 13-18: Metallurgical Sample Drillhole Location



- Sample 1666808 Description – Hole SE-01 (1,633 ft to 1,703 ft).
  - 1.38% Cu head grade, secondary copper sulfides (SecCuS): chalcocite/digenite and covellite (86.4% of total Cu); chalcopyrite (3.9% of total); bornite (2.1% of total); cuprous Fe- oxy/hydroxides (7.2% of total).
- Sample 1666809 Description – Hole SE-02 (1,413 ft to 1,890 ft).
  - 3.47% Cu head grade, SecCuS: chalcocite / digenite and covellite (91.9% of total Cu); cuprous Fe-oxy/hydroxides (3.8% of total); native copper (2.3% of total); chalcopyrite (0.8% of total).
- Sample 1666810 Description – Hole SE-0X (1,627.5 ft to 1,651.0 ft).
  - 0.36% Cu head grade, chalcopyrite (53.3% of total Cu); SecCuS: chalcocite / covellite (24.5% of total); bornite (11.2% of total); cuprous Fe oxy/hydroxides (8.6% of total).
- Sample 1666811 Description – Hole SE-0X (1,674.1 ft to 1,696.9 ft).
  - 0.49% Cu head grade, chalcopyrite (49.4% of total Cu); bornite (33.3% of total); SecCuS: chalcocite/covellite (14.4% of total).

A 110 lb (50 kg) sample of core pieces from SE-01 granite material was also provided for comminution testing (Bond Ball Mill (BMWi) and SMC testing).

CuAS represents 16%-20% of the copper contained in the two higher grade composites (1666808, 1666809). The two lower grade composites (1666810, 1666811) have a lower portion of the contained copper (8%-11%) reporting as acid soluble, but with high levels (HLs) (46%-49%) reporting as primary copper (chalcopyrite). Native copper, which is known to be present at Cactus in minor amounts on occasion, was detected in sample 1666809.

13.5.4 Comminution Scoping Work

The standard JK Drop-Weight test provides specific parameters for use in the JKSimMet Mineral Processing Simulator software. In JKSimMet, these parameters are combined with equipment details and operating conditions to analyze and/or predict SAG/autogenous mill performance. The same test procedure also provides material type characterization for the JKSimMet crusher model.

The SMC Test was developed by Steve Morrell of SMC Testing Pty Ltd (SMCT). The test provides a cost-effective means of obtaining these parameters, in addition to a range of other power-based comminution parameters, from drill core or in situations where limited quantities of material are available. The material specific parameters have been calculated from the test results and are supplied to McClelland in this report as part of the standard procedure.

SMC data for one sample from Cactus Project granite material was developed by Hazen Research for SMC test analysis. The sample was identified as 4655-001. The data were analyzed by JKTech to determine the JKSimMet and SMC Test comminution parameters. SMC Test results were forwarded to SMCT for the analysis of the SMC Test data. Analysis and reporting were completed on 09 March 2021. The results are reported by JKTech in "SMC TEST REPORT JKTech Job No: 21012/P4 Testing Date: February 2021".

The SMC Test results for the 4655-001 sample from Cactus Project are given in Table 13-19. This table includes the average rock density and the DWi (Drop-Weight index) that is the direct result of the test procedure.

Table 13-19: SMC Test Results (Hazen/JKTech 2021)

			Mi Parameters (kWh/t)			
Sample Designation	**DWi (kWh/m3)**	**DWi (%)**	**Mia**	**Mih**	**Mic**	**SG**
4655-001	1.01	2.00	4.40	2.20	1.10	2.63

The values determined for the Mia, Mih and Mic parameters developed by SMCT are also presented in this table. The Mia parameter represents the coarse particle component (down to 750 µm), of the overall comminution energy and can be used together with the Mib (fine particle component) to estimate the total energy requirements of a conventional comminution circuit. The derived estimates of parameters A, b and ta that are required for JKSimMet comminution modelling are given in Table 13-20.

Table 13-20: Parameters Derived by JKTech (JKTech 2021)

Sample Designation	A	b	t_a	SCSE (kWh/t)
4655-001	59.80	4.37	2.57	5.28

Also included in the derived results are the SAG Circuit Specific Energy (SCSE) values. The SCSE value is derived from simulations of a “standard” circuit comprising a SAG mill in closed circuit with a pebble crusher. This allows A*b values to be described in a more meaningful form. SCSE is described in detail in Appendix A.

In the case of the 4655-001 sample from Cactus Project, the A and b estimates are based on a correlation using the database of all results so far accumulated by SMCT.

The Cactus sample A*b = 261.3 and falls in the lowest 2% of the JKTech database values. Note that in contrast to the DWi, a high value of A*b (>80) means that an ore is relatively soft while a lower value (approximately 5-60) means that it is hard. The measured SCSE for the sample tested indicates that the Cactus material is generally soft and readily grindable in a typical SAG/Ball bill circuit with low energy requirements.

In addition, a standard Bond Ball Mill work index test was conducted by McClelland and reported for the Cactus ore sample sent to Hazen. The results of that testing are provided in Table 13-21. Results indicate a medium rating for the sample tested.

Table 13-21: Bond Ball Mill Work Index Testing Results (McClelland 2021)

Ball Mill Work Index	11.29	kW-hr/st
	12.45	kW-hr/mt
Ball Mill Work Index Classification	Medium	

Additional variability testing is required to confirm these initial results for a circuit design.

The influence of particle size on the specific comminution energy needed to achieve a particular t10 value can also be inferred from the SMC Test results. After breaking all 20 particles in a set, the broken product is sieved at an aperture size, one tenth of the original particle size. Therefore, the percent passing mass gives a direct reading of the t10 value for breakage at that energy level. For crusher modelling the t10-Ecs matrix can be derived. This is done by using the size-by-size A*b values that are used in the SMC Test data analysis to estimate the t10-Ecs values for each of the relevant size fractions in the crusher model matrix.

The energy requirements for five particle sizes, each crushed to three different t10 values, are presented in Table 13-22.

Table 13-22: Comminution Energy Requirements (JKTech, 2021)

Sample Designation	Particle Size (mm)														
	14.5			20.6			28.9			41.1			57.8		
	t_{10} Values (%) for Given Specific Energies in kWh/t														
	10	20	30	10	20	30	10	20	30	10	20	30	10	20	30
4655-001	0.05	0.12	0.21	0.05	0.11	0.18	0.04	0.09	0.16	0.04	0.08	0.14	0.03	0.07	0.12

13.5.5 Preliminary Flotation Scoping Work

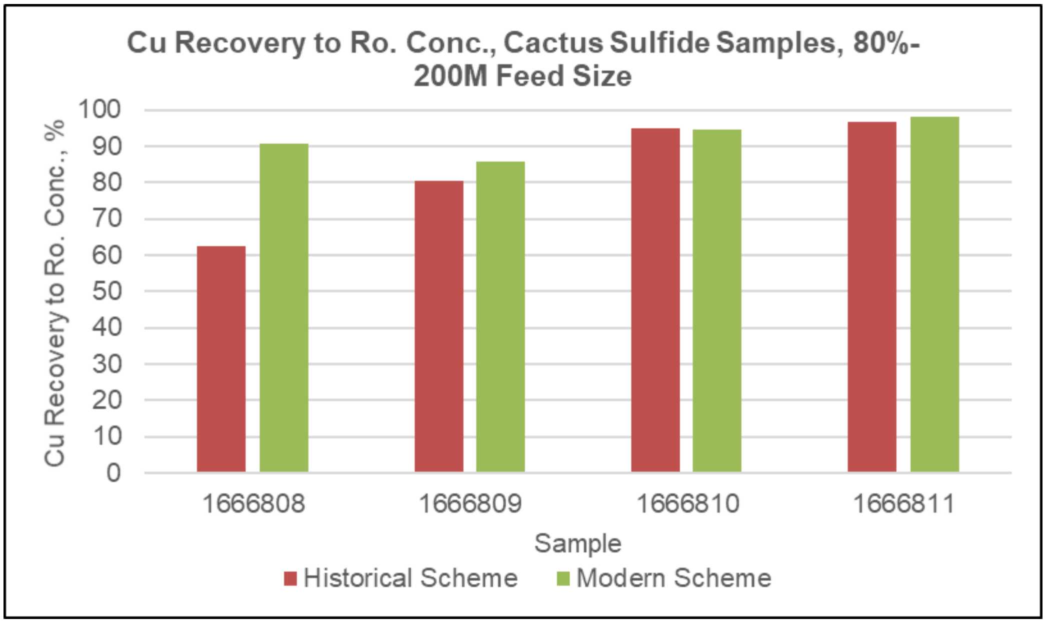
The four composites were tested in a single stage rougher using a 2.2 lb (1 kg) charge. Samples were prepped and ground to a P80 of approximately 200 mesh for all tests, commensurate with the reported grind used by ASARCO.

Two regent schemes were employed, the first to simulate the response using the ASARCO reagent scheme and the second in consultation with Solvay as an initial demonstration of what more modern reagent schemes targeting chalcocite/chalcopyrite could do to simplify and improve upon the ASARCO flowsheet.

The historic scheme used a combination of PAX, Aero-238, Aerofroth 65 and pH 5.5-7.5. The modern scheme was developed in consultation with Solvay and represents and initial option that considers Aero XD-5002/Aero 8944, Aerofroth 65 and pH 10.5-11.5.

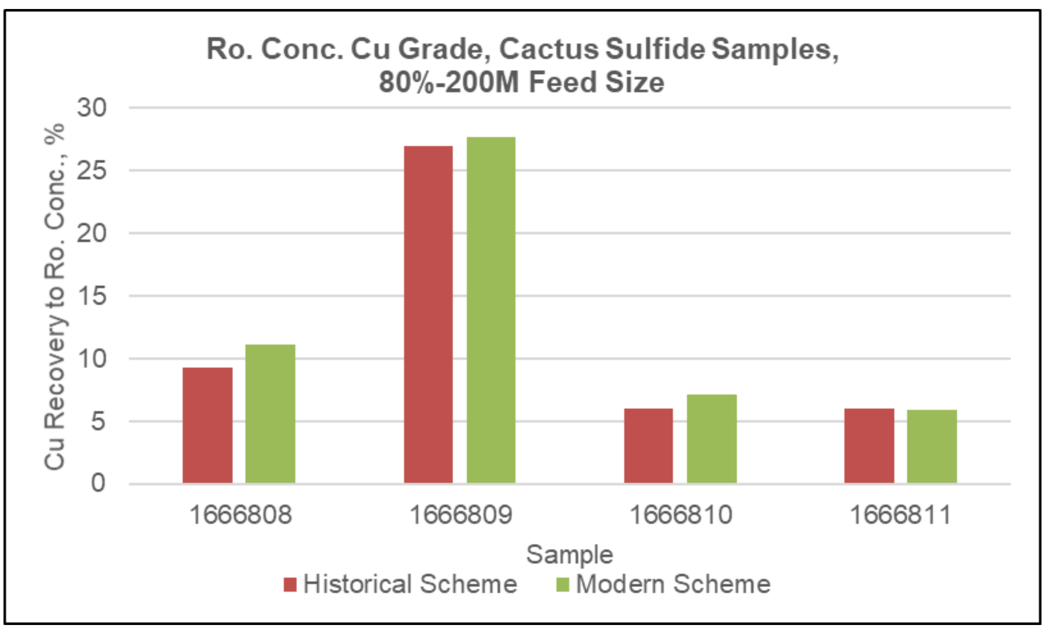
The preliminary rougher copper recoveries obtained is presented in Figure 13-19. Based on the initial testing results, while not optimized, copper flotation recoveries approaching 90% appear to be reasonable, and significant improvement in the oxide copper components (sample 1666808) are apparent.

Figure 13-19: Preliminary Rougher Recovery Results (McClelland 2021)



The associated rougher concentrate grade are also presented in Figure 13-20. These results are typical and higher grade feed resulted in higher grade concentrates. The very high rougher concentrate grade is not typical and may be influenced by the presence of native copper and may also indicate that an improved recovery could be achieved in future testing. These results provide positive starting points for saleable final concentrate grades once locked cycle testing is completed.

Figure 13-20: Rougher Concentrate Grade



13.6 Results Summary and Conclusions

13.6.1 Stockpile Project Metallurgical Testwork

Based on the preliminary scoping testwork completed for Stockpile Project materials, the following observations are provided.

- Copper Recovery exceeded Bottle Roll 90-day predictions for the initial Stockpile Project column testing and should achieve extraction levels more than the predicted 83.3% for the soluble copper components. Based on the results to date a copper recovery for 90% of CuAS and 40% of CuCN for a 90-day leaching cycle is recommended for resource evaluation and economic assessment at this time.

Additional considerations include the following.

- TSol recovery sensitivity showing at over 3/4 inch and P80 particle size of approximately
- 1.5-inch may indicate some oversize crushing could be considered.
- Larger run-of-Stockpile Project testing is required to evaluate the need for crushing particles larger than 1-inch.
- Rapid copper recovery less than 60 days and low CuCN content / impact indicates potential for on-off pad to minimize excess acid consumption and capital investment requirements for oxide ore types.
- Scalability has been considered in extending the timeframe to achieve the column testwork by 50% and employing a 95% extraction efficiency factor to both the CuAS and CuCN average column copper extractions achieved to date, allowing for inefficiencies in the leach solution flows and heap operations. As more information is developed these factors will be reevaluated in future reporting.

Acid consumption exceeded bottle roll expectations for test composites WD-22 and WD-50. A gross acid consumption of 20-40 lb acid per ton leached appears to be required for completion of the leaching process which implies a net acid consumption of 18-21 lb/t for the expected Stockpile Project resource soluble copper grades and 15-18 lb/t for higher copper grade open pit resources.

Additional considerations include the following.

- Acid / water initial leach solution is likely more aggressive than SX raffinate (buffering not realized), ongoing testing will employ leach solutions more like SX raffinate.
- Targeted initial leach solution acid concentration 15 gpl H2SO4 was too high, pregnant leach solution (PLS) pH ≤1.4 indicates that excess acid was applied and apparent for much of the testing period. Future testing will adopt a lower initial acid concentration of 10 gpl H2SO4 as a starting point with additional adjustments as results warrant.

- Possible slow reacting gangue consumption (biotite and limonite) could be problematic for longer term leaching based on the preliminary results. Consideration of a longer duration (96-hour) bottle roll testing will be incorporated in future protocols.

13.6.2 Open Pit Metallurgical Testwork

Open pit column testwork is in progress and results presented herein are indicative in nature only until column tail assays are completed for the sulfide / enriched columns.

Copper recovery for oxide materials appears to be consistent with the Stockpile Project materials tested so far, and copper extraction and acid consumption recommendation should be used for oxide open pit resource evaluation.

Based on the indicative results for sulfide materials, a longer leaching time will be required to achieve copper extraction of 70% to 75% for the soluble copper components. Mineralogy also suggests that gangue encapsulation and pyrite inclusion is present, also indicating a longer leaching time requirement.

Scalability has been considered in extending the timeframe to achieve the column the test work by 100% projected average column copper one-year extractions, allowing for inefficiencies in the leach solution flows and heap operations. As more information is developed these factors will be reevaluated in future reporting.

Historically, ASARCO testing in 1968 suggested a gross acid consumption of approximately 20.8 lb/t for the Sacaton West fresh core material. Gross acid consumption for the materials tested in the column work completed to date ranged from 21 lb/t to 31 lb/t.

Bottle roll tests suggest a net acid consumption of approximately 7 lb/t; however, copper extractions were low due to the mineralogical content. Net acid consumption was highly variable and ranged from 28.5 lb/t to 5.6 lb/t for the columns completed and is generally associated with the sample copper grade. The column result for the open pit oxide column was 5.6 lb/t on a net basis, attributing to the higher copper grade in this sample.

Due to the higher copper content and sulfide mineralization oxidation, the sulfide columns are presently net acid producing. This may be an advantageous feature once sulfide material is mined. For resource evaluations an experienced based long-term net acid consumption of 1 lb/t is recommended as a conservative value for use in current economic evaluations until the current column testing is completed.

13.6.3 Flotation Scoping Metallurgical Testwork

Based on the initial testing results, reasonable concentrator options exist for the Cactus primary copper sulfide material.

- Copper flotation recoveries approaching 90% or better appear to be reasonable.
- Significant improvement in the oxide copper recovery components with modern reagents are apparent which can simplify the prior ASARCO plant design.
- A SAG/Ball milling circuit is the most likely grinding option given the relatively soft material at Cactus. Given the apparent power requirements, relatively low energy costs should also be expected.
- The associated rougher concentrate grades provide positive starting points for saleable final concentrate grades once locked cycle testing is completed.
- No optimization work was completed; the results provide only indicative performance expectations. Locked cycle testing is planned as part of this initial program; however, this testing has not been started or completed.

13.6.4 Deleterious Elements

Preliminary testing has been completed on leach solutions, residues and testwork head samples that do not indicate the presence of constituents that would be deleterious to the proposed process methodology or indicate unexpected environmental impacts.

Head samples for the enriched samples leached were provided by McClelland to PMC Laboratory Ltd for multi-element analysis by 4-acid digest with ICP-AES finish (22 element). A polished block section was systematically scanned in high-resolution particle mapping mode using the Tescan Integrated Mineral Analyser (TIMA) equipped on the Tescan Vega Scanning Electron Microscope to determine the modal composition of the sample and collect more detailed information on the Cu-deportment. These analyses do not indicate the presence of known deleterious elements.

Minor amounts of atacamite (chloride copper mineral) have been historically observed, however no presence has been reported in current sampling. Silver is a known minor constituent of the deposit.

TCLP 8 RCRA metals (As, Ba, Cd, Cr, Pb, Se, Ag, Hg) analysis of final leach residues from the initial stockpile column tests was completed by Western Environmental Testing Laboratory (January 2021) and results included in the McClelland final report (February 2021). Results do not show significant or concerning levels of RCRA elements.

The completed open pit oxide column 4600-01 head sample was submitted by McClelland to ALS USA Inc. for 4-acid digest with ICP (48 element) and trace mercury analysis for initial consideration of potential environmental concerns. Fresh material was deemed to be most representative of the material as mined. No material or unusual levels of potential contaminants or processing concerns were identified in this initial work.

Water chemistry for probable site well make up sources have not been analyzed as part of this work. Prior hydrogeologic characterization completed by Tetra Tech Inc. for the Site Improvement Plan – Sacaton Mine Site, for the ASARCO Multi-State Environmental Custodial Trust (11 March 2019) indicates water sources may contain natural chloride levels up to approximately 120 ppm which may have an impact on bioleaching if confirmed and not mitigated.

14.0 MINERAL RESOURCE ESTIMATE

The Cactus Project resource was estimated in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by CIM council on 29 November 2019 (CIM 2019). The resource estimates for the Project are composed of three parts.

- Cactus Deposits – in situ Cactus West and Cactus East deposits located adjacent to the historical Sacaton pit. This is the first Mineral Resource estimate undertaken under CIM standards for the Cactus Project in situ deposits. The Mineral Resource estimate includes all drilling, geological logging, and historical mapping completed prior to 13 February 2021 and mining depletion of the historical pit mined by ASARCO between 1972 and 1984.
- Cactus Stockpile Project – an historic mineralized stockpile generated as a result of waste dumping from the historic Sacaton pit. Material historically considered as waste included all oxide material, sulfide material considered below the mining COG of 0.3% CuT, and sulfide material above the mining COG but where the oxide component was considered too high. This is an update to the previously reported Mineral Resource estimate undertaken under CIM standards for the Cactus Stockpile Project (dated 01 March 2020) and includes infill drilling to 400 ft spacing. The Mineral Resource estimate includes all drilling, geological logging, historic pit dump information, and topographical updates from rehabilitation work to 04 April 2021.
- Parks/Salyer Deposit - the in situ Parks/Salyer deposit is located to the SW of the historical Sacaton pit and contains mineralization of a similar nature to Cactus. This is the first Mineral Resource estimate undertaken under CIM standards for the Parks/Salyer in situ deposit. The Mineral Resource estimate includes all drilling and geological logging completed prior to 26 September 2022.

All data coordinates are presented in NAD83 ft., Zone 12 truncated to the last six whole digits for easting, and five whole digits for northing. All quantities are given in imperial units unless indicated otherwise. All copper values are presented in percent.

The copper mineralization at the Project was estimated using Vulcan modelling software (v2020.2). Modelling of the geological domains to support the estimate were undertaken by Arizona Sonoran personnel. Grade estimates were undertaken by Allan Schappert, Certified Professional Geologist (CPG #11758) of Stantec.

14.1 Cactus Project Deposits

The inverse distance (ID3) method was used for the estimation of copper grades to the models. Copper estimates were performed on CuT assays and TSol results. TSol results were performed through sequential analysis of the pulp sample with acid soluble analysis followed by cyanide soluble analysis. Results were then added to one another for TSol copper. Validations made use of the nearest neighbor (polygonal) method for statistical review and Discrete Gaussian change of support for grade tonnage smoothing checks.

14.1.1 Resource Drill Hole Database

The Cactus Project drill hole databases are managed in MX-Deposit software. CSV format files were exported from MX-Deposit using a resource specific template for the tables required for the resource database. CSV files were imported into a Vulcan ISIS database using a designated resource import LAVA script. The LAVA script and export template ensured the database was loaded consistently each time. The drill hole database used for the Cactus Project resource estimation was called "cacdrilling_mx_resource_20210213.ddh.isis". The drillhole database used for the Parks/Salyer resource estimation was called "cacdrilling_mx_resource_20220907.ddh.isis".

Lithology logging was used to build broad lithological zones that control where potential mineralization could occur and the assignment of specific gravity to the model. Mineralization logging, in addition to sequential copper assaying and historical mapping, was used to determine the main copper mineral zones that were fundamental to the estimation domains.

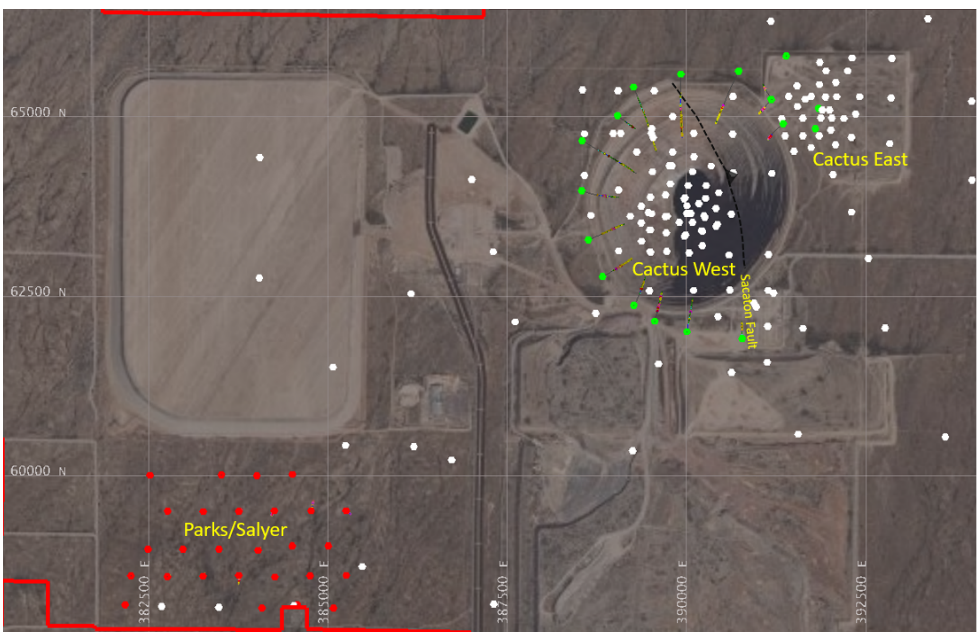
The Cactus and Parks/Salyer drill hole databases can be summarized by the following points.

- The Cactus resource drill hole database contains 201 total holes. This is inclusive of 22 recent drill holes drilled by Arizona Sonoran since 2019. Twenty of these holes were drilled into the Cactus Deposits to support the resource estimate.
- The Parks/Salyer drill hole database contains 31 holes supporting the resource estimate, composed of 27 modern holes drilled by ASCU since 2021 and four historical holes drilled by ASARCO.
- Historic drill holes were drilled vertically with rotary pre-collars through the barren cover and diamond tails through the mineralized zones.
- Most historic drill holes were not downhole surveyed aside from a number of historic holes drilled into the central area of the mineralized zone of the Cactus East deposit and two of the historic Parks/Salyer drillholes.
- Recent drill holes surrounding the pit rim, were drilled using angled diamond drill holes.
- Recent drill holes drilled into the northern expansion of the Cactus East deposit and the Parks/Salyer deposit were drilled vertically.
- All recent holes have been downhole surveyed.

- Samples were assayed on 10 ft (3 m) lengths, except where strong lithological or structural contacts determined a variation in sample length was required.
- All drill holes were logged for lithology, mineralization, alteration, brecciation, and oxidation.
- A significant relogging and re-assaying program was undertaken as part of the recent drilling program to reinstate and/or confirm historical information.

Figure 14-1 plots the drill hole locations within the Cactus Project area including the location of the historical Sacaton pit, which forms part of the Cactus West Deposit, and the NE alluvium dump. The NE alluvium dump outlines the location of the Cactus East deposit. Offsetting the location of the two deposits is the Sacaton Fault which is visible in the eastern wall of the historical pit. The Parks/Salyer deposit is located to the SW of the image adjoining the southern boundary of ASCU's land holdings.

Figure 14-1: Drill Hole Collars and Traces within the Cactus Project – Arizona Sonoran's Recent Cactus Core Drill Holes (Green Circles), Parks/Salyer Core Drill Holes (Red Circles) and ASARCO's Historical Drill Holes (White Circles)



Total Soluble Copper Assays

TSol copper assay information was gained through sequential copper analysis consisting of acid soluble and sequential cyanide soluble assay analysis. From these assays, TSol copper was calculated as the addition of the two sequential assay values. All recent drilling was analyzed for sequential copper analysis. In addition, a large re-assay program was undertaken to verify historic data and provide sequential copper analyses on historic drill holes. As a priority, drill holes influencing the estimation of material adjacent to the historic pit were re-assayed. This program provided good coverage of TSol copper assays throughout the deposit; however, there were a number of drill holes that were not re-assayed.

To maintain the assay relationships of total copper and TSol copper in the oxide and enriched estimated blocks, drill holes containing both assays were analyzed, and a method determined to calculate TSol copper to the samples where it was not currently present. Where a sample was assigned a calculated grade, a flag was set in the Vulcan ISIS database so that a record was maintained of actual assayed values versus calculated values. After analysis, it was determined that at high grades for oxide and enriched (above 0.3% CuT) the TSol copper was high and there was a strong correlation. Below 0.3% CuT, there remained a strong correlation; however, there was more variability than in high grades samples. For missing TSol copper samples with grades of 0.3% CuT or greater, the TSol copper grade was assigned at 95% of the total copper grade for that sample. For missing TSol copper samples with total copper grades below 0.3% CuT, the TSol copper grade was assigned at 90% of the total copper grade for that sample. Calculations were undertaken on the raw drill hole database intercepts prior to compositing.

In future, it is planned to re-assay all samples in oxide, enriched, and the upper portions of primary material to establish full TSol copper grades across the deposit.

Gold, Silver, and Molybdenum

Gold and silver credits in the copper concentrate were awarded to ASARCO when mining the Sacaton pit. Limited data is available relating to gold and silver grades from historic drill hole composites and mill reconciliation reports. Gold and silver are present throughout the deposit but at very low grades. Future work is planned, specifically in the primary material, to improve the knowledge and understanding through re-assay of historic and recent pulps.

This is expected to only provide small incremental value to the Project due to the low grades reported to date.

Similarly, molybdenum is present through the deposit but has only been reported on in limited drill hole composites and some recent drill holes. Future work is planned to re-assay primary material as a potential value addition to the Project.

14.1.2 Geological Modelling

Faults

A number of fault structures define the main fault blocks that control the location and general geometry of mineralization. The Cactus deposits were offset to the NE for up to four miles along a regional listric fault known as the Basement fault. To accommodate extensional movement and block rotation along the Basement fault, NW striking normal faults developed. These created a regular series of horst and graben blocks which were infilled with gravels and conglomerate. The discovery outcrop represents the only outcrop of the Santa Cruz porphyry system at surface. Exploration drilling, and mining of the Sacaton pit, has defined the broad geometries of the mineralized blocks within the Cactus deposit area.

The main fault blocks modelled were defined by the modelling of the individual fault surfaces that form the contacts. The Basement fault was modelled from drill holes that pierce the structure, below this fault there has been no mineralization identified to date. It is sub-horizontal with local undulations and evidence of local offset, likely by later reactivation, along the Sacaton fault. In the Parks/Salyer area the basement fault dips at a long angle to the north-west.

The Sacaton and East faults define the eastern edges of the Cactus West and Cactus East blocks. These represent normal faults that strike approximately 160° and dip between 50°-70° to the east. Blocks were down dropped to the east along these faults. A conjugate set of normal faults, accommodating basement extension, and represented by the fault contact between cover conglomerate and bedrock is known as the west fault. The orientation of this fault varies considerably. In Cactus west, it strikes approximately 340° and dips 25° to the west. In Cactus East, this fault is known as the south fault and the strike and dip is more variable but could generally be defined as striking approximately 085° and dipping 40° to the South. Parks/Salyer is similarly defined by extensional faults creating a horst block. The overall angles of the NE trending normal faults at Parks/Salyer dip at a lower angle Figure 14-2 displays the bedrock zones in red and the related fault contacts defining the bedrock fault block geometries. Individual fault planes were modelled by defining intercept points in drilling and historic interpreted cross section and plan maps. Points were then modelled as surfaces and clipped to one another to define fault block solids. The outer extents of the fault blocks were defined by a generalized alteration halo defined by ASARCO and based on regional exploration drilling (Figure 14-3). As new angled drilling is added, this outer boundary and its controls continue to be refined and present the potential to add more mineralization to the resource within the resource pit limits.

Figure 14-2: NE Oriented Long Section displaying Fault Block Geometries, Facing NW

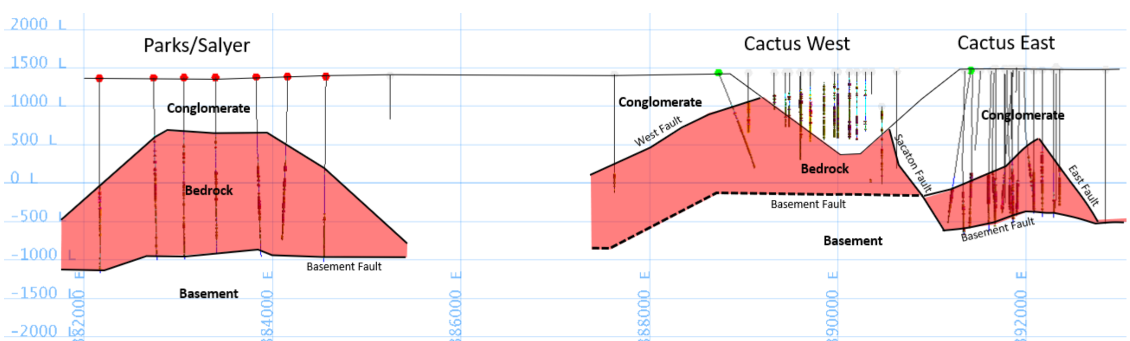
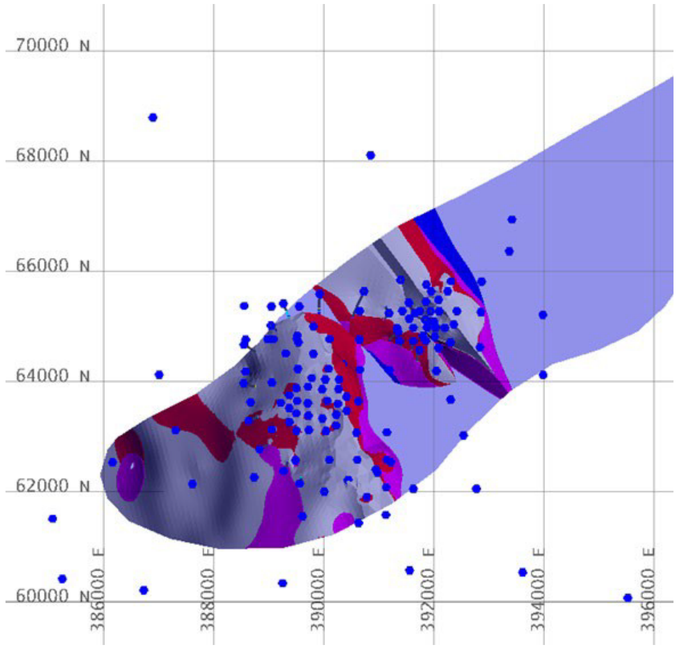


Figure 14-3: Plan View of the Outer Alteration Zone (in Blue) Restriction of Fault Blocks



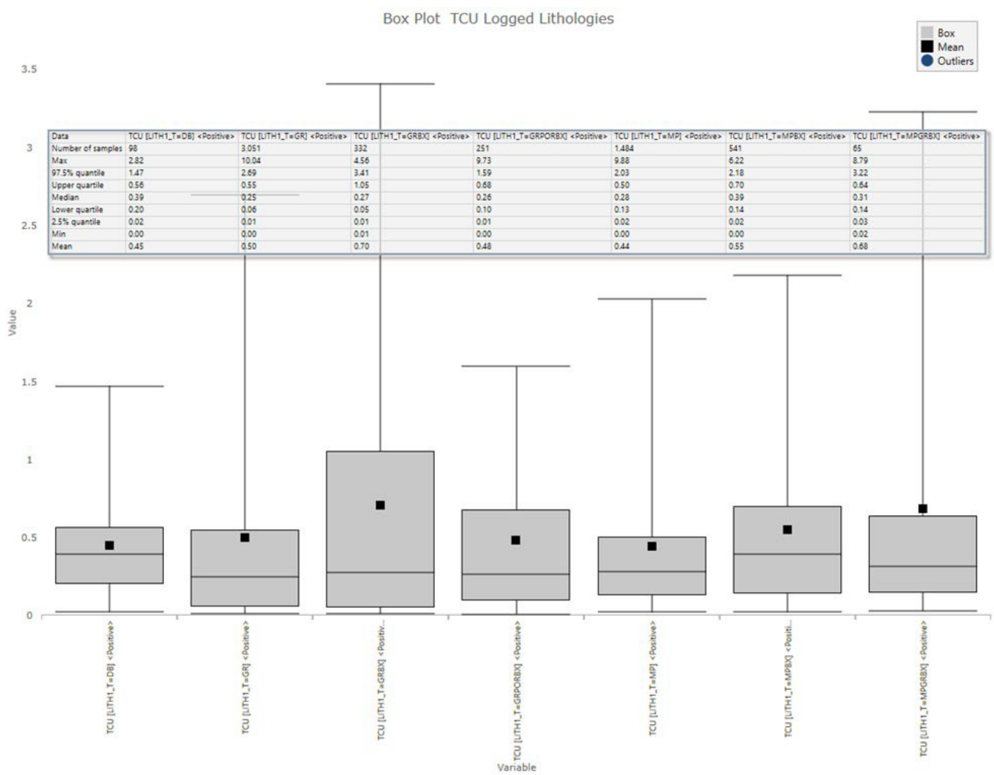
Lithology

Lithology was grouped into multiple domains within the Cactus Project that relate to the presence or absence of mineralization. The main lithological domains modelled are defined in Table 14-1 along with the expected presence or absence of mineralization in that domain. Figure 14-4 displays box plots comparing the CuT distributions of the main logged lithologies within the bedrock. Results show no clear control on grade distributions based on host lithology alone. Figure 14-5 displays a NE-oriented cross section outlined in Figure 14-3 through the Cactus Project, facing north, overlayed with the lithological domains outlined spatially along with the main fault controls. Lithological domains were modelled by combining individually logged lithologies into formations representing the four main lithological domains. Points were then extracted from the drill holes representing the footwall contacts of the alluvium and the conglomerate, in addition to interpretive points being added based on historic cross section and plan maps. Surfaces were modelled from these point sets and the surfaces clipped against the fault block solids to create solid triangulations of the alluvium, conglomerate, and bedrock.

Table 14-1: Lithological Domains Properties

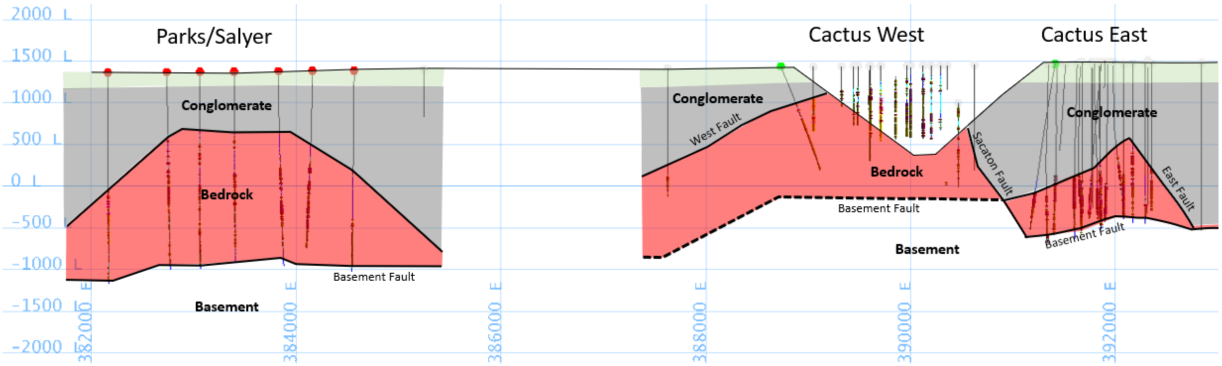
Lithological Domain	Relationship to Mineralization
Alluvium – Quaternary in age.	Non-mineralized
Conglomerate – Tertiary in age.	Non-mineralized
Bedrock units including granite, diabase, and monzonite and quartz monzonite porphyries with varying degrees of brecciation. Oracle granite is of Precambrian age, porphyry intrusions are Laramide in age.	Mineralized
Basement metamorphosed units including the Pinal Schist and metamorphosed granitic, gneissic, and metavolcanic rocks below the Basement fault.	Non-mineralized

Figure 14-4: Box Plots of the Main Logged Lithologies Hosting Mineralization



Data	TCU [LITH1_T=DB] <Positive>	TCU [LITH1_T=GR] <Positive>	TCU [LITH1_T=GRBX] <Positive>	TCU [LITH1_T=GRPORBX] <Positive>	TCU [LITH1_T=MP] <Positive>	TCU [LITH1_T=MPBX] <Positive>	TCU [LITH1_T=MPGRBX] <Positive>
Number of samples	98	3,051	332	251	1,484	541	65
Max	2.82	10.04	4.56	9.73	9.88	6.22	8.79
97.5% quantile	1.47	2.69	3.41	1.59	2.03	2.18	3.22
Upper quartile	0.56	0.55	1.05	0.68	0.50	0.70	0.64
Median	0.39	0.25	0.27	0.26	0.28	0.39	0.31
Lower quartile	0.20	0.06	0.05	0.10	0.13	0.14	0.14
2.5% quantile	0.02	0.01	0.01	0.01	0.02	0.02	0.03
Min	0.00	0.00	0.01	0.00	0.00	0.00	0.02
Mean	0.45	0.50	0.70	0.48	0.44	0.55	0.68

Figure 14-5: NE Oriented Long Section displaying Lithology Zones, Facing NW



Copper Mineral Zones

Of most importance to the estimation of copper grades at Cactus, was the distribution and zonation of the copper mineral zones. Cactus and Parks/Salyer exhibit typical porphyry copper mineral zonation due to the leaching of copper in sulfides at shallow depths with redeposition below the water table to enriched chalcocite and/or covellite copper sulfides. Above the water table, copper oxide minerals formed. Drilling shows the highest grades were typically encountered at the interface of the enriched and oxide zones as a remnant feature of the historic water table level. Contacts between copper mineral zones within the Cactus deposits were generally sharp, with short transitions. In areas where there were transitional zones, this material was typically included as primary mineralization to add conservatism to the treatment of this material for resource reporting and subsequent mine planning. Contact boundaries were identified by the analysis of sequential copper assays and geological logging. Copper mineral zones were modelled within the bedrock lithological domain only. Table 14-2 indicates the main copper mineral domains modelled and their relationship to mineralization. Figure 14-6 displays box plots for the three copper mineral zones highlighting the different CuT distributions between the zones, the limited transitional material evidenced by high solubilities in the oxide and enriched zones, and very low solubility in the primary zone. Figure 14-6 displays the NE cross section outlined in Figure 14-5 with the copper mineral zones of the bedrock overlayed to show the spatial relationships of the zones. Within the bedrock, points were extracted from the drill holes representing the hanging wall contacts of the oxide, enriched, and primary contacts. In addition, interpretive points were added based on historical cross section and plan maps. Surfaces were modelled from these point sets and the surfaces clipped against the bedrock solids to create solid triangulations of the leached, oxide, enriched, and primary copper mineral zones.

Table 14-2: Lithological Domains

Copper Mineral Domain	Relationship to Mineralization
Leached – incorporating the gossanous and leached weathering zones. Cactus West contains multiple phases of leaching.	Poorly mineralized. Copper mineralization typically confined to selvages of oxide enriched, or primary entrapped during subsequent leaching phases.
Oxide	Mineralized with oxide and carbonate copper minerals. Represents potential heap leach mineralization.
Supergene Enriched	Mineralized with secondary chalcocite and covellite. Represents potential heap leach or mill flotation mineralization.
Primary (hypogene)	Mineralized with primary chalcopyrite and pyrite. Represents potential mill flotation mineralization.

Figure 14-6: Box Plots of Copper Grades in Mineralized Zones

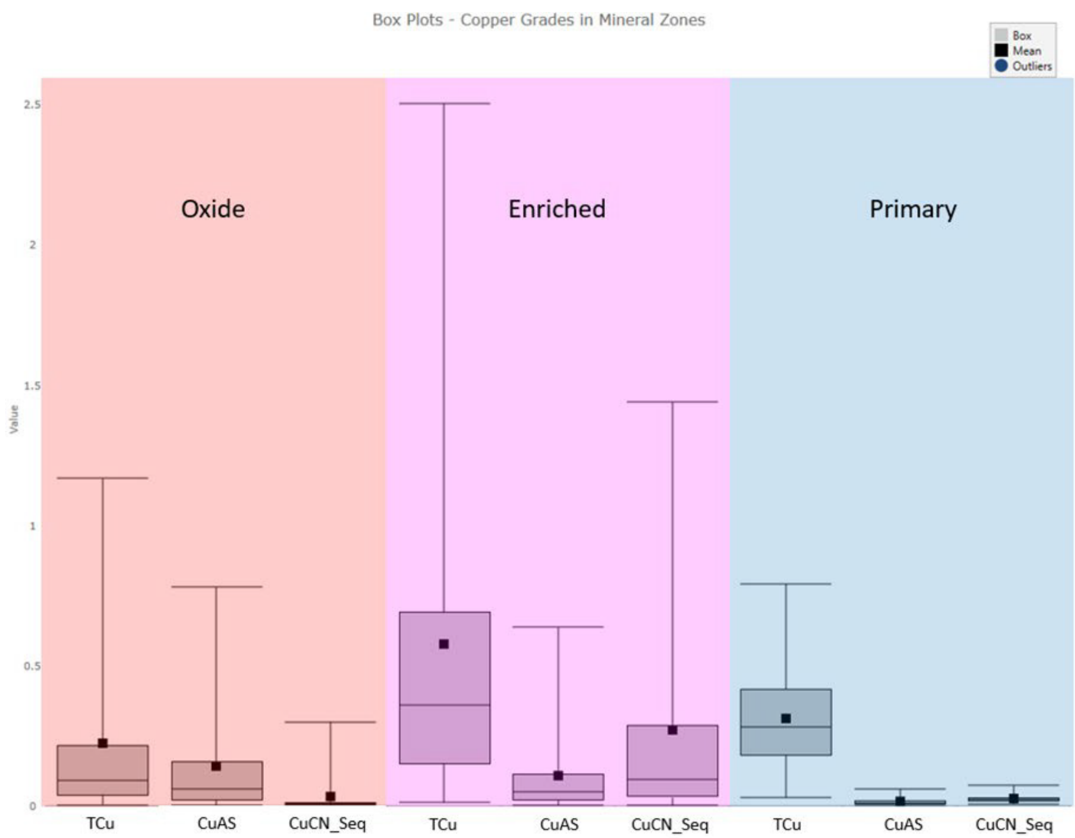
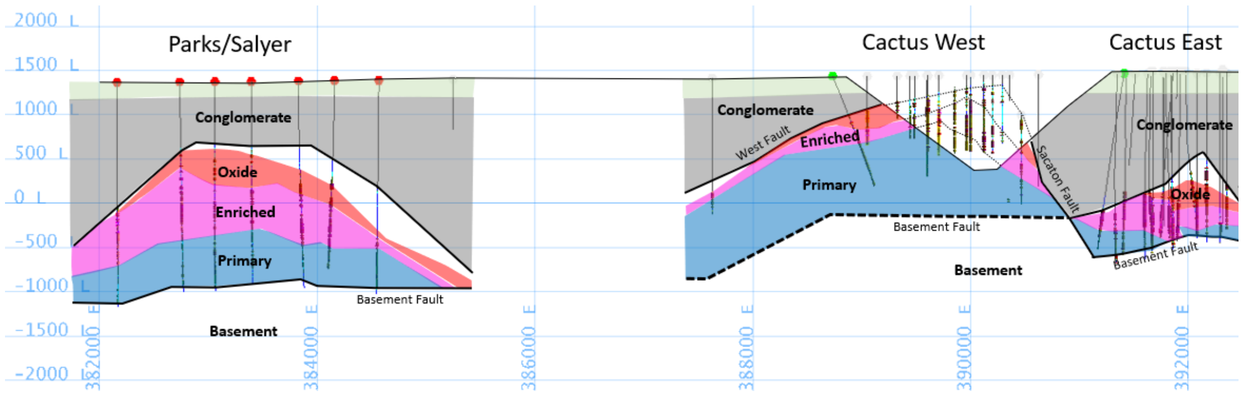


Figure 14-6 displays box plots for CuT and the sequential copper assay components (CuAS and CuCN Seq) to show both the distinct CuT grade distributions defined by the copper mineral zones, the limited transitional material as defined by the high solubilities in the oxide and enriched, and low solubility results in the primary. F shows NE oriented cross section facing NW.

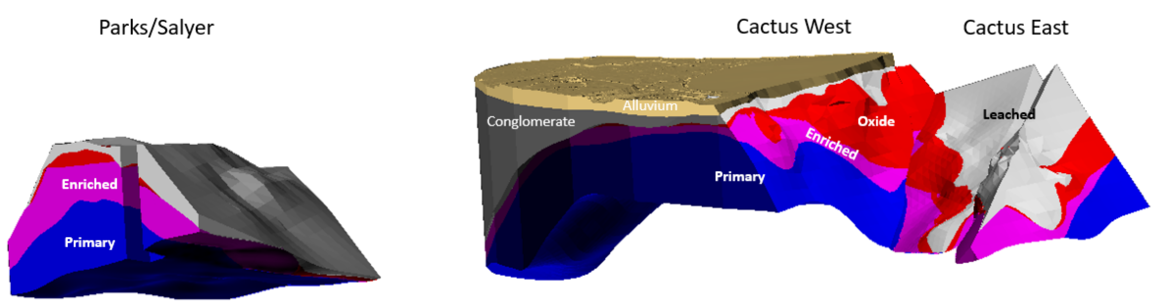
Figure 14-7: Northeast Oriented Cross Section Displaying Copper Mineral Zones, Facing Northwest



14.1.3 Estimation Domains

Final estimation domains were composed of the leached, oxide, enriched, and primary copper mineral zones. Figure 14-8 shows an isometric view of the final copper mineral zones in three dimensions. The alluvium and conglomerate cover have been removed above Cactus East and Parks/Salyer to aid visualization.

Figure 14-8: Isometric View of the Copper Mineral Estimation Domains



14.1.4 Specific Gravity

As of February 2021, historical drill hole logs for the Cactus drilling contained extensive record of specific gravity measurements (1,693 readings). Measurements were undertaken using the wet / dry weight methodology. Values were recorded in metric g/cm3 in the historic logs. To support imperial units and reporting of short tons, the original readings were converted to ft3/t by multiplying the specific gravity value by 0.031213980288072. Variations in specific gravity were recognized between the alluvium, conglomerate, bedrock, and basement zones. Most lithological units within the bedrock contain similar mineralogies. Due to this, the larger differences in specific gravity were deemed a result of the level of weathering of the rock.

The copper mineral zones defined basic zones to encompass different levels of weathering. As such, they were the basis of defining specific gravity average values within the bedrock. Average specific gravity values were calculated and applied based on the copper mineral and lithological domain. Due to the mineralization being disseminated, sulfide content is not highly correlated to specific gravity. Table 14-3 displays the specific gravity values assigned for each domain.

Table 14-3: Specific Gravity Values per Lithological Domain

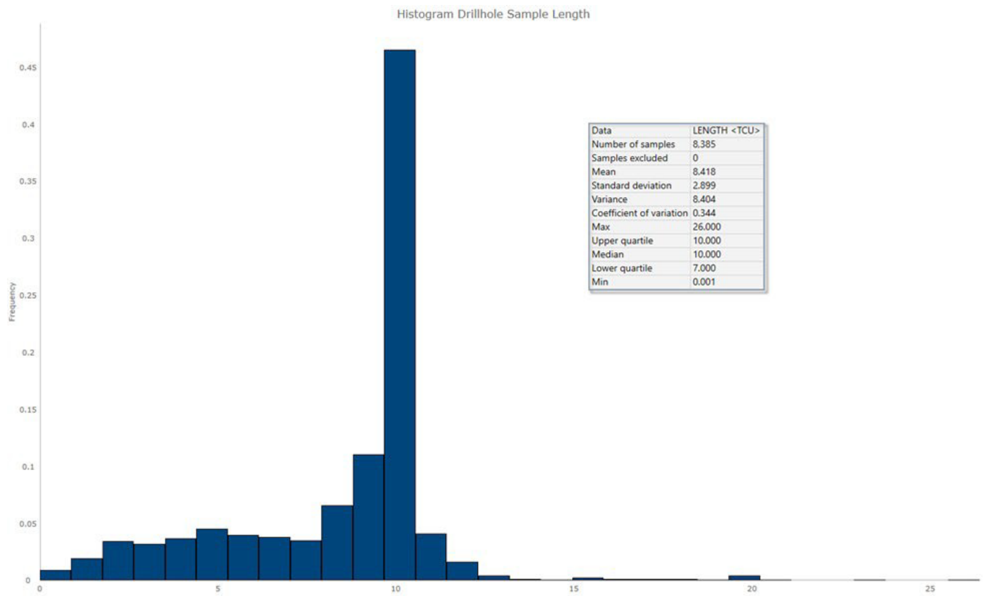
Variable Name	Count	Mean	Standard Deviation	Variance	CV	Max.	Upper Quartile	Median	Lower Quartile	Min
SG_FIN [BOUND=Conglomerate]	12	0.078	0.002	0.000	0.028	0.081	0.080	0.079	0.076	0.074
SG_FIN [BOUND=Enriched]	609	0.081	0.002	0.000	0.029	0.089	0.082	0.080	0.079	0.066
SG_FIN [BOUND=Leached]	84	0.078	0.003	0.000	0.040	0.097	0.078	0.077	0.077	0.072
SG_FIN [BOUND=Oxide]	565	0.079	0.004	0.000	0.051	0.092	0.081	0.079	0.077	0.016
SG_FIN [BOUND=Primary]	423	0.079	0.006	0.000	0.074	0.091	0.081	0.080	0.078	0.017

As of September 2022, further historical drilling density data was available for analysis. Due to this Parks/Salyer densities were updated to reflect the densities measured in Cactus West of 0.08ft3/st for leached, oxide, and primary, and 0.081ft3/st for enriched.

14.1.5 Compositing

Sampling in the drill hole database was historically undertaken on nominal 10 ft samples, except where strong structural or lithological contacts supported a change in this regime. For this reason, the drill hole database was composited to 10 ft lengths with composite lengths cut at the copper mineral contacts, as defined by the triangulation solids. Samples of less than 3 ft at the mineral zone contact were added to the previous composite to avoid having very short composites in the database. This was done to support later grade estimation processes using this database. Figure 14-9 displays the histogram for the drill hole sample lengths within the Cactus resource drill hole database. Over 45% of the samples have an interval length of 10 ft; most of the remaining samples have a shorter interval length. Parks/Salyer was dominantly sampled at 10 ft sample lengths.

Figure 14-9: Histogram of Drill Hole Sample Lengths



14.1.6 Exploratory Data Analysis

Cactus West

In Figure 14-10, CuT and TSol copper were plotted as box plots for the leached, oxide, enriched, and primary domains. Oxide and enriched domains show strong relationships of HLs of TSol copper which is expected of these domains. The primary domain shows a low level of soluble copper as expected. The grade distributions are as expected with the highest-grade domain being the enriched. The oxide domain reports lower grade; however, this domain does contain deeper leaching locally, which leads to the increased skewness of the population. The box plots show very good domain control in separating copper population distributions and material types. Table 14-4 reports the statistics for the main domains in support of the box plot distributions in Figure 14-10.

Figure 14-10: Box Plots of Total Copper and Total Soluble Copper Grades for Cactus West

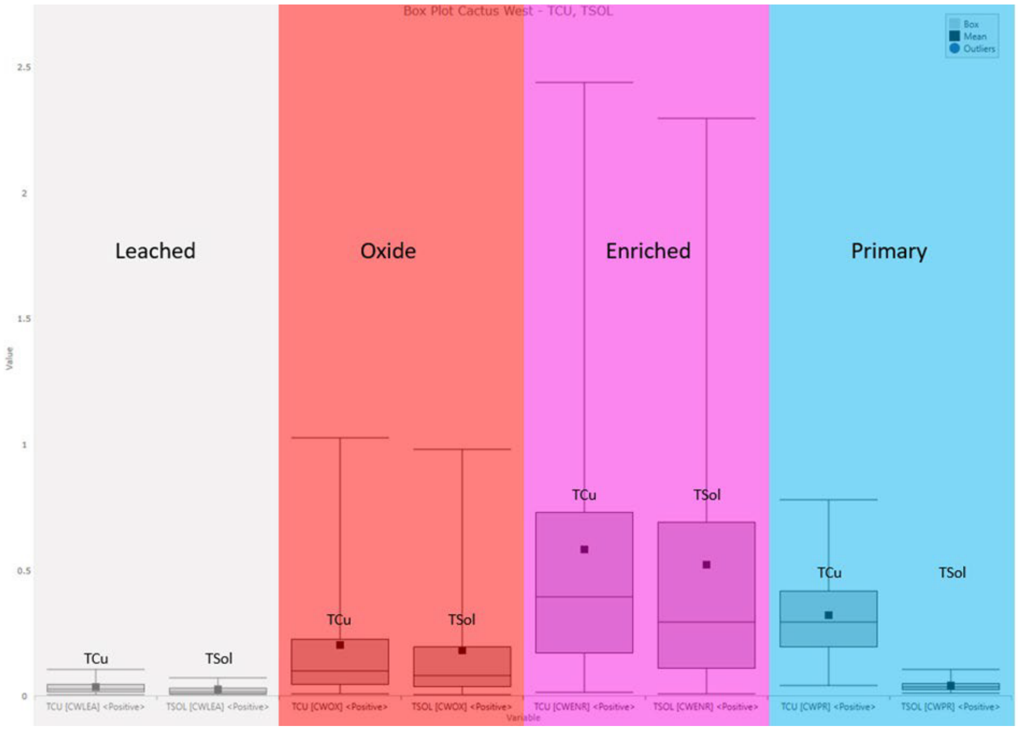


Table 14-4: Cactus West Descriptive Statistics of Total Copper and Total Soluble Copper Grades

Variable Name	Count	Mean	Standard Deviation	Variance	CV	Max.	Upper Quartile	Median	Lower Quartile	Min
CuT [CWLEA] <Positive>	142	0.036	0.030	0.001	0.841	0.182	0.049	0.027	0.014	0.003
TSol [CWLEA] <Positive>	142	0.026	0.023	0.001	0.868	0.133	0.036	0.018	0.009	0.003
CuT [CWOX] <Positive>	1609	0.203	0.312	0.098	1.536	3.521	0.229	0.100	0.045	0.003
TSol [CWOX] <Positive>	1609	0.183	0.301	0.090	1.646	3.519	0.198	0.081	0.036	0.003
CuT [CWENR] <Positive>	1089	0.584	0.643	0.413	1.101	4.890	0.732	0.394	0.170	0.006
TSol [CWENR] <Positive>	1089	0.522	0.630	0.398	1.208	4.888	0.694	0.297	0.111	0.006
CuT [CWPR] <Positive>	1429	0.321	0.186	0.034	0.578	1.526	0.420	0.295	0.195	0.009
TSol [CWPR] <Positive>	820	0.042	0.028	0.001	0.667	0.409	0.061	0.036	0.025	0.008

To confirm the relationship between TSol copper and CuT, scatterplots were plotted for the oxide, enriched, and primary domains (Figure 14-11 through Figure 14-13). For the soluble domains, namely oxide and enriched, the bulk of the TSol copper is expected to plot towards the 45° line, indicating a 1:1 relationship. Samples plotting well away from this line would indicate significant mixing of populations and the potential for significant transitional zones within the mineralization. The bulk of the population does plot close to the 45° line consistent with a low level of transitional material. For the primary domain, the TSol copper plots well away from the 1:1 line, indicating the presence of non-soluble copper as chalcopyrite.

Figure 14-11: Scatter Plots of Total Soluble Copper versus Total Copper within the Oxide Domain for Cactus West

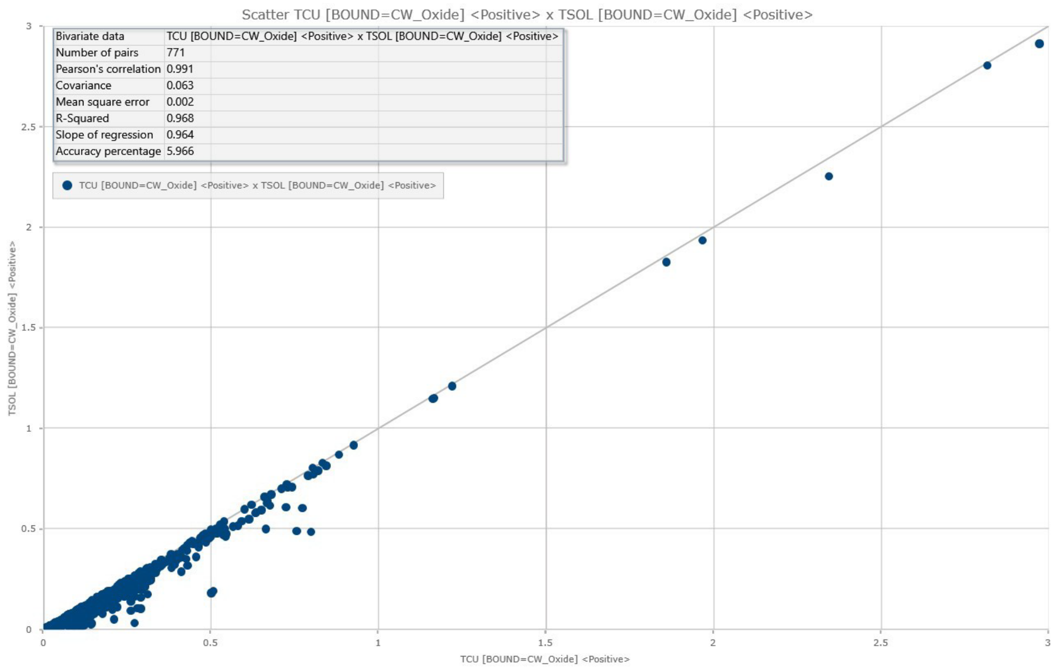


Figure 14-12: Scatter Plots of Total Soluble Copper versus Total Copper within the Enriched Domain for Cactus West

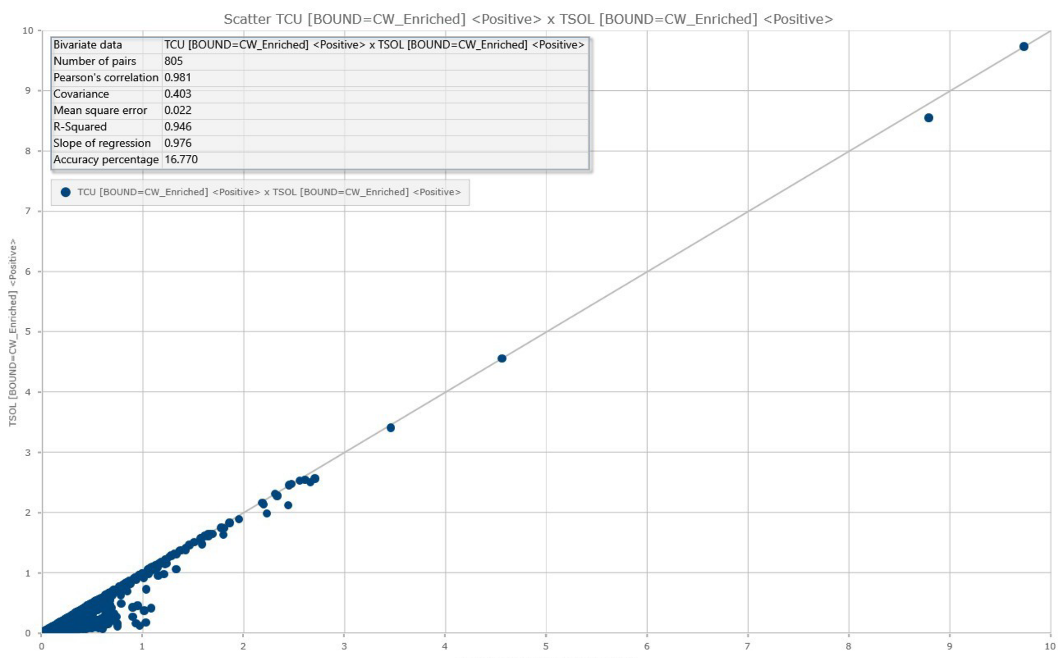
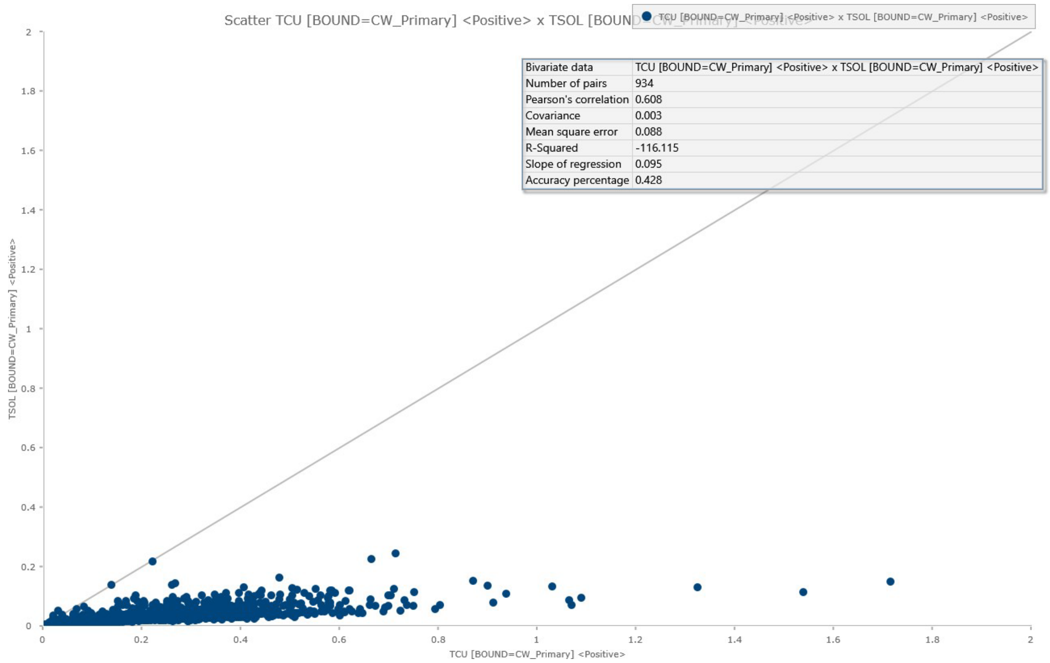


Figure 14-13: Scatter Plots of Total Soluble Copper versus Total Copper within the Primary Domain for Cactus West



For oxide and enriched domains, the bulk of the copper is soluble and plots towards the 45° line indicating a 1:1 relationship with CuT. For the primary domain, as expected, the bulk of the copper is chalcopyrite; therefore, the TSol is low and plots well away from the 45° line.

Due to the different copper mineral species within the copper mineral domains (supported by the different grade distributions) and different mechanisms for precipitation, contacts between the copper mineral domains in Cactus West were treated as hard contacts and therefore contact analysis was not undertaken.

The defined estimation domains show a high degree of control over the copper distributions seen within the Cactus West deposit and are appropriate for use in grade estimation to produce robust estimates of copper grades.

Cactus East

In Figure 14-14, CuT and TSol copper are plotted as box plots for leached, oxide, enriched, and primary domains. Oxide and Enriched domains show strong relationships of HLs of TSol copper which is expected of these domains. The primary domain shows a low level of soluble copper as expected. The grade distributions are as expected with the highest-grade domains being the enriched and oxide. The box plots show very good domain control in separating copper population distributions and material types. Table 14-5 reports the statistics for the main domains in support of the box plot distributions in Figure 14-14.

Figure 14-14: Box Plots of Total Copper and Total Soluble Copper Grades within Copper Mineral Domains for Cactus East

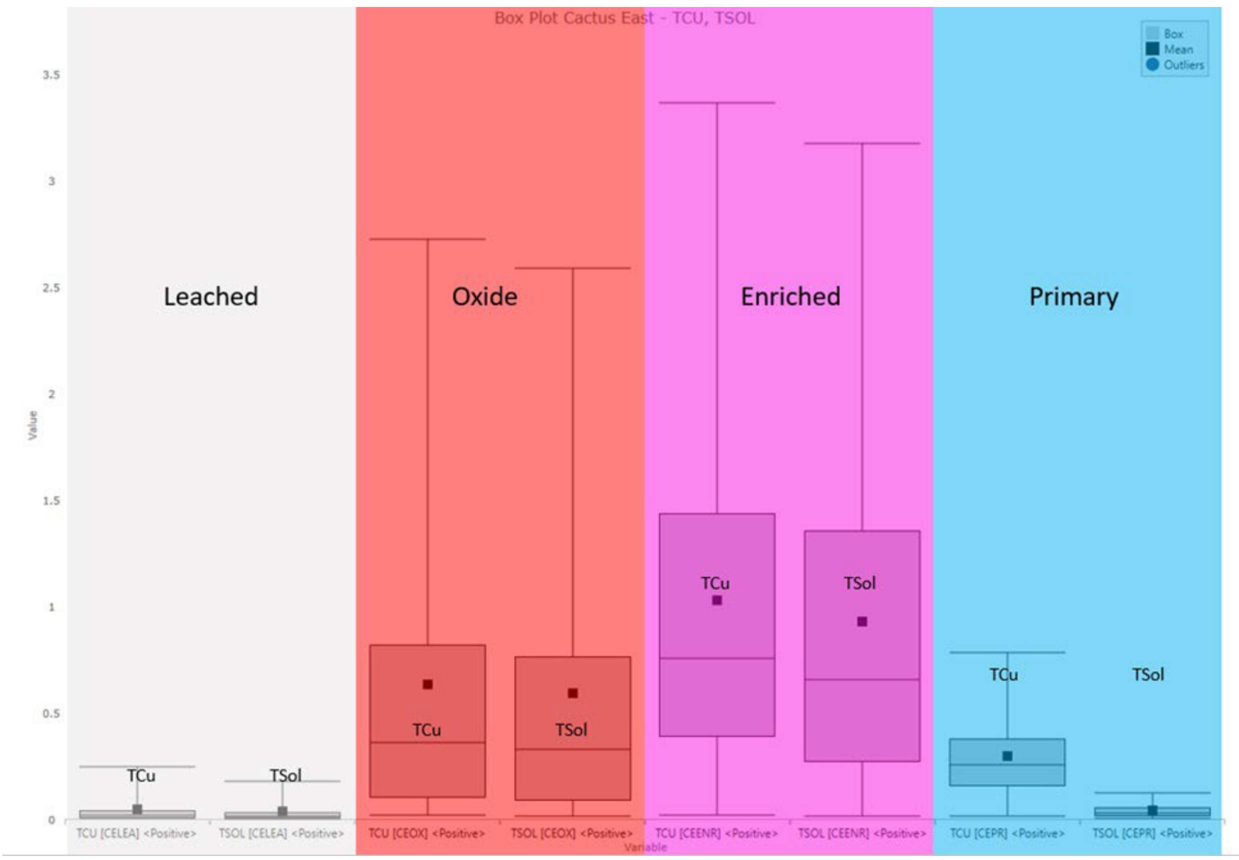


Table 14-5: Cactus East Descriptive Statistics of Total Copper and Total Soluble Copper Grades

Variable Name	Count	Mean	Standard Deviation	Variance	CV	Max.	Upper Quartile	Median	Lower Quartile	Min
CuT [CELEA] <Positive>	148	0.052	0.109	0.012	2.115	0.920	0.047	0.024	0.013	0.007
TSol [CELEA] <Positive>	148	0.041	0.105	0.011	2.566	0.920	0.035	0.018	0.009	0.006
CuT [CEOX] <Positive>	564	0.636	0.762	0.582	1.198	4.823	0.822	0.365	0.105	0.011
TSol [CEOX] <Positive>	564	0.596	0.727	0.529	1.221	4.582	0.777	0.332	0.091	0.006
CuT [CEENR] <Positive>	938	1.033	0.898	0.807	0.870	5.000	1.440	0.762	0.391	0.010
TSol [CEENR] <Positive>	938	0.932	0.880	0.774	0.944	5.000	1.358	0.657	0.273	0.008
CuT [CEPR] <Positive>	499	0.298	0.196	0.039	0.658	1.582	0.381	0.260	0.159	0.003
TSol [CEPR] <Positive>	352	0.044	0.038	0.001	0.877	0.390	0.061	0.032	0.018	0.003

To confirm the relationship between TSol copper and CuT, scatter plots were plotted for the oxide, enriched, and primary domains (Figure 14-15 through Figure 14-17). For the soluble domains, namely oxide and enriched, the bulk of the TSol copper is expected to plot towards the 45° line, indicating a 1:1 relationship. Samples plotting well away from this line would indicate significant mixing of populations and the potential for significant transitional zones within the mineralization. The bulk of the population does plot close to the 45° line consistent with a low level of transitional material. For the primary domain, the TSol copper plots well away from the 1:1 line, indicating the presence of non-soluble copper as chalcopyrite.

Figure 14-15: Scatter Plots of Total Soluble Copper versus Total Copper within the Oxide Domain for Cactus East

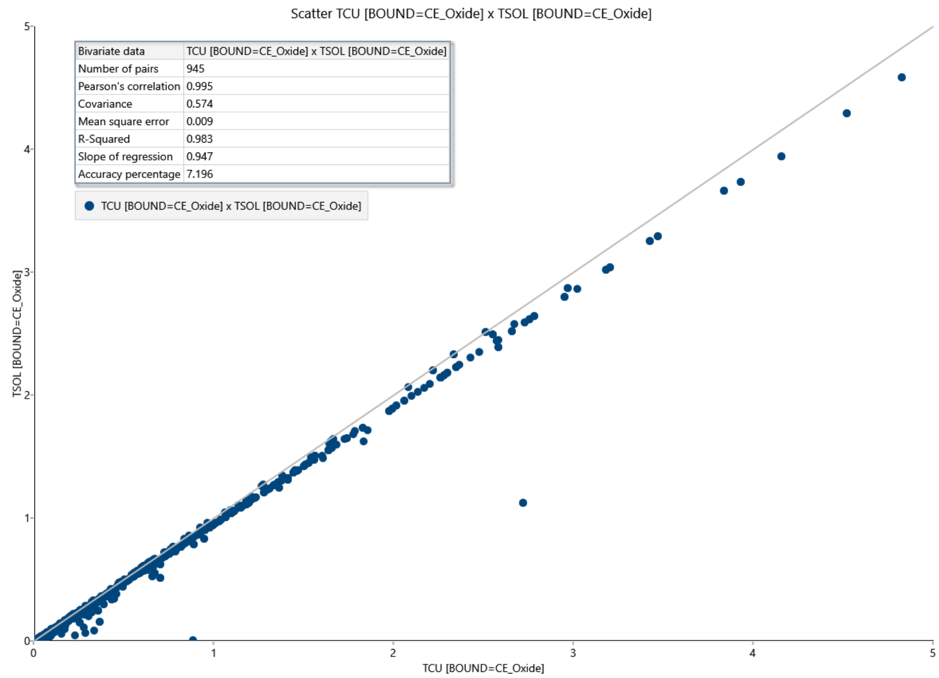


Figure 14-16: Scatter Plots of Total Soluble Copper versus Total Copper within the Enriched Domain for Cactus East

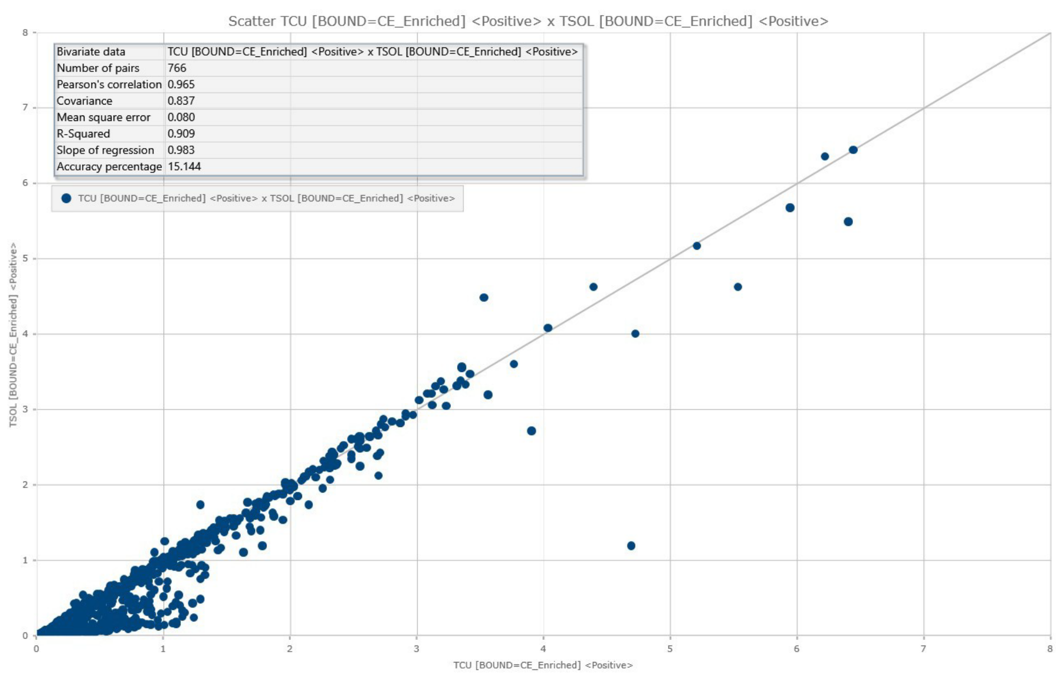
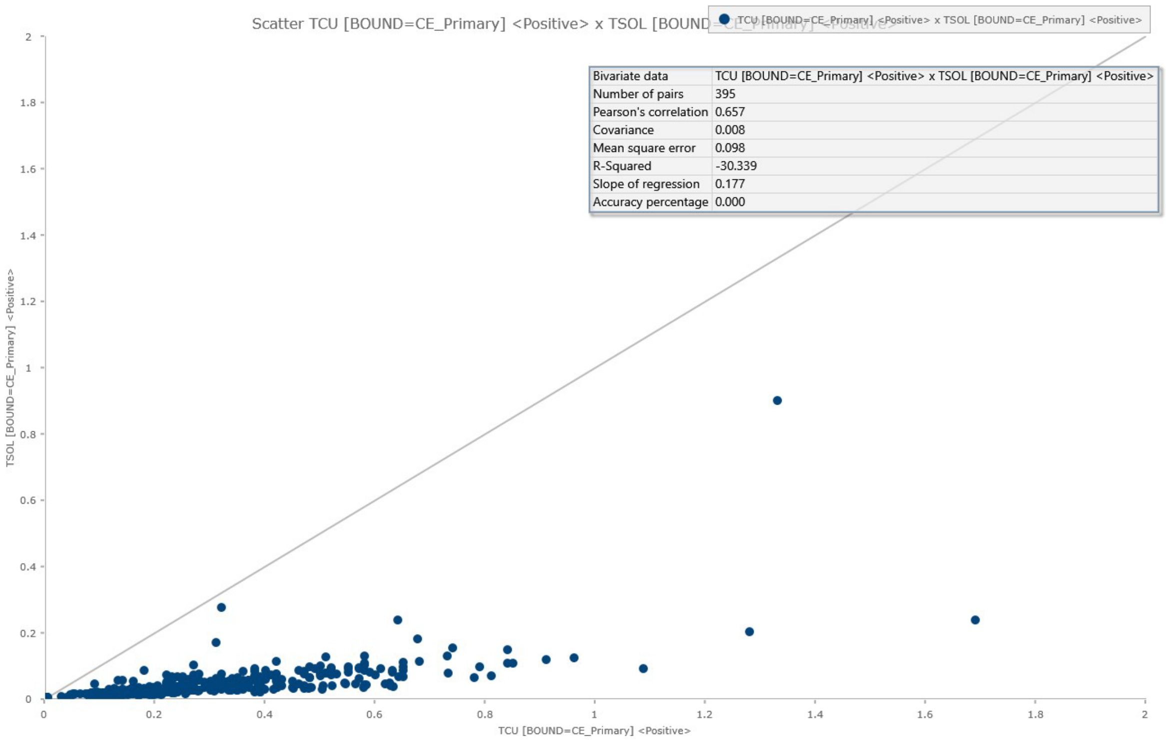


Figure 14-17: Scatter Plots of Total Soluble Copper versus Total Copper within the Primary Domain for Cactus East



For oxide and enriched domains, the bulk of the copper is soluble and plots towards the 45° line indicating a 1:1 relationship with CuT. For the primary domain, as expected, the bulk of the copper is chalcopyrite and therefore the TSol copper is low and plots well away from the 45° line.

Due to the different copper mineral species within the copper mineral domains (supported by the different grade distributions) and different mechanisms for precipitation, contacts between the copper mineral domains in Cactus East were treated as hard contacts and therefore contact analysis was not undertaken.

The defined estimation domains show a high degree of control over the copper distributions seen within the Cactus East deposit and are appropriate for use in grade estimation to produce robust estimates of copper grades.

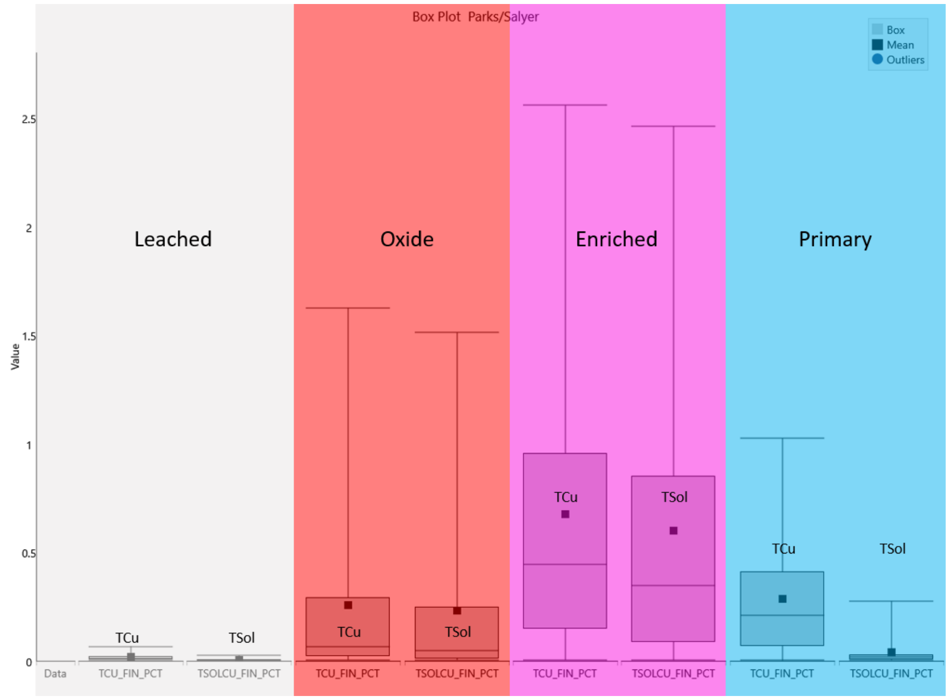
Parks/Salyer

CuT and TSol copper are plotted as box plots for leached, oxide, enriched, and primary domains. Oxide and Enriched domains show strong relationships of HLs of TSol copper which is expected of these domains. The primary domain shows a low level of soluble copper as expected. The grade distributions are as expected with the highest-grade domains being the enriched and oxide. The box plots show very good domain control in separating copper population distributions and material types. Table 14-6 reports the statistics for the main domains in support of the box plot distributions in Figure 14-18.

Table 14-6: Parks/Salyer Descriptive Statistics of Total Copper and Total Soluble Copper Grades

Variable Name	Count	Mean	Standard Deviation	Variance	CV	Max.	Upper Quartile	Median	Lower Quartile	Min
CuT [PSLEA] <Positive>	613	0.021	0.021	0.000	1.017	0.324	0.024	0.016	0.009	0.006
TSol [PSLEA] <Positive>	613	0.009	0.007	0.000	0.778	0.094	0.009	0.006	0.006	0.006
CuT [PSOX] <Positive>	524	0.260	0.425	0.181	1.633	2.551	0.295	0.070	0.027	0.006
TSol [PSOX] <Positive>	524	0.234	0.402	0.161	1.721	2.540	0.252	0.052	0.013	0.006
CuT [PSENR] <Positive>	1367	0.676	0.758	0.575	1.121	7.949	0.957	0.444	0.152	0.006
TSol [PSENR] <Positive>	1367	0.599	0.731	0.534	1.219	7.935	0.852	0.348	0.090	0.006
CuT [PSPR] <Positive>	1287	0.287	0.275	0.076	0.957	1.525	0.413	0.213	0.073	0.006
TSol [PSPR] <Positive>	1287	0.044	0.114	0.013	2.564	1.389	0.033	0.020	0.010	0.006

Figure 14-18: Box Plots of Total Copper and Total Soluble Copper Grades within Copper Mineral Domains for Parks/Salyer



To confirm the relationship between TSol copper and CuT, scatterplots were plotted for the oxide, enriched, and primary domains (Figure 14-19 through Figure 14-21). For the soluble domains, namely oxide and enriched, the bulk of the TSol copper is expected to plot towards the 45° line, indicating a 1:1 relationship. Samples plotting well away from this line would indicate significant mixing of populations and the potential for significant transitional zones within the mineralization. The bulk of the population does plot close to the 45° line consistent with a low level of transitional material. For the primary domain, the TSol copper generally plots well away from the 1:1 line, indicating the presence of non-soluble copper as chalcopyrite.

Figure 14-19: Scatter Plots of Total Soluble Copper versus Total Copper within the Oxide Domain for Parks/Salyer

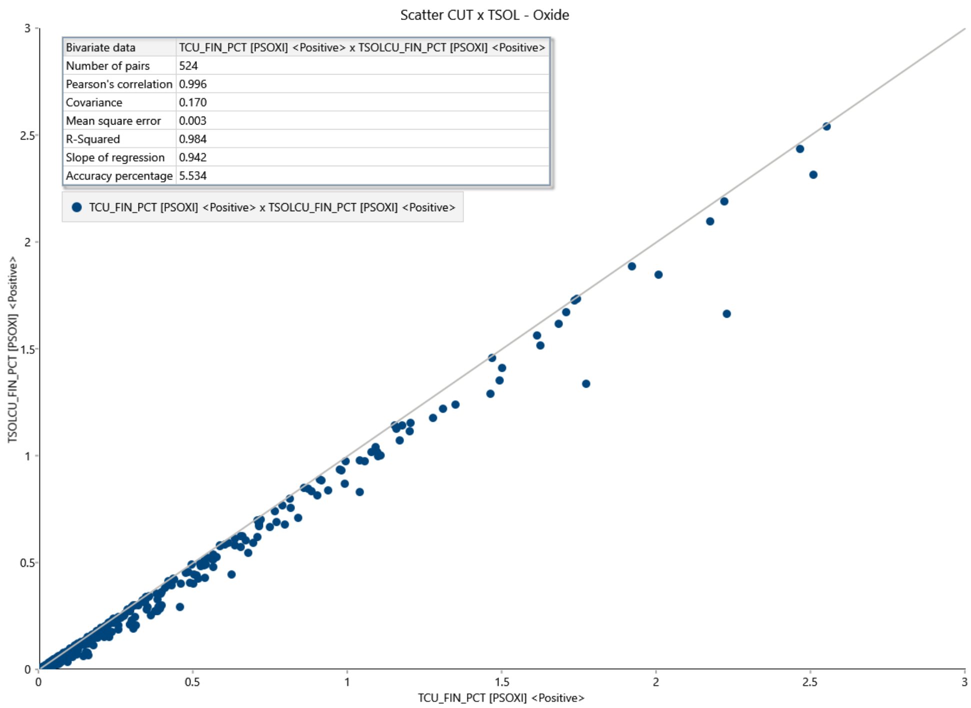


Figure 14-20: Scatter Plots of Total Soluble Copper versus Total Copper within the Enriched Domain for Parks/Salyer

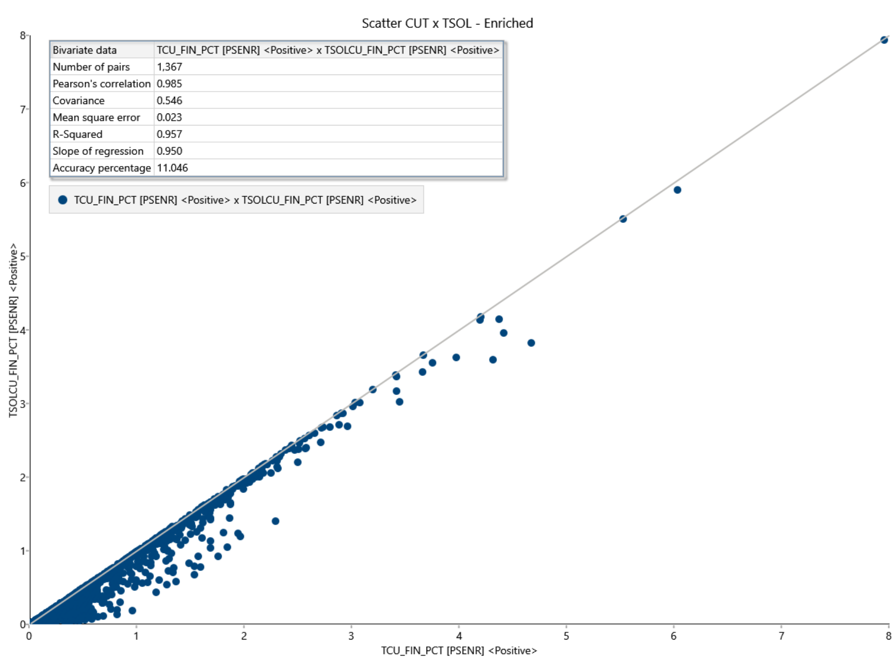
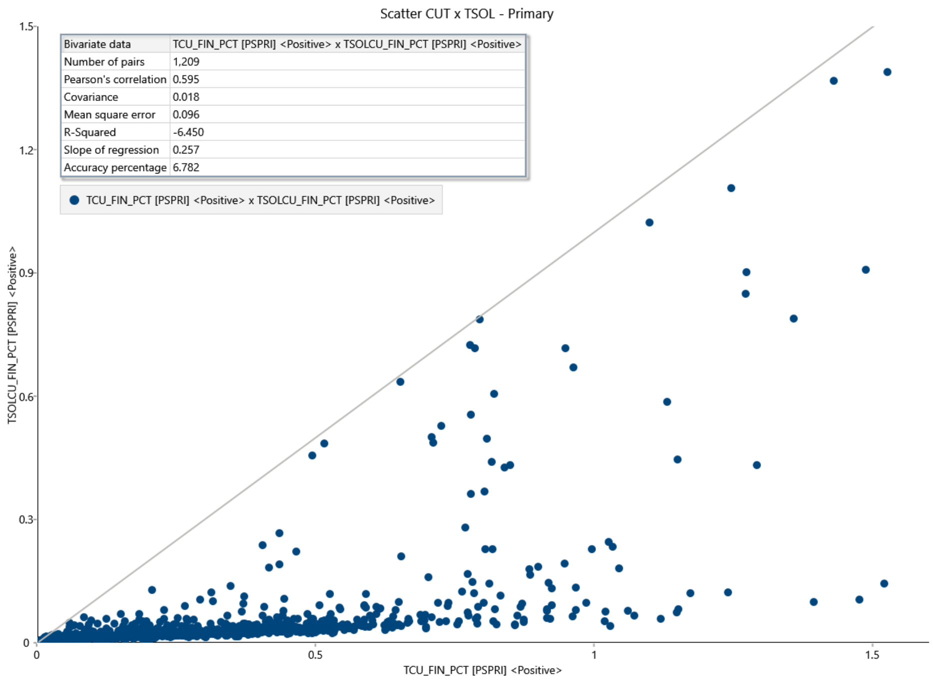


Figure 14-21: Scatter Plots of Total Soluble Copper versus Total Copper within the Primary Domain for Parks/Salyer



For oxide and enriched domains, the bulk of the copper is soluble and plots towards the 45° line indicating a 1:1 relationship with CuT. For the primary domain, as expected, the bulk of the copper is chalcopyrite and therefore the TSol copper is low and plots well away from the 45° line however a transitional zone to the eastern side of the deposit is present, in conjunction with covellite enriched/hypogene mineralization, which is visible in the scatterplot results.

Due to the different copper mineral species within the copper mineral domains (supported by the different grade distributions) and different mechanisms for precipitation, contacts between the copper mineral domains in Parks/Salyer were treated as hard contacts and therefore contact analysis was not undertaken.

The defined estimation domains show a high degree of control over the copper distributions seen within the Parks/Salyer deposit and are appropriate for use in grade estimation to produce robust estimates of copper grades.

14.1.7 Capping

The raw assay data was reviewed to determine if there were sufficient high grades in the various populations to require capping of the high grades during compositing. Histogram and log normal cumulative probability plots were reviewed for CuT and TSol in each of the mineral zones in the Cactus Project resource. Figure 14-22 is a probability plot of CuT showing a good linear plot of values above detection levels on the left side of the chart. The stepped nature of low-grade samples is evidence of the changes in detection limits at the various assay labs used over the years of activity at Cactus. There is a minor break in linearity at 1.6, which transforms to 5% CuT. A review of Figure 14-23, which is a histogram plot of CuT values shows that 5% represents the high-end tail of the grades. A further review of Figure 14-24, which is a box plot of CuT grades, shows that 5% CuT does represent the high end of grades in the deposit. A capping grade of 5% CuT was chosen, with all grades above 5% set to 5% at time of compositing. This only affected 22 intervals in the dataset.

The process was repeated for TSol, which identified 5% TSol as an appropriate capping grade. This affected 20 intervals in the diamond drill database.

Figure 14-22: Log Normal Probability Plot of Total Copper Assays

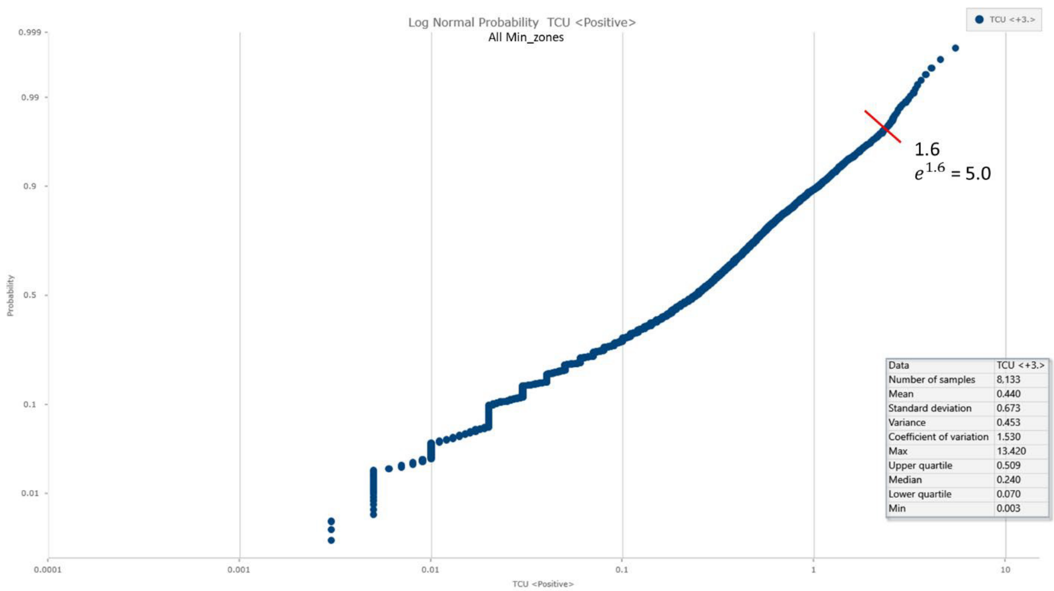


Figure 14-23: Histogram of Total Copper Assays

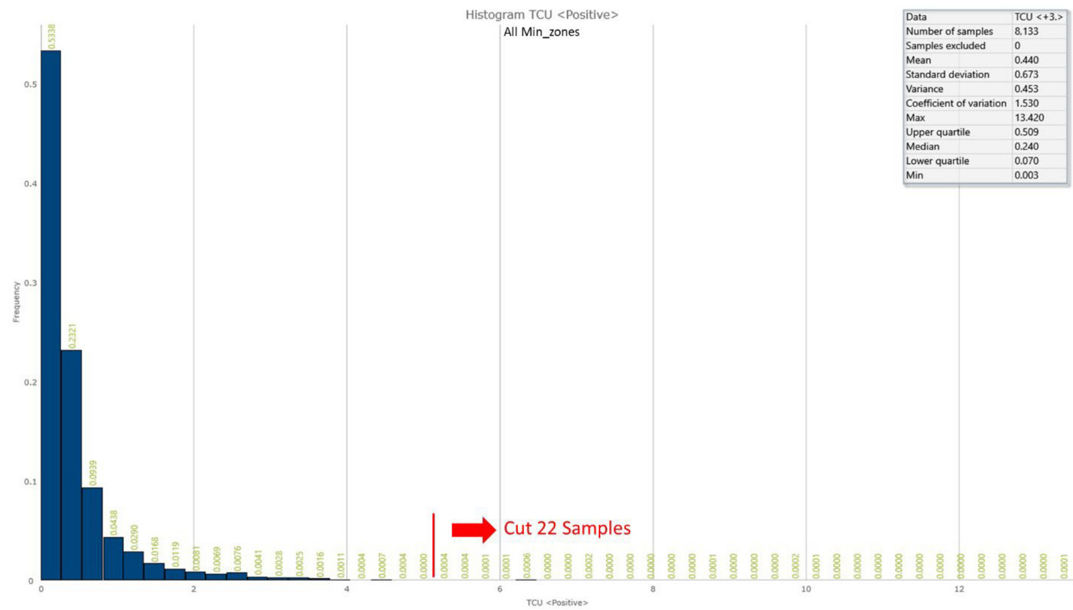
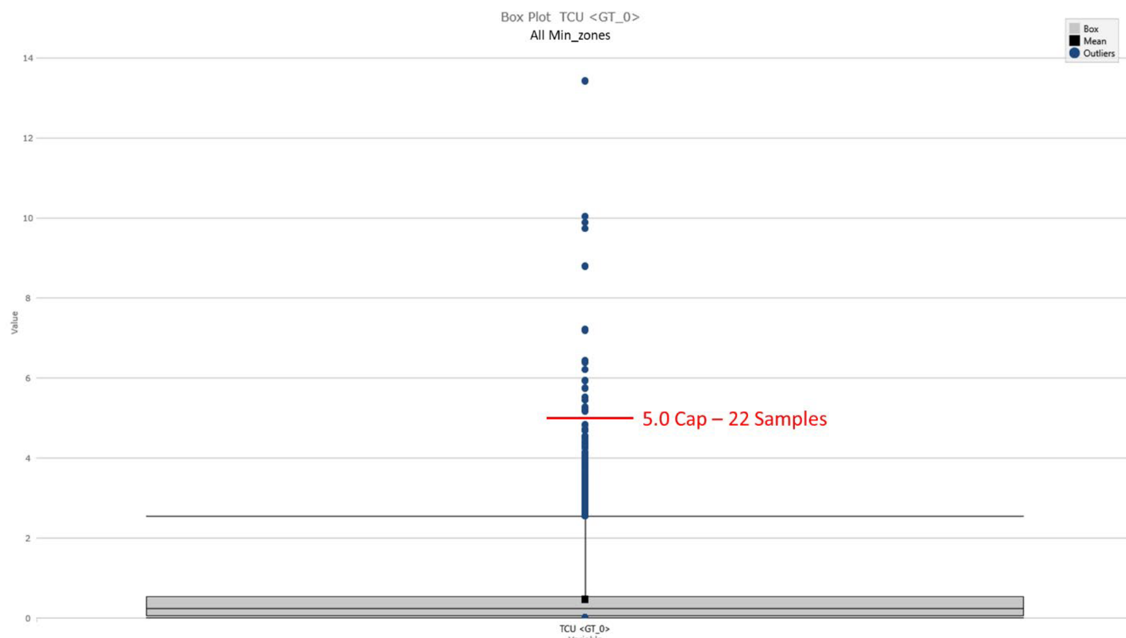


Figure 14-24: Box Plot of Total Copper Assays

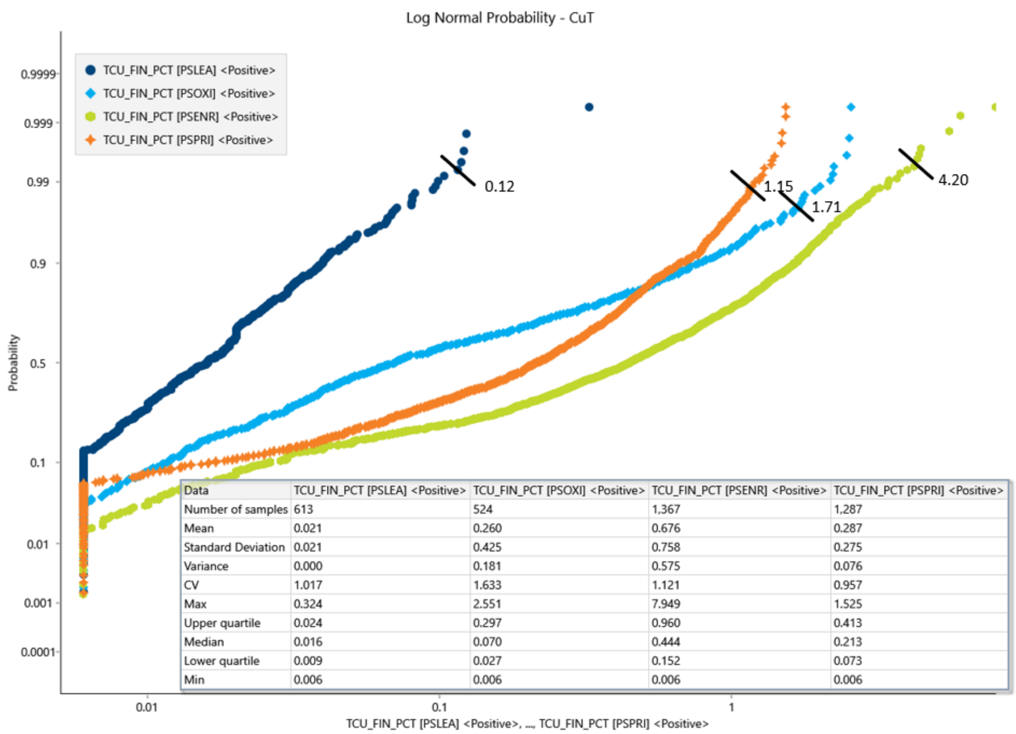


For Parks/Salyer, top cutting was reviewed on a domain basis for both CuT and TSol and are presented in Table 14-7. Populations were reviewed on log normal probability plots to determine top cut levels and are presented in Figure 14-25 to Figure 14-26.

Table 14-7: Capping Levels for Parks/Salyer Estimation Domains

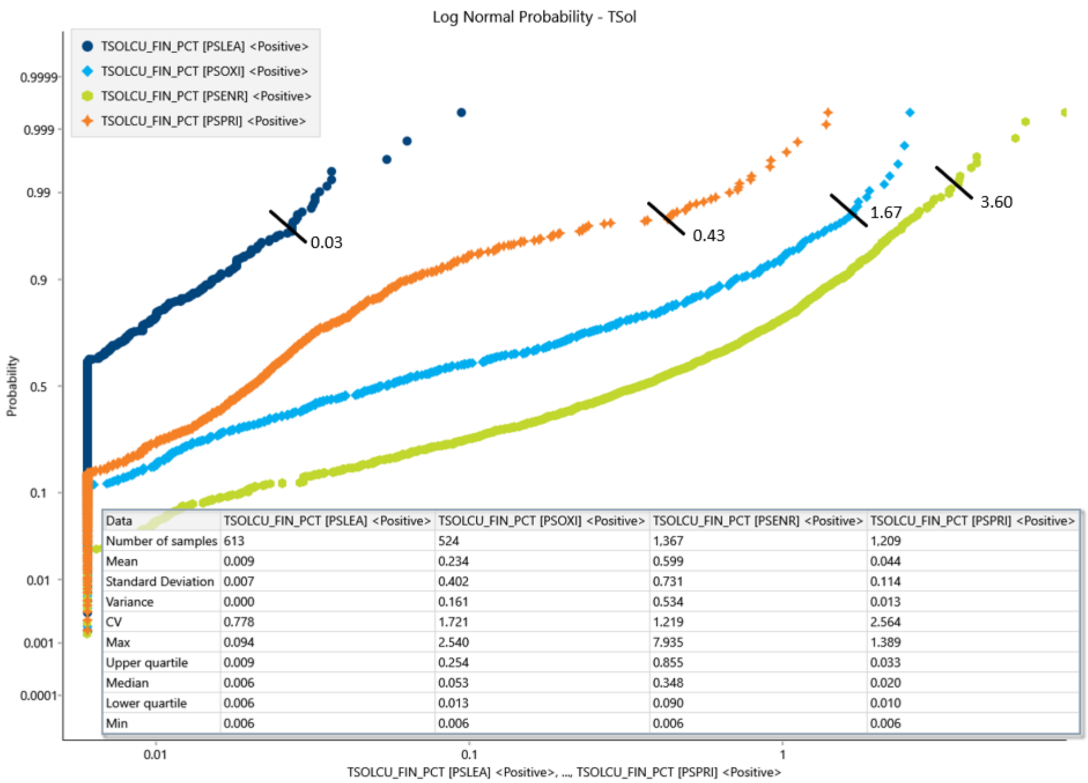
Domain	CuT Top Cut	Samples Cut	TSol Top Cut	Samples Cut
Leached	0.12	2	0.03	11
Oxide	1.71	11	1.67	9
Enriched	4.20	7	3.60	10
Primary	1.15	16	0.43	27

Figure 14-25: Log Normal Probability Plot of CuT for Parks/Salyer Plotting each Estimation Domain and Capping Level Selected



Data	TCU_FIN_PCT [PSLEA] <Positive>	TCU_FIN_PCT [PSOXI] <Positive>	TCU_FIN_PCT [PSENR] <Positive>	TCU_FIN_PCT [PSPRI] <Positive>
Number of samples	613	524	1,367	1,287
Mean	0.021	0.260	0.676	0.287
Standard Deviation	0.021	0.425	0.758	0.275
Variance	0.000	0.181	0.575	0.076
CV	1.017	1.633	1.121	0.957
Max	0.324	2.551	7.949	1.525
Upper quartile	0.024	0.297	0.960	0.413
Median	0.016	0.070	0.444	0.213
Lower quartile	0.009	0.027	0.152	0.073
Min	0.006	0.006	0.006	0.006

Figure 14-26: Log Normal Probability Plot of TSol for Parks/Salyer Plotting each Estimation Domain and Capping Level Selected



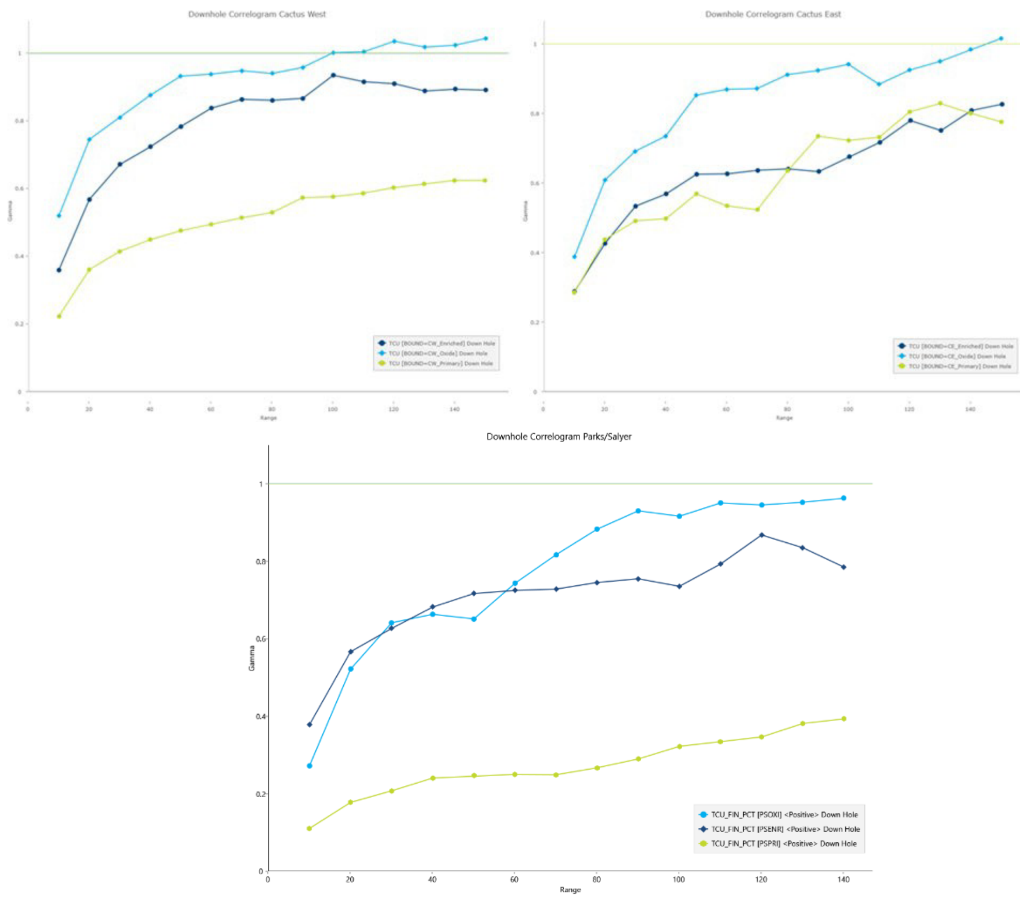
Data	TSOLCU_FIN_PCT [PSLEA] <Positive>	TSOLCU_FIN_PCT [PSOXI] <Positive>	TSOLCU_FIN_PCT [PSENR] <Positive>	TSOLCU_FIN_PCT [PSPRI] <Positive>
Number of samples	613	524	1,367	1,209
Mean	0.009	0.234	0.599	0.044
Standard Deviation	0.007	0.402	0.731	0.114
Variance	0.000	0.161	0.534	0.013
CV	0.778	1.721	1.219	2.564
Max	0.094	2.540	7.935	1.389
Upper quartile	0.009	0.254	0.855	0.033
Median	0.006	0.053	0.348	0.020
Lower quartile	0.006	0.013	0.090	0.010
Min	0.006	0.006	0.006	0.006

14.1.8 Variography

Robust variograms could not be generated from the estimation domains due to the wide spaced nature of the drilling. Variogram modelling represents a future improvement to the modelling once further infill drilling is available. Downhole variograms were calculated and analyzed. All estimation domains support a low nugget effect to the mineralization. Nuggets of enriched and oxide domains were typically between 10%-15% of the total sill. Nuggets of the primary domain were lower, as expected, at between 5%-10% of the total sill.

Figure 14-27 shows downhole variograms for Cactus West (upper left), Cactus East (upper right) and Parks/Salyer (below). Downhole, primary material shows greater continuity than enriched and oxide. Oxide contains the highest variability because of the variable leaching that occurs in the oxide zone. Relationships seen in the downhole variograms show similar nuggets, but more variable continuity which can be related to mechanisms of deposition of the copper minerals and subsequent deeper leaching in Cactus West.

Figure 14-27: Downhole Variograms for the Cactus West (Upper Left), Cactus East (Upper Right) and Parks/Salyer (Below)



14.1.9 Block Model

The block model for Cactus was constructed to encompass the full extents of the Cactus deposits, including additional waste outside the model to support pit optimization work. The block model for Parks/Salyer was constructed to encompass the extents of Parks/Salyer mineralization only. Parent blocks in both models were defined with 20 ft (6 m) by 20 ft (6 m) by 20 ft (6 m) block sizes to support minimum pit selectivity with sub-blocking to honor geological and topographical contacts of 5.0 ft (1.5 m) by 5.0 ft (1.5 m) by 2.5 ft (0.8 m). Table 14-8 outlines the Cactus block model definition parameters. Table 14-9 outlines the Parks/Salyer block model definition parameters.

Table 14-8: Cactus Block Model Definition Parameters

	X	Y	Z
Origin	385,900	60,800	-1,000.0
Bearing / Dip / Plunge	90	0	0.0
Offset Minimum	0	0	0.0
Extent Maximum	9,100	8,100	3,000.0
Parent Block Size	20	20	20.0
Sub-block Block Size	5	5	2.5
Total Blocks			**32,830,478**

Table 14-9: Parks/Salyer Block Model Definition Parameters

	X	Y	Z
Origin	380,500	57,500	-1,500.0
Bearing / Dip / Plunge	90	0	0.0
Offset Minimum	0	0	0.0
Extent Maximum	6,500	4,000	3,000.0
Parent Block Size	20	20	20.0
Sub-block Block Size	5	5	2.5
Total Blocks			**5,931,016**

14.1.10 Estimation Plan

ID3 was selected for the estimation of copper grades to the block model due to the wide drill spacings present in the data, and regular arrangement of the drilling grid. The drill spacing is not sufficient to support the determination of robust variogram models. Smoothing checks in the estimation validation support the use of ID3 as a reasonable approximation of the expected grade tonnage curve supporting open pit and underground COGs for PEA level study. For Cactus West in particular, zones of the oxide and enriched domains contain deeper leached material that could not be defined in the current model based on wide spaced drilling. Using ID3 minimized over-smoothing of the higher grades into the leached zones. For Parks/Salyer, a waste indicator was applied, based on a 0.025% CuT grade, to define deeper leaching within the oxide and enriched zones and these blocks were estimated as part of the leached domain.

With appropriate infill drilling to define more robust variogram models, Ordinary Kriging (OK), with the additional domain controls for the deeper leaching zones applied to Parks/Salyer, should improve the quality of local estimates in the future.

The estimation passes were defined based on the general drill spacings present within the project area. Pass 1 was defined to estimate drilling with approximately 250 ft (76 m) spacing. This drill spacing was planned to target definition of Indicated resources (not applicable to Parks/Salyer estimates). Pass 2 was defined to estimate drilling with an approximately 500 ft (152 m) spacing. This drill spacing was planned to target definition of Inferred Resources. Pass 3 was defined to estimate blocks based on wide spaced drilling, to be used as an exploration tool for future drill planning.

Multiple pass estimation was undertaken with estimation criteria such as the number of samples and search ellipse relaxed with each subsequent pass. Once a block was estimated with a grade, the block was flagged as estimated. Subsequent estimation passes would only see blocks that were not flagged as estimated. Key parameters used in the estimation plan of both the Cactus and Parks/Salyer block models are outlined in Table 14-10 and Table 14-11, respectively. Block grade estimates were undertaken on the parent cell size.

Table 14-10: Key Parameters used in Each Search Pass for Cactus

Pass	Number of Samples			Search Distance, ft (m)		
	Min	Max	Max Per Hole	Major	Semi	Minor
1	5	8	3	300 (91.4)	300 (91.4)	150 (45.7)
2	4	7	3	500 (152.4)	500 (152.4)	250 (76.2)
3	2	6	3	750 (228.6)	750 (228.6)	300 (91.4)

Table 14-11: Key Parameters used in Each Search Pass for Parks/Salyer

Pass	Number of Samples			Search Distance, ft (m)		
	Min	Max	Max Per Hole	Major	Semi	Minor
1	3	8	2	600 (183)	600 (183)	250 (76.2)
2	2	7	3	750 (228.6)	750 (228.6)	300 (91.4)

In addition, the following parameters were applied to the estimate.

- The Cactus West and Cactus East deposits were estimated separately with each treated as a hard domain, therefore only composites within Cactus West could be used to estimate Cactus West blocks and visa-versa.
- Each copper mineral domain was treated as a hard domain.
- The estimate for Cactus was undertaken using three passes. The estimate for Parks/Salyer was undertaken using two passes.
- Un-estimated blocks were assigned a grade of 0.002% CuT.

- The drilling is relatively evenly distributed throughout the deposits, so no de-clustering was applied to the samples. To ensure multiple holes from numerous directions around a block were used in the estimate, the maximum number of samples that could be used from a single hole was set to two or three depending on the estimation pass. In conjunction with the minimum number of samples, this ensured in the passes supporting indicated and inferred estimates that two or more holes were used to estimate each block.
- Grades were capped using a top-cut method. Cap levels were set for CuT and TSol values separately.
- A nearest neighbor was assigned to the blocks during the estimation process for use in validation of the estimate.

Locally, grade continuity can vary due to several factors including the following.

- Structural Controls
- Deeper Leaching Zones
- Historic Water Table Interface

A locally varying search orientation methodology was adopted because of these factors. This ensured that blocks being estimated nearer the contact of the oxide and enriched would see samples nearby that were also near the contact of the oxide and enriched (Figure 14-28) and so forth. The white line within each block displays the orientation vector of the major direction of continuity. This corresponds to the major search direction of the search ellipse at each block. Local search orientation vectors were defined using the most appropriate surfaces relating to each copper mineral domain. Table 14-12 outlines the surfaces used to define orientation vectors in each copper mineral domain.

Figure 14-28: Representative Cross Section View of the Block Model

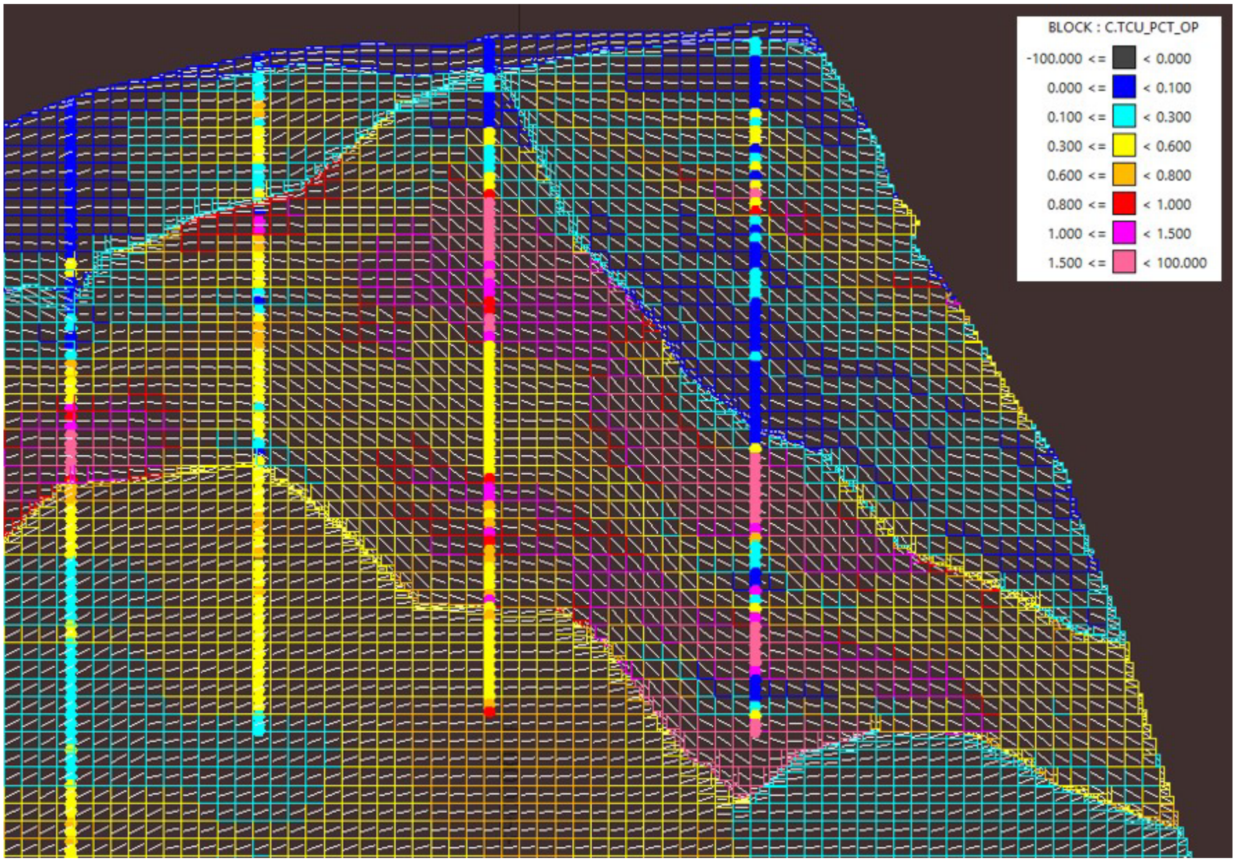


Table 14-12: Domain Surfaces

Domain	Surface(s) Defining Vector Orientation	Description
Leached	South or west fault contact and the top ofoxide	Surfaces define upper and lower contacts of the domainas controls on leaching profile.
Oxide	Top of oxide and top of enriched	Surfaces define upper and lower contacts of the domainas controls on secondary enrichment profile.
Enriched	Top of enriched and top of primary	Surfaces define upper and lower contacts of the domainas controls on secondary enrichment profile.
Primary	South or west fault contact	Contact describes the rotation of the overall fault blockwhich controls broader continuity of primary mineralization.

14.1.11 Mining Depletion

Blocks within the historically mined pit were estimated to aid in validation of the block model estimates. Prior to pit optimization and reporting, the block model grades were depleted from the historic pit using a surveyed pit shell. Due to the presence of water in the bottom of the pit, late-stage pit maps and mining reconciliation were reviewed to determine the ultimate depth of the pit. The pit shell was adjusted below the water level to fully deplete for historic production.

Depletion was achieved by setting a variable (TOPO) to one for all blocks in the model. Then all blocks above the topographic surface and mined pit shell were set to zero. A copy of the SG variable (SG_WHIT) was set in the model. SG_WHIT was set to zero if TOPO was 0.

SG_WHIT was used by Whittle as the density field thus not including any blocks above surface.

No historical mining has been undertaken into either the Cactus East or Parks/Salyer deposits; therefore, no depletion has been applied to these models.

14.1.12 Validations

Validations in this section include the mined material from the historical open pit. Grades reported in this section for Cactus West include material that is no longer present and therefore reported grades should not be considered as representative of the material that is remaining.

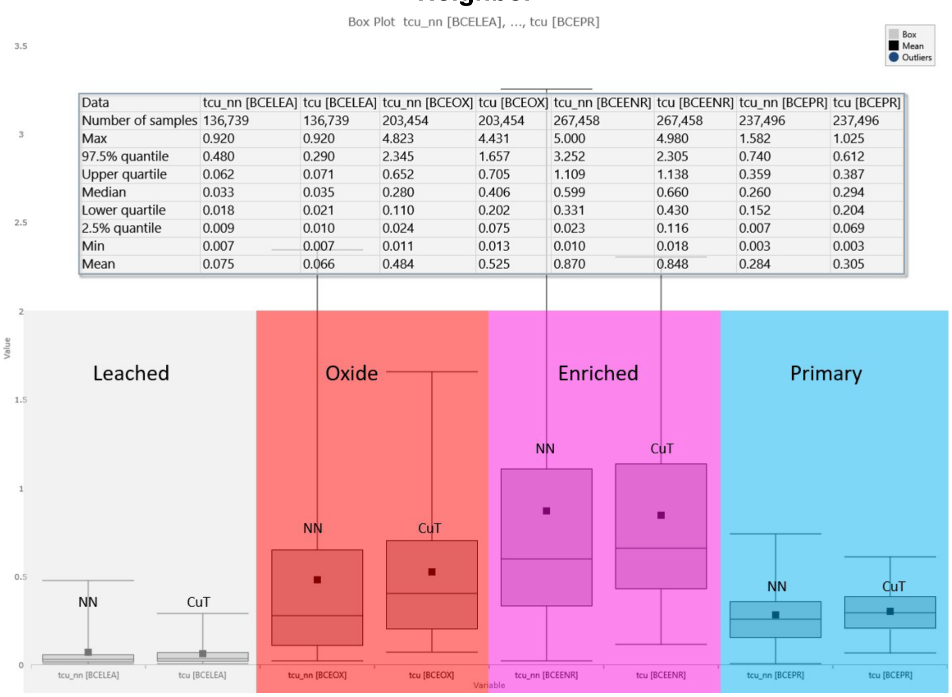
Box Plots

Box plots were created for CuT and TSol copper to compare estimated mean grades and distributions for each domain against the nearest neighbor. Box plots for Cactus West, Cactus East, and Parks/Salyer are presented in Figure 14-29 through Figure 14-34. Comparisons show similar mean grades between the estimated blocks and the nearest neighbor. The adjustment from a nearest neighbor sample support to a block estimate support incurs smoothing (particularly for wider spaced drilling programs). This smoothing is visible in the box plots by the restricted box size within the plots for the estimated blocks versus that of the comparison nearest neighbor plots. No maximum values of the nearest neighbor statistics were reported higher than the planned capping grades, indicating that the top cut was applied to the estimation as planned.

Figure 14-29: Box Plots Comparing the Total Copper for Cactus West Domains Against the Nearest Neighbor

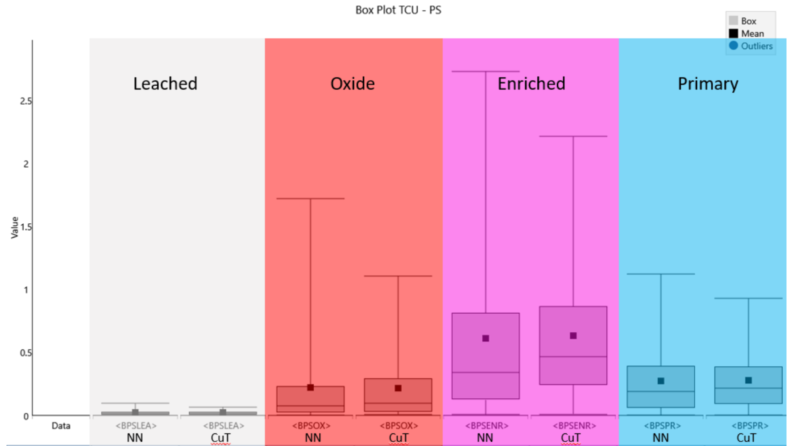
Data	tcu_nn [BCWLEA]	tcu [BCWLEA]	tcu_nn [BCWOX]	tcu [BCWOX]	tcu_nn [BCWENR]	tcu [BCWENR]	tcu_nn [BCWPR]	tcu [BCWPR]
Number of samples	141,070	141,070	462,150	462,150	435,353	435,353	725,508	725,508
Max	0.182	0.182	3.521	2.439	4.890	3.755	1.526	1.223
97.5% quantile	0.148	0.124	0.777	0.659	2.556	1.606	0.666	0.573
Upper quartile	0.058	0.055	0.206	0.222	0.668	0.687	0.374	0.367
Median	0.035	0.036	0.100	0.136	0.366	0.410	0.248	0.262
Lower quartile	0.019	0.024	0.040	0.077	0.169	0.223	0.156	0.186
2.5% quantile	0.005	0.010	0.008	0.019	0.012	0.060	0.041	0.066
Min	0.003	0.003	0.003	0.005	0.006	0.009	0.012	0.030
Mean	0.045	0.042	0.177	0.179	0.562	0.514	0.276	0.280

Figure 14-30: Box Plots Comparing the CuT for Cactus East Domains Against the Nearest Neighbor



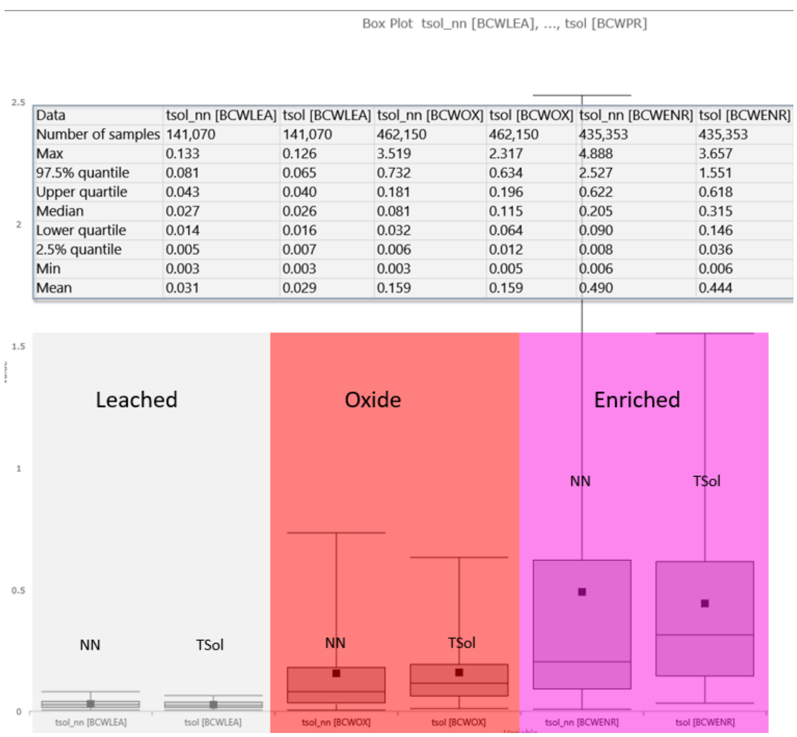
Data	tcu_nn [BCELEA]	tcu [BCELEA]	tcu_nn [BCEOX]	tcu [BCEOX]	tcu_nn [BCEENR]	tcu [BCEENR]	tcu_nn [BCEPR]	tcu [BCEPR]
Number of samples	136,739	136,739	203,454	203,454	267,458	267,458	237,496	237,496
Max	0.920	0.920	4.823	4.431	5.000	4.980	1.582	1.025
97.5% quantile	0.480	0.290	2.345	1.657	3.252	2.305	0.740	0.612
Upper quartile	0.062	0.071	0.652	0.705	1.109	1.138	0.359	0.387
Median	0.033	0.035	0.280	0.406	0.599	0.660	0.260	0.294
Lower quartile	0.018	0.021	0.110	0.202	0.331	0.430	0.152	0.204
2.5% quantile	0.009	0.010	0.024	0.075	0.023	0.116	0.007	0.069
Min	0.007	0.007	0.011	0.013	0.010	0.018	0.003	0.003
Mean	0.075	0.066	0.484	0.525	0.870	0.848	0.284	0.305

Figure 14-31: Box Plots Comparing the CuT for Parks/Salyer Domains Against the Nearest Neighbor



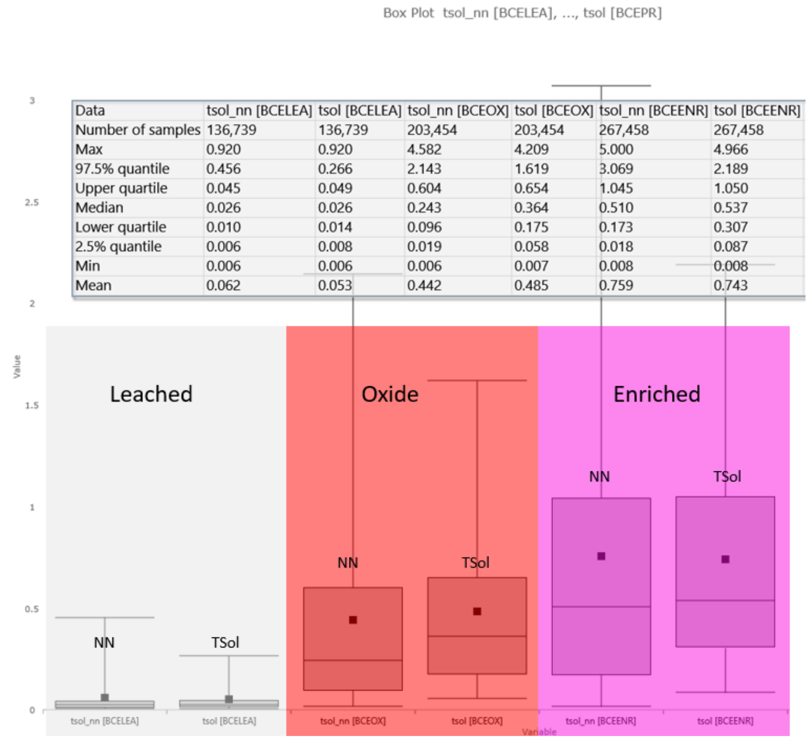
Data	<BPSLEA> NN	<BPSLEA> CuT	<BPSOX> NN	<BPSOX> CuT	<BPSENR> NN	<BPSENR> CuT	<BPSPR> NN	<BPSPR> CuT
Number of samples	795,203	795,203	673,654	684,302	1,203,817	1,203,817	1,220,614	1,220,620
Max	0.120	0.120	1.710	1.710	4.200	4.116	1.150	1.150
97.5% quantile	0.096	0.068	1.710	1.101	2.713	2.202	1.118	0.927
Upper quartile	0.030	0.031	0.234	0.293	0.810	0.863	0.394	0.390
Median	0.019	0.019	0.075	0.099	0.342	0.466	0.192	0.218
Lower quartile	0.010	0.012	0.027	0.030	0.125	0.241	0.064	0.093
2.5% quantile	0.006	0.006	0.006	0.008	0.010	0.012	0.006	0.006
Min	0.006	0.006	0.006	0.006	0.006	0.006	0.006	0.006
Mean	0.024	0.024	0.223	0.219	0.608	0.632	0.273	0.279

Figure 14-32: Box Plots Comparing the Total Soluble Copper for Cactus West Domains Against the Nearest Neighbor



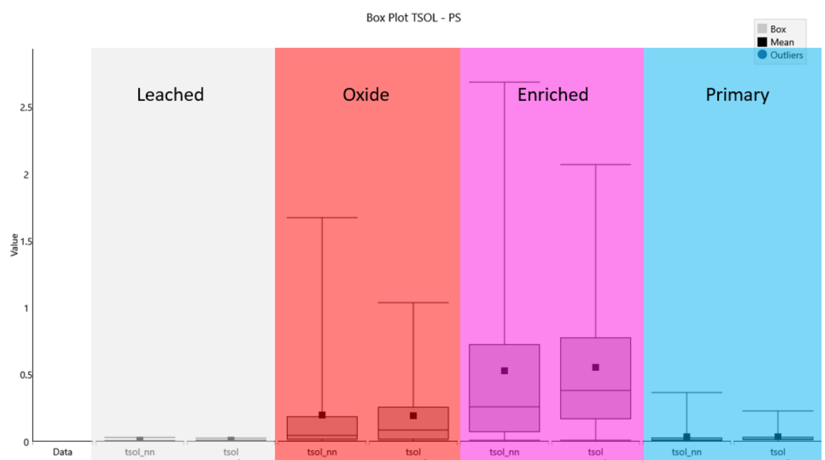
Data	tsol_nn [BCWLEA]	tsol [BCWLEA]	tsol_nn [BCWOX]	tsol [BCWOX]	tsol_nn [BCWENR]	tsol [BCWENR]
Number of samples	141,070	141,070	462,150	462,150	435,353	435,353
Max	0.133	0.126	3.519	2.317	4.888	3.657
97.5% quantile	0.081	0.065	0.732	0.634	2.527	1.551
Upper quartile	0.043	0.040	0.181	0.196	0.622	0.618
Median	0.027	0.026	0.081	0.115	0.205	0.315
Lower quartile	0.014	0.016	0.032	0.064	0.090	0.146
2.5% quantile	0.005	0.007	0.006	0.012	0.008	0.036
Min	0.003	0.003	0.003	0.005	0.006	0.006
Mean	0.031	0.029	0.159	0.159	0.490	0.444

Figure 14-33: Box Plots Comparing the Total Soluble Copper for Cactus East Domains Against the Nearest Neighbor



Data	tsol_nn [BCELEA]	tsol [BCELEA]	tsol_nn [BCEOX]	tsol [BCEOX]	tsol_nn [BCEENR]	tsol [BCEENR]
Number of samples	136,739	136,739	203,454	203,454	267,458	267,458
Max	0.920	0.920	4.582	4.209	5.000	4.966
97.5% quantile	0.456	0.266	2.143	1.619	3.069	2.189
Upper quartile	0.045	0.049	0.604	0.654	1.045	1.050
Median	0.026	0.026	0.243	0.364	0.510	0.537
Lower quartile	0.010	0.014	0.096	0.175	0.173	0.307
2.5% quantile	0.006	0.008	0.019	0.058	0.018	0.087
Min	0.006	0.006	0.006	0.007	0.008	0.008
Mean	0.062	0.053	0.442	0.485	0.759	0.743

Figure 14-34: Box Plots Comparing the Total Soluble Copper for Parks/Salyer Domains Against the Nearest Neighbor

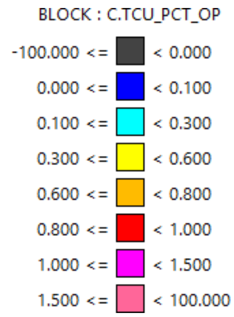


Data	tsol_nn NN	tsol TSol	tsol_nn NN	tsol TSol	tsol_nn NN	tsol TSol	tsol_nn NN	tsol TSol
Number of samples	795,203	795,203	673,654	684,302	1,203,817	1,203,817	1,152,490	1,220,620
Max	0.030	0.030	1.670	1.670	3.600	3.600	0.430	0.430
97.5% quantile	0.030	0.026	1.664	1.031	2.670	2.056	0.362	0.228
Upper quartile	0.012	0.012	0.186	0.257	0.725	0.774	0.030	0.035
Median	0.006	0.008	0.047	0.086	0.258	0.378	0.017	0.020
Lower quartile	0.006	0.006	0.014	0.016	0.071	0.167	0.007	0.010
2.5% quantile	0.006	0.006	0.006	0.006	0.008	0.009	0.001	0.001
Min	0.006	0.006	0.006	0.006	0.006	0.006	0.001	0.001
Mean	0.010	0.010	0.196	0.193	0.527	0.551	0.035	0.036

Visual Validations

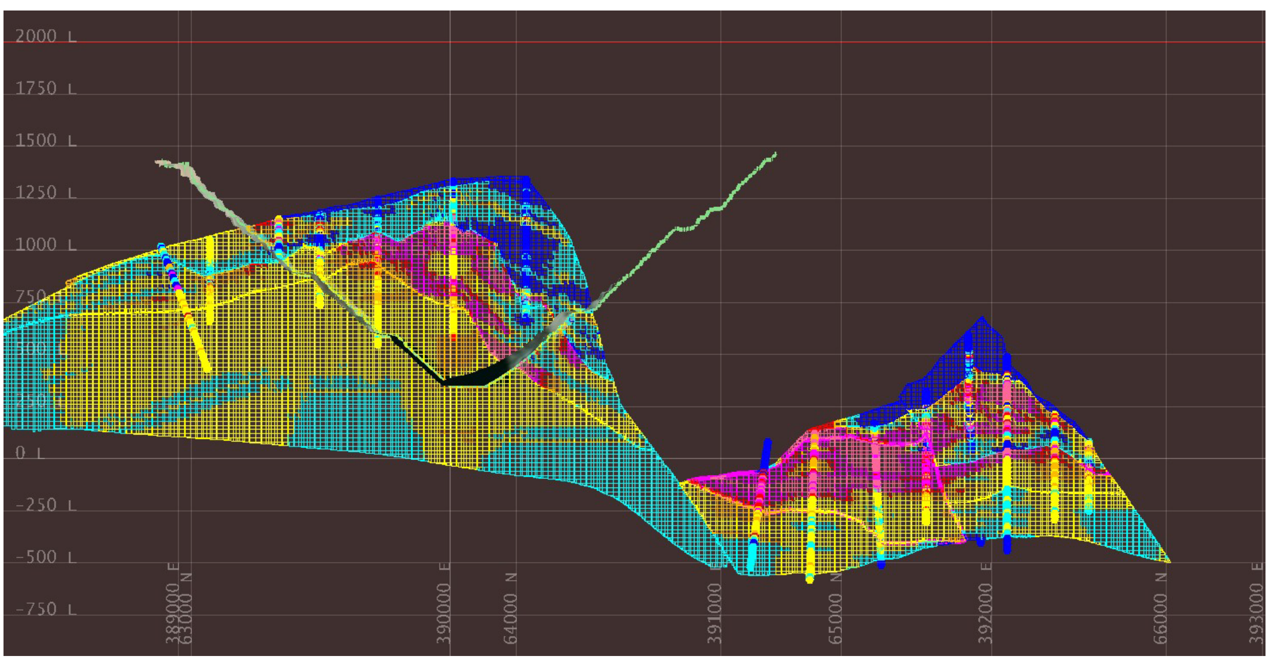
The color legend of Figure 14-35 is applied to all block and composite grade values for comparative purposes. The legend applies to CuT and TSol. Examination indicates appropriate agreement of block grade estimates with the composites. Visual validations confirm the overall grade trends through the copper mineral domains are represented as planned.

Figure 14-35: Legend for Total Copper and Total Soluble Grades



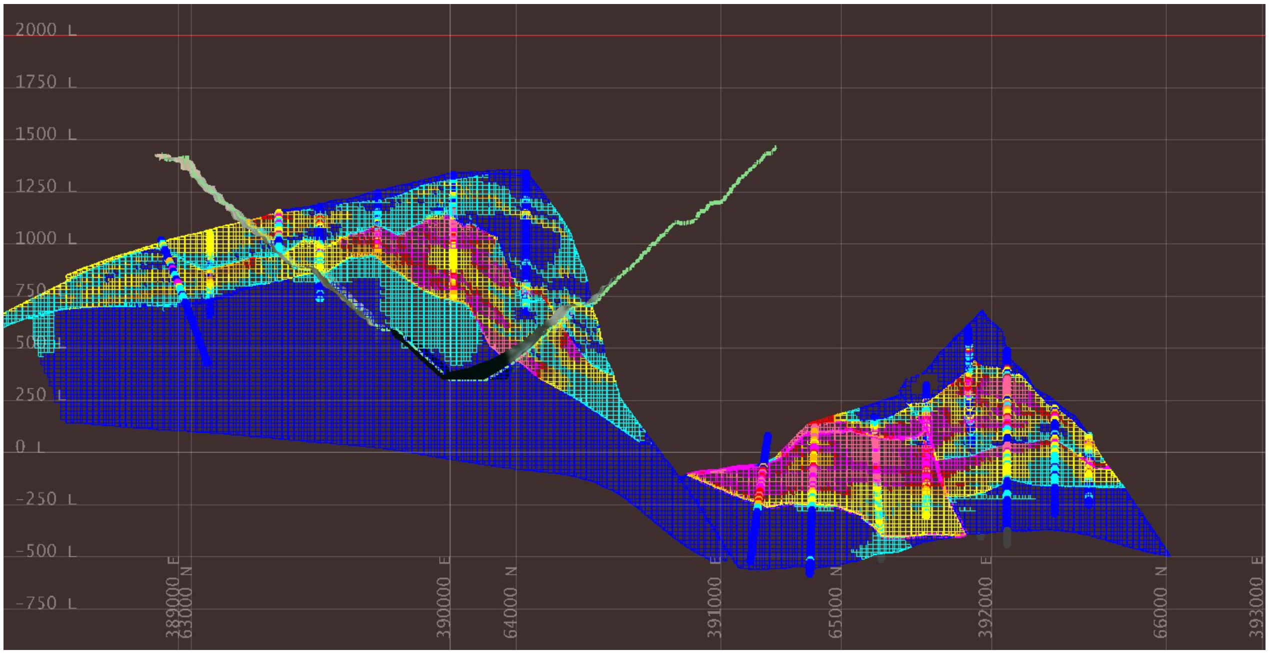
On a local scale, model validation can be confirmed by the visual comparison of block grades to composite grades. A long section through the Cactus East and Cactus West, plus a cross section through each of the Cactus East, Cactus West, and Parks/Salyer deposits, show grade trends through the block model. The first section of each pair shows total copper values, the second shows TSol values. Each section shows the estimated variables with composites superimposed as dots on block grades in Figure 14-36 through Figure 14-43.

Figure 14-36: Long Section through Cactus West and Cactus East, Facing Northwest



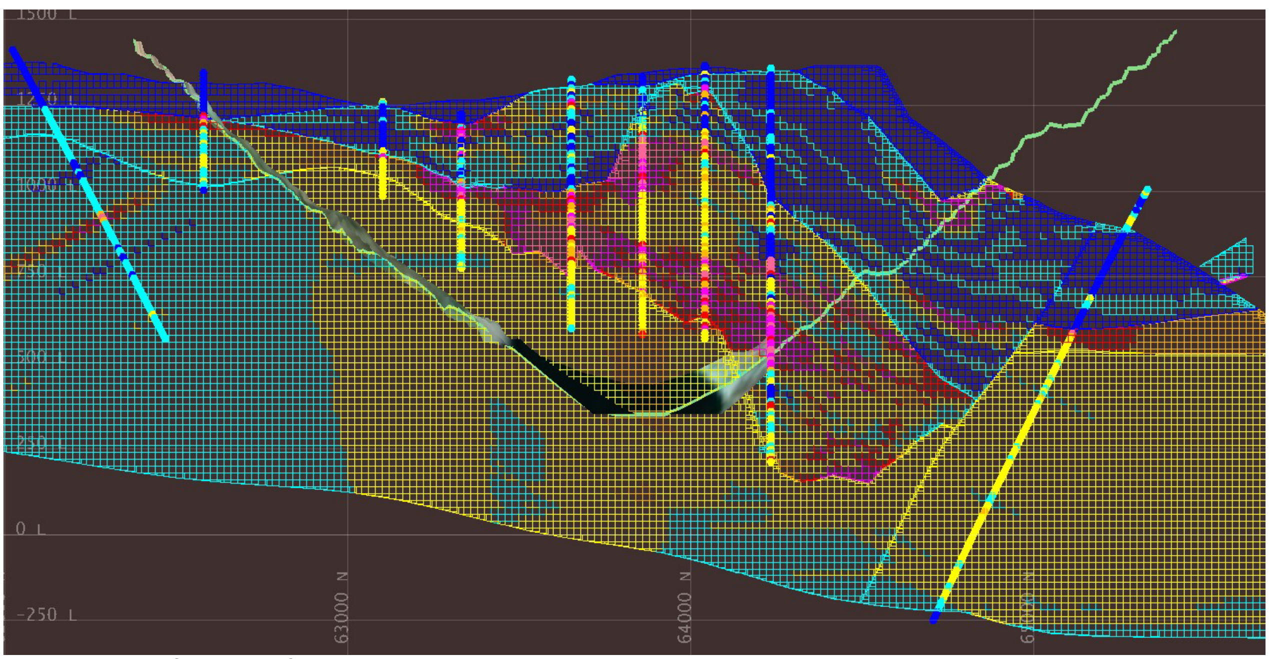
Viewing Total Copper Grades for Both Composites and Block Estimates.

Figure 14-37: Long Section through Cactus West and Cactus East, Facing Northwest



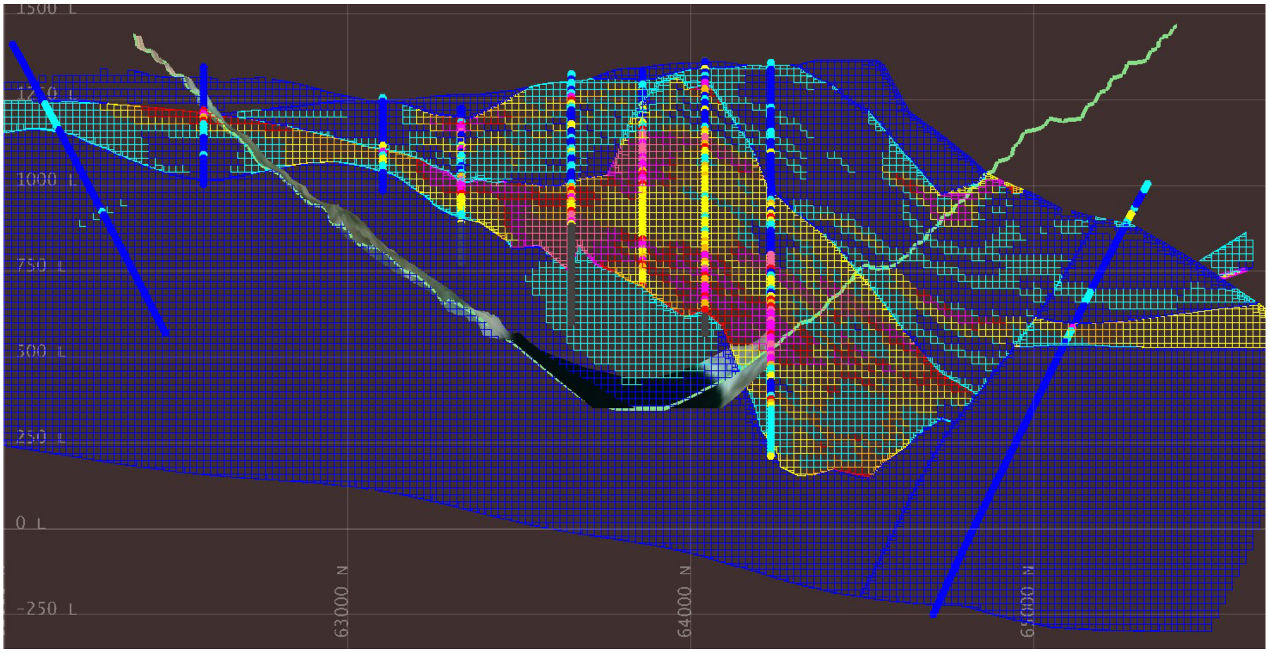
Viewing TSol grades for both composites and block estimates.

Figure 14-38: Cross Section (390000E) through Cactus West, Facing West



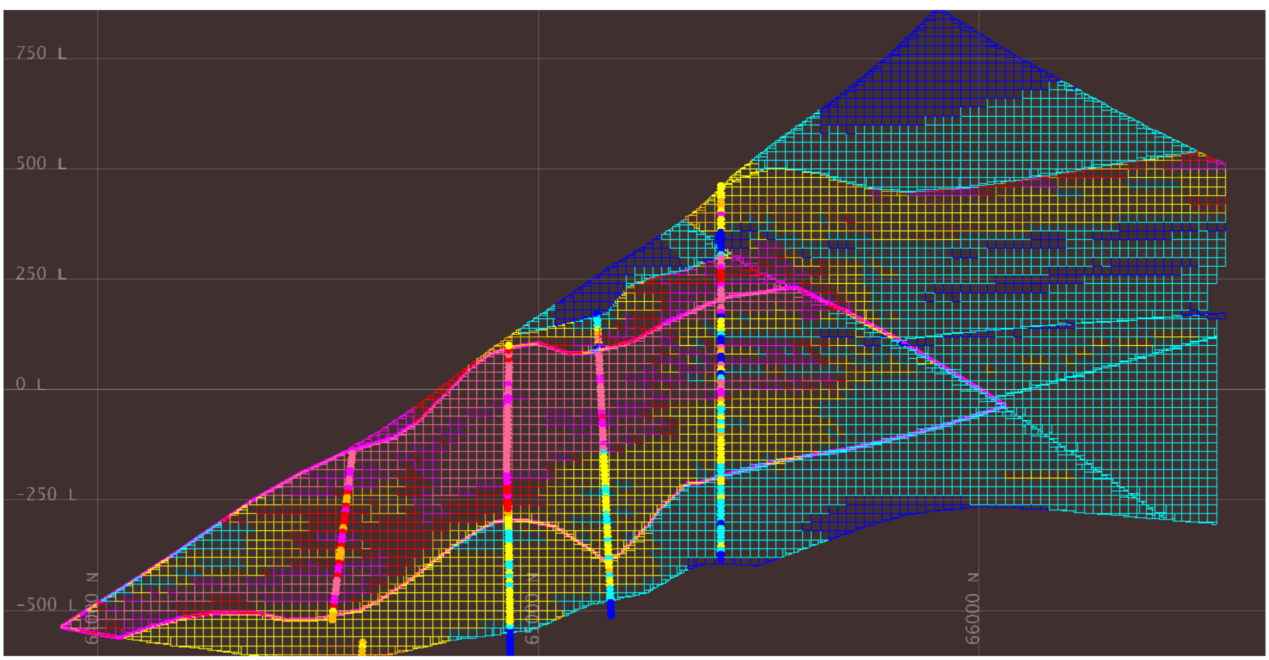
Viewing CuT grades for both composites and block estimates.

Figure 14-39: Cross Section (390000E) through Cactus West, facing West



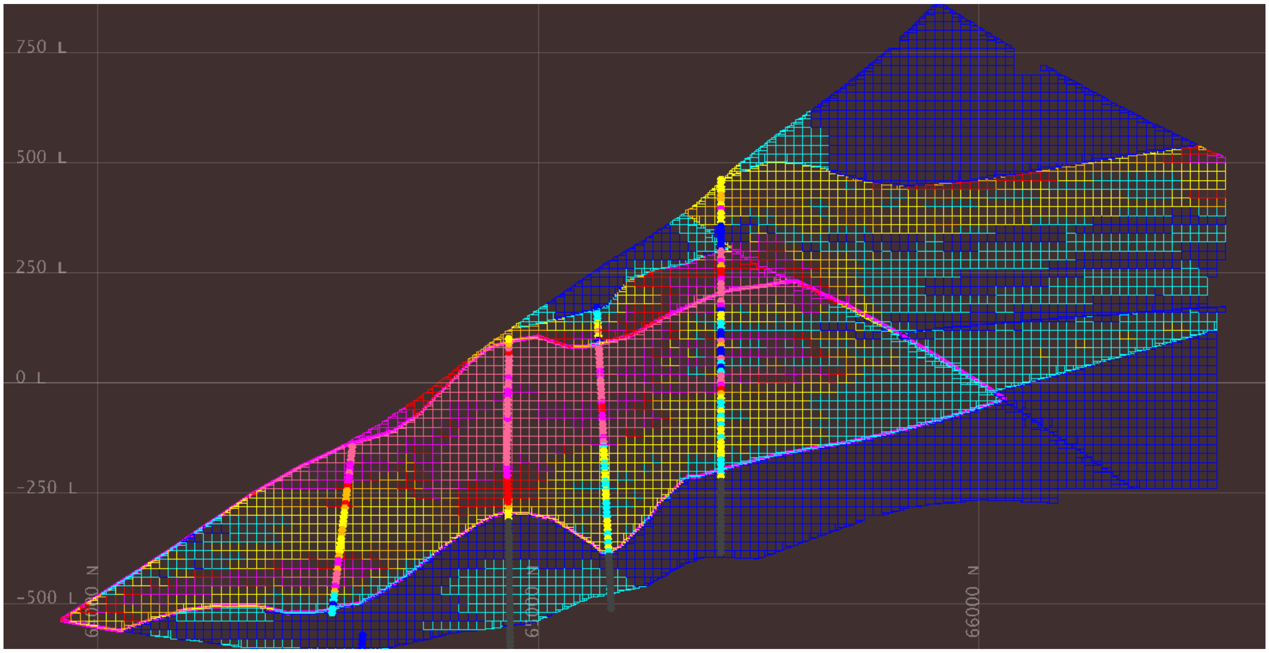
Viewing TSol grades for both composites and block estimates.

Figure 14-40: Cross Section (391550E) through Cactus East, Facing West



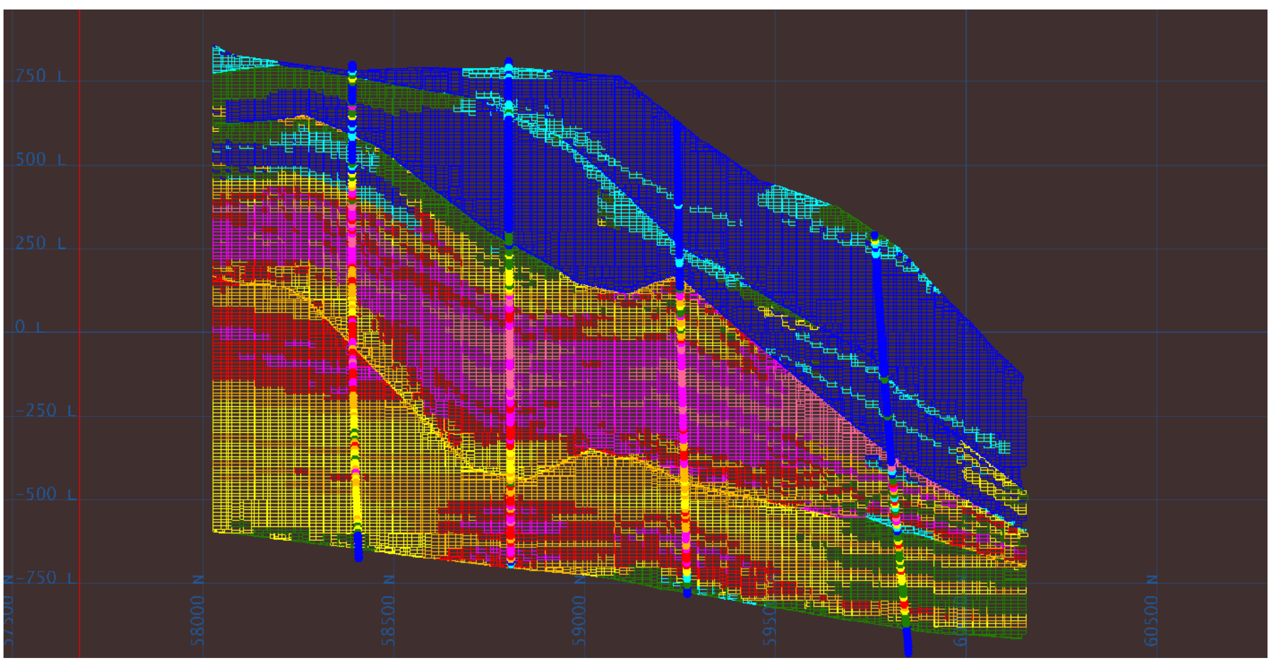
Viewing CuT grades for both composites and block estimates.

Figure 14-41: Cross Section (391550E) through Cactus East, Facing West



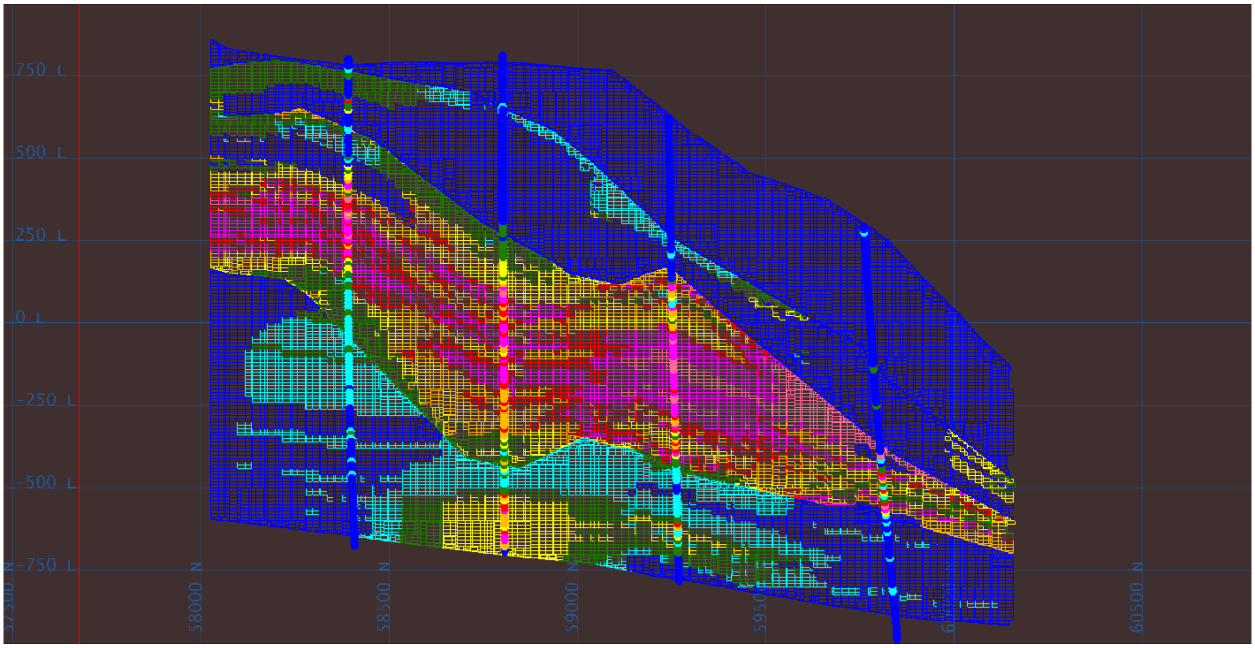
Viewing TSol grades for both composites and block estimates.

Figure 14-42: Angled Cross Section through Parks/Salyer, Facing Northwest



Viewing CuT grades for both composites and block estimates.

Figure 14-43: Angled Cross Section through Parks/Salyer, Facing Northwest



Viewing TSol grades for both composites and block estimates.

Swath Plots

Swath plots were created to compare the grade trends through the Cactus West, Cactus East, and Parks/Salyer deposits between the estimated CuT and TSol against the nearest neighbor models.

Comparisons for CuT and TSol in Cactus West, Cactus East, and Parks/Salyer are shown in Figure 14-44 through Figure 14-46, respectively, for easting (X direction), northing (Y direction), and elevation (Z direction).

Figure 14-44: Swath Plots through Cactus West Comparison with Associated Nearest Neighbor Grade Trends

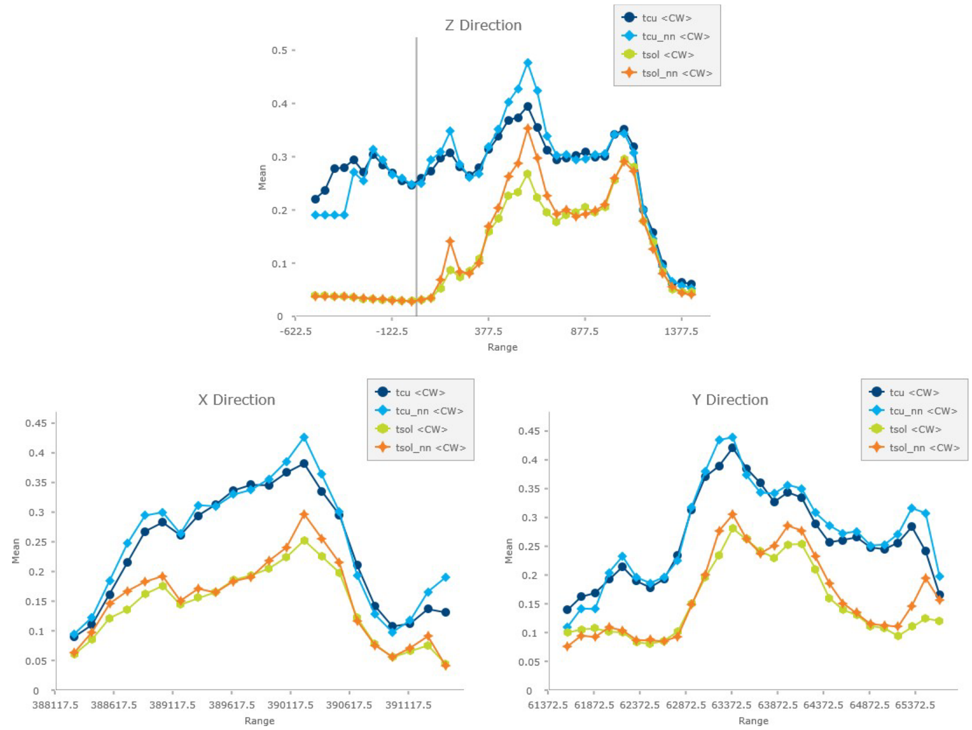


Figure 14-45: Swath Plots through Cactus East Comparison with Associated Nearest Neighbor Grade Trends

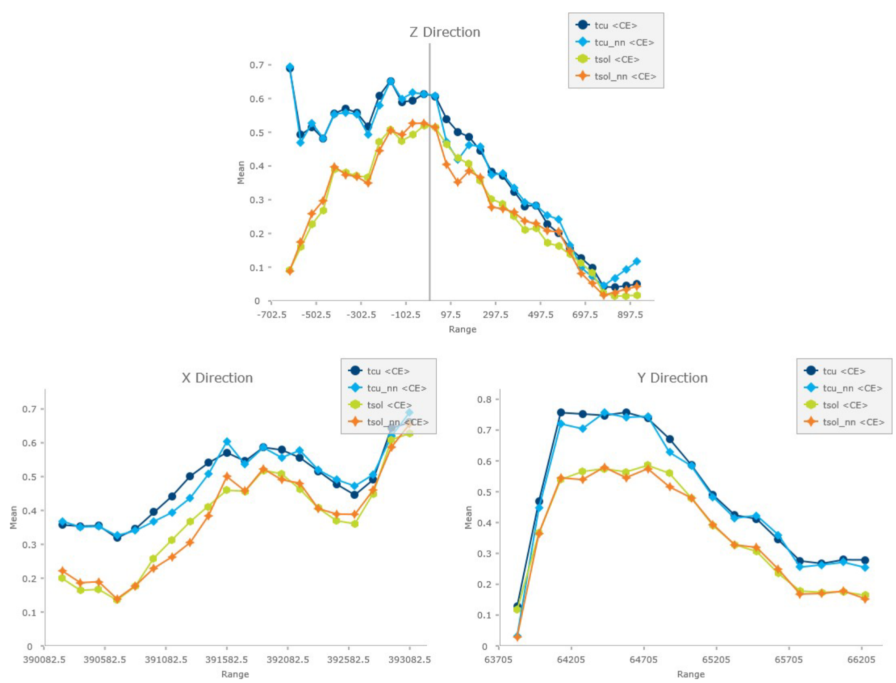
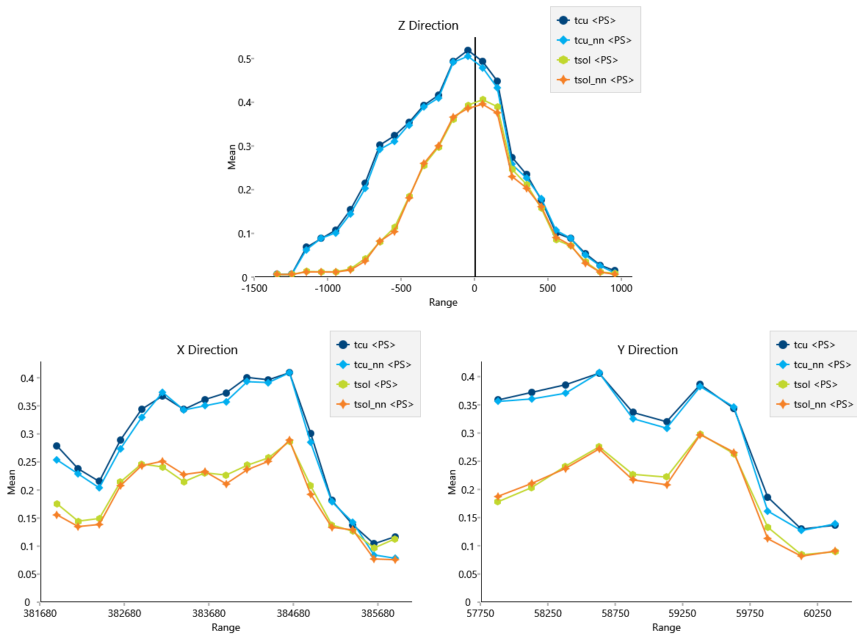


Figure 14-46: Swath Plots through Parks/Salyer Comparison with Associated Nearest Neighbor Grade Trends



There is good consistency in the grade trends defined by both the nearest neighbor values and the estimated block grades for both Cactus West, Cactus East, and Parks/Salyer. In areas of Cactus West where the nearest neighbor grades trend higher, the mean of the estimated grades is a little low, indicating there may be some conservatism in this estimate.

14.1.13 Block Model Regularization

Prior to running the pit optimizer, the Cactus sub-blocked block model was regularized to a new block model with regular block dimensions of 20 ft (6 m) by 20 ft (6 m) by 20 ft (6 m). Estimated grades were averaged to the regular blocks using volume weighted averaging of each of the smaller blocks falling within the larger block. In many cases, the estimated block size was the same as the regularized block size. This regularization process added contact dilution at the boundaries of the copper mineral domains. Table 14-13 outlines the block model parameters which match the Cactus sub-block model entirely except for the application of sub- blocking.

Table 14-13: Cactus Regular Block Model Definition Parameters

	X	Y	Z
Origin	385,900	60,800	-1,000
Bearing/Dip/Plunge	90	0	0
Offset Minimum	0	0	0
Extent Maximum	9,100	8,100	3,000
Parent Block Size	20	20	20
Total Blocks	**27,641,250**		

Smoothing Checks

Change of support smoothing checks were undertaken to measure the appropriateness of the estimated grade tonnage curve in generating a recoverable resource model appropriate to the potential mining method, associated selective mining unit size, and a range of potential economic COGs. Change of support smoothing checks allow the determination of the expected global grade tonnage curve based on a selective mining unit support size (20 ft (6 m) by 20 ft (6 m) by 20 ft (6 m) in this case) and make use of the underlying sample distribution and a model of grade continuity to remap the grade tonnage curve appropriately for that support. Whilst theoretical and global in nature, the change of support grade tonnage curve provides a reasonable measure of the level of smoothing that should be expected in the estimated resource model. The estimation of small blocks from wide spaced drilling is known to over-smooth resource model estimates when reporting against a cutoff. Smoothing checks provide a measure of the level of smoothing to allow tuning of the estimation plan to estimate a grade tonnage curve more appropriate for mine planning purposes. Smoothing checks were performed on the regularized block model to ensure block volume supports were consistent. Smoothing checks for Cactus West, Cactus East, and Parks/Salyer are presented in Figure 14-47 through Figure 14-49, respectively. The smoothing of Cactus East matches the change of support model well with grade, tons, and final metal within 5% for all cutoffs. The smoothing of Cactus West does not match the change of support metal so well, it is reasonable for tons, but much lower with respect to grade. The grade tonnage curve presented is depleted for the mined pit material. It may be that the higher-grade depleted pit material is affecting this comparison which makes the grade appear low. Efforts to increase the grade in the estimate did not provide a significant grade uplift. This may indicate some conservatism in the estimates for Cactus West. It may also be due to depleted higher grades—the cause of this effect, and the true grade tonnage curve, will be confirmed with further infill drilling. The smoothing of Cactus East and Parks/Salyer match the theoretical change of support models well. The smoothing check for Cactus West does not match the change of support model so well, particularly with respect to grade. The grade tonnage curve presented is depleted for the mined pit material. It may be that the higher-grade depleted pit material is affecting this comparison which makes the grade appear low. Efforts to increase the grade in the estimate did not provide a significant grade uplift. Figure 14-47 through Figure 14-49 show change of support smoothing check comparisons.

Figure 14-47: Change of Support Smoothing Check Comparison for Cactus West

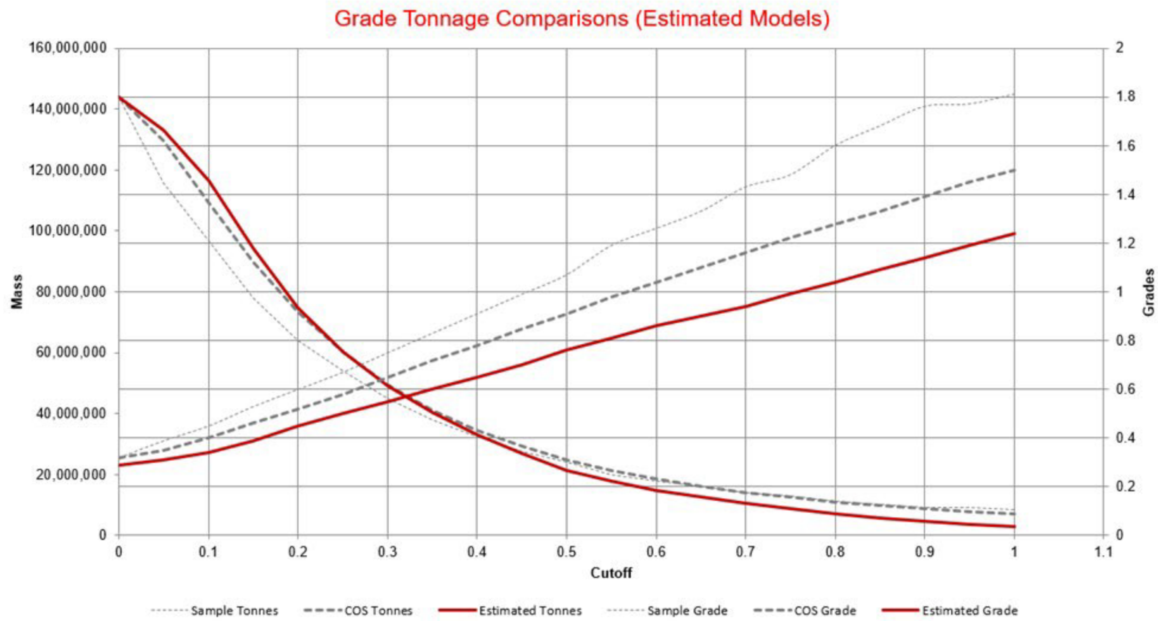


Figure 14-48: Change of Support Smoothing Check Comparison for Cactus East

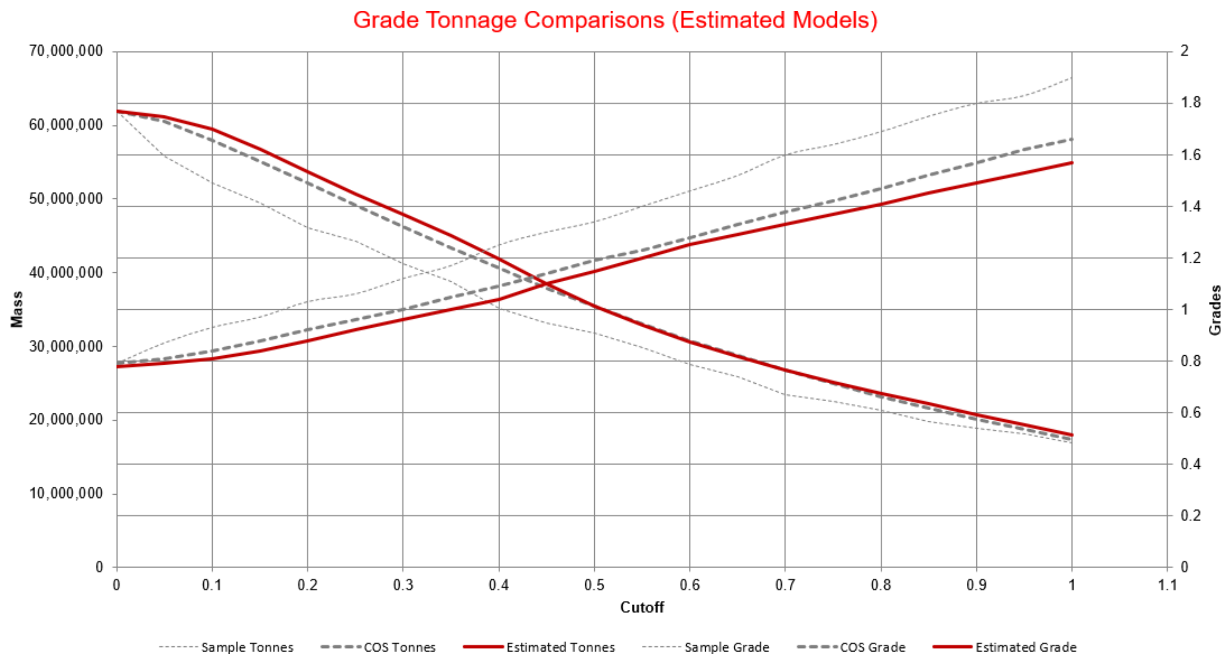
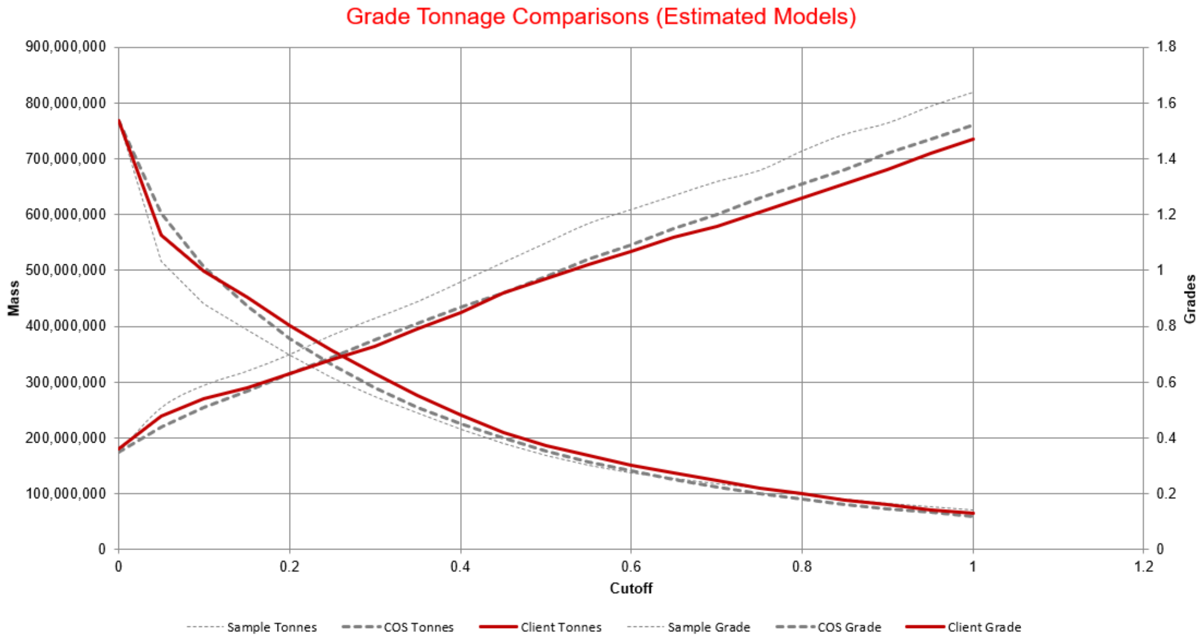


Figure 14-49: Change of Support Smoothing Check Comparison for Parks/Salyer



14.1.14 Resource Classification

Following are the key criteria affecting the classification of Resources for the Cactus and Parks/Salyer deposits.

- Understanding of the geological model and controls on mineralization, drill hole spacing, and the presence of downhole surveys for deeper mineralization such as Cactus East.
- The geological model and its controls on mineralization is generally well understood with the combination of copper mineral zones and sequential copper analyses to confirm relationships.
- Due to more local variation in geology and the current drill spacing, there is no material considered for Measured resources.

Drill spacing within the Cactus and Parks/Salyer deposits were defined with the following in mind.

- Wide exploration drill holes were infilled to 500 ft (152 m) spacing to support initial resource delineation. 500 ft (152 m) spacing was determined to be an appropriate spacing for an Inferred Resource classification. Drilling to 500 ft (152 m) spacing was undertaken both historically, and as part of the resource expansion drilling undertaken by Arizona Sonoran between 2020 and 2022.

- In the higher-grade core of the deposits, further infill drilling was undertaken historically to reduce the drill spacing to 250 ft (72 m) spacing to support resource definition drilling. A 250 ft (72 m) drill spacing is seen as an appropriate spacing to determine an Indicated Resource classification.

In the historic drilling, only a few of the holes within the core of the Cactus East mineralized zone contained downhole surveys. In the early drilling phases of the Project, vertical holes drilled were assumed to not deviate significantly at depth. Later downhole surveying proved this to be untrue, especially as holes got deeper. In areas of the Cactus East deposit where holes did not have downhole surveys, material has been downgraded from Indicated back to Inferred as the accuracy of the drill hole location, and therefore geological contacts and metal, may vary significantly from that modelled.

Basic definition of Inferred and Indicated classifications was defined by the estimation pass in which the blocks were estimated. Blocks estimated in Pass 1 could be assigned to Indicated and blocks estimated in Pass 2 would be assigned to Inferred. A subsequent test pass of the Indicated classification was undertaken using only holes that contained downhole surveys.

From this pass, an interpreted triangulation was created that finalized the classification of Indicated by downgrading areas based on the drill holes supporting it.

For Parks/Salyer, an interpreted triangulation was created to define the classification of Inferred encompassing the drillholes drilled to 500 ft spacing.

14.2 Cactus Stockpile Project

The inverse distance (ID1) method was used for the estimation of copper grades to the model. Copper estimates were performed on CuT, CuAS, and sequential CuCN. TSol results were calculated by adding the estimated CuAS to the CuCN. Validations made use of the nearest neighbor (polygonal) method for statistical and visual review.

14.2.1 Resource Drill Hole Database

The Cactus Stockpile Project drill hole database is managed in MX-Deposit software. CSV format files were exported from MX-Deposit using a resource specific template for the tables required for the resource database. CSV files were imported into a Vulcan ISIS database using a designated resource import LAVA script. The LAVA script and export template ensured the database was loaded consistently each time. The drill hole database used for the Cactus Stockpile Project mineral resource estimation was called cacStockpile Project_mx_resource_20210402.stp.isis.

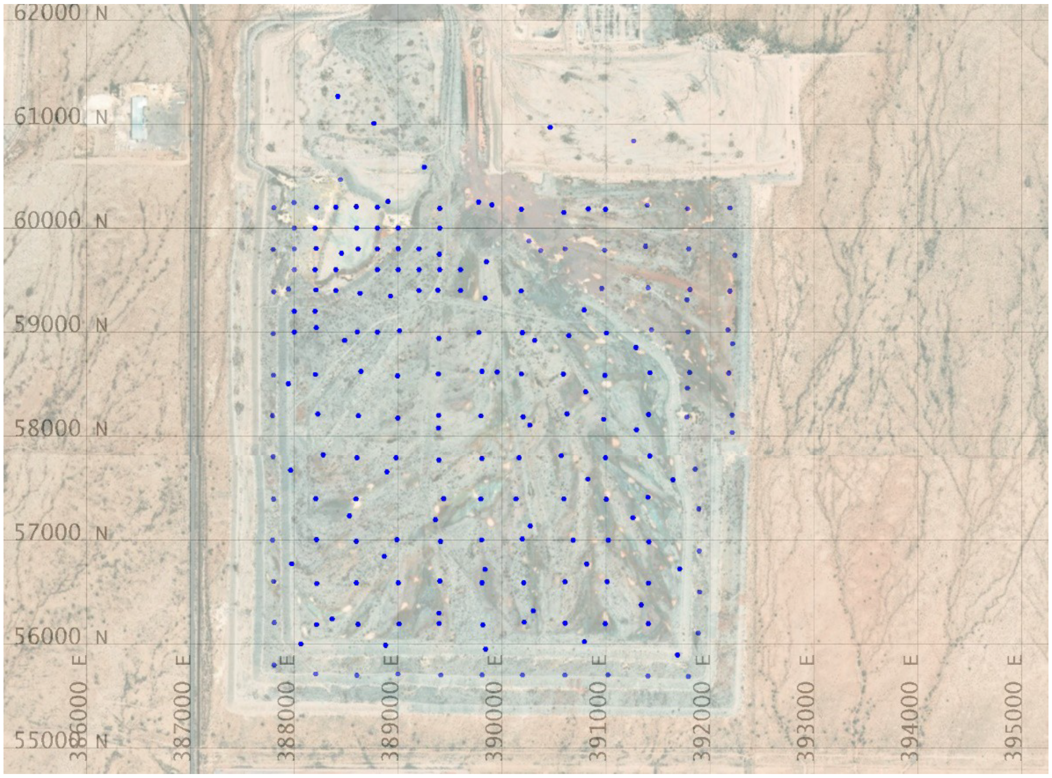
Lithology and mineralization logging was used to define zones for assay. Due to the nature of the dumping schedule and waste handling, logging is not considered as part of the mineral resource estimation process.

The Cactus Stockpile Project drill hole databases can be summarized by the following points.

- All holes within the database were drilled vertically.
- There are no downhole surveys measured as the deepest hole is only 125 ft (38.1 m) and all holes were drilled vertically.
- Drill spacing has been reduced to approximately 400 ft (121.9 m) across the Stockpile Project.
- CuT assays were sampled on 2.5 ft (0.8 m) lengths.
- CuAS and CuCN assays were conducted on 10 ft (3.0 m) composites for the first 40 ft (12.2 m) of the first 55 holes (using the same pulp material as the CuT assays). CuAS and CuCN assays were then conducted on the original 2.5 ft (0.8 m) sample pulps used in CuT assaying for depths greater than 40 ft (12.2 m) downhole of those holes and all parts of subsequent holes.
- The combined table was used in the database to contain the CuAS and CuCN assays and the matching CuT grades. TSol grades were calculated as a validation of the TSol copper grades for comparison against the CuT grade.
- In some zones within the holes there were significant intervals of non-mineralized material (such as conglomerate or alluvium). In these cases, often the intervals were not assayed, a grade of 0.002% CuT (half the detection limit) was applied to these intervals.
- Where an intercept was not assayed, and was not identified as a definitive waste sample, a default value of -99 was assigned.
- Lithology and color were logged for drill hole intercepts to the database. These serve as a guide to identifying non-mineralized zones (grey and tan) against potentially mineralized zones (orange and green). Red and brown logged colors can relate to both mineralized and non-mineralized material within the Stockpile Project.
- Copper mineralization, including copper oxides, were logged.

Figure 14-50 plots the drill hole locations within the Cactus Stockpile Project area. Light colored dumps to the north of the image represent alluvium dumps that have been sterilized by four drill holes as being unmineralized.

Figure 14-50: Drill Hole Collars on the Cactus Stockpile Project



14.2.2 Modelling

Lithology

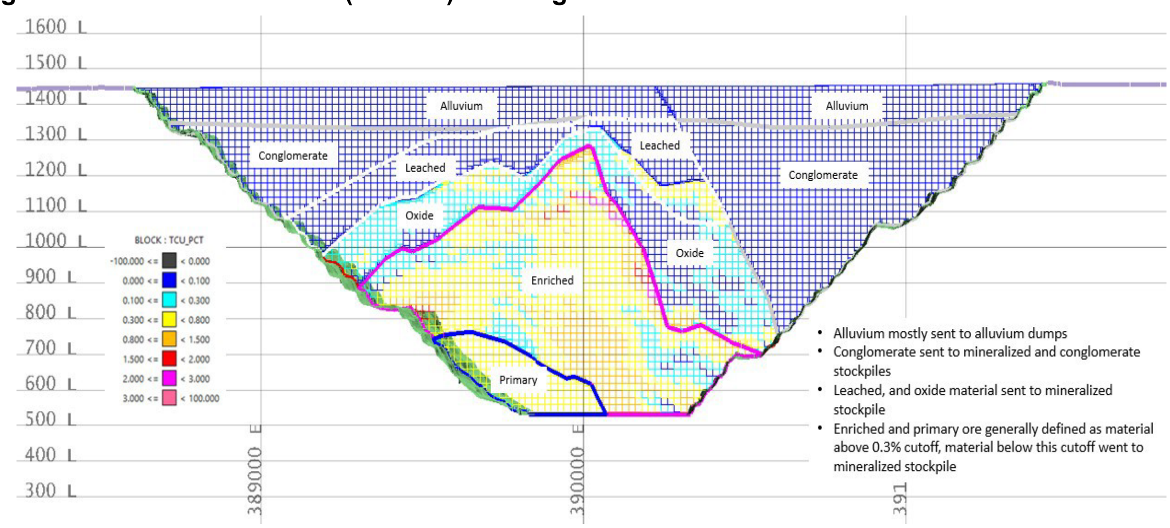
The nature of the mining operations at the historic Sacaton open pit from 1974 through 1984 has led to the dumping of material on the mineralized Stockpile Project where material types are broadly mixed within the Stockpile Project. There is indication, from the 750 ft (229 m) spaced drilling program, that some zones of non-mineralized lithologies such as conglomerate may be identifiable and separable as waste zones. Lithology within the Stockpile Project has no geological context, and as such is not used as any basis for the Stockpile Project mineral resource estimate, except to withhold assaying where broad zones of non-mineralized lithologies were present and assigned a grade of 0.002%. Table 14-14 and Figure 14-51 present the major lithological and porphyry copper alteration material types that represent mineralized and/or non-mineralized material within the Stockpile Project. The host units to mineralization are monzonite porphyry and granite.

Table 14-14: Lithology Codes

Lithological / Alteration Unit	Relationship to Mineralization	Destination*
Alluvium	Non-mineralized	Most material sent to alluvium dumps.
Conglomerate	Non-mineralized	All material sent to either the conglomerate dump or mineralized Stockpile Project.
Leached Zone (monzonite porphyry and granite)	Largely non-mineralized exceptin the case of selvages of mineralization	All material sent to the mineralized Stockpile Project.
Oxide Zone (monzonite porphyry and granite)	Mineralized – copper oxides dominant	All material sent to the mineralized Stockpile Project.
Enriched Zone (monzonite porphyry and granite)	Mineralized – chalcocite dominant	Material above 0.3% Cu sent as ore. Material below 0.3% Cu sent to the mineralized Stockpile Project.
Primary Zone (monzonite porphyry and granite)	Mineralized – chalcopyrite dominant	Material above 0.3% Cu sent as ore. Material below 0.3% Cu sent to the mineralized Stockpile Project.

* Refer to Figure 14-51 for map of destinations

Figure 14-51: Cross Section (64000N) Lithologies and Destinations of Material Mined from the Pit



Stockpile Project Modelling

The mineralized Stockpile Project represents a mixture of material types mined from the pit spatially over time. For this reason, the focus of the modelling was the following.

- Create an accurate topographical surface of the Stockpile Project surface and its base to define the Stockpile Project volume and extents.
- Characterize definitively non-mineralized zones from potentially mineralized zones.
- Define the historical lifts throughout the Stockpile Project that would vertically separate material mined in different time periods.

The topography was modeled from a site-specific Lidar survey undertaken 2018. Lidar data contains fine point resolution to accurately reflect the elevational changes of the topographic surface. The surface was filtered to remove and combine adjacent flat triangles. This improves efficiency of the triangulation for use in modelling with little to no loss in fidelity.

Aside from surface infrastructure such as the Stockpile Projects, dumps, and pits, the topography is generally gently dipping to the south with insignificant drainage channels. The discovery outcrop to the south of the historic Sacaton open pit represents the only natural land feature of any prominence in the local area of the historic mine.

There were two small volume areas on the mineralized Stockpile Project that had been reshaped due to rehabilitation activities since the Lidar was undertaken. These areas were surveyed in the field measuring toe, crest, and spot height observations and the data used to update the Lidar topography locally Figure 14-52 identifies these areas within the mineralized Stockpile Project that were adjusted.

Figure 14-52: Plan View of Mineralized Stockpile Project

Red points indicated the updated survey data acquired to adjust for rehabilitation works undertaken since the lidar survey. The northern surveyed area is locally termed the “bowl” and in the block model is defined as Lift 4.

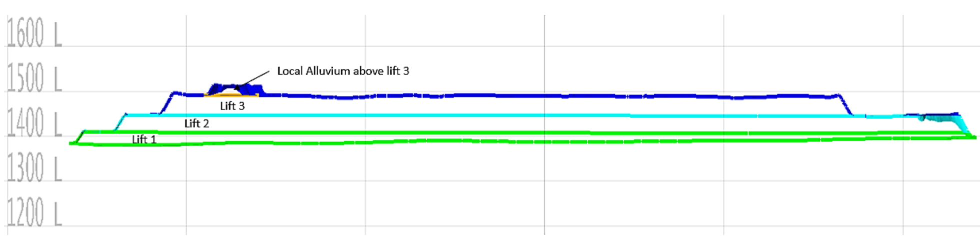
The three lifts of the mineralized Stockpile Project were defined spatially to enable separate treatment of composites and blocks for exploratory data analysis (EDA) and estimation. The lifts were separated by modelling surfaces for the original topography below the Stockpile Project (base of the Stockpile Project), the base of Lift 2, and the base of Lift 3. Lift 4 has been defined as the northern surveyed area in Figure 14-40. It is part of rehabilitation work material from a small historic primary sulfide dump that was recontoured into this zone.

Drilling has shown that the material in the bowl (Lift 4) has oxidized and represents a local high-grade zone of the Stockpile Project.

The base of the Stockpile Project was modeled by clipping out the Stockpile Project extents from the Lidar topographic surface. In most of the sonic drilling, the soil underlying the Stockpile Project was penetrated and the depth of this logged. The base of the Stockpile Project was identified in the holes and used in conjunction with the clipped lidar topography surface to generate a surface representing the original topography pre-Stockpile Project. The current topography was then clipped with this surface to create a new solid representing the full mineralized Stockpile Project volume.

The lifts were separated by defining the planes representing the base of Lift 2 and the base of Lift 3. These surfaces were defined by digitizing points on the outer berms of both levels and then modelling a planar surface using these points. The Stockpile Project solid was then clipped against these surfaces to create three separate solids representing each of the Stockpile Project lifts (see Figure 14-41). The two upper lifts are consistently 40 ft (12 m) in height. Lift 1 is considerably lower in height than the upper lifts due to the gentle dip of the topography from north to south. The height of Lift 1 in the north is approximately 5 ft (1.5 m) increasing to the full 40 ft (12 m) in the south. The vertical exaggeration in Figure 14-53 is set to 250 to aid visualization.

Figure 14-53: Section Through WRD Showing Lifts



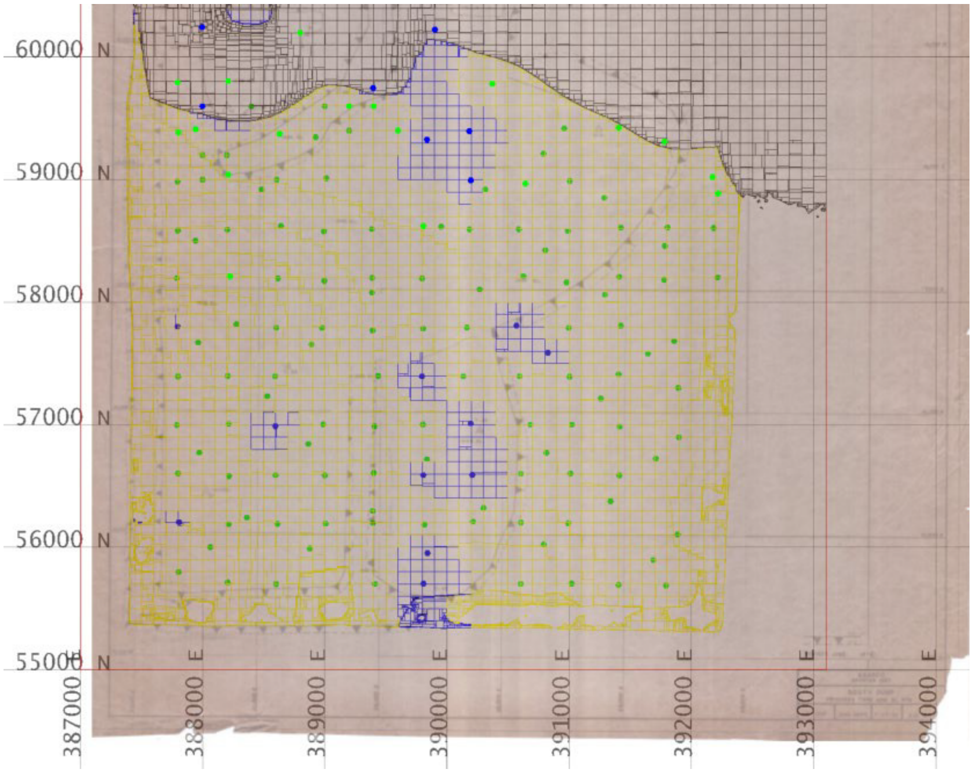
Waste Indicator

To reduce the potential of grade estimation into unmineralized zones a waste indicator model was implemented to identify definitive waste zones within the Stockpile Project. Logged zones of significant non-mineralized material were not sampled and a grade of 0.002% was applied (half the detection limit). Due to the lack of geological controls to the Stockpile Project material and wide drill spacing, composite grades provide a general view to the grade of the Stockpile Project and broader areas within it. However, it is expected that drilling twin holes will show a significant nugget effect laterally and individual drill hole grades are not a good predictor of the grades of the local volume it supports. For this reason, the estimate is highly smoothed with the goal to estimate the global grade tonnage curve and identify broad zones that are mineralized.

With such high smoothing, there is potential to smear metal into areas that are definitively waste. In an extreme case this can create material that may be marginally above cutoff and report to ore. Therefore, an indicator estimation method was required to define definitive waste zones that may have continuity and ensure these blocks were not estimated. This would be most effective in the lower lifts where significant overburden was mined from the pit. This procedure did not limit the grade estimation itself from defining waste areas where low grades prevailed in the composites.

An indicator estimation method was used to assign the mineralized extents to the block model so that these could be estimated separately from definitive waste areas. In Figure 14-54, the Stockpile Project blocks are shown color coded according to its indicator estimation. Blocks defined by the estimation as potentially mineralized are colored yellow, definitive waste areas are colored blue. The estimation is based on composite grades, which are displayed as dots for reference. Composites are colored yellow if their CuAS value is above 0.01%, blue if their value is below. CuAS grades were used as this indicates the readily leachable material which is most likely to support mineralization that could be economic for heap leaching.

Figure 14-54: Plan View (1405L) Showing the Indicator Defining Zones of Consistent Waste Intercepts



The indicator method was assigned to the block model as follows.

- For CuAS, a mineralized composite for Stockpile Project purposes was defined as a sample having a grade greater than 0.01% CuAS.
- Each composite was assigned a 1 if its grade was above the specified threshold, or a 0 if its grade was below.
- These 1 and 0 values were estimated into the Stockpile Project blocks using the lifts as separate estimation domains for composite selection. This results in an estimated value between 0 and 1 being assigned to each block – this value represents the probability that the block is mineralized above 0.01% CuAS.
- If a block had a probability of greater than 50% (or 0.5) then it was determined to be potentially a mineralized block. If the value was less than 0.5, the block was assigned as waste material.
- Blocks defined as part of the mineralized material were estimated for grade separately from blocks defined as waste. The mineralized estimate may use any sample within the lift of the block, the waste blocks were not estimated and were automatically assigned grades of 0.002% for CuAS, CuCN, and CuT. Selection of all samples to estimate the potentially mineralized blocks adds a level of conservatism to the estimate which takes into account that wide spaced drilling does not define these material contacts well.

The indicator ensured high grades from mineralized areas could not be used to estimate adjacent areas determined as waste.

The use of an indicator complements both the grade estimation and the capping thresholds used in the grade estimation since high grades are only used to estimate potentially mineralized areas of the Stockpile Project. CuAS, CuCN, and CuT used the same indicator to determine which blocks could be estimated as potentially mineralized.

14.2.3 Estimation Domains

Final estimation domains were based on the combination of the dump lift and the waste indicator discussed in Section 14.2.2. No grades were estimated into zones defined as definitive waste.

14.2.4 Specific Gravity

Due to the unconsolidated nature of the Stockpile Project material, measuring bulk density can be problematic. At this stage of the Project, no direct bulk density measurements have been undertaken for the Stockpile Project material. Stantec has significant experience in stockpile analysis and the use of relevant bulk densities appropriate to these material types.

A density value of 0.0649 t/ft3 was assigned to the mineralized stockpile based on recommendations by Stantec. A density value of 0.0468 t/ft3 was assigned the alluvium defined stockpiles.

The bulk densities recommended are supported by the measured densities from the Cactus West units mined in the historical pit (Table 14-15) and expected relative proportions of material types sent to the mineralized Stockpile Project.

Table 14-15: Lithologic Unit Densities

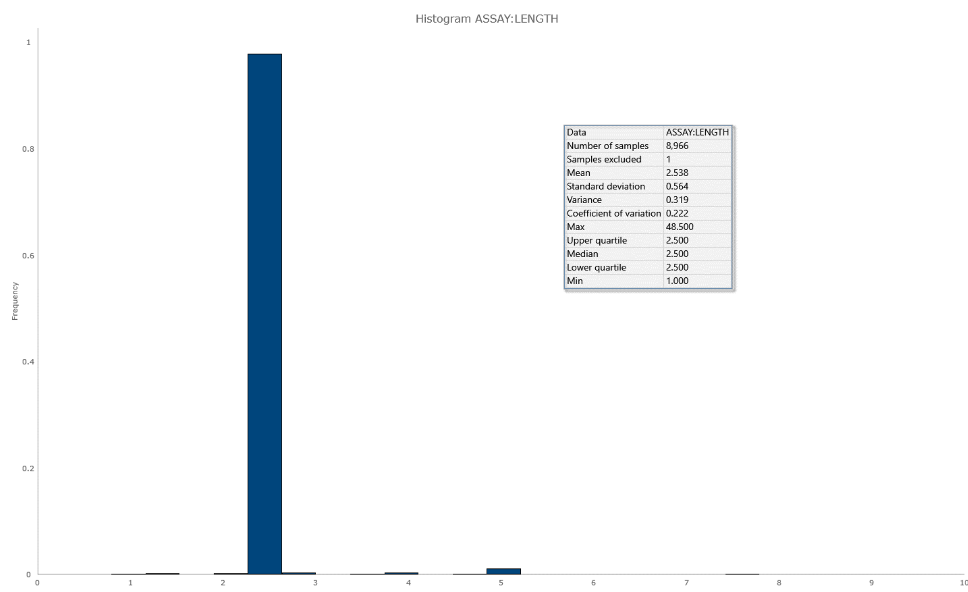
Material	Density (t/ft^3)	Source
Alluvium	0.0468	Typical for dry, sandy soils.
Conglomerate	0.0812	Typical for sedimentary rocks.
Leached	0.0793	Mean of 75 samples taken from historical Sacaton west and east deposits.
Oxide	0.0780	Mean of 98 samples taken from historical Sacaton west deposit.
Enriched	0.0808	Mean of 198 samples taken from historical Sacaton west deposit.
Primary	0.0799	Mean of 143 samples taken from historical Sacaton west deposit.

Based on the densities in Table 14-11 and the relative volumes of the material types sent to the mineralized Stockpile Project a density of the in situ rock of 0.0759 t/ft3 can be assumed. When in situ rock is broken and placed on a Stockpile Project, its volume increases by a factor known as the swell factor (30% is a common industry standard applied). As the Stockpile Project sits, compaction occurs over time and the Stockpile Project volume will decrease by a factor known as the compaction factor (recommended between 5%-15% by Porter and Bleiwas, 2003). Applying a 30% swell factor to the in situ rock density and a 10% compaction factor gives a final bulk density of the Stockpile Project of 0.0649 t/ft3.

14.2.5 Compositing

The drillhole intercepts were composited to 10 ft (3.0 m) composite lengths for CuAS, CuCN, and CuT. The stockpile was built in three vertical lifts of approximately 40 ft (12.1 m) height (Figure 14-55). Due to this, the separate lifts vertically represent different periods of time in the mining sequence. Composites were split at the modelled lift contacts and the lifts were flagged to the composites. Where a composite was generated at less than half the composite length (5 ft [1.5 m]), it was combined into the previous 10 ft (3.0 m) composite to ensure short length composites were not generated. Sample grades with values of -99 were ignored during compositing.

Figure 14-55: Histogram of Drill Hole Sample Lengths



The stockpile designation was flagged to the composites as were the bench levels that could define future 20 ft (6 m) working mining benches for the Stockpile Project. TSol was back calculated to the composites as the addition of CuAS and CuCN.

14.2.6 Exploratory Data Analysis

CuT grades represent the total copper present within the drilled intercept. Copper mineralization in the form of chalcopyrite, mostly present in the primary zone, typically leaches poorly using acid-based heap leaching processes. To measure the expected leachable copper, sequential copper analysis was undertaken by first leaching the sample using acid to attain the CuAS, and then leaching the residue with cyanide to attain the CuCN. CuAS assays are expected to account for the copper content of the copper oxides and up to half of the chalcocite. CuAS assays also account for the readily leachable component of the copper within the sample. CuCN assays will account for the copper content of any covellite and the remainder of the chalcocite. This copper is still leachable by acid solutions or bio-solutions, but recovery will be slower and less effective (lower recoveries over a longer period, up to two years). TSol is calculated as the addition of CuAS and CuCN as a measure of the total leachable copper grade for the composite.

Univariate statistics were calculated for the mineralized material of the stockpile for CuAS, CuCN, TSol, and CuT and results were reported for the entire stockpile and by individual lifts. The summary statistics are shown in Table 14-16. It can be seen from this table that mean grades decrease down through the stockpile lifts. This is consistent with the scheduled waste dumping from the historical open pit where considerably more mineralized waste is expected to have been mined later in the mine life which would position this material in the upper levels of the mineralized stockpile.

Table 14-16: Lift Drill Hole 10 ft Composite Statistics for CuT, CuAS, CuCN, and TSol

Variable name	Count	Mean	Standard deviation	Variance	CV	Max	Upper quartile	Median	Lower quartile	Min
TCU_FIN_PCT [MATERIAL=stockpile]	1580	0.140	0.131	0.017	0.932	1.609	0.190	0.108	0.056	0.002
TCU_FIN_PCT [LIFT=4]	18	0.334	0.169	0.029	0.507	0.668	0.432	0.348	0.224	0.003
TCU_FIN_PCT [LIFT=3]	492	0.174	0.139	0.019	0.800	1.609	0.240	0.151	0.080	0.002
TCU_FIN_PCT [LIFT=2]	781	0.123	0.114	0.013	0.929	1.235	0.160	0.096	0.052	0.002
TCU_FIN_PCT [LIFT=1]	289	0.118	0.133	0.018	1.130	1.495	0.151	0.091	0.043	0.002
CUAS_FIN_PCT [MATERIAL=stockpile]	1580	0.097	0.099	0.010	1.024	1.344	0.130	0.072	0.030	0.002
CUAS_FIN_PCT [LIFT=4]	18	0.245	0.139	0.019	0.567	0.574	0.306	0.238	0.168	0.003
CUAS_FIN_PCT [LIFT=3]	492	0.125	0.106	0.011	0.846	0.993	0.172	0.104	0.051	0.002
CUAS_FIN_PCT [LIFT=2]	781	0.083	0.082	0.007	0.997	0.780	0.108	0.061	0.027	0.002
CUAS_FIN_PCT [LIFT=1]	289	0.078	0.108	0.012	1.374	1.344	0.101	0.053	0.023	0.002
CUCN_SEQ_FIN_PCT [MATERIAL=stockpile]	1580	0.023	0.044	0.002	1.883	0.692	0.024	0.012	0.006	0.002
CUCN_SEQ_FIN_PCT [LIFT=4]	18	0.059	0.036	0.001	0.614	0.122	0.086	0.061	0.029	0.003
CUCN_SEQ_FIN_PCT [LIFT=3]	492	0.026	0.039	0.002	1.524	0.587	0.031	0.017	0.008	0.002
CUCN_SEQ_FIN_PCT [LIFT=2]	781	0.021	0.046	0.002	2.172	0.692	0.021	0.011	0.006	0.002
CUCN_SEQ_FIN_PCT [LIFT=1]	289	0.021	0.042	0.002	1.966	0.371	0.019	0.009	0.005	0.002
TSOLCU_FIN_PCT [MATERIAL=stockpile]	1580	0.120	0.123	0.015	1.022	1.580	0.163	0.087	0.041	0.002
TSOLCU_FIN_PCT [LIFT=4]	18	0.305	0.163	0.027	0.535	0.646	0.398	0.305	0.208	0.006
TSOLCU_FIN_PCT [LIFT=3]	492	0.151	0.130	0.017	0.861	1.580	0.202	0.133	0.064	0.002
TSOLCU_FIN_PCT [LIFT=2]	781	0.104	0.108	0.012	1.041	1.132	0.136	0.075	0.036	0.002
TSOLCU_FIN_PCT [LIFT=1]	289	0.100	0.124	0.015	1.247	1.372	0.125	0.067	0.030	0.002

Figure 14-56 is a scatter plot produced to compare the CuAS grades to the TSol grades on a composite basis. This indicates the presence of the readily leachable copper within the TSol copper population. The closer a composite value plots to the 45° grey line, the higher the proportion of readily leachable copper present within that composite. The bulk of the samples plot close to the grey line indicating that much of the soluble copper should leach well.

Figure 14-56: Scatter Plot of CuAS Against TSol Showing the Prevalence of Rapidly Leachable Copper within the Soluble Copper Composites

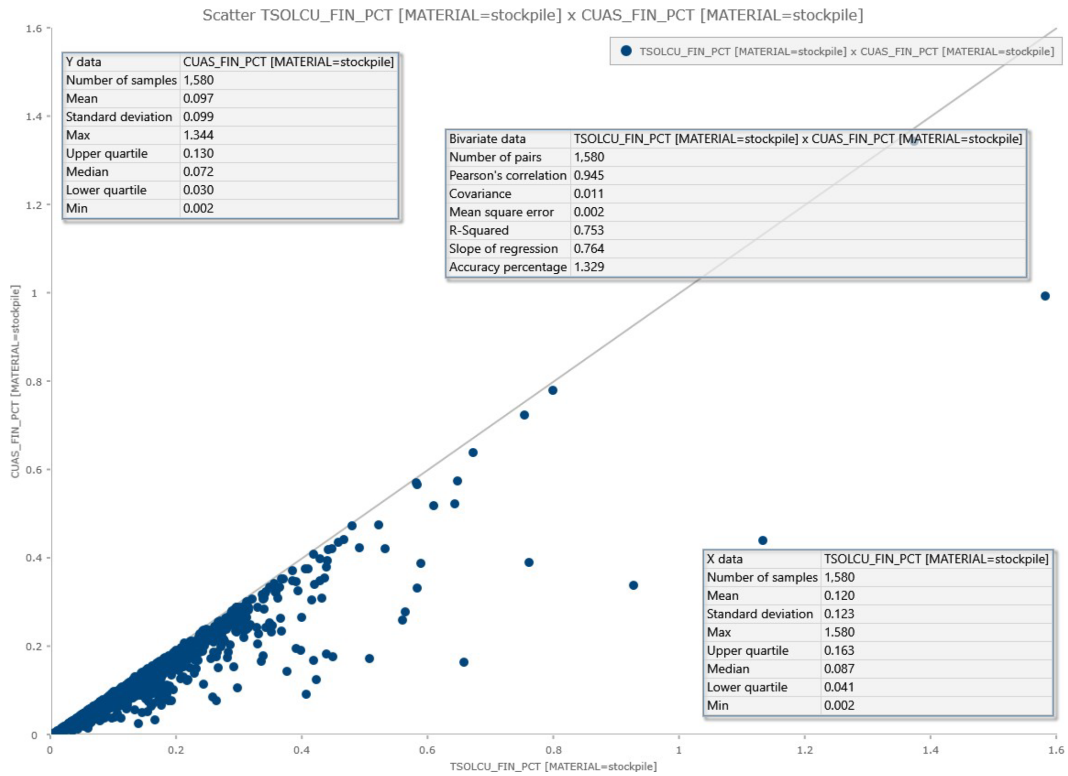


Figure 14-57 is a scatter plot produced to compare the TSol grades to the CuT grades on a composite basis. This indicates the presence of the leachable copper within the CuT population. The closer a composite plot is to the 45° grey line, the higher the proportion of leachable copper present within that composite. The bulk of the samples plot close to the grey line indicating that much of the CuT is in a mineralogy that is leachable. Copper that is not leachable in the analysis undertaken is expected to be chalcopyrite primary mineralization and for the purposes of metallurgy will not be recoverable.

Figure 14-57: Scatter Plot of TSol Against CuT Showing the Prevalence of Leachable Copper within the CuT Composites

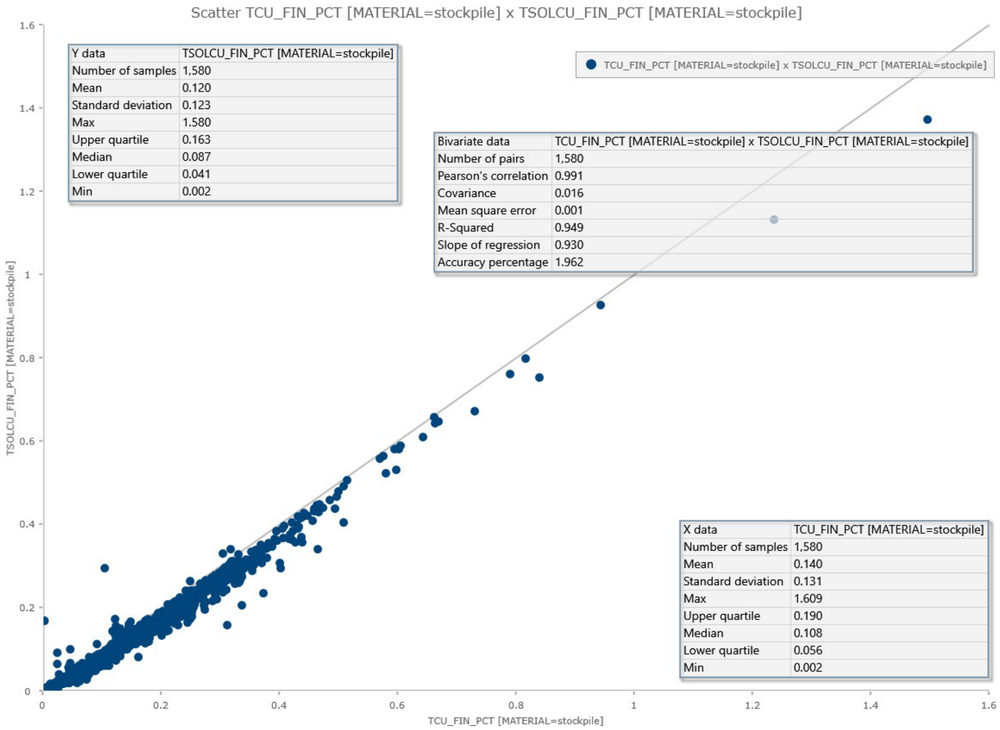


Figure 14-58 is a scatter plot produced to compare the CuCN grades to the CuAS grades on a composite basis. This indicates if there is a relationship between assay distributions that should be honored in the grade estimation stage. The closer the composites plot to a straight line, the stronger the evidence that there is for a relationship between the grades that should be honored in the block estimation. The plot indicates that there is little relationship at the composite level between these two grade datasets and that therefore they can be treated independently.

Figure 14-58: Scatter Plot of CuCN Against CuAS Showing Little Correlation Between Readily Leachable Copper Grades and Slow Leaching Copper Grades

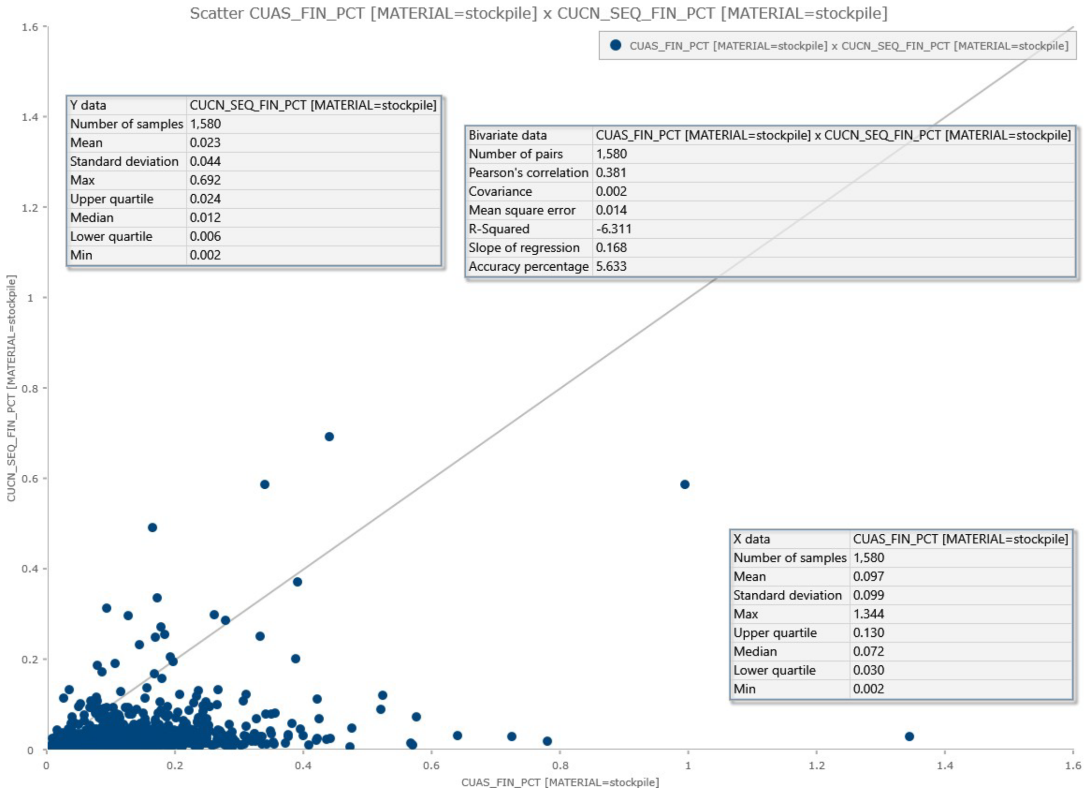
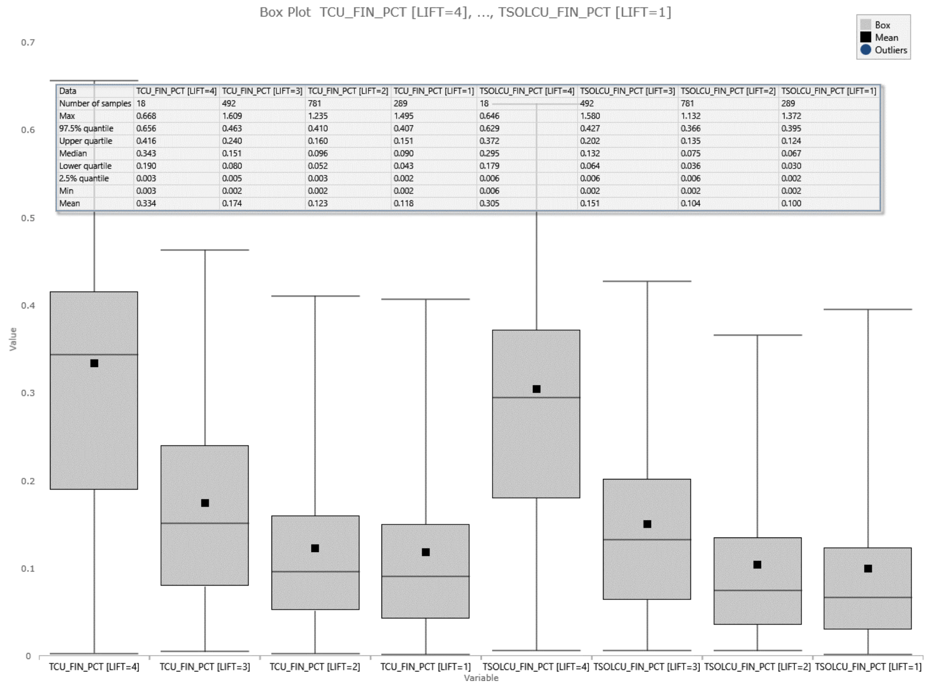


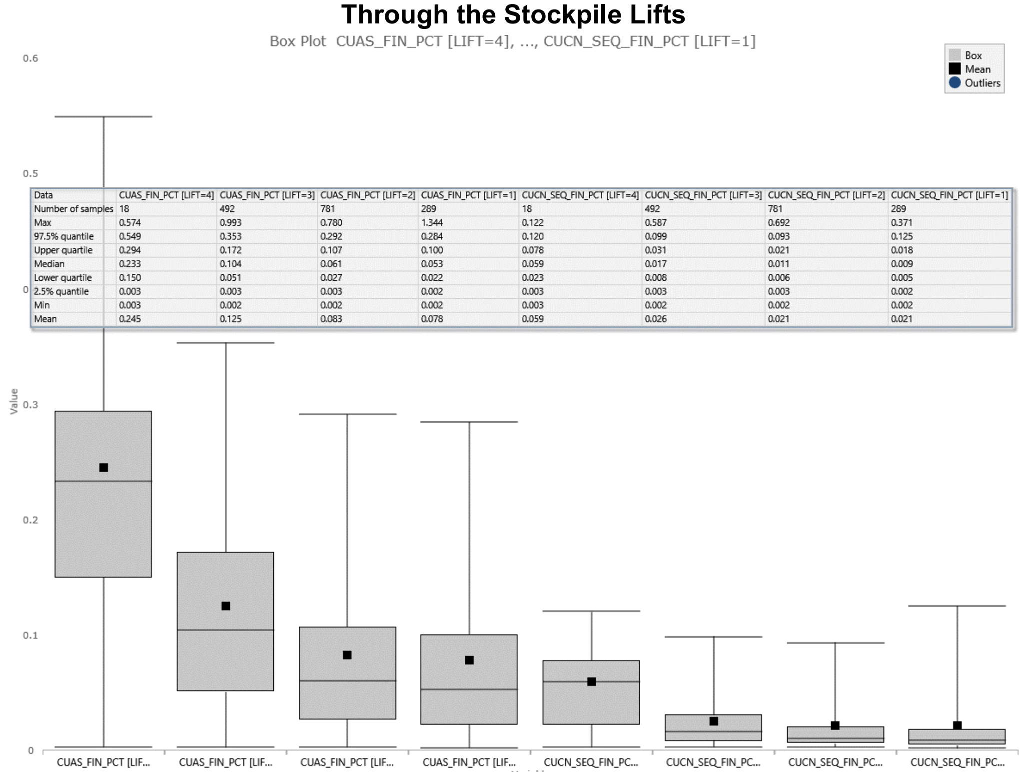
Figure 14-59 and Figure 14-60 show box plots created for CuT, TSol, CuAS, and CuCN grouped by lift within the stockpile. The box plots show the clear relationship of decreasing grade moving down through the lifts from Lift 3 to Lift 1. This supports the waste dumping schedule history and indicates that it is better to estimate the separate lifts independently using only assays within that same lift. It also highlights the significant proportion of copper that is present in a readily leachable form signified by the CuAS grade distribution versus the copper that will leach more slowly signified by the CuCN grade distribution.

Figure 14-59: Box Plots for CuT and TSol Grouped by Lift Showing the Grade Reduction Down Through the Stockpile Lifts



Data	TCU_FIN_PCT [LIFT=4]	TCU_FIN_PCT [LIFT=3]	TCU_FIN_PCT [LIFT=2]	TCU_FIN_PCT [LIFT=1]	TSOLCU_FIN_PCT [LIFT=4]	TSOLCU_FIN_PCT [LIFT=3]	TSOLCU_FIN_PCT [LIFT=2]	TSOLCU_FIN_PCT [LIFT=1]
Number of samples	18	492	781	289	18	492	781	289
Max	0.668	1.609	1.235	1.495	0.646	1.580	1.132	1.372
97.5% quantile	0.656	0.463	0.410	0.407	0.629	0.427	0.366	0.395
Upper quartile	0.416	0.240	0.160	0.151	0.372	0.202	0.135	0.124
Median	0.343	0.151	0.096	0.090	0.295	0.132	0.075	0.067
Lower quartile	0.190	0.080	0.052	0.043	0.179	0.064	0.036	0.030
2.5% quantile	0.003	0.005	0.003	0.002	0.006	0.006	0.006	0.002
Min	0.003	0.002	0.002	0.002	0.006	0.002	0.002	0.002
Mean	0.334	0.174	0.123	0.118	0.305	0.151	0.104	0.100

Figure 14-60: Box Plots for CuAS and CuCN Grouped by Lift Showing the Grade Reduction Down Through the Stockpile Lifts



Data	CUAS_FIN_PCT [LIFT=4]	CUAS_FIN_PCT [LIFT=3]	CUAS_FIN_PCT [LIFT=2]	CUAS_FIN_PCT [LIFT=1]	CUCN_SEQ_FIN_PCT [LIFT=4]	CUCN_SEQ_FIN_PCT [LIFT=3]	CUCN_SEQ_FIN_PCT [LIFT=2]	CUCN_SEQ_FIN_PCT [LIFT=1]
Number of samples	18	492	781	289	18	492	781	289
Max	0.574	0.993	0.780	1.344	0.122	0.587	0.692	0.371
97.5% quantile	0.549	0.353	0.292	0.284	0.120	0.099	0.093	0.125
Upper quartile	0.294	0.172	0.107	0.100	0.078	0.031	0.021	0.018
Median	0.233	0.104	0.061	0.053	0.059	0.017	0.011	0.009
Lower quartile	0.150	0.051	0.027	0.022	0.023	0.008	0.006	0.005
2.5% quantile	0.003	0.003	0.003	0.002	0.003	0.003	0.003	0.002
Min	0.003	0.002	0.002	0.002	0.003	0.002	0.002	0.002
Mean	0.245	0.125	0.083	0.078	0.059	0.026	0.021	0.021

14.2.7 Capping

Grade capping for CuAS, CuCN, and CuT was applied to the composites at the estimation stage using a top cut method. Composite grades above this threshold were reset to the threshold level during the estimation process.

Capping levels were determined using the industry standard log normal probability plot method. Analysis of the upper end of the log probability distributions identified the threshold at which point the distribution loses consistency. This indicates that grades above this level are inconsistent with the population characteristics and therefore represent metal at risk in the estimation process.

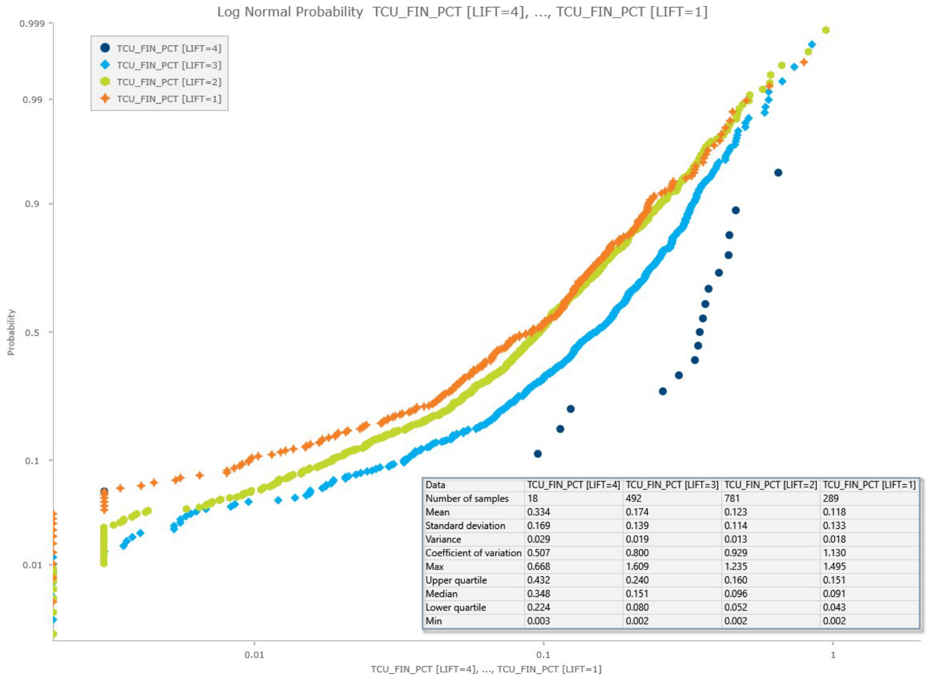
Log normal probability plots were generated per lift to define applicable capping levels within each lift. Due to the grade distribution differences between lifts a single threshold defined for the global population was not appropriate.

Table 14-17 shows the capping levels determined for CuAS, CuCN, and CuT per lift. For CuAS and CuT, capping levels decrease down through the lifts as expected from the underlying data distributions (see Figure 14-61 through Figure 14-63). Lift 4 represents only a very limited dataset with its own characteristics.

Table 14-17: Capping Threshold Values Applied per Lift to the Estimation of CuT, CuAS, and CuCN

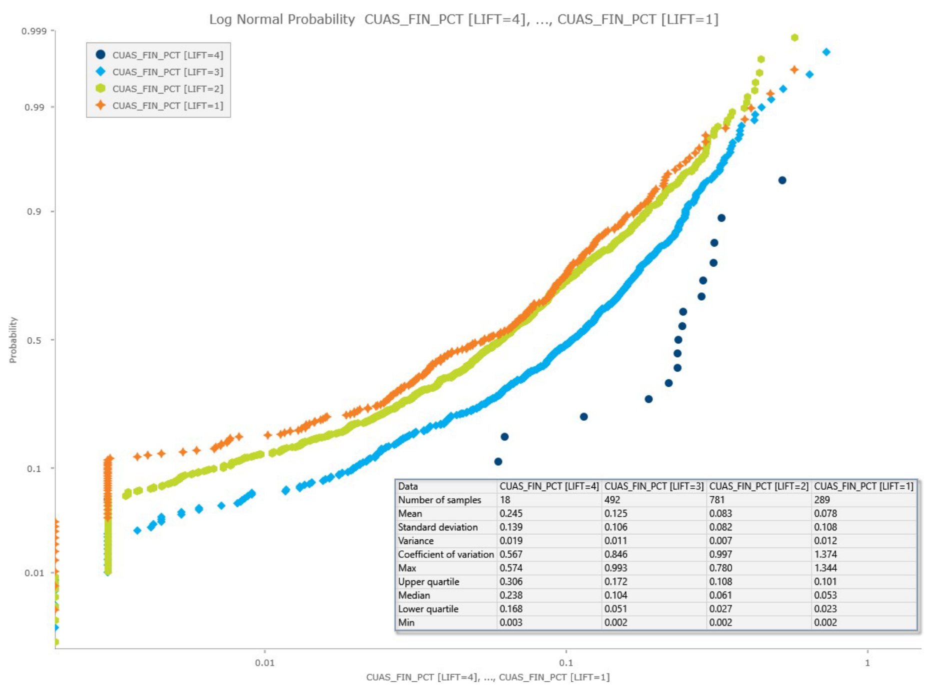
Lift	CuT	CuAS	CuCN
Lift 4	0.46	0.33	0.11
Lift 3	0.51	0.38	0.19
Lift 2	0.48	0.35	0.20
Lift 1	0.45	0.29	0.21

Figure 14-61: Log Normal Probability Plot of Total Copper Assays Grouped by Lift



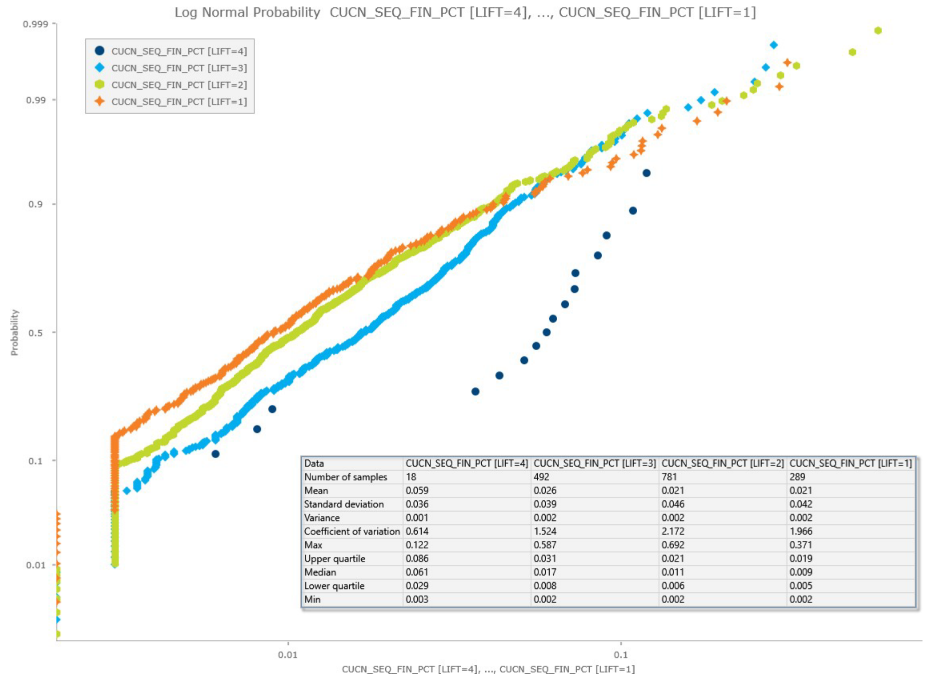
Data	TCU_FIN_PCT [LIFT=4]	TCU_FIN_PCT [LIFT=3]	TCU_FIN_PCT [LIFT=2]	TCU_FIN_PCT [LIFT=1]
Number of samples	18	492	781	289
Mean	0.334	0.174	0.123	0.118
Standard deviation	0.169	0.139	0.114	0.133
Variance	0.029	0.019	0.013	0.018
Coefficient of variation	0.507	0.800	0.929	1.130
Max	0.668	1.609	1.235	1.495
Upper quartile	0.432	0.240	0.160	0.151
Median	0.348	0.151	0.096	0.091
Lower quartile	0.224	0.080	0.052	0.043
Min	0.003	0.002	0.002	0.002

Figure 14-62: Log Normal Probability Plot of CuAS Assays Grouped by Lift



Data	CUAS_FIN_PCT [LIFT=4]	CUAS_FIN_PCT [LIFT=3]	CUAS_FIN_PCT [LIFT=2]	CUAS_FIN_PCT [LIFT=1]
Number of samples	18	492	781	289
Mean	0.245	0.125	0.083	0.078
Standard deviation	0.139	0.106	0.082	0.108
Variance	0.019	0.011	0.007	0.012
Coefficient of variation	0.567	0.846	0.997	1.374
Max	0.574	0.993	0.780	1.344
Upper quartile	0.306	0.172	0.108	0.101
Median	0.238	0.104	0.061	0.053
Lower quartile	0.168	0.051	0.027	0.023
Min	0.003	0.002	0.002	0.002

Figure 14-63: Log Normal Probability Plot of CuCN Assays Grouped by Lift



Data	CUCN_SEQ_FIN_PCT [LIFT=4]	CUCN_SEQ_FIN_PCT [LIFT=3]	CUCN_SEQ_FIN_PCT [LIFT=2]	CUCN_SEQ_FIN_PCT [LIFT=1]
Number of samples	18	492	781	289
Mean	0.059	0.026	0.021	0.021
Standard deviation	0.036	0.039	0.046	0.042
Variance	0.001	0.002	0.002	0.002
Coefficient of variation	0.614	1.524	2.172	1.966
Max	0.122	0.587	0.692	0.371
Upper quartile	0.086	0.031	0.021	0.019
Median	0.061	0.017	0.011	0.009
Lower quartile	0.029	0.008	0.006	0.005
Min	0.003	0.002	0.002	0.002

14.2.8 Variography

Variogram modelling is inappropriate for use with material that is not in situ as there is no geological context or expected continuity due to the material being dumped to the pile inconsistently.

14.2.9 Block Model

The mineralized dump represents an area approximately 5,100 ft (1,554 m) north-south by 5,000 ft (1,524 m) east-west (see Figure 14-38). The height of the material in the stockpile is approximately 65 ft (19.8 m) in the far north, increasing to 120 ft (36.6 m) on the south end. The Stockpile Project block model was constructed using a 100 ft (30.5 m) × 100 ft (30.5 m) × 20 ft (6.1 m) parent block size (XYZ), with sub-blocking to 2.50 ft (0.8 m) × 2.50 ft (0.8 m) ×0.25 ft (0.08 m) to accurately reflect the mineralized stockpile volume. The 20 ft (6 m) block height was incorporated to reflect the planned bench heights that would be utilized to potentially mine the stockpile (two benches per lift). Table 14-18 displays the key block definition parameters.

Table 14-18: Block Model Definition Parameters

	X	Y	Z
Origin	387,000.0	55,000.0	1,345.00
Bearing / Dip / Plunge	90.0	0.0	0.00
Offset Minimum	0.0	0.0	0.00
Extent Maximum	6,100.0	7,500.0	200.00
Parent Block Size	100.0	100.0	20.00
Sub-block Block Size	2.5	2.5	0.25
Total Blocks	**2,002,613**		

The mineralized stockpile material was assigned a material type of stockpile. There is one small volume alluvium dump located on top of Lift 3. These blocks were set to a material type of alluvium. Stacked material immediately to the north of the mineralized stockpile was also incorporated into the block model extents and assigned a material type of alluvium. The blocks below the original topographic surface and below the stockpile at depth were assigned a material type of soil. Block model volumes were compared against the input triangulation volumes to ensure the block model sub-blocking schema satisfactorily reflected the volumes of each lift and the total mineralized stockpile. Results are reported in Table 14-19.

Table 14-19: Block Model Volumes Compared to Triangulation Volumes

Material	Lift	Block Volume	Triangulation Volume	Difference
alluvium	4w	4,794,359	4,794,581	0.0%
alluvium	5w	426,019,308	425,837,464	0.0%
stockpile	1	471,215,184	471,245,855	0.0%
stockpile	2	870,090,722	870,181,531	0.0%
stockpile	3	609,880,878	610,583,418	-0.1%
stockpile	4	8,470,794	8,471,564	0.0%
Total		**2,390,471,245**	**2,391,114,414**	**0.0%**

The lifts were designated to the block model with lift numbers of 1, 2, and 3. An area on the north end called the bowl must be backfilled with historical sulfide material and has been designated as Lift 4. Alluvium dumps were assigned similar lift numbers but with a suffix to delimit them from the mineralized lifts easily (i.e., 4w, 5w).

Twenty-ft (6.1-m) benches were assigned into the blocks based on the bench within which the block sits. These were aligned with the lift elevations.

14.2.10 Estimation Plan

For each lift in the mineralized stockpile, CuT, CuAS, and CuCN values were estimated using the Inverse Distance to the Power of 1 (ID1) method. Due to the characteristics of the dumping schedule for the stockpile and the wide spaced drilling, a HL of smoothing was implemented as individual composites may not represent the volumes adjacent to them that they are estimating.

Significant parameters used in the copper estimates included the following.

- Stockpile lifts were treated as hard domains, therefore only composites within the same lift were used to estimate a block in that lift.
- The estimation was undertaken using two passes. The first pass focused estimating the 400 ft (122 m) drill spacing which covers the bulk of the Stockpile Project. The second pass filled out the estimates throughout the mineralized part of the Stockpile Project.
- A minimum number of six composites and a maximum number of 12 composites were used to estimate a block.
- Cell de-clustering was run on the composite database based on a 400 ft (121.9 m) × 400 ft (121.9 m) grid. De-clustering weights were used at the estimation stage to de- cluster the samples being used in the estimate.
- Only blocks with a mineralized indicator probability of 0.5 could be estimated for grade (based on a 0.01% CuAS indicator). All other blocks were assigned a default grade of 0.002%.
- Un-estimated blocks were automatically assigned a grade of 0.002%.
- To ensure multiple holes from numerous directions around a block were used in the estimate, the maximum number of samples that could be used from a single hole was set to 3. In conjunction with the minimum number of samples, this ensured at least two holes were required to estimate a block.
- The search ellipse was set to 500 ft (152.4 m) × 500 ft (152.4 m) × 30 ft (9.1 m) for the first pass. The search ellipse was set to 1,000 ft (304.8 m) × 1,000 ft (304.8 m) × 30 ft (9.1 m) for the second pass.
- Grades were capped using a top cut method. Cap levels were set on a per lift basis.
- A nearest neighbor value was assigned to the blocks during the estimation process for use in validations of the estimate.

14.2.11 Mining Depletion

There was no depletion applied to the mineralized stockpile as no mining has taken place. Updates were made to the topographic surface as discussed in Section 14.2.2 which removed some overburden alluvium from the stockpile and added some mineralized material to the bowl area as part of rehabilitation earthworks that were undertaken by the Trust.

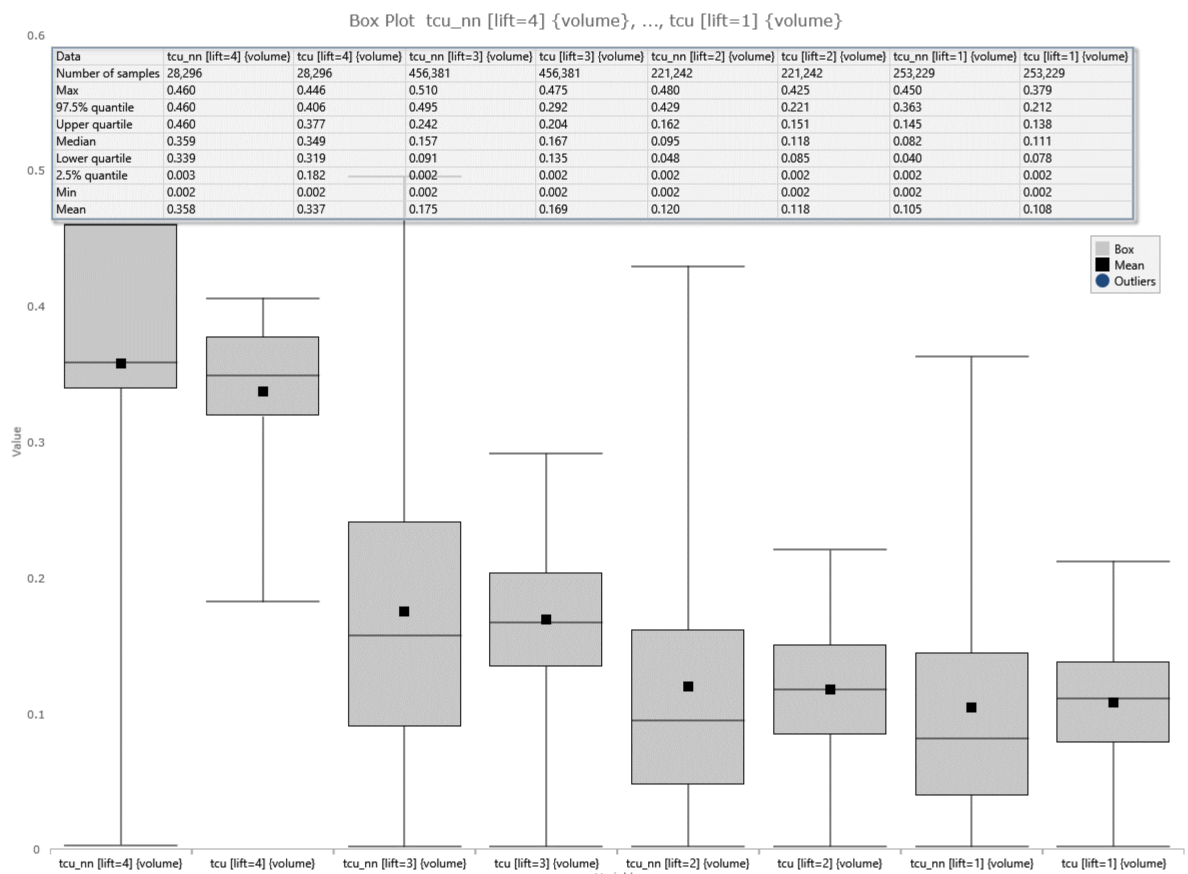
14.2.12 Validations

The main set of validations consist of comparisons against a nearest neighbor and are composed of box plots, visual validations, and swath plots.

Box Plots

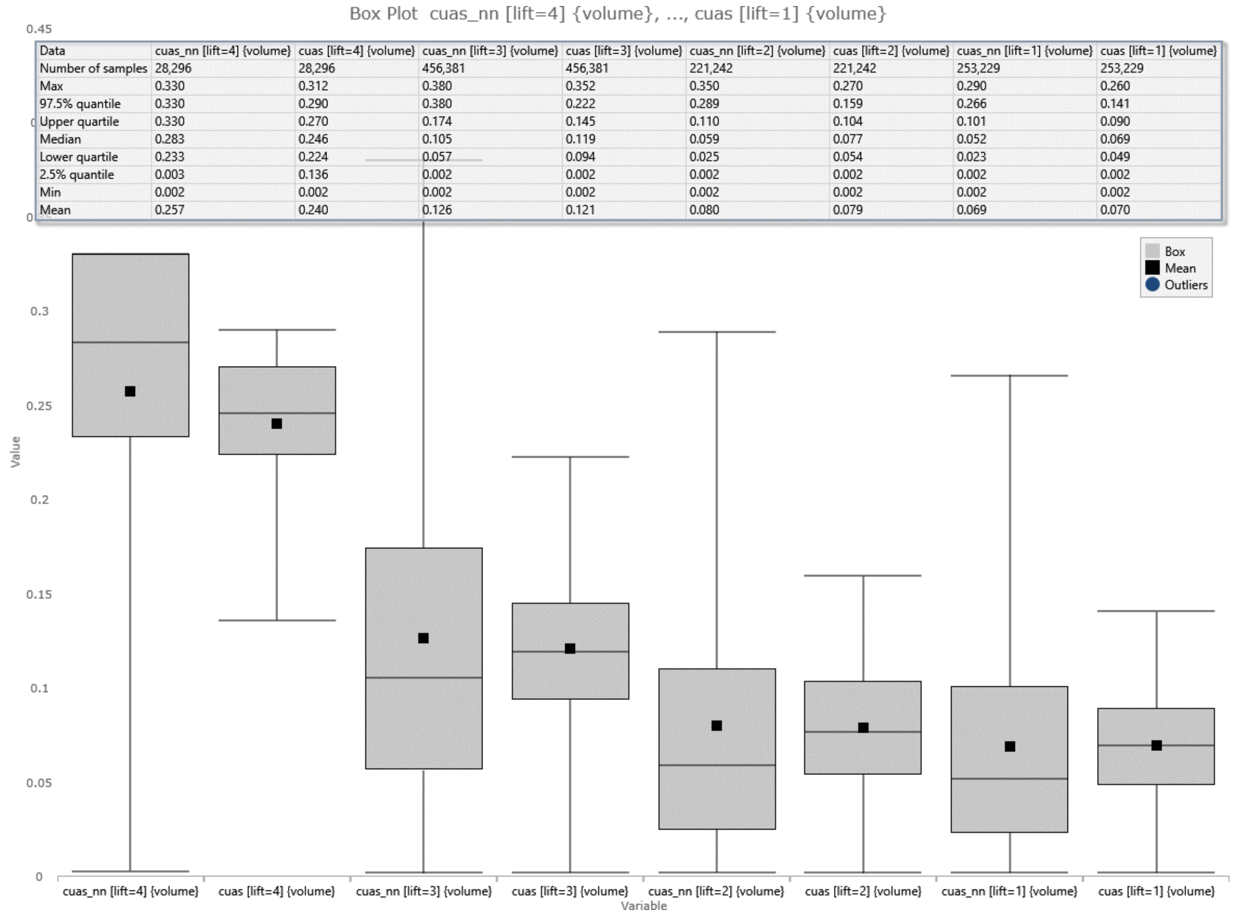
Box plots were created for CuT, CuAS, and CuCN mean grades and distributions within each lift to compare against the nearest neighbor (representing declustered composites) in Figure 14-64 through Figure 14-66, respectively. All comparisons show very similar mean grades between the estimated blocks and the nearest neighbor. The adjustment from a nearest neighbor sample support to a block estimate support incurs significant smoothing (particularly for wider spaced drilling programs and where smoothing is a planned feature of the model such as for the Stockpile Project estimate). This smoothing is visible in the box plots by the restricted box size within the plots for the estimated blocks versus that of the comparison nearest neighbor plots. No maximum values of the nearest neighbor statistics are reported higher than the planned top cuts, indicating that the top cut was applied to the estimation as planned.

Figure 14-64: Box Plots Comparing CuT for the Cactus Stockpile Project Against the Nearest Neighbor Grouped by Lift



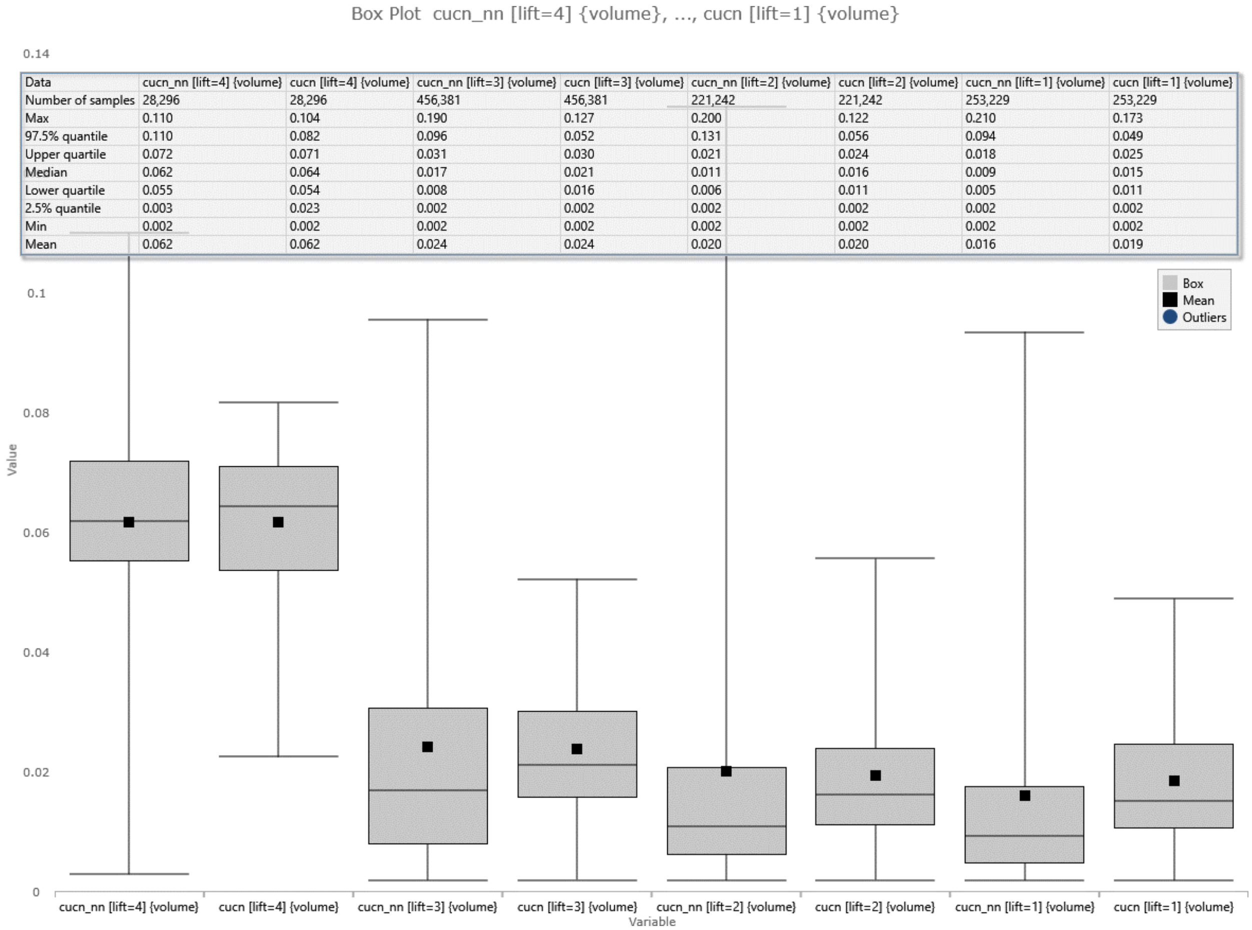
Data	tcu_nn [lift=4] {volume}	tcu [lift=4] {volume}	tcu_nn [lift=3] {volume}	tcu [lift=3] {volume}	tcu_nn [lift=2] {volume}	tcu [lift=2] {volume}	tcu_nn [lift=1] {volume}	tcu [lift=1] {volume}
Number of samples	28,296	28,296	456,381	456,381	221,242	221,242	253,229	253,229
Max	0.460	0.446	0.510	0.475	0.480	0.425	0.450	0.379
97.5% quantile	0.460	0.406	0.495	0.292	0.429	0.221	0.363	0.212
Upper quartile	0.460	0.377	0.242	0.204	0.162	0.151	0.145	0.138
Median	0.359	0.349	0.157	0.167	0.095	0.118	0.082	0.111
Lower quartile	0.339	0.319	0.091	0.135	0.048	0.085	0.040	0.078
2.5% quantile	0.003	0.182	0.002	0.002	0.002	0.002	0.002	0.002
Min	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002
Mean	0.358	0.337	0.175	0.169	0.120	0.118	0.105	0.108

Figure 14-65: Box Plots Comparing CuAS for the Cactus Stockpile Project Against the Nearest Neighbor Grouped by Lift



Data	cuas_nn [lift=4] {volume}	cuas [lift=4] {volume}	cuas_nn [lift=3] {volume}	cuas [lift=3] {volume}	cuas_nn [lift=2] {volume}	cuas [lift=2] {volume}	cuas_nn [lift=1] {volume}	cuas [lift=1] {volume}
Number of samples	28,296	28,296	456,381	456,381	221,242	221,242	253,229	253,229
Max	0.330	0.312	0.380	0.352	0.350	0.270	0.290	0.260
97.5% quantile	0.330	0.290	0.380	0.222	0.289	0.159	0.266	0.141
Upper quartile	0.330	0.270	0.174	0.145	0.110	0.104	0.101	0.090
Median	0.283	0.246	0.105	0.119	0.059	0.077	0.052	0.069
Lower quartile	0.233	0.224	0.057	0.094	0.025	0.054	0.023	0.049
2.5% quantile	0.003	0.136	0.002	0.002	0.002	0.002	0.002	0.002
Min	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002
Mean	0.257	0.240	0.126	0.121	0.080	0.079	0.069	0.070

Figure 14-66: Box Plots Comparing CuCN for the Cactus Stockpile Project Against the Nearest Neighbor Grouped by Lift

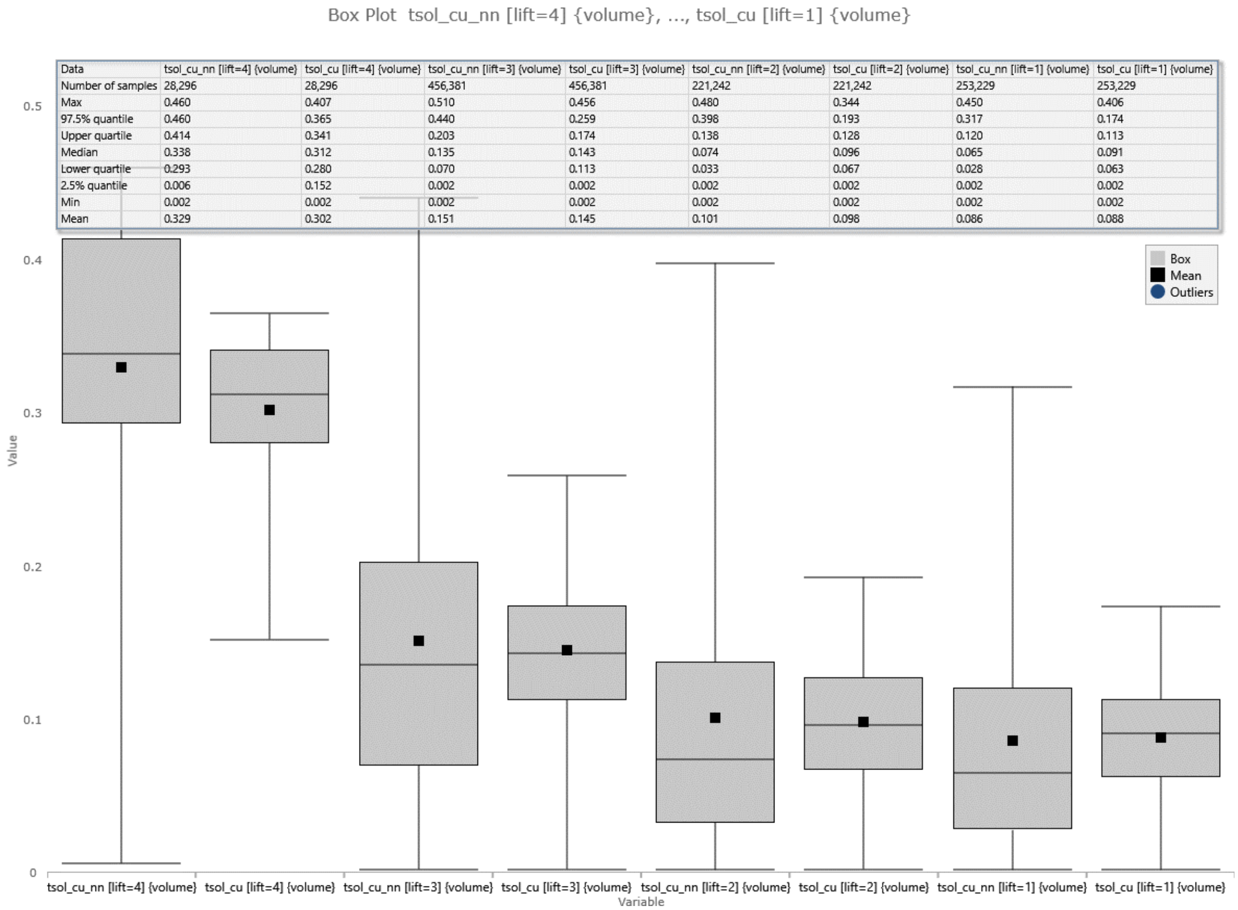


Data	cucn_nn [lift=4] {volume}	cucn [lift=4] {volume}	cucn_nn [lift=3] {volume}	cucn [lift=3] {volume}	cucn_nn [lift=2] {volume}	cucn [lift=2] {volume}	cucn_nn [lift=1] {volume}	cucn [lift=1] {volume}
Number of samples	28,296	28,296	456,381	456,381	221,242	221,242	253,229	253,229
Max	0.110	0.104	0.190	0.127	0.200	0.122	0.210	0.173
97.5% quantile	0.110	0.082	0.096	0.052	0.131	0.056	0.094	0.049
Upper quartile	0.072	0.071	0.031	0.030	0.021	0.024	0.018	0.025
Median	0.062	0.064	0.017	0.021	0.011	0.016	0.009	0.015
Lower quartile	0.055	0.054	0.008	0.016	0.006	0.011	0.005	0.011
2.5% quantile	0.003	0.023	0.002	0.002	0.002	0.002	0.002	0.002
Min	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002
Mean	0.062	0.062	0.024	0.024	0.020	0.020	0.016	0.019

As an independent check on the grade estimates, box plots were created for TSol mean grades and distributions within each lift to compare against the nearest neighbor.

Figure 14-67 shows the box plots and confirms similar mean grades and smoothed distributions in line with the composite distributions.

Figure 14-67: Box Plots Comparing TSol for the Cactus Stockpile Project Against the Nearest Neighbor as an Independent Cross Check Grouped by Lift



Data	tsol_cu_nn [lift=4] {volume}	tsol_cu [lift=4] {volume}	tsol_cu_nn [lift=3] {volume}	tsol_cu [lift=3] {volume}	tsol_cu_nn [lift=2] {volume}	tsol_cu [lift=2] {volume}	tsol_cu_nn [lift=1] {volume}	tsol_cu [lift=1] {volume}
Number of samples	28,296	28,296	456,381	456,381	221,242	221,242	253,229	253,229
Max	0.460	0.407	0.510	0.456	0.480	0.344	0.450	0.406
97.5% quantile	0.460	0.365	0.440	0.259	0.398	0.193	0.317	0.174
Upper quartile	0.414	0.341	0.203	0.174	0.138	0.128	0.120	0.113
Median	0.338	0.312	0.135	0.143	0.074	0.096	0.065	0.091
Lower quartile	0.293	0.280	0.070	0.113	0.033	0.067	0.028	0.063
2.5% quantile	0.006	0.152	0.002	0.002	0.002	0.002	0.002	0.002
Min	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002
Mean	0.329	0.302	0.151	0.145	0.101	0.098	0.086	0.088

Visual Validations

On a local scale, model validation can be confirmed by the visual comparison of block grades to composite grades. The color legend of Figure 14-68 is applied to all block and composite grade values for comparative purposes. A plan view and long section of each of the estimated variables showing composites superimposed as dots on block grades is shown in Figure 14-68 through Figure 14-74. The legend applies to CuT, CuAS, and CuCN. Examination indicates appropriate agreement of block grade estimates with the composite grades considering the level of smoothing that has been built into the model. Visual validations confirm the overall grade trends through the stockpile are represented as planned.

Figure 14-68: Legend for all Copper Grade Sections

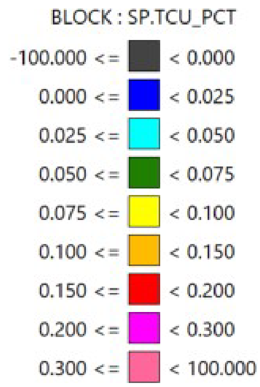
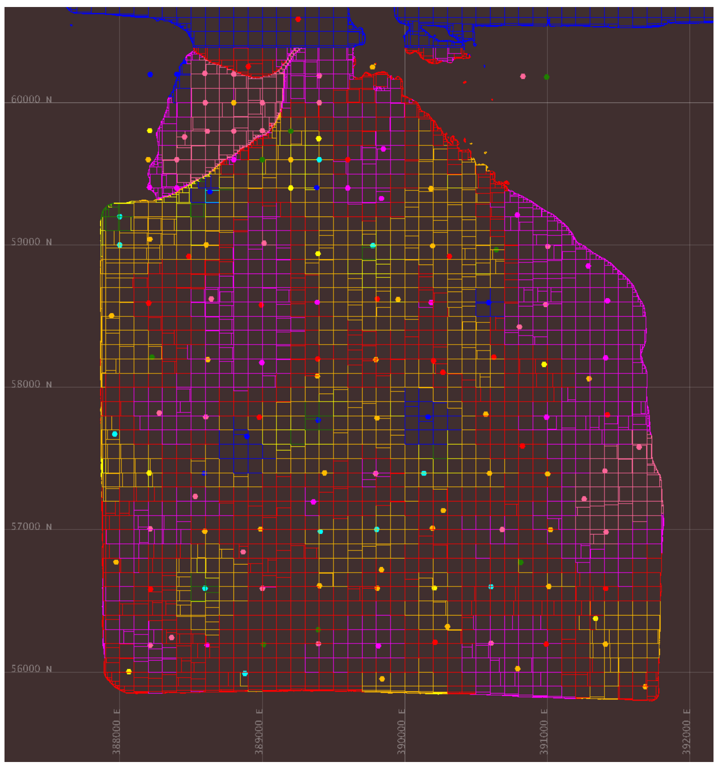
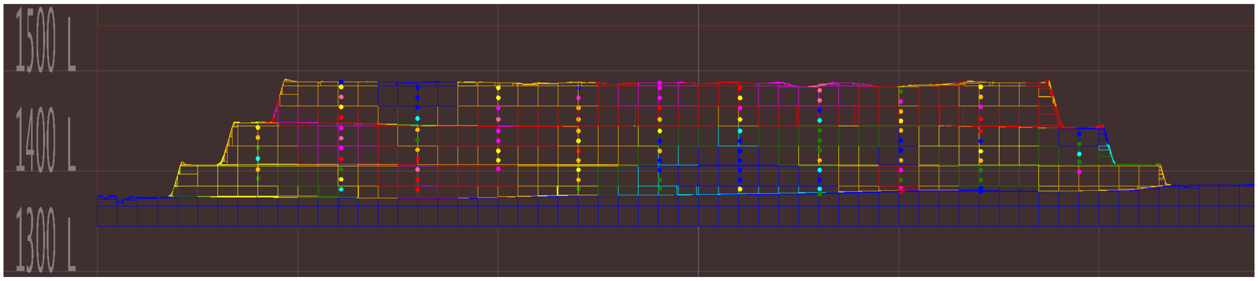


Figure 14-69: Plan View Lift 3 (1445) for CuT Grade Comparing Blocks to Sample Composites



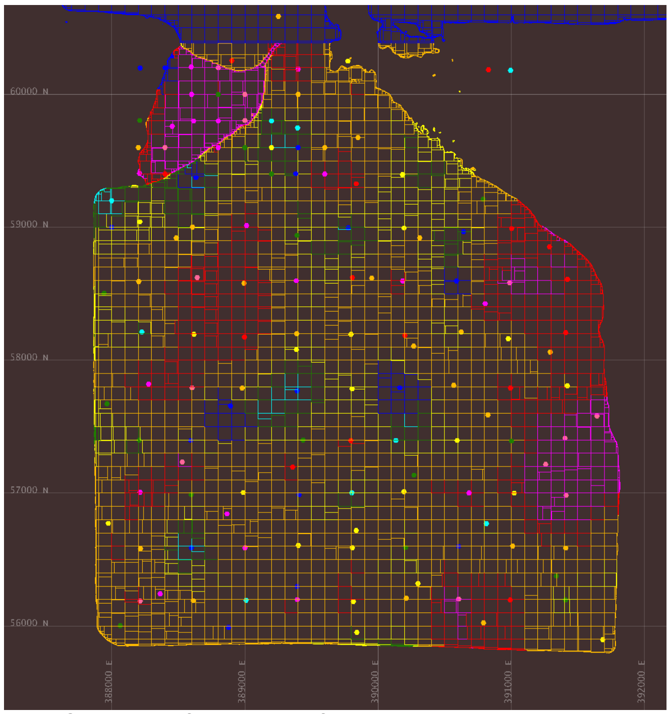
Note: Clipping is 5 ft either side of the section.

Figure 14-70: Cross Section view (56600N) for CuT Grade Comparing Blocks to Sample Composites



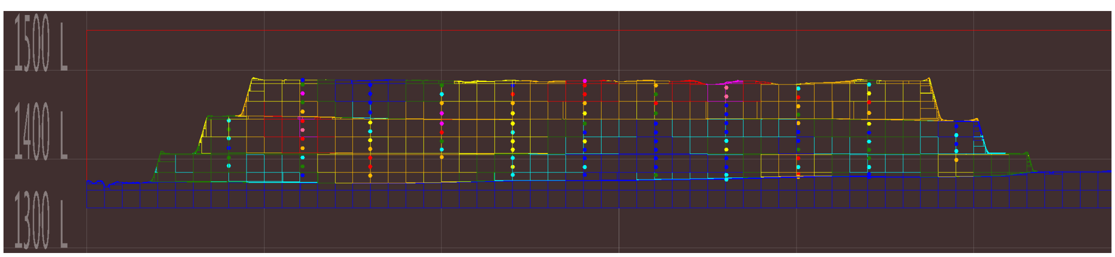
Note: Clipping is 200 ft either side of the section. Vertical exaggeration is set to 500.

Figure 14-71: Plan View Lift 3 (1445) for CuAS Grade Comparing Blocks to Sample Composites



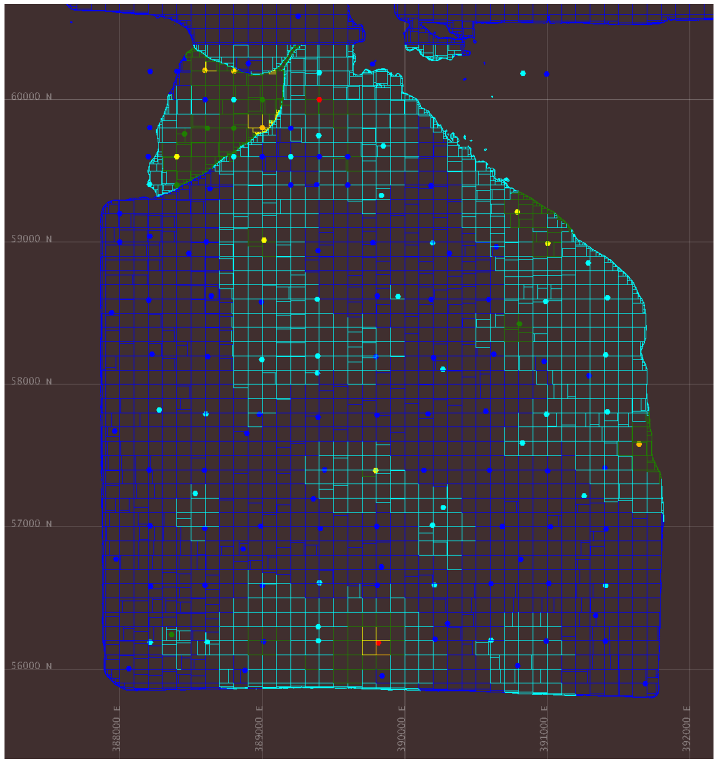
Note: Clipping is 5 ft either side of the section

Figure 14-72: Cross Section View (56600N) for CuAS Grade Comparing Blocks to Sample Composites



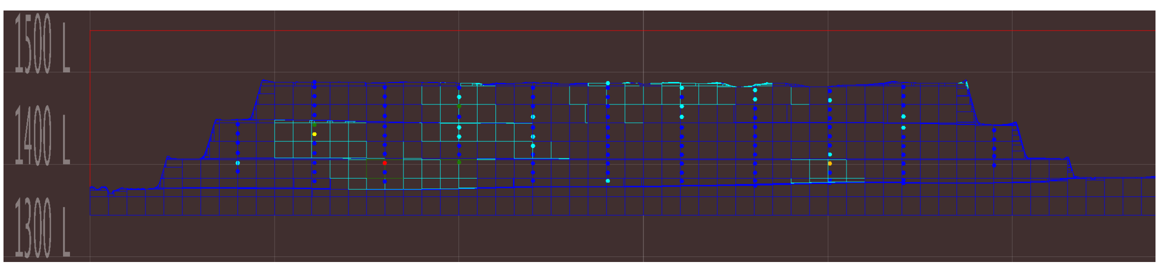
Note: Clipping is 200 ft either side of the section. Vertical exaggeration is set to 500.

Figure 14-73: Plan View Lift 3 (1445) for CuCN Grade Comparing Blocks to Sample Composites



Note: Clipping is 5 ft either side of the section.

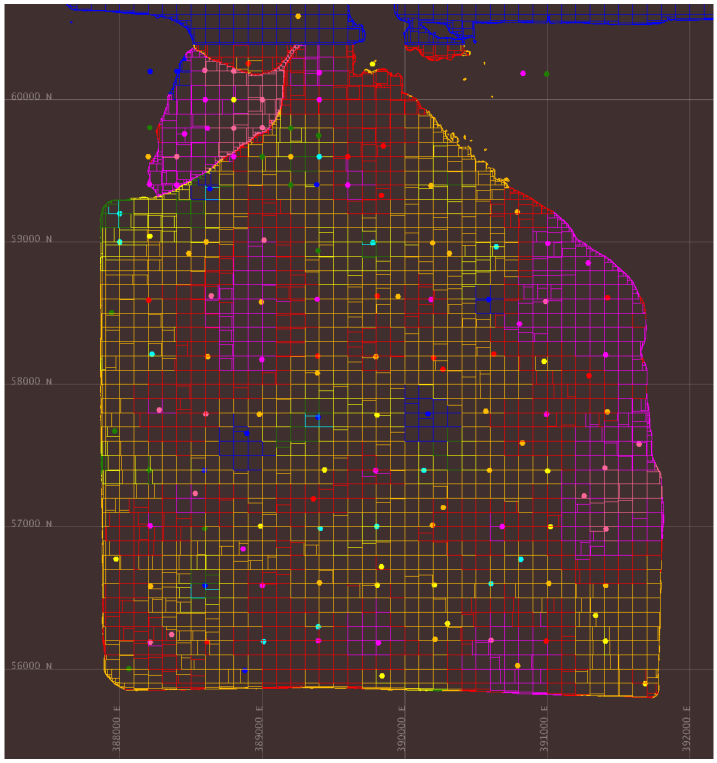
Figure 14-74: Cross Section View (56600N) for CuCN Grade Comparing Blocks to Sample Composites



Note: Clipping is 200 ft either side of the section. Vertical exaggeration is set to 500.

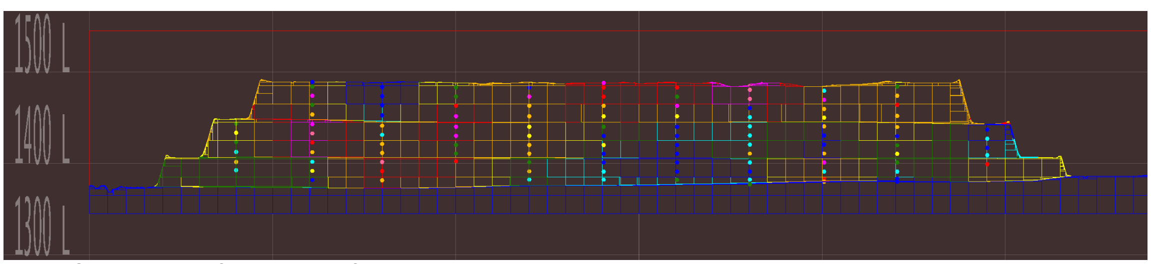
As an independent check on the grade estimates, a visual comparison of block grades to composite grades was also performed for the TSol grades (see Figure 14-75 and Figure 14-76). Examination confirms appropriate agreement and that overall grade trends are represented as planned.

Figure 14-75: Plan View Lift 3 (1445) for TSol Grade Comparing Blocks to Sample Composites



Note: Clipping is 5 ft either side of the section.

Figure 14-76: Cross Section View (56600N) for TSol Grade Comparing Blocks to Sample Composites



Note: Clipping is 200 ft either side of the section. Vertical exaggeration is set to 500.

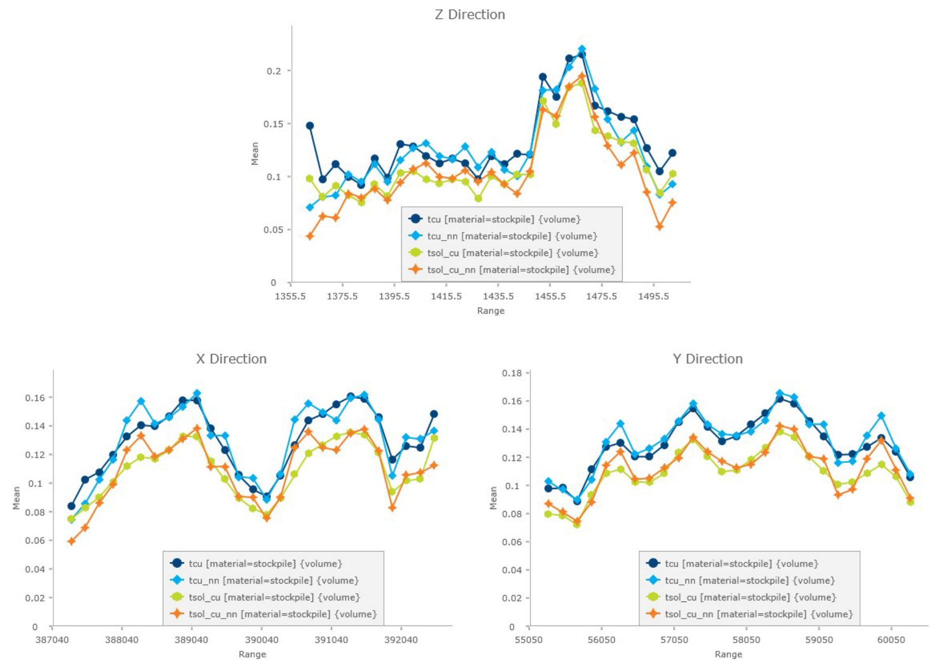
Swath Plots

Swath plots were created to compare the grade trends through the mineralized stockpile between the estimated CuT, CuAS, and CuCN against the nearest neighbor model. As an independent check on the estimates, swath plots were also generated for TSol.

Comparisons for CuAS and CuCN are shown in Figure 14-77. Comparisons for CuT and TSol are shown in Figure 14-78.

Figure 14-77: Swath Plots through Cactus West Comparison with Associated Nearest Neighbor Grade Trends

Figure 14-78: Swath Plots through the Cactus Stockpile Project with Associated Nearest Neighbor Grade Trends



14.2.13 Resource Classification

The drill spacing for the Cactus Stockpile Project has been reduced from approximately 750 ft (229 m) to 400 ft (122 m) spacing. Due to the nature of the dumping of material to the stockpile and inherent variability, at this drill spacing the mineral resource classification remains at an Inferred status. Of particular note is that through the process of halving the drill spacing and tripling the number of drill holes, there has been little change to the grade tonnage curve and global resource from that previously reported in 2020.

14.3 Resource Reporting

14.3.1 Resource Cutoff Grades

To meet a REEEE requirement, as stated in CIM 2019 Best Practices, COGs were applied to both a potential open pit across the Cactus deposit and a potential underground mine at depth in Cactus East.

Conceptually, copper from oxide and enriched material in the open pit would be recovered in a heap leach. Therefore, COGs in the amenable oxide and enriched zones were based on TSol assays. COGs for the sulfides in the primary material were based on CuT assays. High-level cost analysis for the Cactus open pit suggested COGs of 0.096% TSol for the oxides, and 0.098% TSol for the enriched material. A cutoff of 0.205% CuT was applied to the primary material to be stockpiled for potential recovery in a flotation mill. A Whittle pit was run using these parameters and the reported resource is for material within that pit.

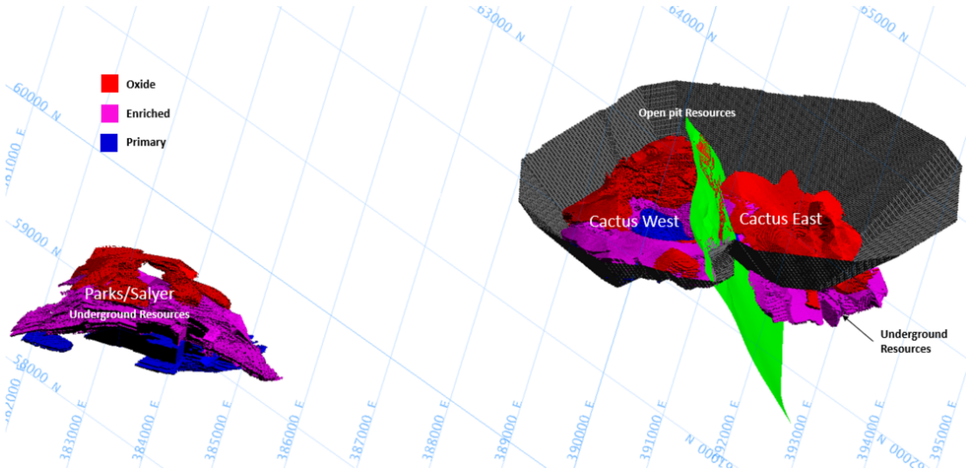
Additional mineral resources outside of the Whittle pit in Cactus East have the potential to be amenable to underground mining. High-level analysis of the material yielded cutoffs of 0.560% TSol for the oxides and 0.700% TSol for the enriched. The primary had a 0.700% cutoff applied to the CuT grade for potential recovery in a flotation mill.

Mineral resources for Parks/Salyer were also determined based on its amenability to underground mining. Due to the resources for Parks/Salyer having an effective date of 26 September 2022, a higher copper price of US$3.75/lb, was used in determining the COGs for underground mining. High-level analysis of the material yielded cutoffs of 0.495% TSol for the oxides and 0.600% TSol for the enriched. The primary had a 0.586% cutoff applied to the CuT grade for potential recovery in a flotation mill.

Stockpile Project mineral resources were defined using a COG of 0.095% TSol.

Figure 14-79 displays an oblique image of the Cactus open pit and underground resources for Cactus West and Cactus East as defined by the Whittle pit shell (gray) and underground COG. Red areas indicate oxide resources, magenta areas indicate enriched sulfide resources, and blue areas indicate primary sulfide resources. The sacaton fault which offsets the Cactus West and Cactus East ore bodies is defined in green.

Figure 14-79: Oblique Image Displaying Open Pit and Underground Resources for Cactus West, Cactus East, and Parks/Salyer and Material Types



14.3.2 Resource Table

Table 14-16 details the breakdown of resources for Cactus West and Cactus East by mineral zone and classification within the Whittle pit. Table 14-17 and Table 14-18 have the same breakdown for the potential underground mineral resources for Cactus East and Parks/Salyer. Table 14-19 shows the combined total of the two previous tables.

Table 14-20: Cactus West and Cactus East Open Pit Indicated and Inferred Resource

Material Type	Tons (kt)	CuT (%)	TSol (%)	Contained Copper (klb)
Indicated				
Oxide	27,000		0.512	275,900
Enriched	39,200		0.822	643,800
Total Leachable	66,200		0.696	919,700
Primary	75,700	0.338		511,900
Total Indicated	141,900	0.505		1,431,600
Inferred				
Oxide	51,600		0.268	282,000
Enriched	48,100		0.405	390,100
Total Leachable	99,700		0.334	672,100
Primary	110,000	0.344		756,600
Total Inferred	**209,700**	**0.339**		**1,428,700**

Note: Refer to Table 14-17 for appliable notes to the open pit resource parameters and assumptions. Totals may not add up due to rounding.

Table 14-21: Cactus East Underground Indicated and Inferred Resource

Material Type	Tons(kt)	CuT(%)	TSol(%)	Contained Copper (klb)
Indicated				
Oxide	4,400		0.844	74,200
Enriched	3,300		1.101	72,000
Total Leachable	7,700		0.954	146,200
Primary	2,200	0.767		33,800
Total Indicated	9,900	0.912		180,000
Inferred				
Oxide	10,900		0.718	157,200
Enriched	7,000		1.136	158,500
Total Leachable	17,900		0.881	315,700
Primary	1,300	0.7624	0.091	20,200
Total Inferred	**19,200**	**0.873**		**335,900**

Note: Refer to Table 14-17 for appliable notes to the underground resource parameters and assumptions. Totals may not add up due to rounding.

Table 14-22: Parks/Salyer UG Inferred Resource

Material Type	Tons (kt)	CuT (%)	TSol (%)	Contained Copper (klb)
Inferred				
Oxide	14,100		0.827	233,700
Enriched	101,200		1.100	2,227,200
Primary	28,300	0.804		454,400
Total Inferred	143,600	1.015		2,915,400

Note: Refer to Table 14-17 for appliable notes to the underground resource parameters and assumptions. Totals may not add up due to rounding.

Table 14-23: Cactus Stockpile Project Inferred Resource

Tons (Kt)	CuT (%)	TSol (%)	CuAS (%)	CuCN (%)	CuT Metal (Klb)	TSol Metal (Klb)
Inferred						
77,400	0.169	0.144	0.118	0.026	262,100	223,500

Note: Refer to Table 14-17 for appliable notes to the stockpile resource parameters and assumptions. Totals may not add up due to rounding.

Table 14-24: Cactus Project Total Indicated and Inferred Resource

Material Type	Tons(kt)	CuT(%)	TSol(%)	Contained Copper (klb)
Indicated				
Total Leachable	73,900		0.723	1,065,200
Total Indicated	151,800	0.531		1,610,700
Inferred				
Total Leachable	310,400		0.590	3,663,700
Total Inferred	449,900	0.544		4,894,200

Notes:
1. CuT means total copper and TSol means total soluble copper as the addition of sequential acid soluble and sequential cyanide soluble copper assays. Tons are reported as short tons.
2. Cactus and Stockpile Resource estimates have an effective date of 31 August 2021 and use a copper price of US$3.15/lb. The assumptions in respect of the Cactus and Stockpile Resource estimates are as stated in the PEA titled "Arizona Sonoran Copper Company, Inc. Cactus Project, Arizona, USA Preliminary Economic Assessment" with an effective date of August 31, 2021; Parks/Salyer Resource estimate has an effective date of 26 September 2022 and uses a copper price of US$3.75/lb.
3. Technical and economic parameters defining resource pit shell: mining cost US$2.45/t; G&A US$0.55/t, and 44°-46° pit slope angle.
4. Technical and economic parameters defining underground resource: mining cost US$28.93/t, and G&A representing 7% of direct costs.
5. Technical and economic parameters defining processing: Heap leach (HL) processing cost including selling US$1.77/t; HL recovery 83% of CuT; mill processing cost US$8.50/t.
6. For Cactus: Variable cutoff grades were reported depending on material type, potential mining method, and potential processing method. Oxide material within resource pit shell = 0.096% TSol; enriched material within resource pit shell = 0.098% TSol; primary material within resource pit shell = 0.205% CuT; oxide underground material outside resource pit shell = 0.56% TSol; enriched underground material outside resource pit shell = 0.70% TSol; primary underground material outside resource pit shell = 0.70% CuT.
7. For Parks/Salyer: Variable cut-off grades were reported depending on material type associated potential processing method. Oxide underground material = 0.495% TSol; enriched underground material = 0.60% TSol; primary underground material = 0.586% CuT.
8.For the stockpile: There is a reasonable probability of eventual economic extraction of this resource using sulfuric acid leaching and SX/EW recover at a TSol cutoff of 0.095%
9. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, sociopolitical, marketing, or other relevant factors.
10. The quantity and grade of reported inferred mineral resources in this estimation are uncertain in nature and there is insufficient exploration to define these inferred mineral resources as an indicated or measured mineral resource; it is uncertain if further exploration will result in upgrading them to an indicated or measured classification.
11. Total may not add up due to rounding.

15.0 MINERAL RESERVE ESTIMATES

This section is not applicable.

16.0 MINING METHODS

The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as at 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021.

This study involves moving leachable oxide and chalcocite material to leach pad facilities from three different sources. They are an existing, historical low-grade stockpile (Stockpile Project) located on surface, a traditional open pit operation and an underground mine operation. The remaining primary, or chalcopyrite resource, is not considered in this report. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report.

16.1 Geotechnical

Geotechnical analysis was considered for the open pit and underground only.

For the open pit operation, the maximum pit slope face angles were evaluated in terms of kinematic stability, in relation to pit slope orientation versus the predominant orientations of geologic structure.

Underground stope versus pit wall stability is evaluated in terms of differential stress magnitude in the pit wall versus rock strength, at different stand-off distances. Stope sizing is evaluated in terms of rock mass stability versus excavation dimensions.

16.1.1 Data

Data used in the evaluations are primarily RQD, structural orientations, strength properties, and excavation geometry. Oriented drill core was collected and logged for RQD and structural orientation from two drill holes, ECW-010 and ECE-016. Oriented drill core facilitates the evaluation of specific interactions of structural and geometric orientations for stability evaluation. This improves quality of design and implementation by identifying geometric limits to stability. Figure 16-1 shows the collar locations of ECW-010 and ECE-016 in reference to the existing pit. Figure 16-2 is a general representation of major structural geology around the existing pit.

Figure 16-1: ECW-010 and ECE-016 Collar Locations

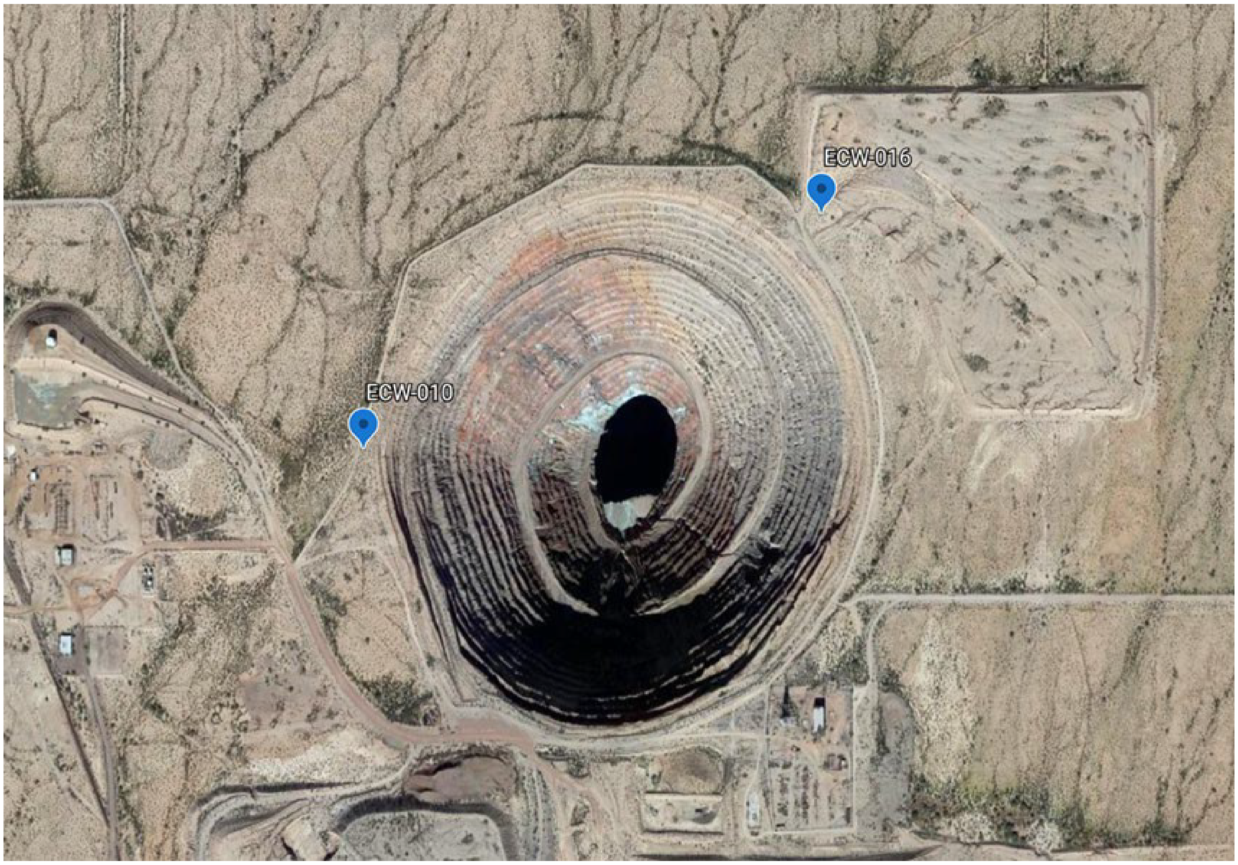
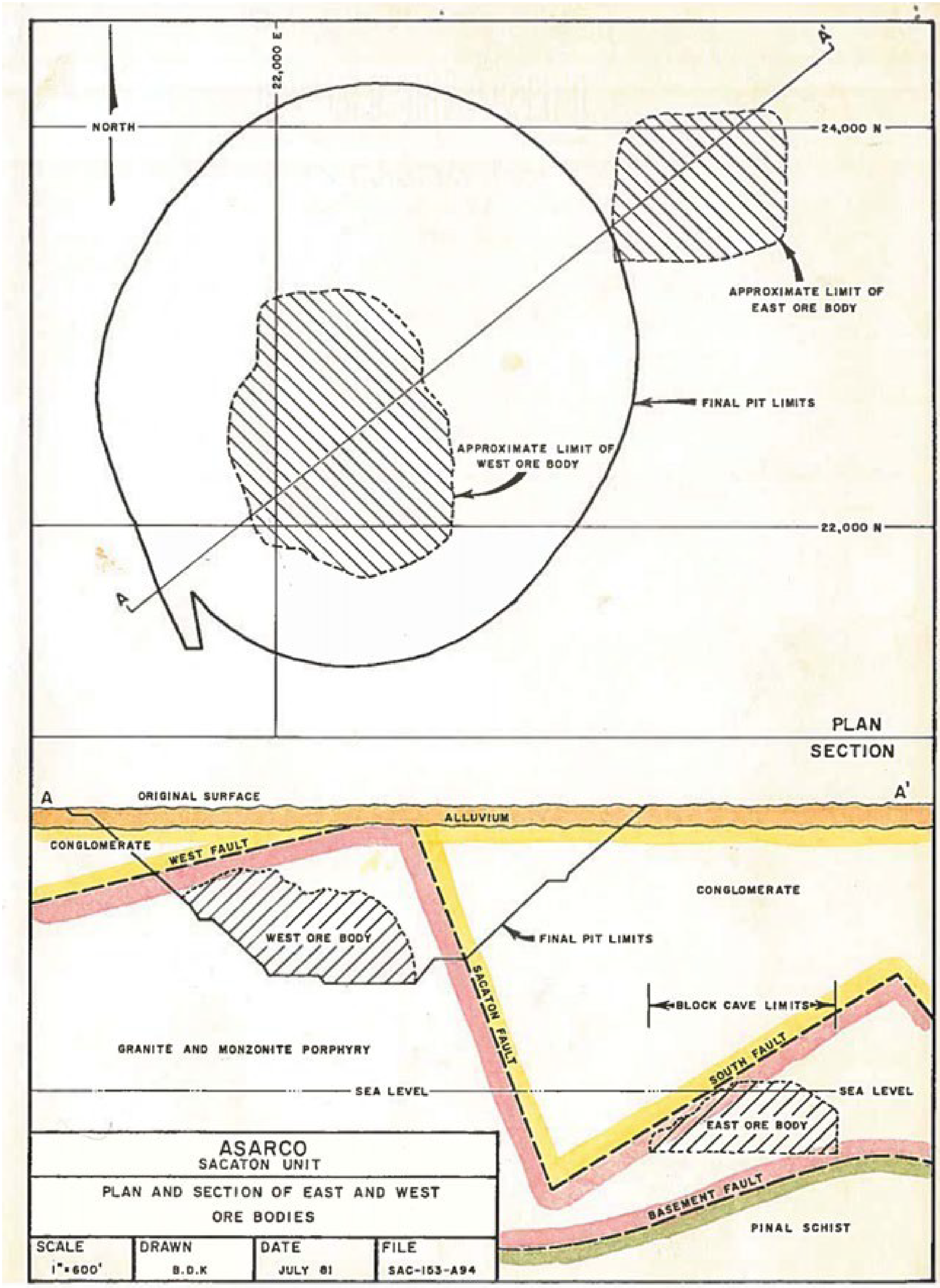


Figure 16-2: Major Geologic Structures



Drill hole ECW-010 data is most representative of rock conditions to be expected during pit expansion. This drill hole is collared at the pit's western pushback and intercepts ground to be mined. Drill hole ECE-016 data represents conditions to be encountered at the underground expansion, due to proximity at the eastern mineralized zone. For these studies, RQD is primarily used to validate open stope dimensions.

Aside from the rock engineering properties, kinematic analysis of pit wall stability is based mostly on structural and pit wall orientations. These were derived from measurements of structure intercepted by drilling in relation to the drill hole orientation, then normalized to give the true orientations of structures.

Figure 16-3 shows structures logged from ECW-010. Most structural planes dip to the west/northwest at shallow angles. Due to similarities in orientations, there is little wedge bearing potential of major wedges.

Figure 16-3: ECW-010 Structural Orientations

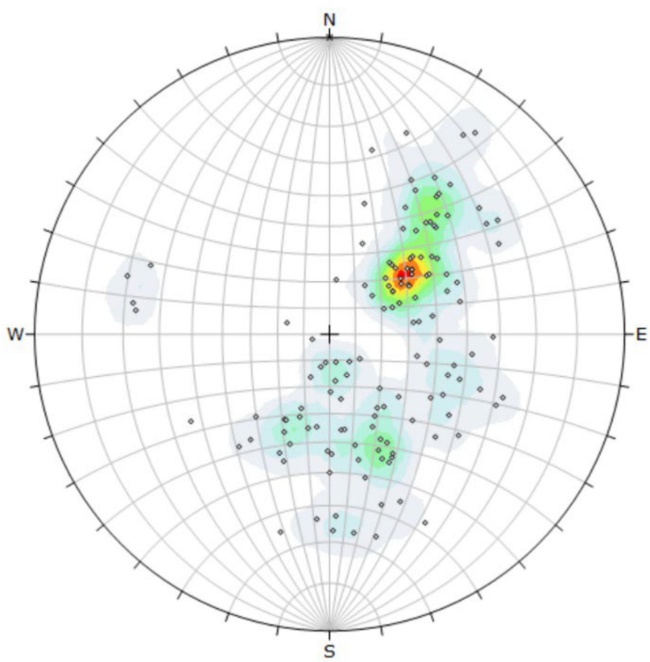
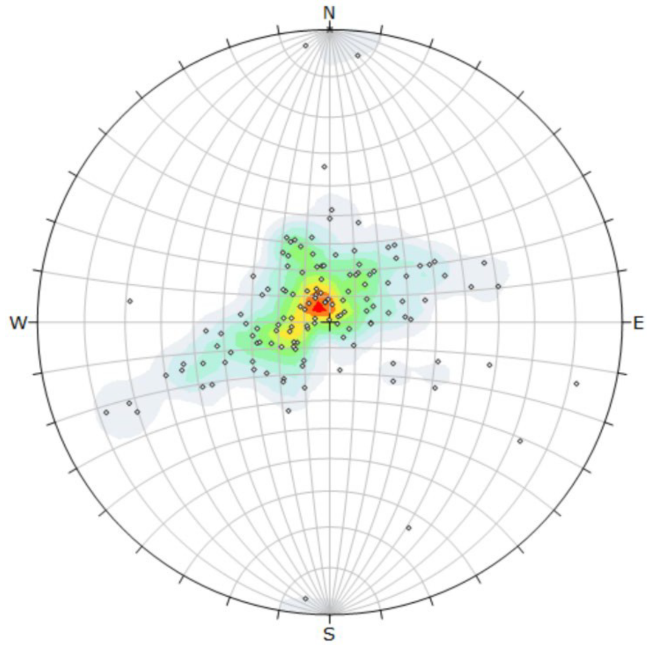


Figure 16-4 shows structures logged from ECE-016. Most structural planes are horizontal. Since kinematic analysis is applied to pit wall stability, ECW-010 data is most representative of the pit expansion and ECE-016 data is disregarded for surface stability analysis.

Figure 16-4: ECE-016 Structural Orientations



16.1.2 Pit Pushback Slope Analysis

The batter angle of each bench is the maximum pit slope face angle. This is determined through the kinematic interaction of slope and structural orientations, which differs from the overall pit slope angle which also considers bench geometry. The batter angles are not designed to a specified factor of safety, rather the batter angles are evaluated in terms of reliability, to introduce flexibility in terms of stability requirements, stripping, stand-up time, and risk tolerance. Reliability is defined by stable structural combinations for a given pit wall orientation out of the total structural combinations defining a wedge or a sliding plane.

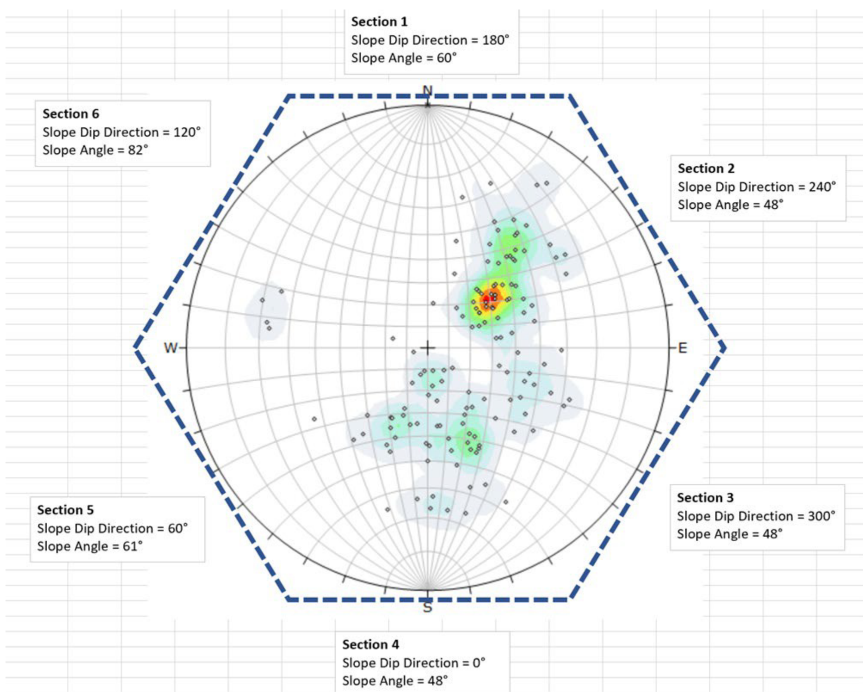
Kinematic analysis is a stereonet exercise where failure potential is determined through three general mechanisms: planar sliding, sliding wedge failure, and toppling. Toppling is ruled out since ECW-010 structures typically have shallow dip angles, leaving planar sliding and sliding wedge failure mechanisms considered.

Six slope orientation segments are considered. Each orientation range represents a 60° segment of a full 0°-360° degree range. The maximum batter angles of each segment are determined at 95% reliability. The Kinematic analysis for each section is shown in Appendix B. The results are maximum batter angles from planar sliding and sliding wedge failure analysis (Table 16-1). The shallower batter angle determined through both methods is the maximum batter angle, for each face orientation (Figure 16-5).

Table 16-1: Maximum Slope Angles for the Pit Pushback

Conglomerate		Failure Modes	
Section	Dip Direction	Planar Sliding	Wedge Analysis
1	180°	60°	79°
2	240°	48°	49°
3	300°	49°	48°
4	0°	48°	50°
5	60°	61°	78°
6	120°	82°	90°

Figure 16-5: Maximum Slope Angle Versus Slope Dip Direction



Results show that slopes facing north and west have the shallowest allowable batter angles. Planar structures and plane intersections are dipping down into the pit for these slopes.

Shallower slope face angles provide greater sliding resistance since a greater normal force is exerted on the sliding plane. Much steeper slope angles designed to 95% reliability are possible for slopes facing eastward. This is due to predominant structural orientations dipping into the pit wall and away from the pit in these segments. Figure 16-6 shows the suggested batter angles along the pit's western pushback, based on analysis results.

Figure 16-6: Suggested Batter Angles for Western Pit Expansion

16.1.3 Pit Wall Pillar Stability

Underground stope versus pit wall stability is evaluated in terms of differential stress magnitudes in the pit wall in relation to rock strength, at different stand-off distances of mined stopes. A numerical modeling approach is taken, which is discussed in Appendix C.

Model results indicate that with a minimum pillar thickness of 115 ft (35 m) to 164 ft (50 m), the pillar is not subject to fracturing due to stress magnitudes during mining. For initial design purposes, a minimum pillar thickness of 131 ft (40 m) should be maintained. As mine planning progresses and additional geotechnical data becomes available through rib mapping and definition drilling, each stope is to be reviewed individually to mitigate stability challenges due to rock mass, pillar thickness, and excavation dimensions. Stope panel length management and backfill practice will require special attention. In some cases, a different extraction method (i.e., drift and fill) may be more appropriate.

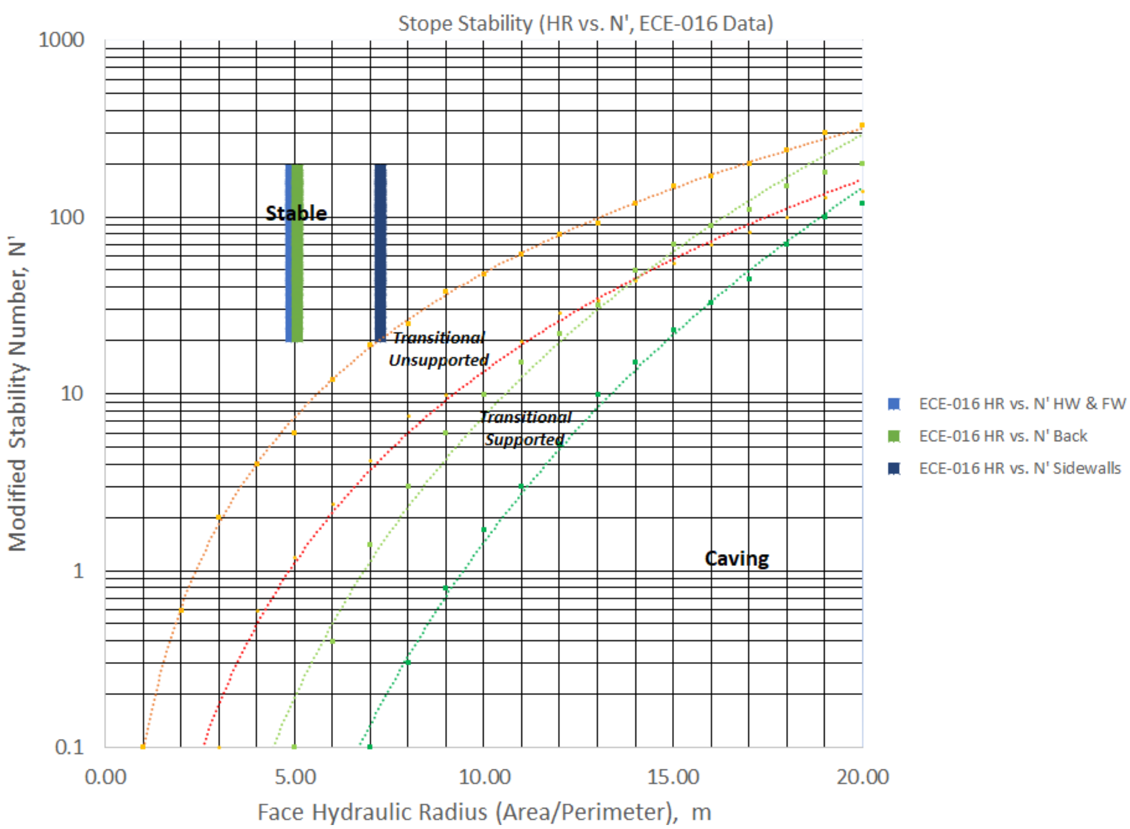
16.1.4 Stope Stability

Planning and maintaining stable stoping practice is key for pit wall and pit bottom stability, where stoping occurs in proximity. Stope stability is evaluated in terms of rock mass quality versus excavation dimensions, applied to a stability graph known as the Mathews-Potvin Stability Graph Method and derived from global empirical data representing stability of large underground excavations (Hutchinson & Diedrichs, 1996). Data was primarily taken from ECE-016, which due to location is believed to be most representative of ground conditions at the Eastern mineralized zone.

Stope stability analysis is presented in Appendix D, where ground conditions are represented by a modified stability index N' and excavation dimensions are represented by the hydraulic radii (HR) for each face of the stope, where HR is the face's area: perimeter ratio.

Figure 16-7 shows where the stope defined by the proposed design dimensions falls in relation to the Mathews-Potvin stability curves.

Figure 16-7: Mathews-Potvin Stability Graph for Stope Sizing



The proposed maximum underground primary stope lengths of 100 ft (30 m) are acceptable for initial design and long range planning purposes though are subject to adjustment typically at a semi-annual, quarterly, or monthly basis based on operating performance. Length adjustments are based on the validation of N' values through rib mapping at the drill and mucking levels of each stope, as these sill cuts are mined. Rib mapping should occur in cycle or as soon as each sill cut is mined to completion to have new data available to update N' and find a revised HR and stope length. This allows sufficient time for modifications to the mine plan based on stope length adjustments. Any secondary or tertiary stope lengths are not restricted if the backfill attains the calculated strength for an infinitely long stope side wall. Additional geotechnical drilling and analysis should be performed in the next stage of work to broaden geotechnical understanding and validate the stope design dimensions.

The stopes adjacent to the pit wall pillar should be mined very late in the LOM to minimize the time after the pit wall is undercut. Minimizing the pit wall pillar's underground exposure is also achieved by managing the stope panel sizes, minimizing stope cycle times, and optimizing backfill quality and placement. The stability of the pit wall pillar should be monitored visually continuously and through instrumentation (i.e., extensometers) and periodically survey (i.e., laser scans). Recommendations for stopes nearest the pit wall include stope size management, expedient backfill placement, monitoring, and a trigger action response plan if monitoring thresholds are exceeded, or considering alternate mining method like drift and fill, where smaller volumes are excavated.

16.2 Surface Mining

The open pit and Stockpile Project will be a truck and loader / shovel mining method. This section describes the methodology behind the development of the open pit.

16.2.1 Mine Optimization Method and Parameters

The pit optimization used to establish the most optimal ultimate pit limit was run on the most recent Cactus geological resource model. GEOVIA Whittle was used to conduct the pit optimization using the pseudoflow optimization method. Both indicated and inferred mineral resource categories were included for reporting of the pit quantities. The pit optimization only considered leachable material, which are the oxide copper minerals and enriched ore. A list of all the input parameters used during pit optimization is shown in Table 16-2.

Table 16-2: Pit Optimization Inputs

Item	Value	Units
Mining Unit Cost (In Situ Hard Rock)	2.50	$US/t
Overburden Mining Cost	1.88	$US/t
Mining Recovery	95%	%
Mining Dilution	5%	%
Oxide Leaching Cost	2.21	$US/t
Oxide Leaching Recovery	90%	%
Enriched Leaching Cost	1.25	$US/t
Enriched Leaching Recovery	72%	%
Selling Price	3.15	$US/lb
Selling Cost	0.04	$US/lb
Discount Rate	8%	%
Mining Rate	30,000,000	tpa
Leaching (Processing) Rate	15,000,000	tpa
Mining Width	150	ft

Mining cost, mining recovery, and dilution were based on Stantec's internal cost database. The cost of mining through overburden is assumed to be overall 75% of the cost of mining through hard rock because drill and blast activities would not be required.

Processing cost, selling cost, and production rate parameters were provided by Arizona Sonoran and Samuel Engineering. The overburden mining cost was applied to material above the 1,340 ft (408 m) elevation using available geological info.

The primary pit optimization goal was to achieve recovered copper to the plant at 22,000 tons per annum (tpa) for the first 6 years, after which recovered metal production was ramped up to 35,000 tpa.

This allowed for a reduction in initial capital requirements as well as allowing a practical ramp-up period for mineralized material production from the pit.

The historic open pit was modeled as part of the natural topography of the surrounding area, ensuring that none of the mined-out rock or mineralized material were included in the pit optimization. Grades were applied on a whole block basis to all blocks designated as Measured, Indicated, or Inferred. Blocks classified as a potential resource were considered to be waste rock.

Pit optimization was conducted using the total economic copper grade (TECu item). The TECu item represents the grade of soluble copper for oxidized and enriched material. All non-leachable material (typical sulfidic mineralized material that would require grinding and flotation) was sent to a primary stockpile for potential processing in future. The sulfide material is also called 'prime' or 'primary' in some parts of the text.

Another important component for deriving the optimal pit was the geotechnical characteristics of the In Situ rock. For this purpose, the pit was divided into six slope sectors based on results of a preliminary kinematic analysis (see Section 16.1.2 and Appendix B). The overall slope angles are based on batter (bench face) angles from the kinematic analysis as well as a common bench height of 60 ft (18.3 m), and a minimum catch bench width of 26 ft (7.9 m). The slope sectors as they were applied in the pit optimization are shown in Figure 16-8 and Table 16-3.

Figure 16-8: Slope Sectors Shown Over the Ultimate Optimized Pit

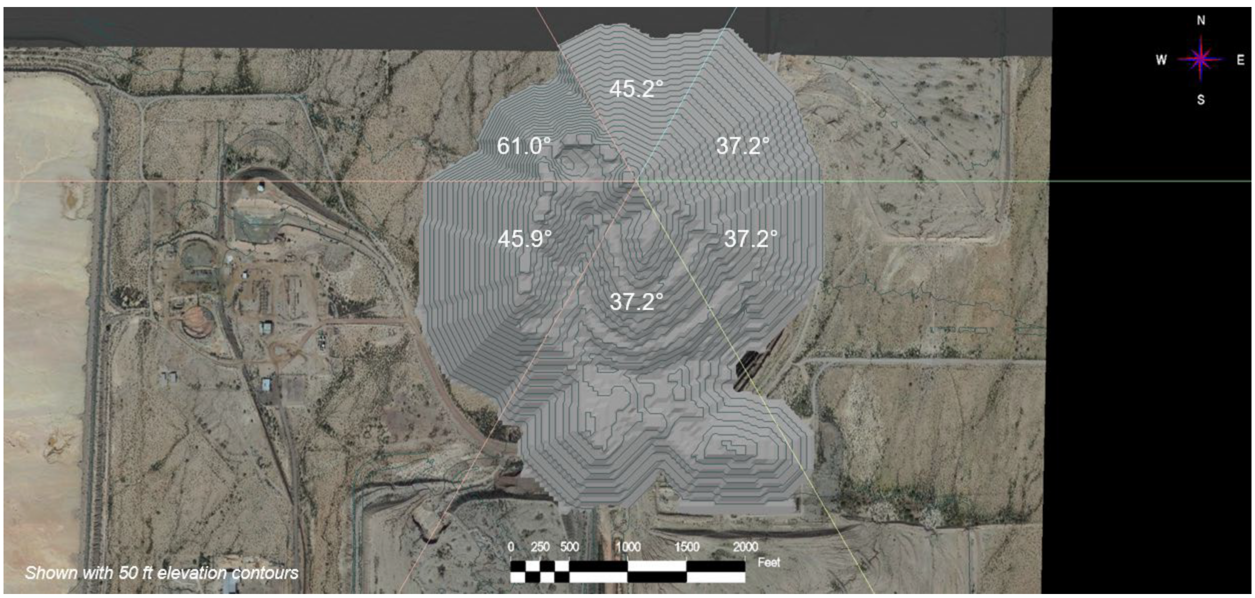


Table 16-3: Slope Sector Parameters

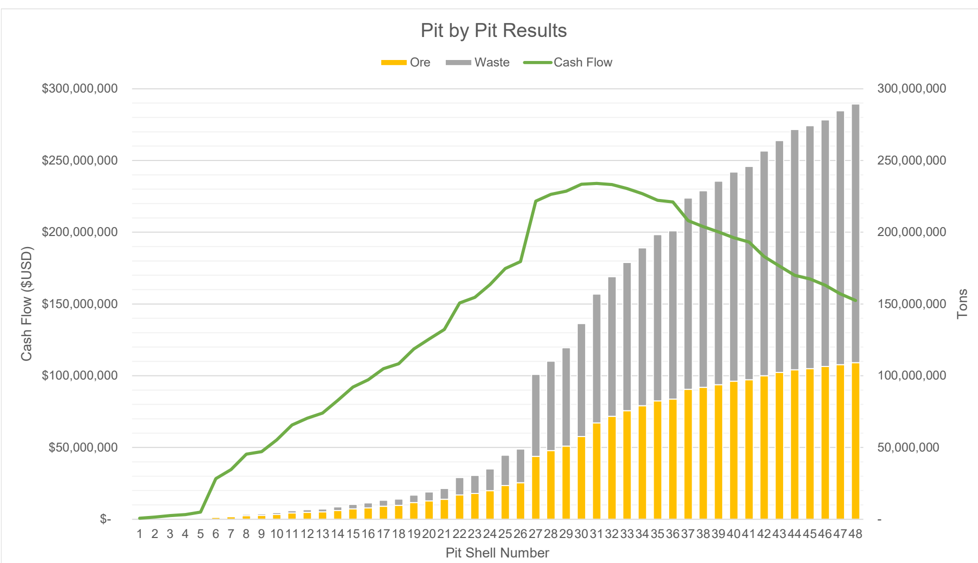
Sector	Slope Dip Direction		Bench Face Angle	Catch Bench Width	Overall Slope Angle
	Lower Bound	Upper Bound			
1	150	210	60	26.3	45.2
2	210	270	48	26.2	37.2
3	270	330	48	26.2	37.2
4	330	30	48	26.2	37.2
5	30	90	61	26.3	45.9
6	90	150	82	26.2	61.0

The pit optimization was conducted from revenue factors 0.10-1.30 in increments of 0.02. The pit-by-pit graph for all pits with a positive cashflow (revenue factors 0.10-1.06) are shown in Figure 16-9. Pit shells were selected based on maximizing the project cash flow and discounted cash flow from the open pit operation.

To ensure sufficient width for equipment in terms of safety and productivity, a minimum mining width was applied to the selected pit shells (Pits 26 and 31). Final tonnage figures will vary when compared to the schedule due to the application of this minimum mining width.

Tonnages from the selected pit shells after applying the minimum mining width are shown in Table 16-7.

Figure 16-9: Pit-by-Pit Optimization Results



COGs of 0.096% and 0.098% copper for oxidized and enriched material, respectively, were calculated based on the processing costs, recovery, and dilution parameters in Table 16-2.

16.2.2 Mining Equipment

Since the open pit operation has a limited life (see Table 16-8 for production schedule) it is envisioned that the mining operation will be operated with a contract mining fleet. This will increase the unit operating cost to some extent but will reduce mining capital requirements significantly.

Based on the required production rate, the primary equipment fleet will consist of a fleet of rigid dump trucks in the 100-150 short ton range. Loading equipment will consist of at least two digging units in the mine, assisted by a wheel loader. The sizing of these machines will be determined by the specifications of the haul truck fleet as well as the actual rock conditions.

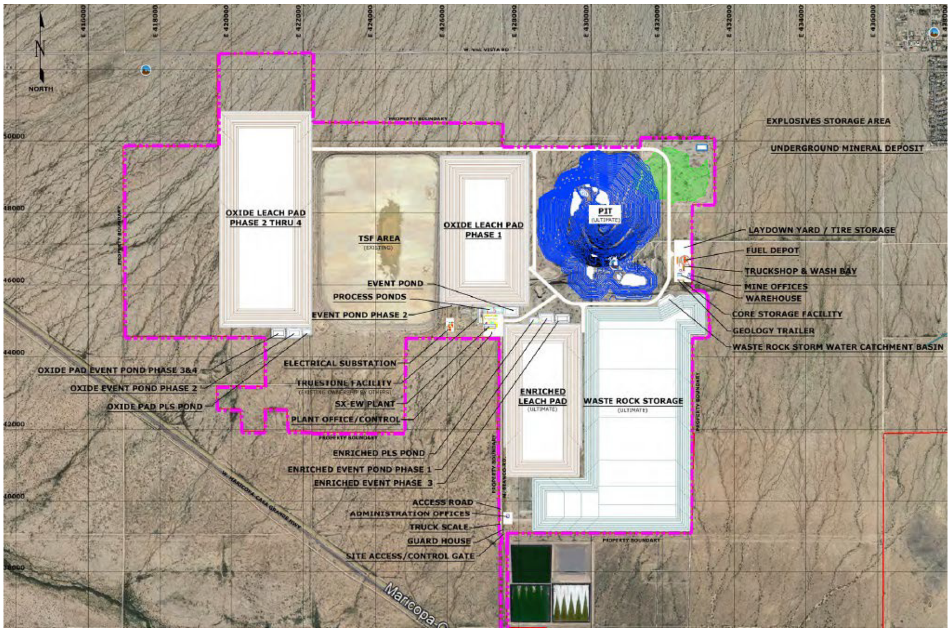
The primary fleet will be complemented by a fleet of ancillary machines consisting of at least two track dozers, one road grader, one wheel dozer, one water truck and drill and blast equipment.

16.2.3 Mining the Stockpile Project

Per the Stockpile Project PEA dated 10 March 2020 and given the cycle routing between Stockpile Project and open pit mining, it is envisioned that a separate, smaller fleet would be used to mine the Stockpile Project. This allows for direct haul to leach pad facilities while keeping larger mining equipment dedicated to overburden stripping of the pit.

Haul trucks will travel approximately 8,000 ft (2,438 m), on average, to the leach pad, with waste materials rehandled within the current Stockpile Project footprint. The haul trucks will use a maintained dirt haulage road to move material to the leach pad, placing material in lifts (refer to Figure 16-10).

Figure 16-10: Site General Arrangement



Material will be removed from the Stockpile Project from a series of sequenced production faces. After each cycle, each face will be sampled, and those samples will be sent to the lab for sample preparation and assays. The face will sit stagnant until the CuAS results are received from the lab, then the material will be directed to the correct dump point on the waste pile or leach pad. Three or more production faces will be in rotation to allow time for the assay checks without disturbing a continual feed to the leach pad.

Total waste tonnage per lift is illustrated in Table 16-4. As material is identified as leach pad feed or waste through sampling and assaying, short range mine planning activities will be updated regularly as new information is available to reduce the amount of waste re-handle.

Table 16-4: Total Waste Tonnage Per Lift

Lift No.	Leach Material (t)	Strip Ratio	Waste (Mt)	Waste/Tons to be Removed	
				% Waste	Mt
4	0.563	0.024	0.013	100%	0.013
3	36.000	0.160	5.800	60%	3.600
2	30.200	0.910	27.700	30%	8.000
1	14.700	1.130	16.600	68%	11.500
Total Material	**81.200**	**0.620**	**50.100**	**45%**	**22.800**

As illustrated in Table 16-4, all waste encountered in Lift 4 (upper lift) is required to be handled to the designated waste area to ensure subsequent lifts are available for mineralized material release.

For Lift 3, approximately 60% of the waste material will be required to be re-handled to a designated waste area. For Lift 1 (lower lift), material that is under any waste that is left in place from Lift 2 will be sampled using short range mine planning activities coordinating with mine operations to develop drop-cuts as required. This will drop the Lift 3 mining elevation to expose the material to be sampled, assayed, and kriged. Subsequent leach pad feed determinations that, at a minimum, meet COG criteria will determine if the waste material from Lift 2 will be required to be moved to access the leach pad feed in Lift 1. It is currently envisioned that approximately 50% of the modeled waste will be required to be excavated to allow leach pad feed extraction below from Lift 1.

Any waste that is encountered in Lifts 2 and 1 will ideally be left in place except for material that may need to be removed for optimizing haulage and reducing operating costs. It is currently estimated that approximately 30% of this waste will need to be handled and placed in the designated waste area.

All activities will be performed by a contractor; therefore, modifications to this method may include equipment selection changes and discharge changes.

16.2.4 Pit Processing Cutoff Grade Refinement

The copper processing COG calculation was further refined after the pit shells were chosen for the schedule. Further refinement included the acid and cyanide portion split along with updated recovery factors for each of the acid and cyanide portion. The processing cost is used as proxy for COG, which is used to determine whether to send rock to the plant or waste dump. As such, the material only needs to cover the processing cost and other directly attributable cost to be sent to the plant. The mining cost does not impact this decision.

Copper processing COG calculations for the oxide and enriched material are summarized in Table 16-5.

Table 16-5: Processing Cutoff Grade Calculation Parameters

Item	Oxide Material	Enriched Material
Dilution Factor	1.05	1.05
Process Cost ($US/t)	2.21	1.25
Acid Portion (%)	80.7%	27.3%
Acid Recovery (%)	90.0%	90.0%
Cyanide Portion (%)	19.3%	72.7%
Cyanide Recovery (%)	72.0%	72.0%
Value ($US/t)	6,300	6,300
Selling Cost ($US)	80	80
Net Value ($US)	6,220	6,220
COG (Cu%)	**0.043%**	**0.027%**

The equation for the processing COG for both the oxide and enriched material are as follows. The primary material follows an overall recovery estimate as shown in Table 16-6.

Table 16-6: Recovery and Value of Primary Material

Item	Primary Material
Dilution Factor	1.05
Process Cost ($US/t)	8.50
Overall Processing Recovery (%)	92%
Value ($US/t)	6,300
Selling Cost ($US)	80
Net Value ($US)	6,220
COG (Cu%)	0.156%

16.2.5 Pit Resource

Applying the refined COG to the exported pit shell from Whittle, the HxGN Reserve Tool calculated the resources as listed in Table 16-7. A 5% dilution and a 95% mining recovery, consistent with the same assumptions used in Section 16.2.1, is incorporated in Table 16-7.

Table 16-7: Material Class and Grade

Resource Class	Diluted Tons	Diluted Grade (%)
Mineralized Material –Indicated		
Oxide	9,720,000	0.1875
Enriched	7,480,000	0.4386
Primary (Non-Leachable)	1,240,000	0.3108
Indicated Subtotal	18,440,000	0.2976
Mineralized Material –Inferred		
Oxide	37,100,000	0.1843
Enriched	15,660,000	0.4181
Primary (Non-Leachable)	580,000	0.3179
Inferred Subtotal	53,340,000	0.2544
Total Leachable Material	69,960,000	0.2642
Total Material	71,780,000	0.2655
Waste	101,890,000	-
Total Material Mined	173,670,000	-

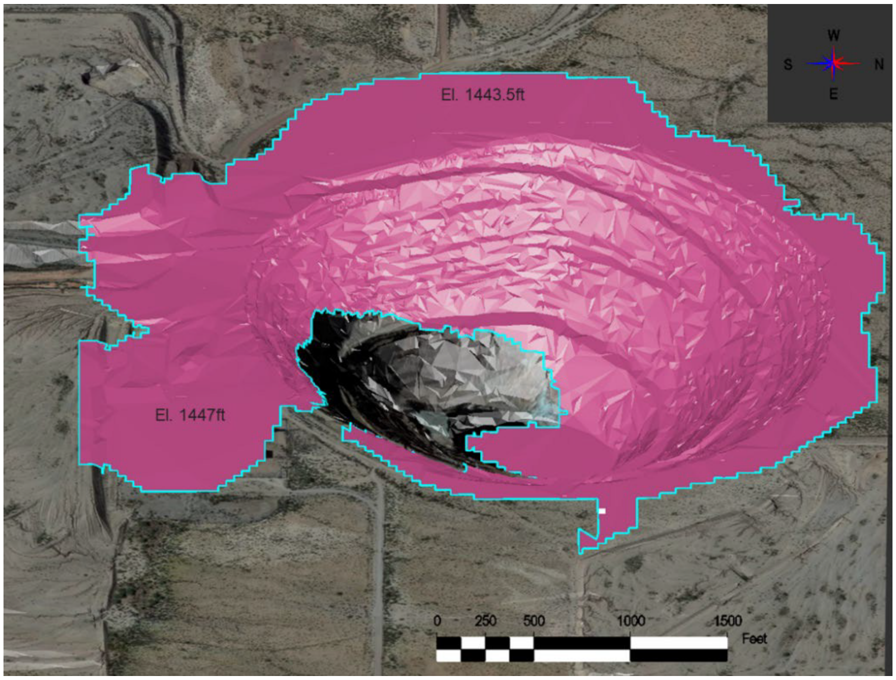
16.2.6 Pit Sequence

The pit material movement directs the various material to its associated location as follows.

- Oxide and Enriched Mineralized Material → Leach Pad
- Primary Mineralized Material → Primary Stockpile
- Waste Storage Facility

The two selected pit shells from Section 16.2.1 were then imported into HxGN Mine Sight Schedule Optimizer to produce a pit sequence and schedule. The pit shells are considered as Phase 1 and Phase 2, with Phase 2 being the ultimate pit shell. Although the widths to accommodate have been incorporated, a detailed pit design has not been completed for this report. As such, access and benches have not been designed into the pit sequence and schedule. Mining benches are assumed to be 30 ft (9 m) high. The initial surface is shown in Figure 16-11.

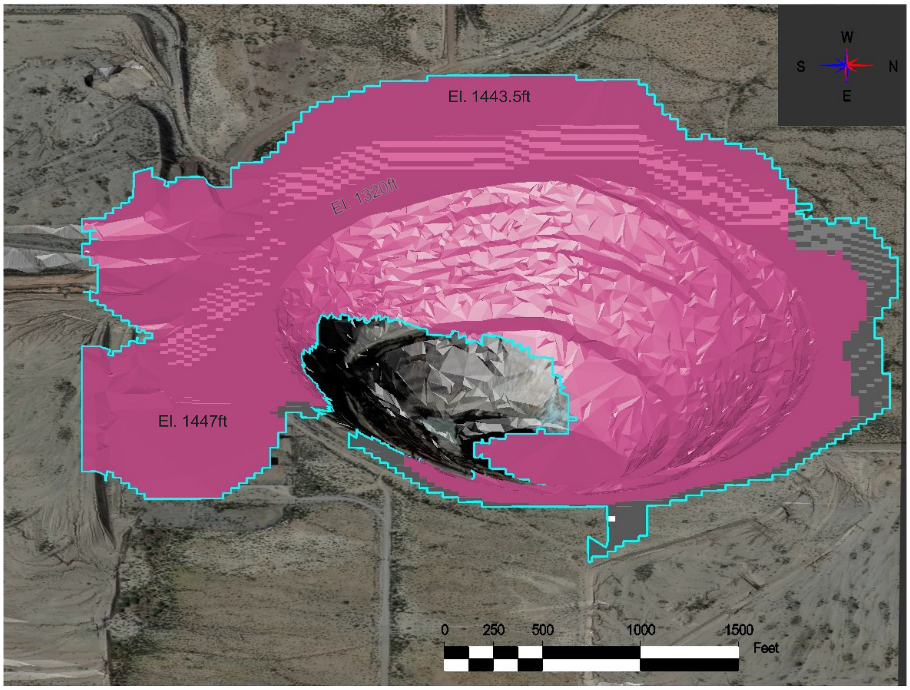
Figure 16-11: Mine Layout at Commencement



Year 1 (2023)

Pre-stripping of waste occurs in Year 1 (2023) at 13.9 million tons with minimal mineralized material release (0.4 million tons). Mineralized material tons mentioned in this year and all subsequent years have a dilution factor of 5% incorporated. Phase 1 pit shell is mined to elevation 1,320 ft (402 m). Figure 16-12 illustrates the end of period pit for Year 1 (2023).

Figure 16-12: Open Pit at End of Y1

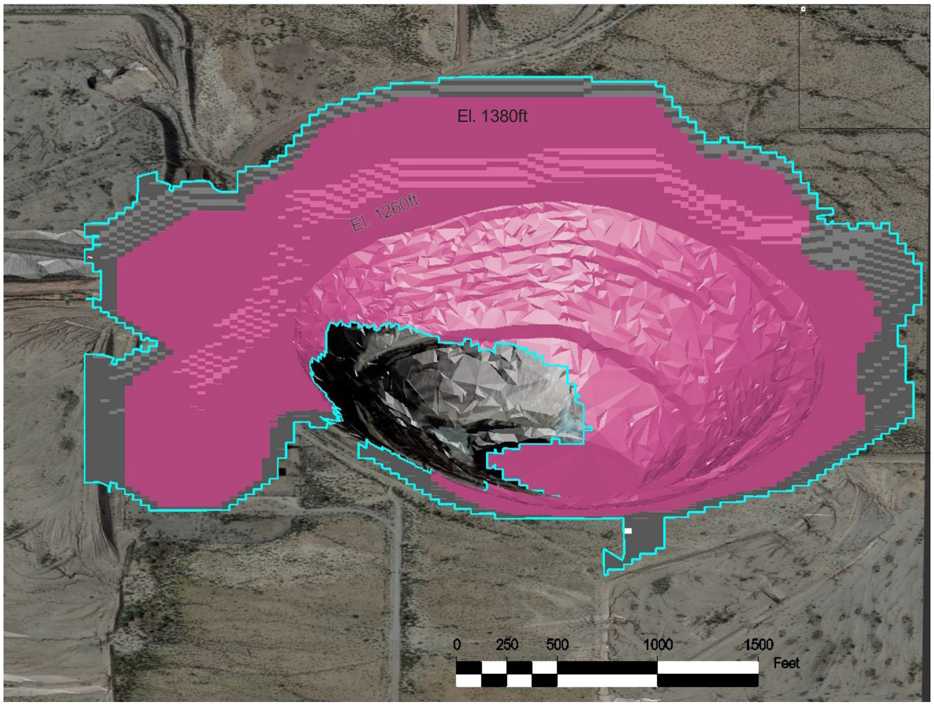


Year 2 (2024)

Waste removal ramps up to 17.5 million tons. Mineralized material release increases to 1.29 million tons as Phase 1 is mined to elevation 1,260 ft (384 m) and Phase 2 is mined to elevation 1,380 ft (421 m). A total of nine vertical benches are mined. This vertical advance rate (or sinking rate) was chosen to reflect the difficulty of mining the pit geometry.

Figure 16-13 illustrates the end of the period for the pit in Year 2 (2024).

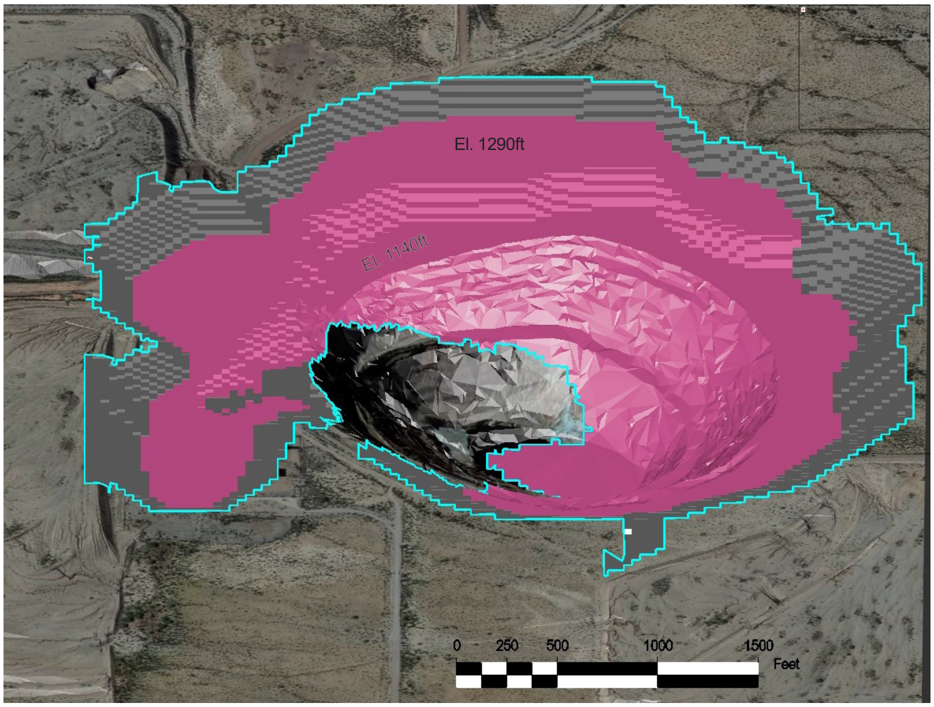
Figure 16-13: Open Pit at End of Y2



Year 3 (2025)

Peak waste removal is reached with 18.6 million tons for the year. Mineralized material release increases to 3.1 million tons. Phase 1 is mined to elevation 1,140 ft (347 m) and Phase 2 is mined to 1,290 ft (393 m). Due to the difficulty of the geometry, a total of nine vertical benches are mined. Figure 16-14 illustrates the end of the period for the pit in Year 3 (2025).

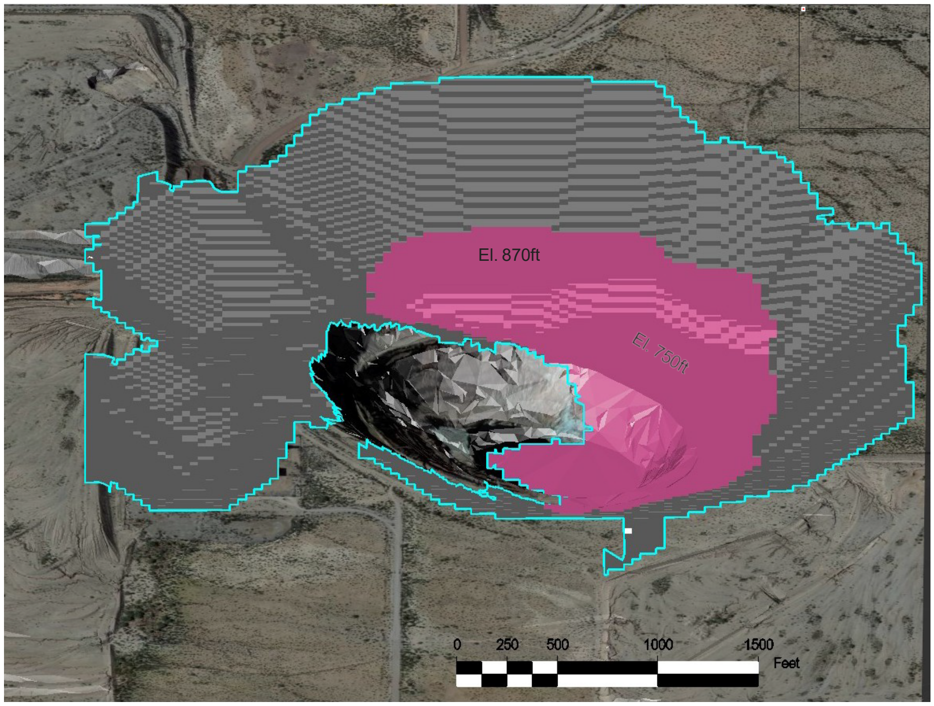
Figure 16-14: Open Pit at End of Y3



Year 4-7 (2026-2029)

As waste removal tons decreases mineralized material release increases. Average waste tons in Year 4-7 (2026-2029) is 11.7 Mtpa and the average mineralized material tons release is 8.4 Mtpa. Phase 1 is mined to elevation 750 ft (229 m) and Phase 2 is mined to 870 ft (265 m). Vertical mining is capped at nine benches where applicable. Figure 16-15 illustrates the end of the period for the pit in Year 7 (2029).

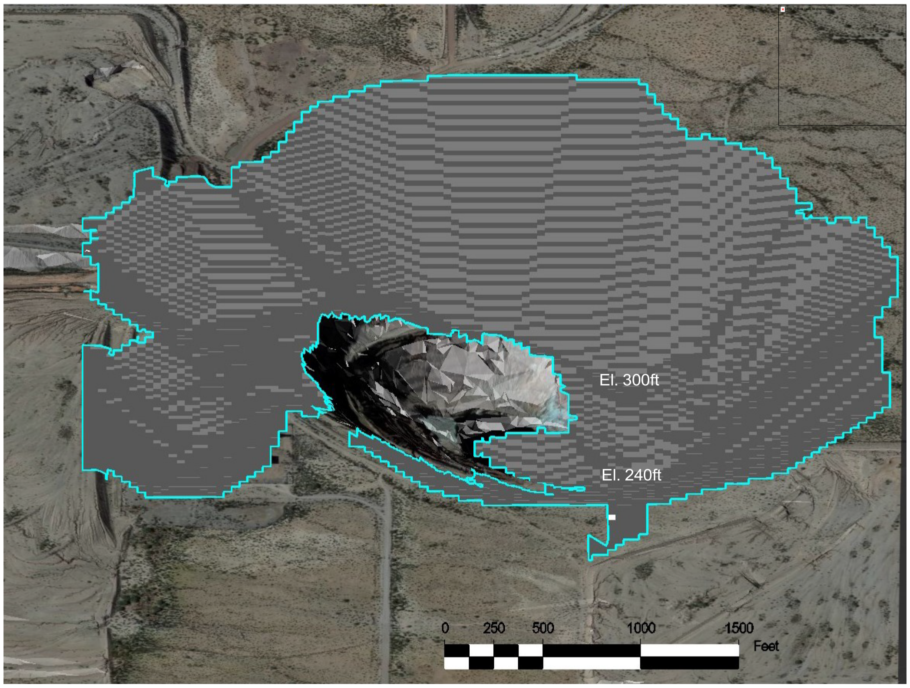
Figure 16-15: Open Pit at End of Y7



Year 8-13 (2030-2035) End of Pit

Waste rock mining during this period is minimal since the majority of material is ore. Average waste removal is 0.9 Mtpa and average mineralized material release is 5.2 Mtpa. Pit mining is completed to Elevation 240 ft (73.2 m). Figure 16-16 illustrates the completion of the pit in Year 13 (2035).

Figure 16-16: Open Pit at End of Y13



16.2.7 Pit Schedule

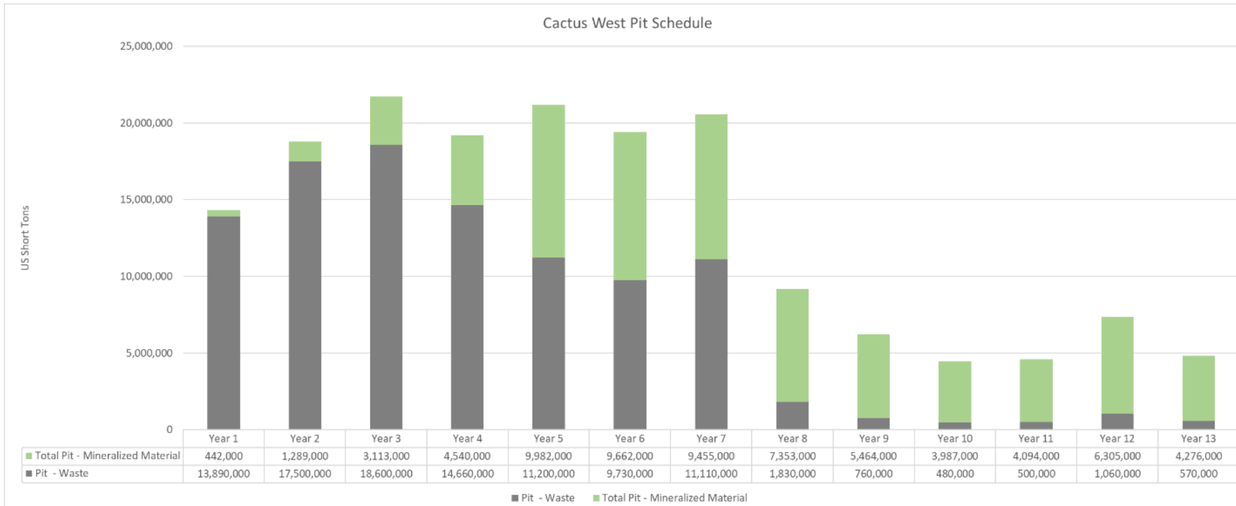
Table 16-8 shows the pit release schedule by material. Enriched mineralized material and oxide mineralized material are directly delivered to the plant while primary mineralized material is stockpiled. All waste material will go to the waste storage facility. See Section 16.4 for the full combination schedule with the underground and external stockpile components.

Table 16-8: Mine Plan

		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	Total
Pit - Enriched - Mineralized Material	Mineralized Material Tons	42,000	459,000	1,023,000	2,390,000	1,432,000	3,362,000	2,735,000	903,000	1,194,000	2,397,000	3,014,000	1,715,000	2,466,000	**23,132,000**
	TECU Grade* (%)	0.2060	0.3958	0.3711	0.2560	0.3621	0.3346	0.3596	0.6018	0.6229	0.6011	0.4805	0.4299	0.4473	**0.4248**
	Contained Cu (t)	90	1,800	3,800	6,100	5,200	11,200	9,800	5,400	7,400	14,400	14,500	7,400	11,000	**98,090**
	Recoverable Cu (t)	70	1,400	2,900	4,700	4,000	8,700	7,600	4,200	5,700	11,100	11,100	5,700	8,500	**75,670**
Pit - Oxide - Mineralized Material	Mineralized Material Tons	400,000	830,000	2,090,000	2,150,000	8,550,000	6,300,000	6,720,000	6,450,000	4,270,000	1,590,000	1,080,000	4,590,000	1,810,000	**46,830,000**
	TECU Grade* (%)	0.1107	0.0901	0.1039	0.1253	0.1593	0.1864	0.1933	0.2072	0.2279	0.2167	0.2144	0.2005	0.2294	**0.1850**
	Contained Cu (t)	440	700	2,200	2,700	13,600	11,700	13,000	13,400	9,700	3,400	2,300	9,200	4,200	**86,540**
	Recoverable Cu (t)	380	600	1,900	2,300	11,800	10,200	11,200	11,600	8,400	3,000	2,000	8,000	3,600	**74,980**
Total Pit - Mineralized Material	Mineralized Material Tons	442,000	1,289,000	3,113,000	4,540,000	9,982,000	9,662,000	9,455,000	7,353,000	5,464,000	3,987,000	4,094,000	6,305,000	4,276,000	**69,962,000**
	TECU Grade* (%)	0.1198	0.1990	0.1917	0.1941	0.1884	0.2380	0.2414	0.2556	0.3143	0.4478	0.4103	0.2629	0.3551	**0.2642**
	Contained Cu (t)	500	2,600	6,000	8,800	18,800	23,000	22,800	18,800	17,200	17,900	16,800	16,600	15,200	**185,000**
	Recoverable Cu (t)	450	2,000	4,800	7,000	15,800	18,900	18,800	15,800	14,100	14,100	13,100	13,700	12,100	**150,650**
Pit – Primary - Stockpile	Mineralized Material Tons	0	0	0	0	100	74,300	40,200	126,000	116,500	209,300	516,700	237,400	504,700	**1,825,200**
	TECU Grade* (%)	0.0000	0.0000	0.0000	0.0000	0.4245	0.3609	0.3063	0.2851	0.3134	0.3164	0.3027	0.3405	0.3099	**0.3131**
	Contained Cu (t)	0	0	0	0	0	300	100	400	400	700	1,600	800	1,600	**5,900**
	Recoverable Cu (t)	0	0	0	0	0	247	113	330	336	609	1,439	744	1,439	**5,257**
Pit - Waste	Waste Tons	13,890,000	17,500,000	18,600,000	14,660,000	11,200,000	9,730,000	11,110,000	1,830,000	760,000	480,000	500,000	1,060,000	570,000	**101,890,000**

Figure 16-17 depicts the schedule of mineralized material and waste release.

Figure 16-17: Mining Schedule



	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
Total Pit - Mineralized Material	442,000	1,289,000	3,113,000	4,540,000	9,982,000	9,662,000	9,455,000	7,353,000	5,464,000	3,987,000	4,094,000	6,305,000	4,276,000
Pit - Waste	13,890,000	17,500,000	18,600,000	14,660,000	11,200,000	9,730,000	11,110,000	1,830,000	760,000	480,000	500,000	1,060,000	570,000

16.2.8 Pit Schedule Risks

Some risks are associated with the assumptions made for the schedule. The schedule is based on the exported pit shells, selected for optimal value, and not on detailed pit designs. This may result in an increase of waste tons and/or a decrease of mineralized material tons once pit design benching and access are included.

The mining sequence using the pit shell phase geometry may not be optimal for productive mining results. The pit shell does aim for a minimum 120 ft (37 m) working width but there may be areas that are narrower than 120 ft (37 m) due to the rough nature of the pit shells. Further study and more detailed design are needed to increase confidence in the mining production rate.

16.3 Underground Mining

The remaining mineral resource available in Cactus East was evaluated as an underground mine. For the purposes of this evaluation, the underground mining method will be TLS.

16.3.1 Mine Design – Development and Production

Table 16-9 lists all the key parameters used for the underground development design and TLS.

Table 16-9: Mine Design Criteria

Mine Development	Value	Units
Designed Maximum Gradient	+/- 13	%
Minimum Curve Radius	50	ft
Development – Mineralized Material and Waste	15 × 15	ft
Min Raisebore Length	225	ft
Internal Ventilation Drop Raises	8	ft
Internal Vent Raise Minimum Angle	75	degrees
Surface Ventilation Raises	16.5	ft
Egress Raise	Same as Vent Raise	ft
Minimum Pillar between Drifts	$2^1/_2$	
Transverse Longhole Stoping	**Value**	**Units**
Average Standoff Distance FW Drift to Mineralized Zone	65	ft
Sublevel Interval	75	ft
Stope Length (along strike)	50	ft
Open Stope Height	75	ft
Minimum Stope Width (FW to HW)	30	ft
Maximum Width of Open Stope (FW to HW)	75	ft
Drill Drift (Top Cut) Dimensions	15 × 15	ft
Stope Sequencing	Primary / Secondary	
Stope Access Drift Spacing (along sublevel)	100	ft

16.3.2 Mining Sequence

The top of the underground deposit, Cactus East, is roughly 800 ft (244 m) below the surface and extends an additional 1,000 ft (305 m) vertically. The deposit averages 800 ft (244 m) in thickness, from hanging wall to footwall. TLS with cemented rockfill (CRF) for primary stopes and unconsolidated rockfill (URF) for secondary stopes, was selected as the preferred mining method. The secondary stopes will be partially filled with CRF to build the bulkhead on the lower sill and then the remaining void can be filled with URF. The mining unit cost used is inclusive of this type of backfilling.

The mine plan is expected to ramp up to an initial production rate of 3,500 tpd and reach a daily production of 7,000 tpd for several years before end of mine life. To achieve this production rate, the deposit will be split into two mining horizons. Given the size of the deposit, both laterally and vertically, each mining horizons will be capable of 3,500 tpd.

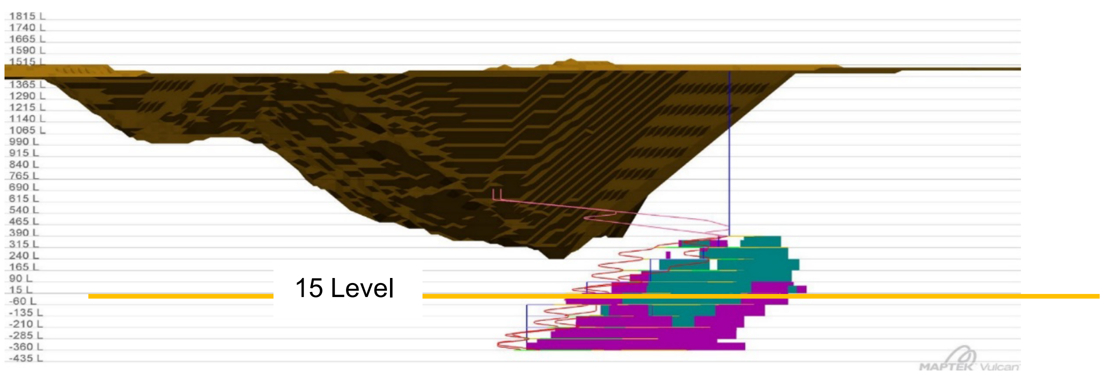
To access the underground mine, twin declines will be developed from the wall of the new open pit. Due to the high daily production rate required, the declines will use one way traffic to minimize traffic congestion. Preproduction development will excavate the twin declines down to the center of the deposit and split to opposite ends of the deposit.

Once the top sublevel is established, the main ventilation raise can be driven to surface. Dual internal ramps will be driven down to the midpoint of the deposit (15 Level).

The 15 Level will define the first horizon. Ventilation from the initial vent raise will be carried down through the sublevels from the top level to the first horizon. Production of the initial stopes will begin once the ventilation circuit is established. All the mined-out stopes on the 15 Level will be filled with CRF to establish a sill pillar and separate the two mining horizons within the mine. While production mining on the 15 Level begins, development of the two internal ramps will continue to the lowest level where the second mining horizon can begin.

Figure 16-18 shows the underground mine design with the two mining horizons.

Figure 16-18: Underground Life-of-Mine Design – Long Section – Looking West



16.3.3 Underground Cutoff Grade

The copper COG calculations for the oxide, enriched and primary material are summarized in Table 16-10.

Table 16-10: Cutoff Grade Parameters

Item	Transverse Stoping Oxides –Leach	Transverse Stoping Sulfides (Enriched) – Leach
Copper Price ($/lb)	$3.15	$3.15
Copper Refining Cost ($/lb)	$0.04	$0.04
Mining ($/t)	$28.93	$28.93
Crushing and Process ($/ore ton)	$2.21	$1.25
G&A ($/ore ton)	$2.05	$2.05
Surface Haulage ($/ore ton)	$0.30	$0.30
Royalty %	3.2%	3.2%
Copper Recovery (%)	90.00%	72.00%
Copper Payable (%)	99.90%	99.90%
COG (Cu%)	**0.60%**	**0.73%**

The underground COG (CuT) of 0.85% Cu was chosen to optimize the mine life by targeting the higher grade material within the underground resource.

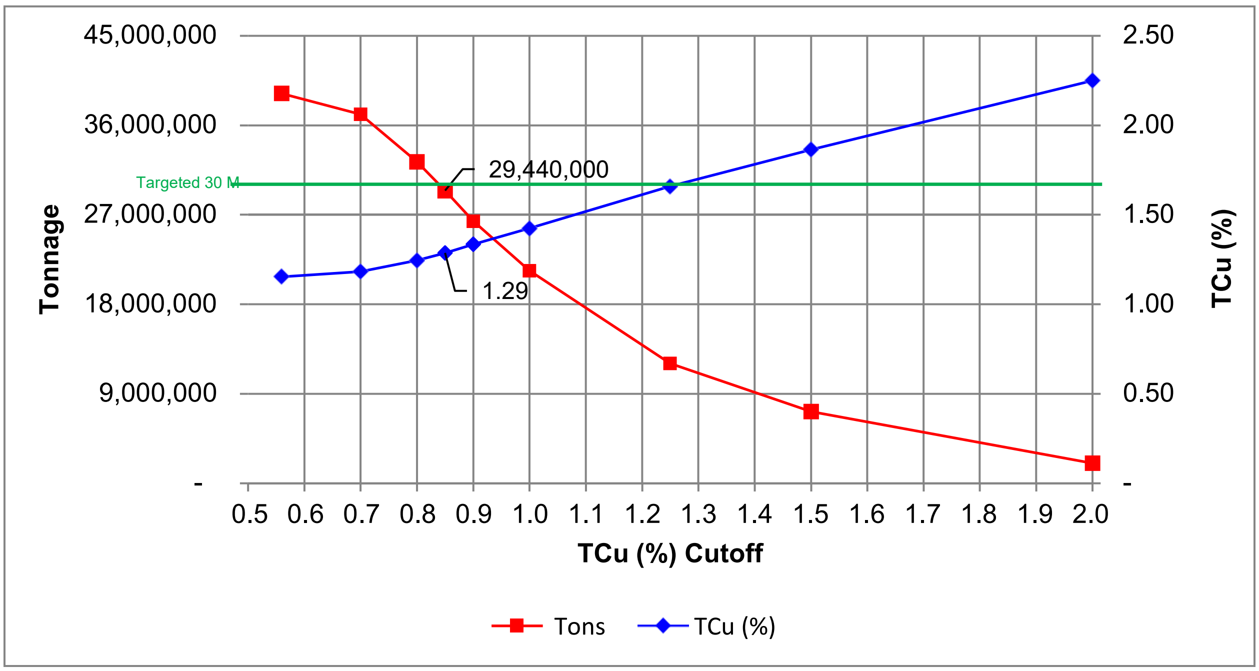
16.3.4 Underground Resource

Using Vulcan Mining Stope Optimizer software (MSO), transverse stope shapes were generated for the oxide and enriched material at their respective COGs. A grade sensitivity analysis was run on the generated stopes to further optimize the grade and tonnage combination. Table 16-11 and Figure 16-19 show the results of the analysis.

Table 16-11: Stope Grade Sensitivity Result Table

CuT Cutoff (%)	Tons	Cu Grade (%)
0.56	39,200,000	1.15
0.70	37,115,000	1.18
0.80	32,330,000	1.25
0.85	**29,440,000**	**1.29**
0.90	26,360,000	1.34
1.00	21,400,000	1.43
1.25	12,080,000	1.66
1.50	7,220,000	1.86
2.00	2,030,000	2.25

Figure 16-19: Stope Grade Sensitivity Grade Tonnage Curve

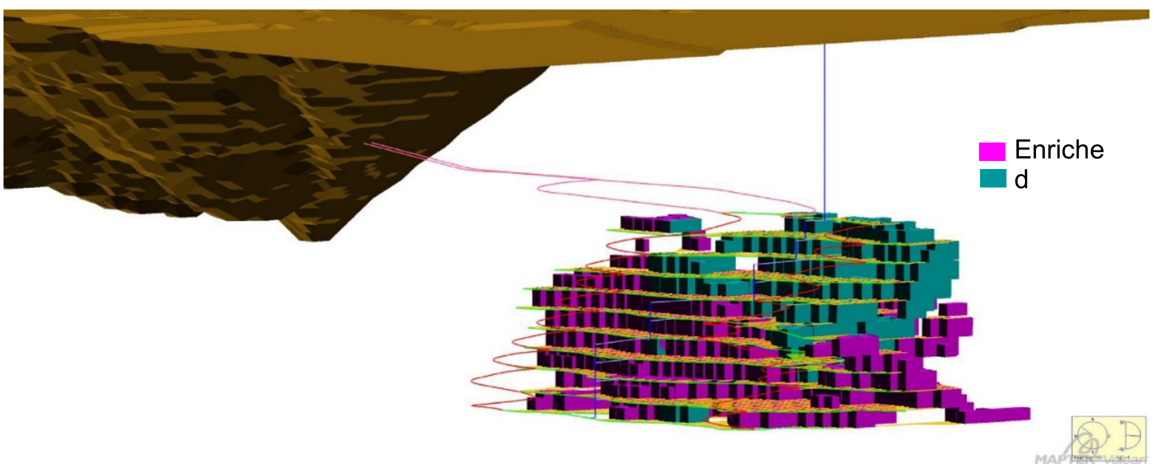


The 0.85% cutoff was chosen as the base case for this study as it was closest to the 30 million tons requirement for the underground deposit while optimizing cash flow for the underground resource. The 30 million tons was based on processing constraints, production rate and mine life. The resulting LOM development designs and production schedule were based on the 0.85% cutoff scenario. Table 16-12 lists the 0.85% Cu cutoff tons and grade by rock type. Figure 16-20 illustrates the enriched and oxide stopes, with development, within the underground deposit. For the scoping level study, no mineralized material loss or dilution factors were determined for the underground mine.

Table 16-12: 0.85% Cu Cutoff by Rock Type

Rock Type	Tons	CuT Grade (%)
Enriched	22,580,000	1.33
Oxide	6,860,000	1.15
Total	**29,440,000**	**1.29**

Figure 16-20: Underground Life of Mine Design – Isometric



16.4 Combined Life of Mine Schedule

Figure 16-21 and Figure 16-22 graphically show the overall combined mineralized material and waste tons and combined LOM schedule, respectively.

Figure 16-21: Combined Mineralized Material and Waste Breakdown

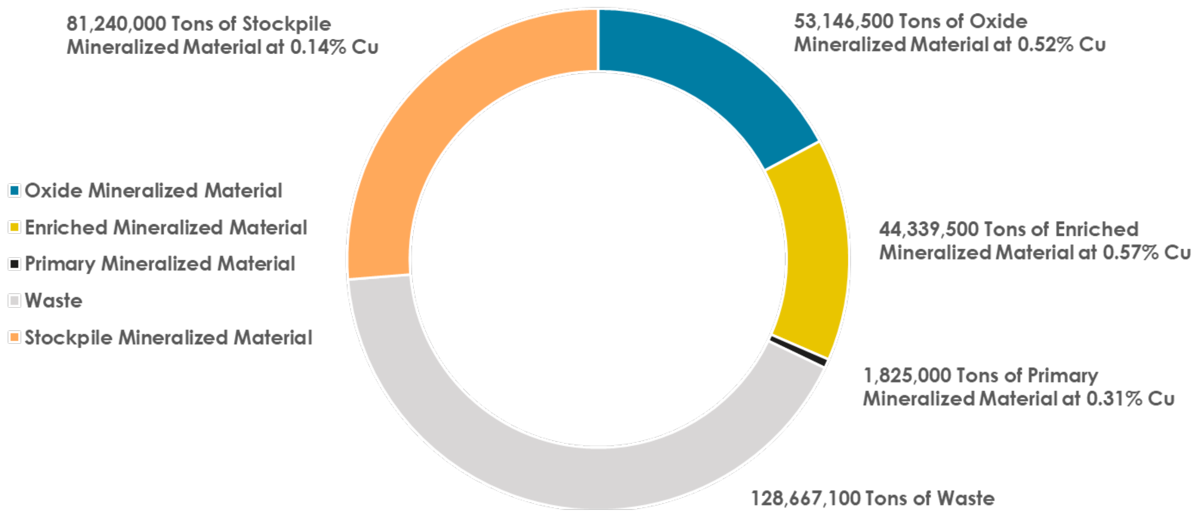
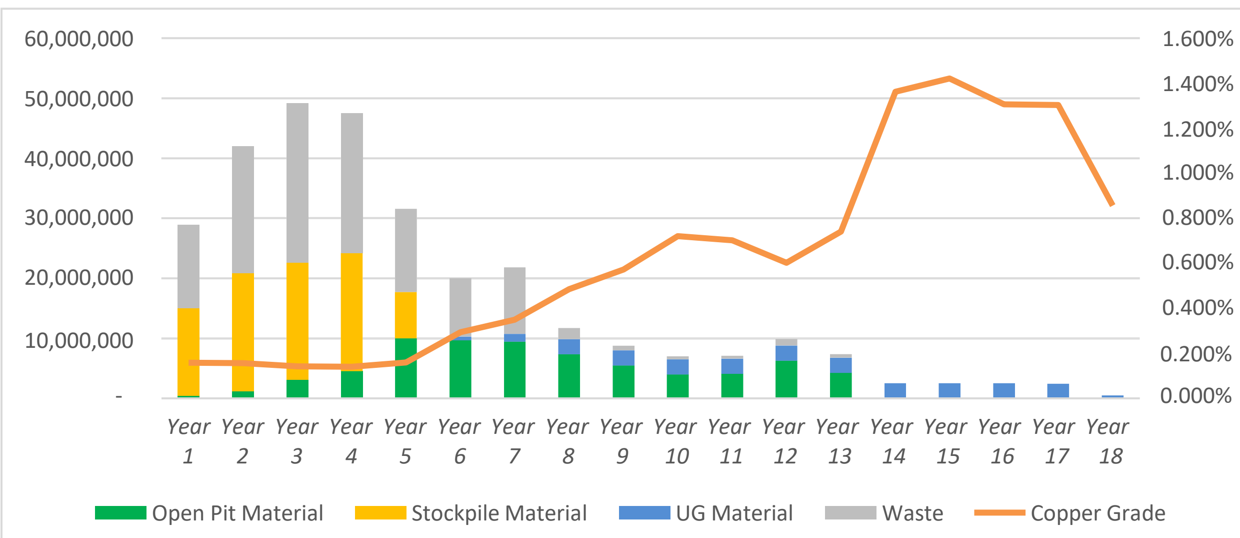
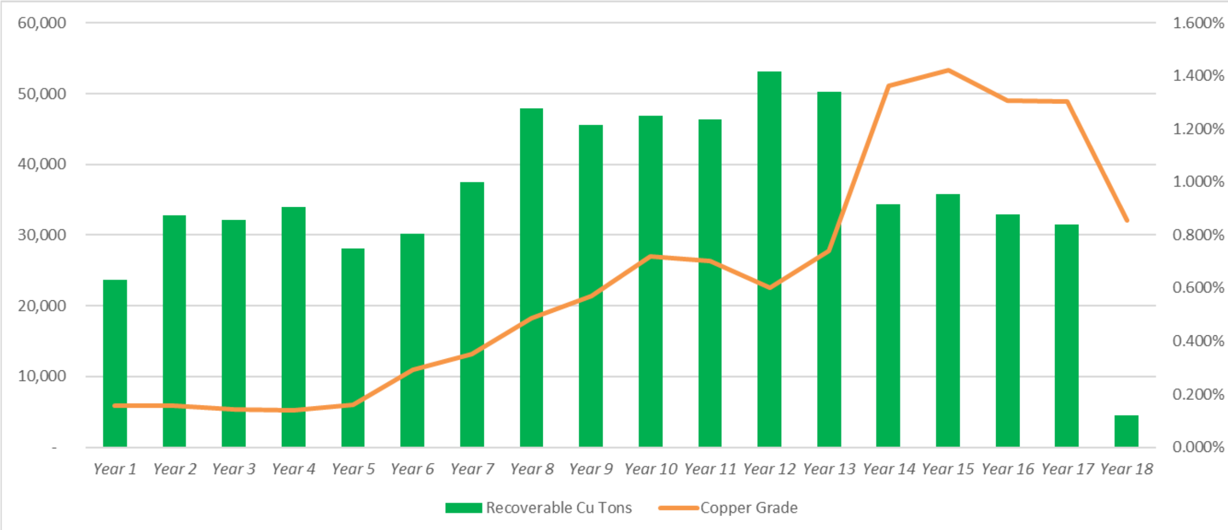


Figure 16-22: Combined LOM Production Schedule



The combined recoverable copper tons LOM schedule is shown in Figure 16-23. A maximum 50,000 t of mined recoverable copper metal per year was used as a target for the LOM scheduling.

Figure 16-23: Combined Mined Recoverable Copper Tons Schedule



Appendix E shows the combined open pit, Stockpile Project, and underground LOM schedule.

17.0 RECOVERY METHODS

The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as at 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report.

17.1 Process Plant Description and Flow Sheet

Potential resources considered in this report are related to the existing mine Stockpile Project built during the development and operation of a copper open pit and milling facility from 1974 to 1984, an expansion of the existing open pit and a new underground mine extension to extract deeper resources in the deposit.

The potential resource processing sources include oxide and sulfide enriched material containing primarily copper mineralization. The materials are believed to be suitable for treatment in a conventional ROM heap leach, SX and EW process facility to produce copper cathodes at LME Grade A quality standards ASTM B115-10 – Cathode Grade 1.

Oxide resources will be processed on a lined leach pad suitable for H2SO4 leaching and enriched materials will be processed on a second leach pad employing bioleaching and H2SO4 technology.

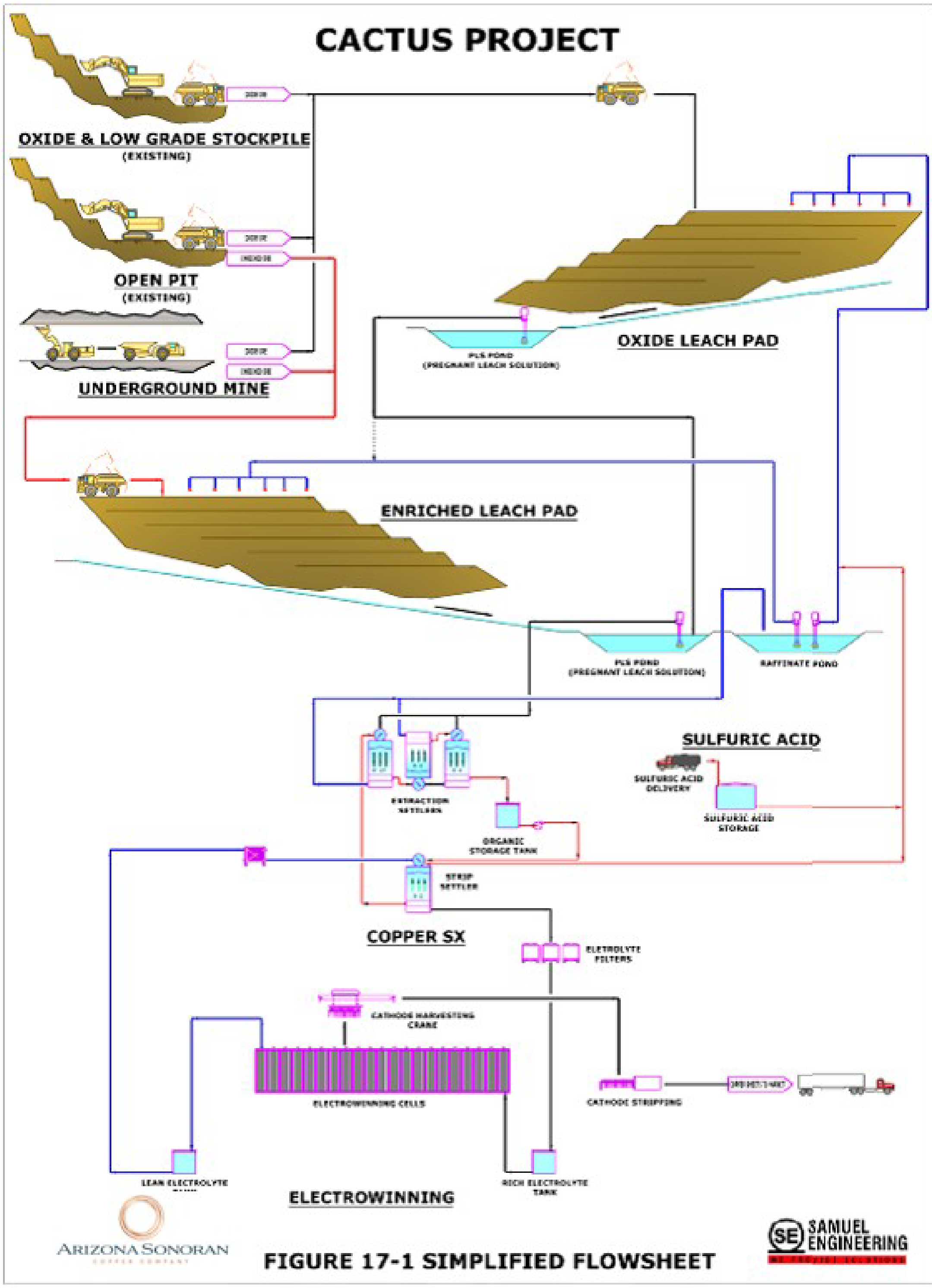
A summary of the resources considered for processing by source is provided in Table 17-1.

Table 17-1: Potential Leach Materials Distribution

MiningSource	Material Type	Leach Material (t)	Grade % TSol (% Cu)	LeachableCu (t)	Distribution Percent	
					Material	Cu
StockpileProject	Oxide	82,331,000	0.141	116,279	100	100
Open Pit	Oxide	46,810,000	0.190	88,939	67	48
	Enriched	23,131,000	0.420	97,150	33	52
Underground	Oxide	6,317,000	1.180	74,271	23	21
	Enriched	21,208,000	1.260	274,597	77	79
Total	**Oxide**	**135,458,000**	**0.203**	**274,705**	**75**	**43**
	Enriched	**44,339,000**	**0.822**	**364,371**	**25**	**57**
	Total	**179,797,000**	**0.355**	**639,076**		

The initial Project envisions oxide processing from resources mined from the existing Stockpile Project with oxide and enriched resources leached as the open pit and underground resources are developed and mined. The conceptual flow diagram for the processing facilities included in the Project is presented in Figure 17-1. A modularized plant design and construction is considered.

Figure 17-1: Processing Facilities (Conceptual Flow Diagram)



17.1.1 Heap Leaching

Leach material mined from the Stockpile Project and new mining operations will be placed in 20 ft (6 m) lifts on lined heap leach pads depending on an oxide or enriched designation based on soluble copper sequential assay.

Oxide material mined from the Stockpile Project is expected to be relatively fine (approximately 80% -1-inch based on bulk sampling) and freshly mined material from open pit and underground operations will be blasted to a -4-inch top size. The initial oxide materials pad is 8.5 million ft2 (790 thousand m2) to hold approximately 40 million tons of leach material, approximately 2-3 years of mined material. Initial leach material is predominantly coming from the Stockpile Project with some open pit contribution as pre-stripping activities are initiated.

As enriched material is encountered in sufficient quantities, a second leach pad will be constructed for this material. A leach pad to hold approximately 6 million tons of enriched materials is planned for operation in Year 2 to allow for sufficient materials to be mined and will be built as part of the initial project installations. The capacity of the enriched pad is sufficient for the initial 5-6 years of material feed.

Material will be “as mined” from the new mining operations with no additional crushing or handling and stacked with mine trucks using an end dumping methodology. Mine blasting protocols will be evaluated to ensure a minimal occurrence (10%-15%) of plus 4-inch materials.

Placement of materials on the leach pads will be by truck dump and push methods. Surfaces will be ripped, and cross ripped to a depth of 6 ft (2 m) to minimize surface compaction and surface permeability degradation. Fresh materials will be placed over previously leached materials, in 20 ft (6 m) lifts. The height of the leach material on the pad will eventually reach 200 ft (61 m) in overall height. The leaching sequence for the oxide and enriched pads is planned as follows in Table 17-2.

Table 17-2: Average Leach Cycle Times by Material Type

Leach Cycle Component	Oxide Leach Pads (days)	Enriched Leach Pads (days)
Pad Loading	14	14
Surface Preparation / Piping	7	7
Active Solution Application	90	180
Drain Down and Decommissioning	9	9
Minimum Total Cycle Time	120	210

Leaching solutions, containing dilute H2SO4 (5-10 g/L H2SO4) will be pumped and applied to the top of each lift and allowed to percolate though the copper leach material. Solution application is planned to be by a combination of sprinklers and drip emitters. The planned solution application rate for oxide materials is approximately 0.01 gpm/ft2. The solution application rate planned for enriched materials is 0.005 gpm/ft2 allowing for slower, bioleaching of sulfide minerals.

Copper recovery from the leached materials considered is presented in Table 17-3.

Table 17-3: Average Copper Recovery by Material Type

Leach Material Component	Net Copper Recovery
Oxide Materials	
CuAS Copper Content	90%
CuCN Copper Content	40%
Enriched Materials	
CuAS Copper Content	90%
CuCN Copper Content	72%

Since mineralized material placement occurs over a year's time in the mine production plan, the last quarter of the year (3 months) is not expected to contribute to the production in the year mined. Recovery has been shifted to the following year to account for the placement and preparation time required in the current estimations.

The expected materials placement and copper production schedule is presented in Table 17-4.

Table 17-4: Leach Pad Plan and Estimated Copper Production

TOTAL			Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Mining Rate	Tons/y	178,627,940		15,002,000	20,919,000	22,643,000	24,230,000	17,713,940	10,292,005	10,721,316	9,873,043	7,983,981	6,506,962	6,613,997	8,824,997	6,795,979	2,519,996	2,520,027	2,519,956	2,418,515	529,226	-
CuAS	%Cu			0.129%	0.124%	0.107%	0.101%	0.113%	0.168%	0.215%	0.321%	0.192%	0.184%	0.173%	0.253%	0.218%	0.326%	0.342%	0.390%	0.649%	0.151%	
Contained	Tons Cu	305,147		19,414	25,949	24,247	24,442	20,030	17,298	23,069	31,666	15,302	11,944	11,460	22,355	14,808	8,208	8,617	9,833	15,705	798	-
CuCN	%Cu			0.028%	0.032%	0.034%	0.038%	0.045%	0.125%	0.134%	0.164%	0.379%	0.537%	0.528%	0.348%	0.522%	1.035%	1.079%	0.915%	0.653%	0.705%	
Contained	Tons Cu	341,619		4,149	6,674	7,692	9,287	8,013	12,852	14,419	16,234	30,298	34,956	34,940	30,745	35,492	26,092	27,183	23,067	15,798	3,729	-
CuSOL	%Cu			0.158%	0.157%	0.142%	0.140%	0.159%	0.293%	0.350%	0.485%	0.571%	0.721%	0.702%	0.602%	0.740%	1.361%	1.421%	1.306%	1.303%	0.855%	
Contained	Tons Cu	647,553		23,723	32,833	32,120	33,902	28,108	30,150	37,488	47,900	45,600	46,900	46,400	53,100	50,300	34,300	35,800	32,900	31,503	4,527	-
Copper Recovered to Cathode																						
CuAS Recovery				68%	84%	92%	90%	95%	94%	84%	84%	114%	96%	91%	79%	101%	108%	89%	87%	82%	510%	
Recovered	Tons Cu	274,632		13,104	21,884	22,205	21,954	19,020	16,183	19,464	26,565	17,454	11,505	10,423	17,668	15,026	8,873	7,663	8,576	12,813	4,073	180
CuCN Recovery (on-off pad)				31%	42%	50%	54%	66%	62%	65%	62%	62%	70%	72%	73%	69%	78%	70%	74%	75%	123%	
Recovered	Tons Cu	234,112		1,281	2,776	3,856	5,001	5,267	7,979	9,437	10,060	18,878	24,307	25,122	22,375	24,357	20,251	19,139	16,975	11,786	4,593	671
Copper Recovered	Tons Cu	508,744		14,385	24,660	26,061	26,954	24,287	24,162	28,900	36,625	36,332	35,813	35,545	40,043	39,383	29,124	26,803	25,551	24,599	8,666	851
				61%	75%	81%	80%	86%	80%	77%	76%	80%	76%	77%	75%	78%	85%	75%	78%	78%	191%	

17.1.2 SX/EW Processing Plant

The PLS from the heap leach ponds will be pumped for processing in a copper SX/EW plant capable of nominally producing 22,000 tpa of copper cathodes (design maximum of approximately 25,000 tpa) with a design PLS flow to the SX units of up to 3,000 gpm and grade at approximately 4.1 g/L Cu based on an average 92% CuT recovery from PLS to cathodes. The plant layout and critical equipment design will allow for easy expansion to 32,000 tpa production (35,000 tpa maximum) in future.

The design basis for the Cactus SX/EW process plant is a modular facility. Metalex Technologies (METALEX), a company based in Santiago, Chile, designs and supplies small, modular, relocatable standard SX/EW plants for the recovery of copper was contacted for preliminary equipment sizing and costs for this PEA.

METALEX plants are designed to have a low capital cost and be easily transportable, with everything fitting onto trucks or containers for easy transportation of equipment. Materials of construction and equipment sizing for the facility will generally be based on shop fabricated fiberglass reinforced plastic (FRP), high density polyethylene (HDPE), chlorinated polyvinyl chloride (CPVC) or similar materials.

METALEX has based the SX/EW equipment for Cactus on the designs from two other operating facilities, Benkala Copper Mine, and Andacollo. METALEX has endeavored to combine the best features of each to provide Arizona Sonoran with a package with that maximizes amount of preassembly that can be done, thereby minimizing the time needed onsite for field installation.

The SX plant is designed to process up to 3,000 gpm of PLS and be operated in a series-parallel configuration with a single stage of stripping (E1×E2×E1P×1S). Two minutes mixing time per mixer-settler unit is anticipated. No wash stages or after-settlers are anticipated or included in the current design. A loaded organic tank and diluent storage tank are collocated with the SX mixer settlers.

The initial EW plant construction will be 22,000 tpa copper production able to accommodate a maximum designed production up to 25,000 tpa of copper cathodes (production Years 1-7). A future expansion to 35,000 tpa copper cathodes production expansion with a maximum production up to 40,000 tpa of copper cathodes is also considered in the design to accommodate higher grade open pit and underground materials in future (production Years 8 to 18).

Copper EW is expected to require 36 cells, constructed of polymer concrete, and containing 87 cathodes (25 ft2 plating area per cathode) and 88 anodes each, operating in series and connected to two parallel rectifier transformer units (32 kA/100 VDC). Expected current efficiency is 92% operating at a nominal 28 A/ft2 current density (design 32 A/ft2). Cathode stripping from the permanent stainless-steel blanks will be done is a stripping machine that is of a semi-automatic, robotic design.

An addition of 18 EW cells in a new building annex is contemplated for the future expansion with a single rectifier transformer unit installed compatible with the initial units.

Copper cathode bundles of up to 4,500 lb to 5,500 lb each will be sampled, weighed, labeled, and strapped then placed in a secure area for pick up by a copper broker for transport and sale.

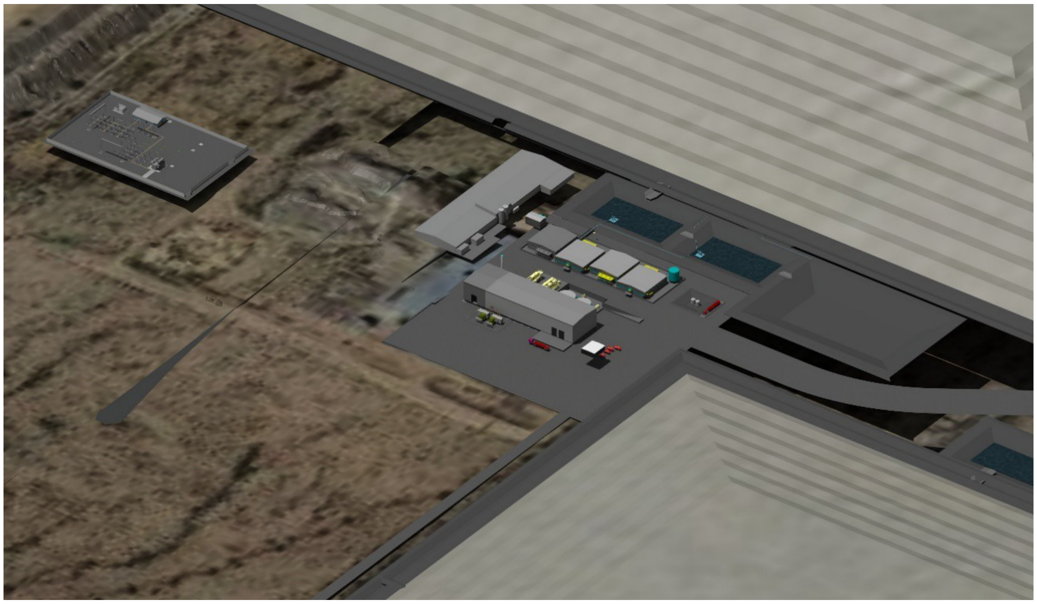
The EW operation will be housed in a pre-engineered steel building fitted with an overhead crane for copper production material handling. Siding will be fabric or fiberglass construction. The process are office and process control room will be located in a prefabricated building including a small wet laboratory for process control assays and mine grade control Stockpile Project sample assays.

The facilities also include a tank farm area composed of electrolyte solution tanks, electrolyte filters, crud handling system, and a solution management holding tank.

An administration / control building located near the site entrance will consist of a new prefabricated, double-wide structure.

The general layout of the proposed processing facilities is shown in Figure 17-2.

Figure 17-2: Plant Area General Layout



The leaching system at the proposed Arizona Sonoran Cactus Project is intended for a conventional heap leach built over time in 20 ft (6.1 m) lifts to a maximum elevation of approximately 200 ft (61.0 m) over a period of 4 years. The Arizona Sonoran pad is in a gently sloping terrain NW to SE and considered as a flat pad base (less than 2% grade) arrangement for design purposes. Pad ultimate height is not considered extreme for design purposes. ADEQ Best Available Demonstrated Current Technology (BADCT) general principles and prescriptive requirements are applied in the design as a minimum.

The oxide leach pad will be constructed in two phases. The footprint of the initial leach pad area is about 4,000 ft (1,219.2 m) × 2,250 ft (685.8 m) = 9.0 million ft2 (0.84 km2) total and will support approximately 57 million tons leach material. The initial build out (Phase 1) will be in two sub-phases and would be roughly 42% of the total 135.5 million tons of oxide material to be mined.

The remaining material will be placed on a second oxide pad area west of the existing tailings facility initially constructed in Year 4 (44.8 million tons, 9.3 million ft2 [0.86 km2]), with incremental additions in Year 7 (20.1 million tons, 5.2 million ft2 [0.48 km2]) and Year 10 (12.3 million tons, 1.3 million ft2 [0.12 km2]) of the mine life.

The first phase of the enriched material leach pad will also be constructed in the area made available by mining of the north end of the Stockpile Project area in Year 2-3 of the operations. The footprint of the initial enriched leach pad area is about 2,200 ft (670.6 m) × 1,000 ft (304.8 m) = 2.2 million ft2 (0.2 km2) total and will support approximately 5.3 million tons leach material. The initial build out (Phase 1) would be roughly 12% of the total 44.3 million tons of enriched material to be mined.

The remaining enriched material will be placed on subsequent extensions of the initial pad area constructed in incremental additions extending south in Year 6 (22.4 million tons, 5.1 million ft2 [0.47 km2]) and Year 12 (16.6 million tons, 2.0 million ft2 [0.19 km2]) of the mine life.

When necessary, solution stacking of PLS between from the oxide to enriched leach pads will be employed manage both overall PLS flowrates and optimal pH in the SX plant.

Figure 17-3 shows the initial processing facilities to be constructed. Leach pad design considered is conceptually shown in Figure 17-4, is assumed to be a double-lined system consisting of a single, 60 mil HDPE primary liner with a compacted soil secondary liner. The soil liner will be a low-permeability soil layer (Kd = 1 × 10-6 cm/sec hydraulic conductivity rating), compacted amended soil approximately 12-inch in depth (built in two 6-inch layers) consisting of a non-gap graded particle size distribution minus 3/8 inch material with a greater than or equal to P30 of -200 mesh content. Existing site alluvium is expected to meet these requirements.

In addition to the first phase of the oxide leach pad, there are three ponds that would also need to be constructed to initiate operations: the SX Raffinate Pond (270 ft [82.3 m] × 190 ft [57.9 m]), the PLS Pond (270 ft [82.3 m] × 190 ft [57.9 m]) and an Event Pond (600 ft [182.8 m] × 320 ft [97.5 m]). The three ponds will be situated below the leach pad and leach solution will flow by gravity downhill via collection ditches that will discharge into the lined storage ponds.

In addition to the first phase of the enriched leach pad, there are two ponds that would also need to be constructed to initiate operations: the PLS Pond (300 ft [91 m] × 190 ft [58 m]) and an event pond (440 ft (134.1 m) × 290 ft (88.4 m)). The three ponds will be situated below the leach pad and leach solution will flow by gravity downhill via collection ditches that will discharge into the lined storage ponds.

The second phase of the oxide pad will require a PLS pond for that area. Subsequent pad area expansions at both the oxide and enriched pads will also include additional storm water pond capacity construction.

The order of precedence for pond volumes is designed as PLS, raffinate, and storm water whereby fluids from the leaching system (largest inflow contributor) report first to the PLS pond and when/if this pond is full a spillway directs the flows to the raffinate pond and for extreme events (e.g., 4.85-inch 100 year / 24-hour storm event 1) a spillway directs flows to the storm water pond.

All ponds are designed with a 2:1 slope on the sides in an inverted pyramid frustum shape. Pond depths are 30 ft (9.1 m). A 2 ft (0.6 m) freeboard is assumed for all ponds. The normal operating volume of the two processing ponds (PLS and raffinate) is 50% full by effective height based on pond inflows under normal operations.

Figure 17-5 shows the initial leach pads, process, and site facilities. Figure 17-6 shows the incremental expansions required over the course of the mining activities and Figure 17-7 shows the ultimate leach pad area and infrastructure required. Figure 17-8 provides a perspective view of the ultimate facilities envisioned.

U.S. Department of Commerce Global Summary of the Year 1960 – 1985, National Centers for Environmental Information, NOAA

- National Oceanic & Atmospheric Administration, National Environmental Satellite, Data, and Information Service, Location: Elev: 1400 ft. Lat: 32.8875° N Lon: -111.7147° W, Station: CASA GRANDE, AZ US USC00021306 and Western Regional Climate Center data 1898-2009 (updated 2012).

Figure 17-3: Schematic Layout, Process Plant

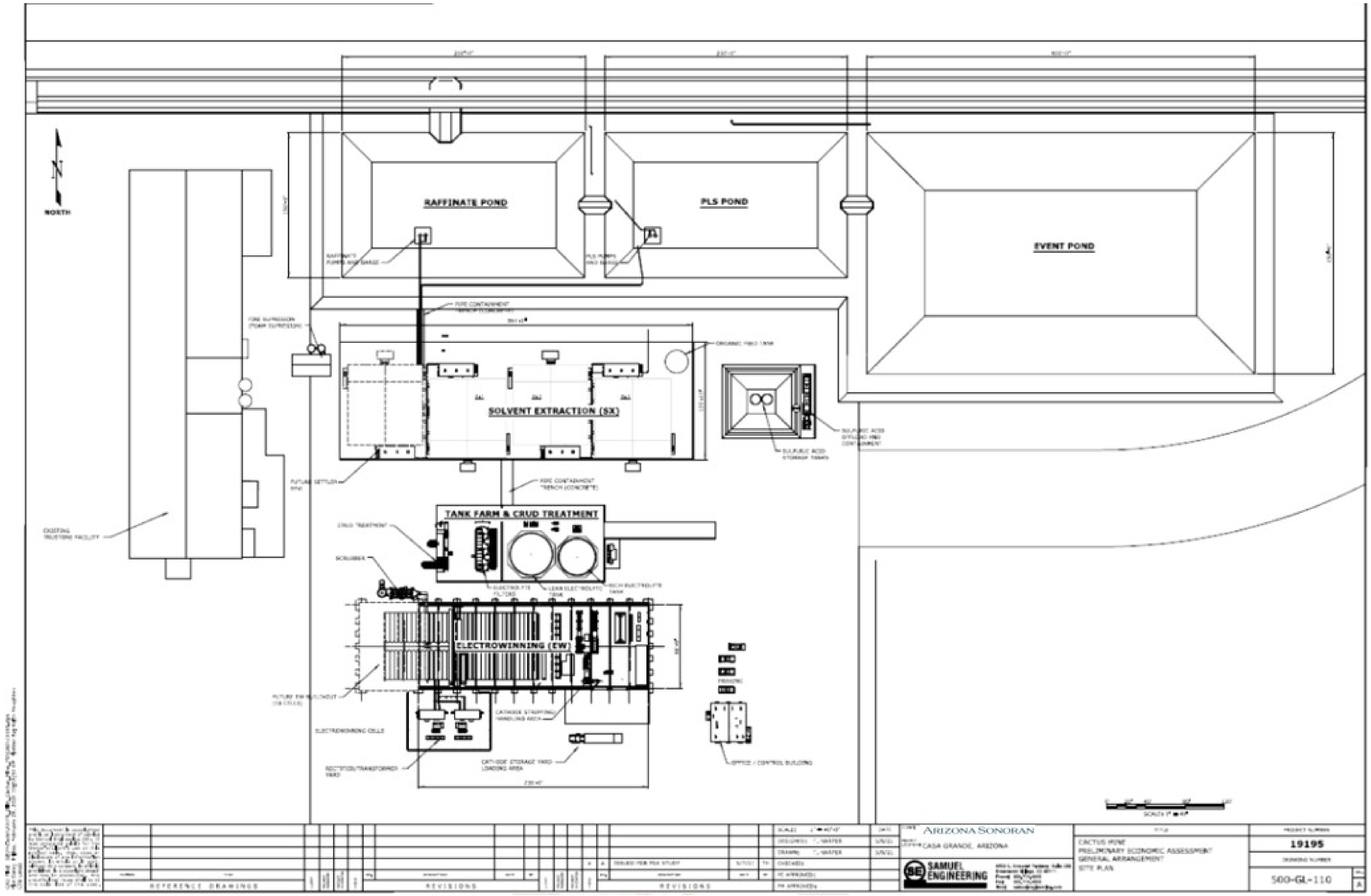


Figure 17-4: Typical Leach Pad Design

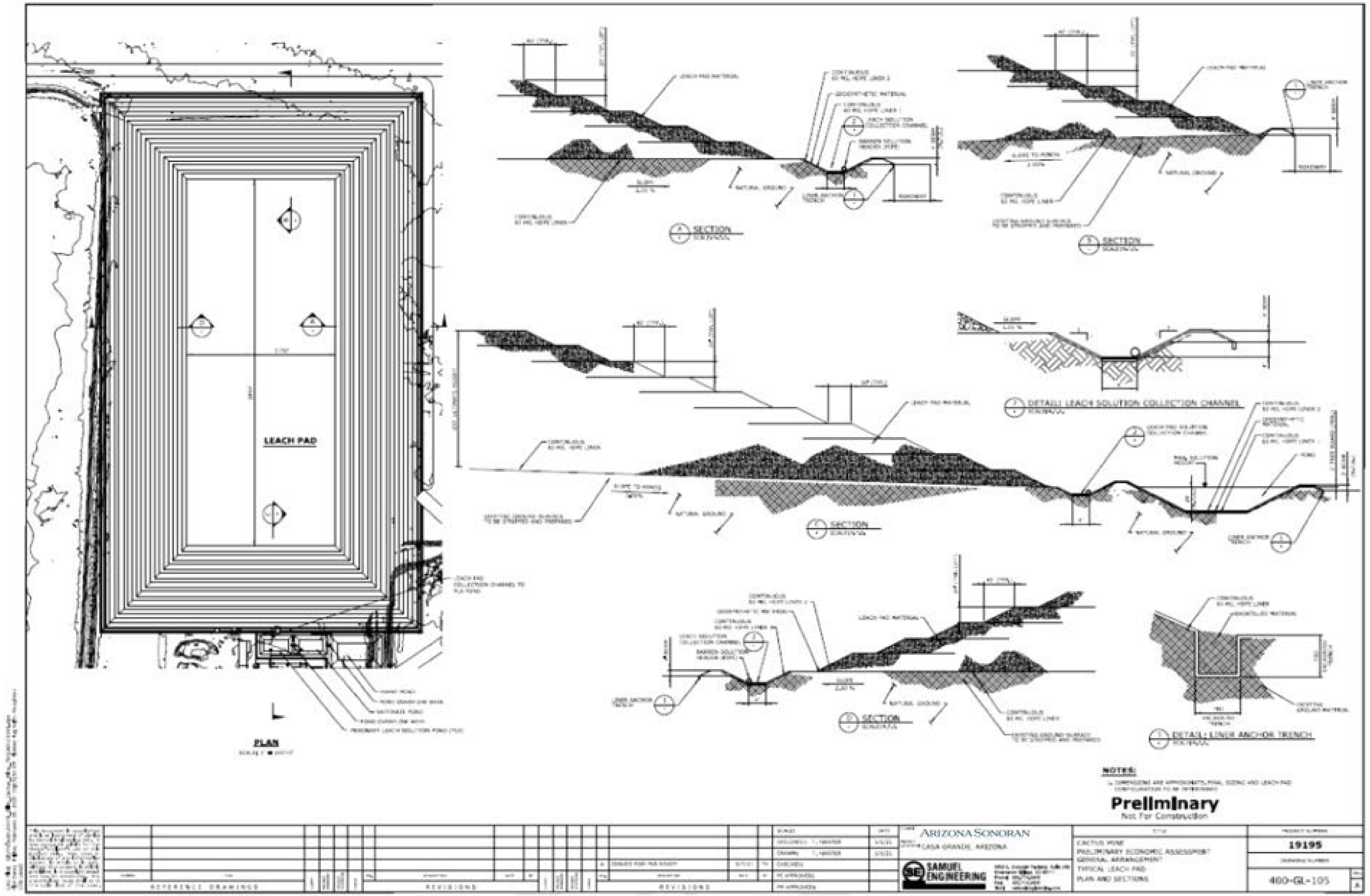


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Figure 17-5: Plot Plant Proposed Phase I Execution

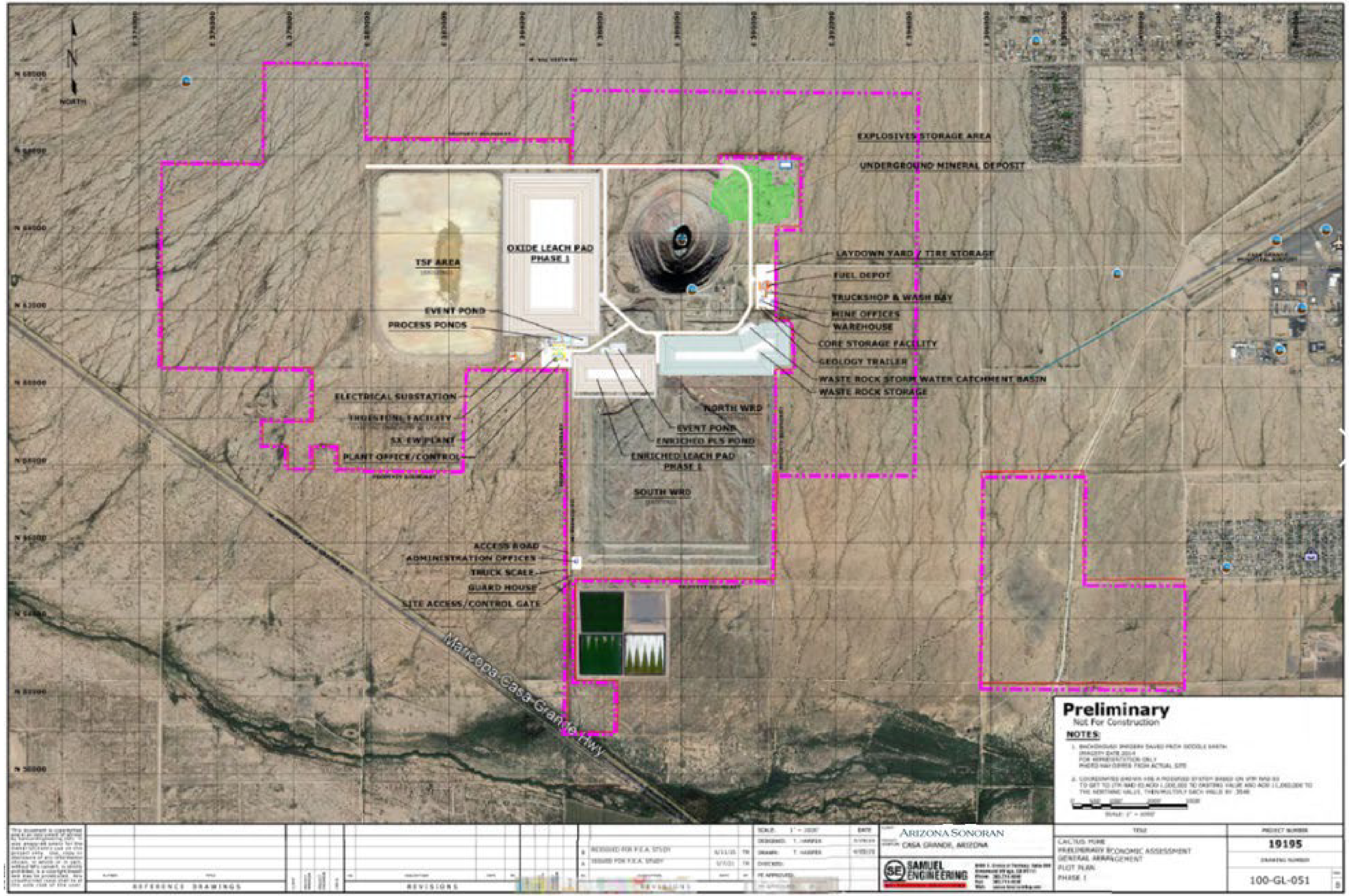

Figure 17-6: Phase 2 and 3 Site Plot Plan

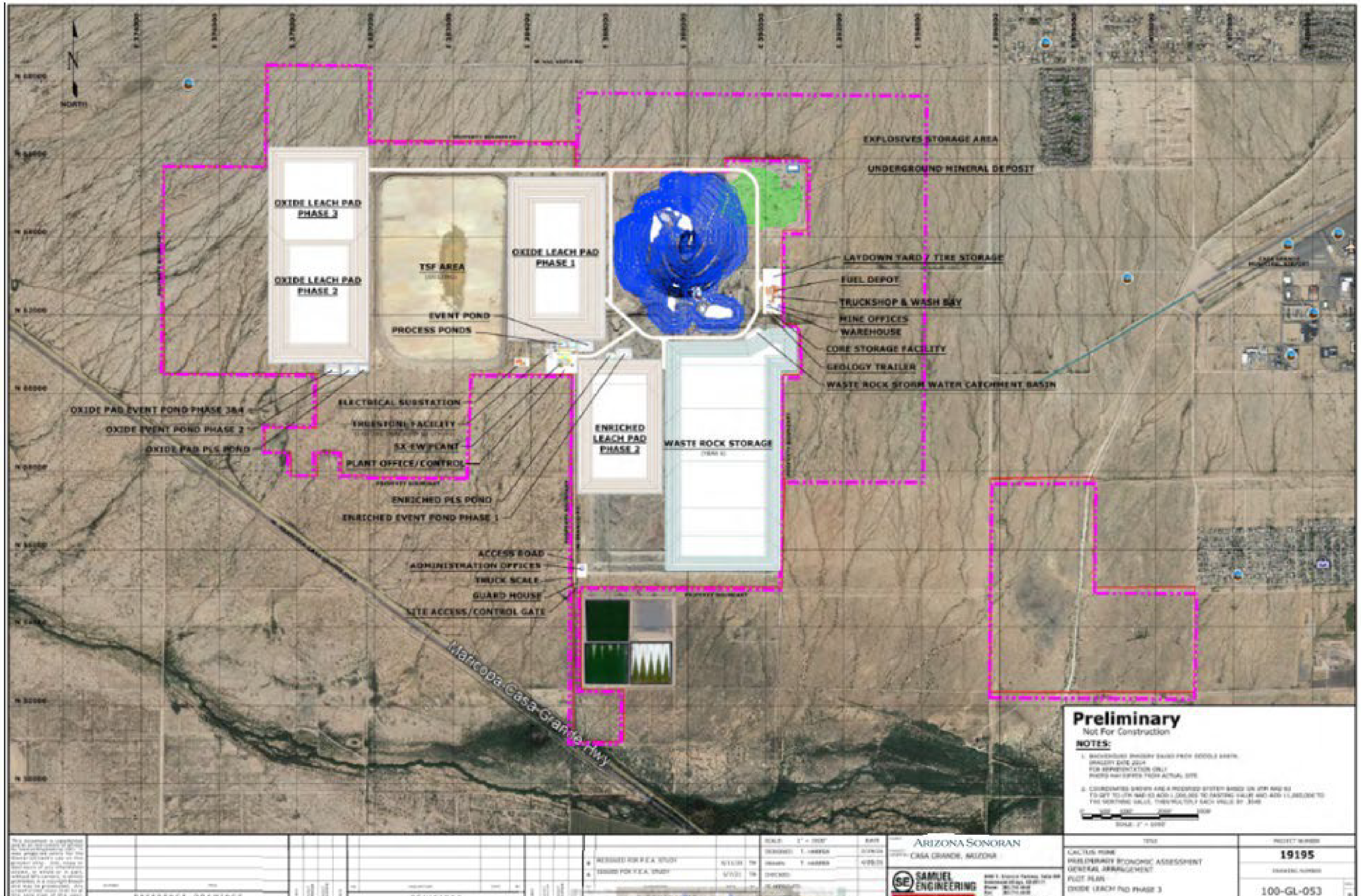

Figure 17-7: Ultimate Life-of-Mine Facilities Site Plot Plan

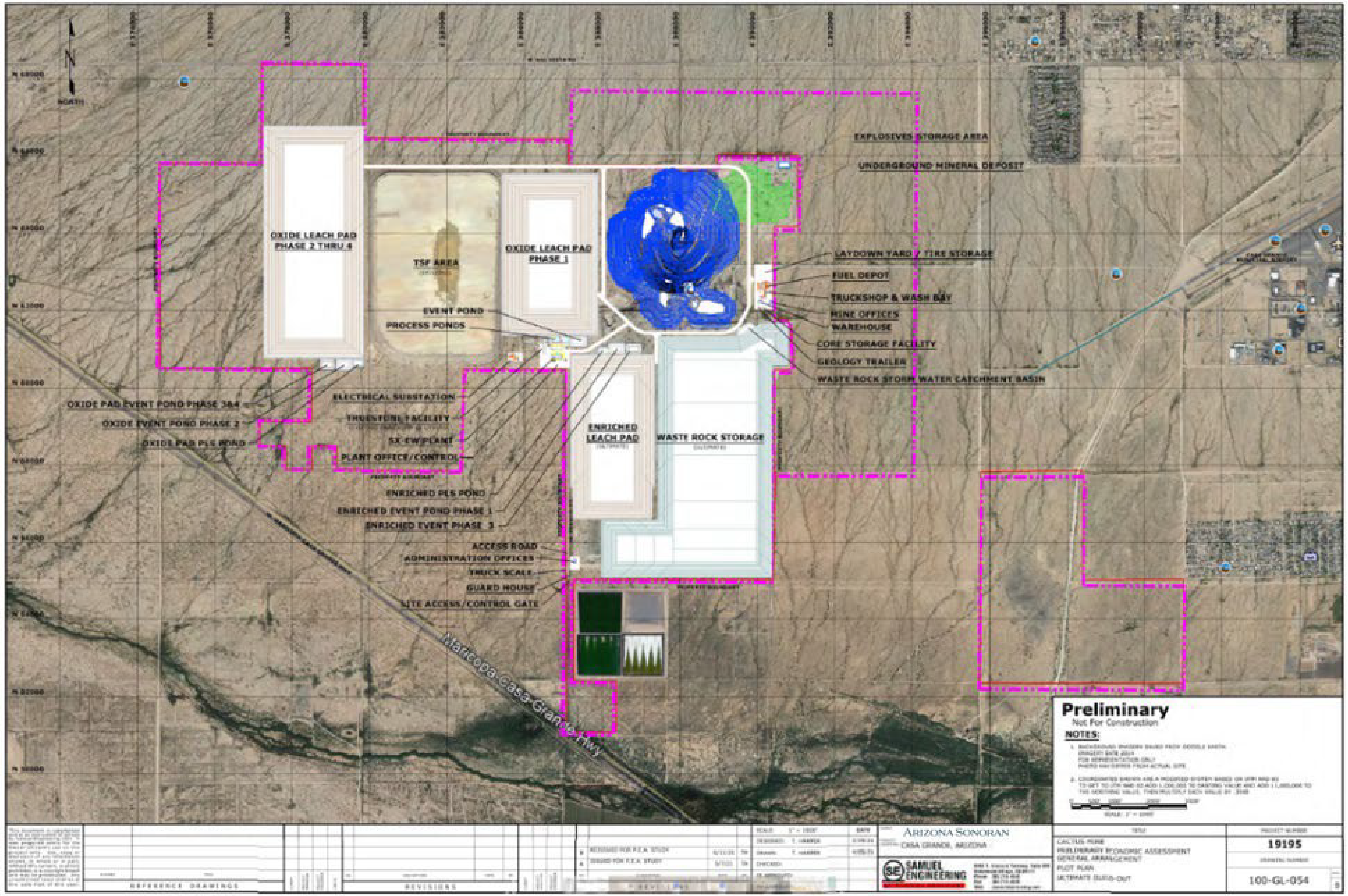

Figure 17-8: Perspective View Ultimate Facilities (Plot Plan)

17.2 Reagents, Water, and Power

Projected reagent and operating consumables requirements for the project are summarized as follows.

- Energy: 1.50 kWh/lb Cu produced.
- Makeup fresh water: 658-951 gpm (including dust control).
- Sulfuric Acid: 300 tpd.
- Leaching: 13.4-3.4 lb/ton leached net of SX/EW credits.
- SX reagents.
- Extractant: 637-890 lb/d (289-404 kg/d).
- Diluent: 150-200 gallons/d.
- EW Reagents.
- Cobalt sulfate: 0.05 lb/t Cu produced.
- Guar: 0.01 lb/ton Cu produced.
- Mist suppressant: FC-1100.

17.2.1 Water

The estimated average water requirements for the Cactus Project at average full production rates is provided in Table 17-5.

Table 17-5: Project Average Fresh Water Usage

Water Usage Source	Quantity LOM Average	Quantity Year 1-4 Average	Units
Evaporation (Net) Pads and Ponds	274	274	gpm
	144,014,000	144,014,000	gal/y
Ore Consumption (Average)	349	642	gpm
	183,593,000	337,435,000	gal/y
Dust Control	100	100	gpm
	52,530,000	52,530,000	gal/y
Misc. Process and Human Usage	15	15	gpm
	7,884,000	7,884,000	gal/y
Additions (Rain and Acid)	(80)	(80)	gpm
	(42,048,000)	(42,048,000)	gal/y
Average Annual Totals			
Flowrate	658	951	gpm
Gallons per Year	345,845,000	499,846,000	gal/y
Acre-Feet per Year	1,061	1,534	afy

Moisture retention is estimated to be 7% by weight for the mineralized material in the leaching areas. This is calculated from an initial moisture content of approximately 3% by weight and a terminal moisture content of 10% after leaching and complete drain down.

Mineralized material to the leach pad area will average from 55,000 tpd to 30,000 tpd. Water retained in leached mineralized material is calculated to be 924,000 gal/d to 504,000 gal/d = 642 gpm to 350 gpm.

Evaporation losses (Life-of-Mine Average)

- The active heap leach area will be irrigated with a combination of sprinklers and drip emitters at the 3,000 gpm flowrate across the 2,500,000 ft2 (232,258 m^2) on a continuous 24-hour basis. Based on experience with this type of equipment and system, an average evaporation of 9% of the total flow to the pad has been considered for the evaporative losses in the leach pad and trench areas. The expected evaporative losses in the leach pad areas are calculated as 3,000 gpm × 9% = 270 gpm.
- The four ponds considered in the Project design are the two PLS ponds for the oxide pad and enriched pad, raffinate pond, and storm water collection pond. Pond evaporation losses are calculated from the reported 2 average annual pan evaporation of 107.4-inch per year (8.95 ft/y) at an average 50% operating volume of the PLS and raffinate ponds.

Water supply is already available via buried pipeline to the property boundary as a result of prior mining and commercial operations. The property, at present, have groundwater rights associated with mining activities.

- Type 2 Non-Irrigation Grandfathered Right No. 58-100706.0004. This right includes 136 afy.
- Permit to Withdraw Groundwater for Mineral Extraction and Metallurgical Processing Permit No. 59-233782.0000. This permit allows Arizona Sonoran the rights to 3,600 afy for 50 years for Heap leach mining activities, dust control and processing at the Cactus Project site. The effective date of permit is 14 April 2021, and the Expiration Date of Permit is 14 April 2070.

The two owned water rights allow for 3,736 afy. Currently, there are five wells / locations that water could be pumped from, these are Well 1, Well 2, Well 5, Well 6, and the prior ASARCO Production Shaft. Additional locations may need to be identified for water production depending on facility layout and future needs.

If needed additional requirements could be met in two ways.

- Purchase of water from the Gila River Water Storage, LLC (GRWS) resources in the Pinal Active Management Area (AMA).
- Mine dewatering credits as the project is developed in the future.

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[1] U.S. Department of Commerce Global Summary of the Year 1960 – 1985, National Centers for Environmental Information, NOAA - National Oceanic & Atmospheric Administration, National Environmental Satellite, Data, and Information Service, Location: Elev: 1400 ft. Lat: 32.8875° N Lon: -111.7147° W, Station: CASA GRANDE, AZ US USC00021306; Western Regional Climate Center data 1898-2009 (updated 2012).

If needed additional requirements could be met in two ways.

- Purchase of water from the GRWS resources in the Pinal AMA.
- Mine dewatering credits as the project is developed in the future.

The Pinal AMA covers approximately 4,000 square miles in central Arizona and consists of five sub-basins with unique groundwater underflow, storage, and surface water characteristics. These sub-basins are Maricopa-Stanfield, Eloy, Vekol Valley, Santa Rosa Valley, and Aguirre Valley. New on-site metering (GRWS), storage and distribution systems will be required for the project for use of these resources.

17.2.2 Electric Power

Approximately 11 MW of power will be required for the initial and 14.3 MW for the expanded project site process facilities as shown in Table 17-6.

Table 17-6: Projected Process Average Electric Power Usage

Process Area	Process Years 1-6		Process Years 7-17	
	kWh/lb	kWh/y	kWh/lb	kWh/y
EW	1.00	5,303	1.00	7,422
SX/TF	0.45	2,387	0.45	3,340
Utilities / Misc	0.15	796	0.15	1,113
Leaching	0.45	2,400	0.32	2,400
Total Power	**2.05**	**10,866**	**1.92**	**14,275**

17.2.3 Sulfuric Acid

The heap leach acid consumption estimate varies with the tonnage rates processed, types or materials leached (oxide and enriched) and the recovered copper content (grade). The expected H2SO4 consumption in years 1-6 is high, at approximately 300 tpd on a 100% basis. Acid consumption in years 7-17 is much different at 14 tpd due to significantly higher copper grades and enriched (sulfide) mineralized material comprising approximately half the material leached overall.

Acid will be delivered in 20 t tanker trucks and off-loaded into two site acid storage tanks of approximately 5,000 gal capacity each. Initially an average of 15 tanker trucks per day will be required to be received and offloaded per day, approximately two per hour to avoid night-time operations.

A mine average for each significant period of time and plant production is included based on a gross acid consumption for all materials and net of copper production credits is included in Table 17-7.

Table 17-7: Average Acid Consumption Heap Leach Operations

Production Timeframe	Units		Comments
Initial Project Years 1-6 (average)	lb H_2SO_4/ton ore	13.4	Year 1-6 mining rate (17.8M mineralizedmaterial tpa).
	lb H_2SO_4/lb Cu	4.69	Average Annual Cu Production 23,230 t.
	tons H_2SO_4/d	298	Truck delivered (20 t tanker).
	tons H_2SO_4/yr	109,000	98% Acid.
Project Years 7-17 (average)	lb H_2SO_4/ton ore	3.4	Year 7-17 mining rate (6.1M mineralizedmaterial tpa).
	lb H_2SO_4/lb Cu	0.15	Average Annual Cu Production 32,500 t.
	tons H_2SO_4/d	14	Truck delivered (20 t tanker).
	tons H_2SO_4/yr	5,110	98% Acid.

An additional 1-2 tpd acid is expected to satisfy electrolyte bleed make-up and all other SX/EW requirements. Most, if not all, of this acid would report to the raffinate pond and be used in the leaching operation.

18.0 PROJECT INFRASTRUCTURE

The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as at 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report.

18.1 Mining and Maintenance

The mining operations are anticipated to be contracted to a local company experienced in larger scale earthmoving. Given the proximity to major infrastructure in Casa Grande, the contractor may bring temporary facilities onto the site to facilitate their operations and maintenance activities self-sufficiently on the project site. A specific contractor plan has not been developed. This will be similar to the facilities set up on-site as part of the recent reclamation effort.

Waste material will either be set aside in the Stockpile Project as the material is mined or taken to existing nearby waste dumps on site. Although a detailed mine plan and sequence has not yet been developed, it is expected that most of the waste material will remain in the current Stockpile Project area.

18.2 Leach Pad / Ponds Preliminary Location / Conceptual Design

The oxide leach pad will be constructed in two phases. The footprint of the initial leach pad area is about 4,000 ft (1,219.2 m) × 2,250 ft (685.8 m) = 9.0 million ft2 (0.84 km2) total and will support approximately 57 million tons leach material. The Phase 1 pad would be roughly 42% of the total 135.5 million tons of oxide material to be mined. The initial build out (Phase 1a) would be roughly 45% of the Phase 1 total with the remaining 55% to be completed in Year 3 of plant operations. The capital cost estimate includes only Phase 1a of the leach pad with a base footprint of roughly 2,100 ft (640 m) × 2,250 ft (685.8 m) or 4.8 million ft2 (0.45 km2), which will support approximately 25 million tons. The final build (Phase 1b) out would occur in Year 3 of the operations.

Leach pad and pond design will be compliant with ADEQ BADCT applicable guidelines and prescriptive requirements. The leach is assumed to be a double-lined system consisting of a 60-mil non-textured HDPE liner with a compacted soil secondary liner. Process solution ponds are assumed to be constructed with a triple-lined system consisting of two 60-mil non-textured HDPE liners with a compacted soil tertiary liner and integrated leak detection between the HDPE liners.

The remaining life-of-mine material will be placed on a second oxide pad area west of the existing tailings facility initially constructed in Year 4 (44.8 million tons, 9.3 million ft2 [0.86 km2]), with incremental additions in Year 7 (20.1 million tons, 5.2 million ft2 [0.48 km2]) and Year 10 (12.3 million tons, 1.3 million ft2 [0.12 km2]) of the mine life.

The first phase of the enriched material leach pad will be constructed in the area made available by mining of the north end of the Stockpile Project area. In Year 2-3 of the operations. The footprint of the initial enriched leach pad area is about 2,200 ft (685.8 m) × 1,000 ft (304.8 m) = 2.2 million ft2 (204 thousand m^2) total and will support approximately 5.3 million tons leach material. The initial build out (Phase 1) would be roughly 12% of the total 44.3 million tons of enriched material to be mined.

In addition to the first phase of the oxide leach pad, three ponds need to be constructed to initiate operations: the SX raffinate pond (270 ft [82.3 m] × 190 ft [57.9 m]), the PLS pond (270 ft [82.3 m] × 190 ft [57.9 m]), and an event pond (600 ft [182.9 m] × 320 ft [97.5 m]). The three ponds will be situated below the leach pad and leach solution will flow by gravity downhill via collection ditches that will discharge into the lined storage ponds.

In addition to the first phase of the enriched leach pad, two ponds need to be constructed to initiate operations: the PLS pond (300 ft [91.4 m] × 190 ft [57.9 m]) and a 100 year event pond (440 ft [134.1 m] × 290 ft [88.4 m]). The two ponds will be situated below the leach pad and leach solution will flow by gravity downhill via collection ditches that will discharge into the lined storage ponds.

An allowance is included for installation of monitoring wells.

18.3 Other Facilities Considerations

The maximum height of all site facilities was considered due to the site's proximity to the existing Casa Grande Municipal Airport that is owned and operated by the City of Casa Grande. A draft airport master plan currently includes proposals for a 4,750 ft (1,447.8 m) southwesterly extension of the existing runway for a total ultimate runway length of 8,400 ft (2,560 m). In addition, the plan considered construction of new exit taxiways, and a new 3,650 ft (1,112.5 m) parallel runway located north and west of the existing runway.

Federal Aviation Administration (FAA) requirements are outlined in the Federal Aviation Act of 1958, as amended and pursuant to 49 U.S.C. Section 46301(a). A summary of the relevant Federal Aviation Regulations (FAR) Part 77 Section 77.9 is provided as follows.

§ 77.9 — Any person/organization who intends to sponsor any of the following construction or alterations must notify the Administrator of the FAA.

- Any construction or alteration exceeding 200 ft above ground level.
- Any construction or alteration within 20,000 ft of a public use or military airport that exceeds a 100:1 surface from any point on the runway of each airport with at least one runway more than 3,200 ft.
- Within 10,000 ft of a public use or military airport which exceeds a 50:1 surface from any point on the runway of each airport with its longest runway no more than 3,200 ft.
- Within 5,000 ft of a public use heliport which exceeds a 25:1 surface.
- Any highway, railroad or other traverse way whose prescribed adjusted height would exceed that above noted standards.
- When requested by the FAA.
- Any construction or alteration located on a public use airport or heliport regardless of height or location.

Figure 18-1 through Figure 18-4 demonstrate the phasing of leach pads, waste dumps, and ultimate site configuration.

Figure 18-1: Initial Leach Pad Construction, Waste Dump, and Site Configuration

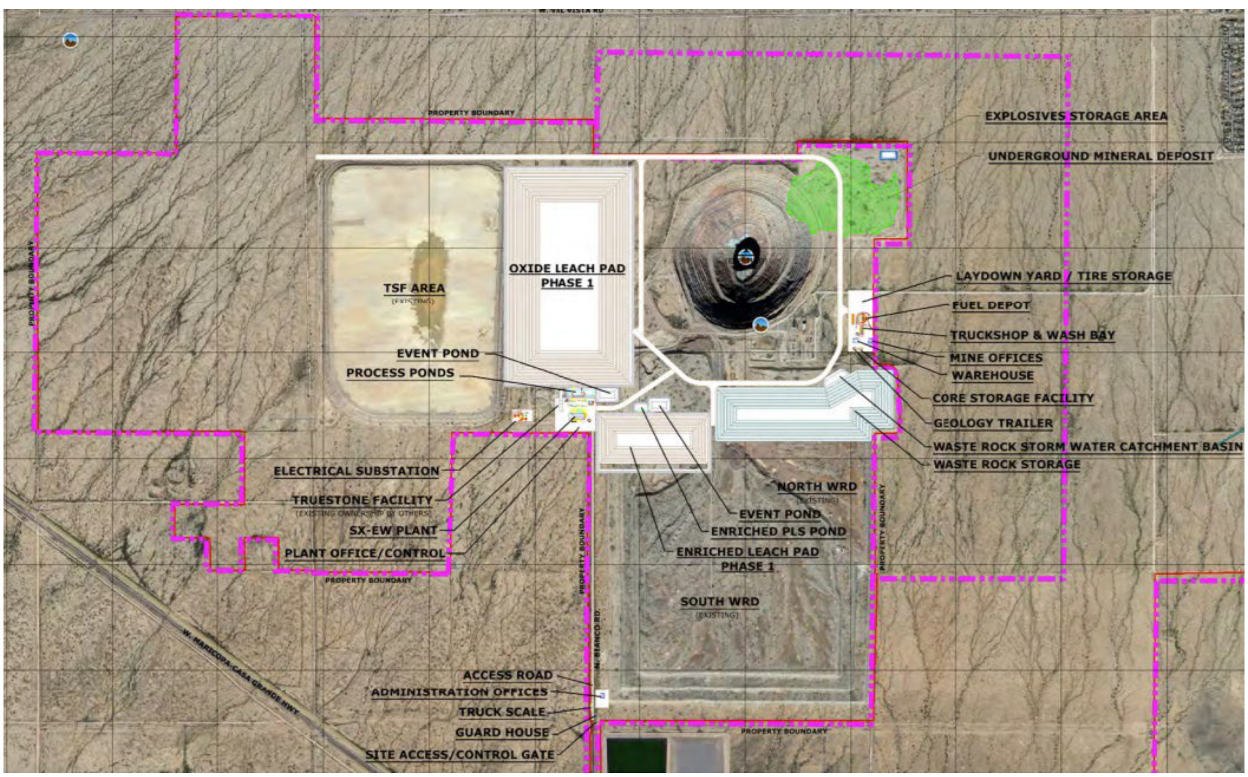


Figure 18-2: Phase 2 Leach Pad Construction, Waste Dump, and Site Configuration

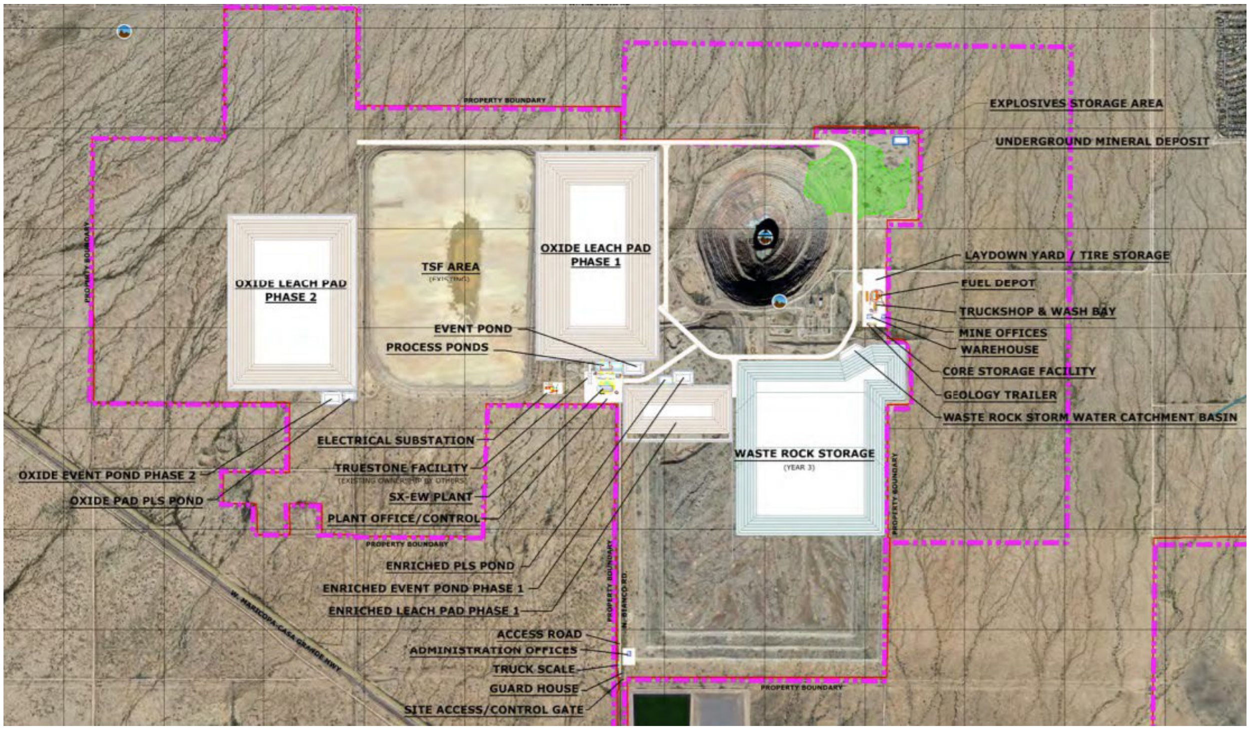


Figure 18-3: Phase 3 Leach Pad Construction, Waste Dump, and Site Configuration

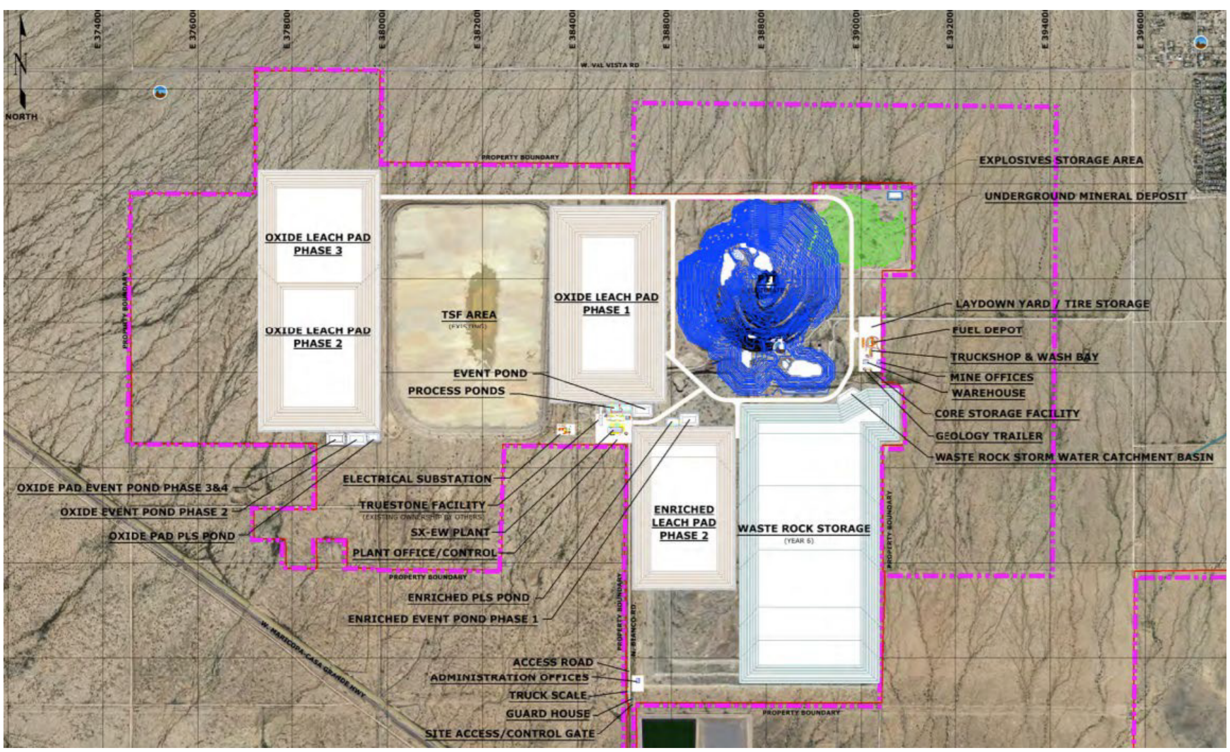
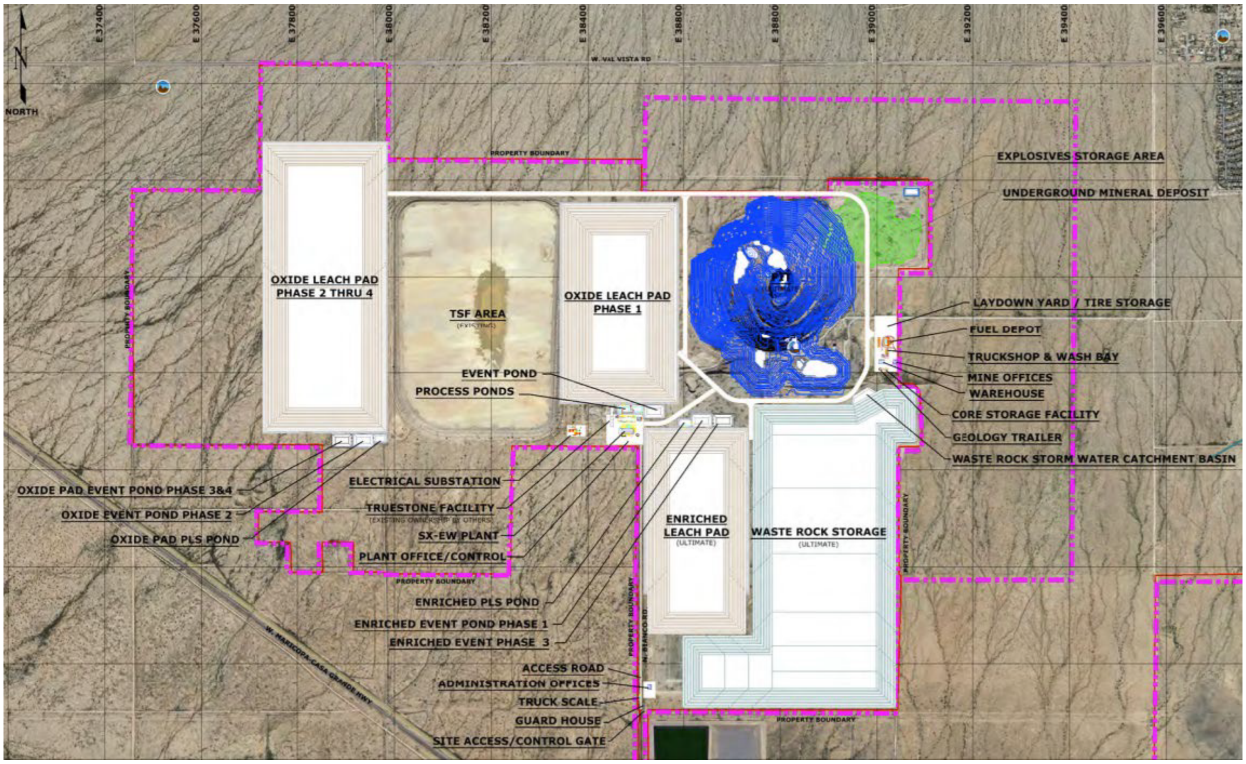


Figure 18-4: Ultimate Configuration



18.4 Site Buildings / Maintenance Shops / Administration Building

Given the proximity to the city of Casa Grande, limited non-process facilities are required. The project will require minimal buildings and shops in light of the existing infrastructure, contract mining and minimal site-based staffing required.

The SX/EW plant site offices, control room and security will be housed in a single prefabricated building located on-site near the main gate and process plant facilities. A 200 ft (61.0 m) × 400 ft (121.9 m) building is included for these purposes.

The EW process will be enclosed in a prefabricated steel structure with space allowed for minor maintenance activities and materials storage. Additional storage of materials will be provided within the fenced area near the plant and in shipping containers repurposed from the delivery of materials and equipment to site. The abandoned TruStone facility may also be considered in future for additional maintenance, warehousing, and other uses.

Arizona Sonoran maintains a corporate office in Tempe, Arizona, from the Project site for administrative staff not required to be regularly on site.

Mine support infrastructure has been assumed to be provided by the selected contract mining company as required and locations have been identified for those potential facilities within the property boundaries.

18.5 Process Buildings

A new SX/EW facility will be constructed inside the fenced area of an abandoned process building known as the TruStone facility. The area has been cleared and graded and was previously used for parking or laydown.

There are existing access roads to the facilities along with a rail spur that dead-ends in front of the plant across the access road, although it is not currently connected to the main line. There are no current plans to reconnect or use the rail line.

An incoming utility powerline is connected to an existing substation owned by Arizona Power System that was originally used to power the TruStone facility. This substation will be used to power the new SX/EW facility and other project loads. No work is currently planned on the electrical system upstream of the low-side connection to the main transformer.

The EW operation will be housed in a pre-engineered building fitted with an overhead crane for copper production material handling. Siding will be fiberglass, PVC-coated fabric, or protected steel.

The administration / control building will consist of a new, prefabricated, double-wide structure.

18.6 Acid Supply and Storage (Truck or Rail)

Acid will be provided by a local broker, delivered to site in bulk 3,300 gal (25 t) acid truck / trailers. Tanker trucks will be off-loaded to a mild steel site storage tank located in the SX/EW tank farm area with a nominal capacity of 60,000 gal (2 days nominal usage).

Approximately eight to nine trucks will be received and off-loaded per day.

Consideration will be given in the future to refurbishing the existing rail spur connecting the site with the Union Pacific Railroad Line approximately 3.7 miles south of the site and delivery by 100 t railway cars.

18.7 Water Supply and Distribution

Water rights totaling 3,736 afy have been secured via an historic Grandfathered Water Rights Type 2 Non-Irrigation grandfather rights (Certificate 58-100706.0005) for 136 afy and a Mineral Extraction Process Permit No. 59-233782.000 from the ADWR for 3,600 afy. Water will be sourced from two offsite wells, No. 1 and No. 2, and two onsite wells, No. 5 and No. 6. The Mineral Extraction and Process Permit is permitted for 50 years and will serve as water supply requirements for the life of the Project.

Additionally, process makeup water can also be sourced from open pit dewatering and the existing flooded shaft constructed and abandoned by ASARCO.

Potable water is available on the project site (servicing the prior TruStone and ASARCO facilities) for the minor potable usage requirements.

18.8 Power Supply and Distribution

Power is available to an existing 115 kV substation at site. Arizona Public Service (APS) will provide power via existing 115 kV power transmission lines owned by APS run from its Casa Grande substation to the existing substation on the site located about 400 ft (121.9 m) west from the planned processing plant location.

The site substation has not been evaluated but it is operational and serviced the prior ASARCO mine operations and more recently the TruStone production facility (now closed) next to the proposed SX/EW plant location.

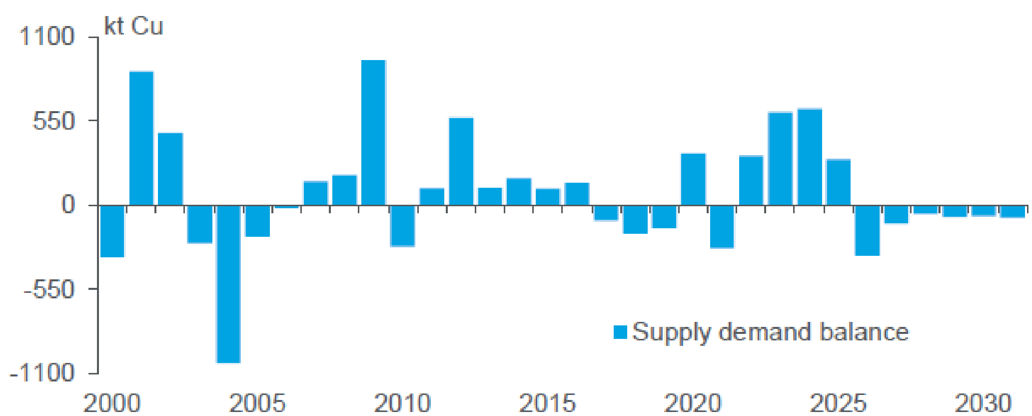
Expected average annual power costs are $0.058/kWh (including demand charges) based on preliminary discussion with APS and a new customer services rate for a Small General Service Plan (non-residential).

19.0 MARKET STUDIES AND CONTRACTS

The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as at 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report.

As demonstrated by various market studies and research reports, the COVID-19 pandemic which emerged in 2020 has had a significant impact on copper supply and demand fundamentals. According to Wood Mackenzie, the fall in demand in 2020 was less than expected while the 2021 recovery looks set to be better than originally anticipated. Last year's China led recovery has been reinforced this year with progress on vaccinations and continued efforts by governments to boost growth via large stimulus policies across major markets. As per Wood Mackenzie's publication titled Copper 2021 update to 2040, a long-term structural deficit remains in place. Beyond 2025, an anticipated shortfall in global copper supply will emerge as the pace of supply growth slows relative to demand. Prices are expected to trade higher in reaction to these anticipated deficits and as accumulated inventories are drawn down and consumed. This should provide sufficient confidence to encourage producers to reactivate shuttered mines and undertake incremental expansions, mine life extensions and eventually develop projects that are needed to maintain a reasonable long term market balance. Figure 19-1 provides a view of the global supply-demand balance to 2030.

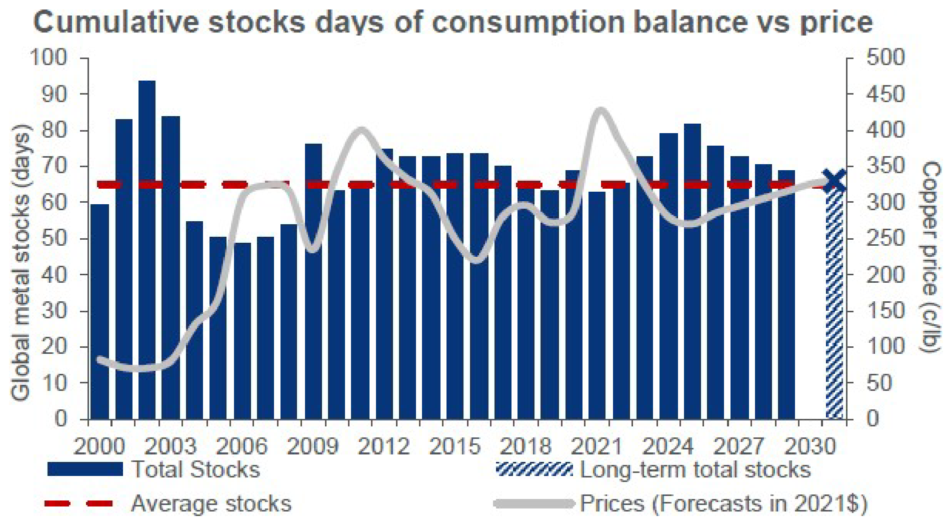
Figure 19-1: Global Supply-Demand Balance



Chinese demand continues to underpin the shorter-term fundamentals, providing good price support. Green infrastructure spending is expected to contribute significantly to medium-term demand, as countries and governments move toward adherence to climate change goals and electrification.

Inventories continue to be depleted by general copper consumption, compounded by global energy and electrification initiatives. Although there are expected to be short term supply surpluses, there is potential for a longer-term deficit from 2025-2026 onwards (refer to Figure 19-2Figure 19-2).

Figure 19-2: Cumulative Stocks Days of Consumption Balance Versus Price



According to Wood Mackenzie, “A deficit opens from 2026 onwards, unless projects currently in our probable and possible categories are developed. We now estimate that the supply gap will stand at ~5.4 Mt by 2031. This assumes conversion of 70% of brownfield probable projects and 50% greenfield probable projects into the base case, plus an allowance for mine life extensions.”

This theoretical supply gap reflects not only the revisions to Wood Mackenzie’s global refined demand outlook but also adjustments to the scrap volumes that should be available to smelters and refineries over the long term. Taking all these factors into account, they remain confident that their long-term incentive price of US$7275/t (US$3.30/lb) in constant 2021 US dollars should be sufficient to close this supply gap to maintain market equilibrium and retain a reasonable market balance over the next decade.

Given Arizona Sonoran’s anticipated development plans, it is positioned well to benefit from these developments in the medium term with stable pricing through to 2025 and robust pricing in the longer term.

There are no material contracts relevant to Arizona Sonoran at this stage.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as at 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report.

20.1 Environmental Studies

Several documents were reviewed to provide an indication of the existing environmental conditions at the Cactus property near Casa Grande, Pinal County, Arizona. These reports included the following.

- Notice of Disposal Form Submitted by ASARCO Inc. to the Arizona Department of Health Services that Formally Closed the Sacaton Facility from the Existing Groundwater Protection Permit 10 January 1985.
- CERCLA Site Investigation: Ecology and Environmental, Inc. 14 November 1985.
- Sacaton Site Characterization Plan (SIP): Tetra Tech, dated 01 May 2017 and 11 March 2019.
- Technical Memo – Initial Hydrogeologic Characterization Study: Tetra Tech, dated 21 December 2017.
- Demolition Completion Report Sacaton Mine Site: Tetra Tech, dated 11 March 2019.
- Environmental Baseline Survey for Sacaton Mine: Engineering and Environmental Consultants Inc. 28 October 2019.
- Site Improvement Plan (SIP) Construction Completion Report Addendum 1: Tetra Tech, dated 14 February 2020.
- ADEQ Water Quality Division: Monitoring & Assessment. (http://www.azdeq.gov/environ/water/assessment/index.html).

Review of historical water quality data collected from 1972 through the present identified sulfates, nitrates, and fluoride exceedances over Arizona drinking water standards at various locations throughout the site. Other constituents found in the soils and ground water were as follows.

- Aluminum
- Antimony
- Arsenic
- Barium

- Beryllium
- Cadmium
- Chromium
- Copper
- Lead
- Manganese
- Mercury
- Molybdenum
- Nickel
- Selenium
- Silver
- Thallium
- Uranium
- Vanadium
- Zinc

No environmental fatal flaws that would materially impede the advancement of the project have been identified. Prior due diligence with the State of Arizona has indicated that the soil and groundwater at the site is highly mineralized and contaminated with heavy constituents such as arsenic, chromium, selenium, and zinc, and therefore is unfit for domestic, livestock, or agricultural use. These constituents were not the result of any mining activity in the area, but are related to the younger, geologic activity in the region. The open pit from ASARCO's mining contains water with high mineralization and a very low pH.

20.2 Permitting

The Cactus property consists mostly of private surface and mineral rights, with the exception of 2 Arizona State Land Department Leases (ASLD) (parcel number: 502-25-7020 Prospecting Permit # 008-122116-00 and parcel number: 503-26-7000 Prospecting Permit # 008-121173-00-100). Permitting for an operation on private and ASLD lands will require the following major permits and certifications, already issued or in progress.

- Dust Permit Pinal Air Quality Control Permit (permit obtained).
- AZPDES permits (construction and Multi-Sector General Permit) (permit obtained for both the Mine Facility and the TruStone Facility). In Q2 of 2022 a new AZPDES was granted, this permit eliminated the TruStone Facility and incorporated that area into the new mine permit. LTF No. 95924 ID No. AZMS95924.
- ADWR Permit to Withdraw Groundwater for Mineral Extraction and Metallurgical Processing Permit No. 59-233782.0000. This permit allows Arizona Sonoran the rights to 3,600 afy for 50 years for Heap leach mining activities, dust control and processing at the Cactus Project site. The effective date of permit is 14 April 2021, and the expiration date of Permit is 14 April 2070.

- ADEQ APP and Amended APP: Both APP applications have been accepted pending bond submittal.
- US Army Corp of Engineers (USACE) Approved Jurisdictional Determination (AJD) 404: On 11 February 2022 USACE issued the signed AJD for the site.
- Pinal Air Quality Control Industrial Permit (applied for in October 2022).
- Arizona State Mine Inspector Mined Lands Reclamation Permit (applied for in October 2022).
- An estimate of $1.5 million will be required for the initial reclamation bond based on the initial construction plan and prior estimates for site closure for the Stockpile Project. An additional $3.5 million is estimated to be required to close the planned facilities and bonding will be adjusted as new facilities are added, particularly the Phase 2 leach pad. Closure funding is expected to be supplemented by resale of the modular SX/EW plant and other infrastructure and equipment, with an estimated salvage value of $5 million.
- Special Land Use Permit (SLUP) for use of State Surface to construct facilities for the mining operation: Application Number: 023-123266-03-100 (Approved Contract signed and sent back to Arizona State Lands Department) Permit Number: 23-123266-03.

Further permitting will be required and modification of existing permits to account for the final operational and mine plan to be adopted and to reflect processing and other facilities.

Figure 20-1 outlines the major permits required as a precursor for project construction along with expected current timing. An approximate total of $0.5 million is required to complete these permitting activities.

Figure 20-1: Permitting Plan

The following additional permits will be required pursuant to a construction decision.

- Arizona Department of Agriculture – NOI to Clear Land
- Pinal County – Mining Construction Permits
- ADEQ – Above-Ground Tank Storage
- FCC – Radio License, Wireless Communication

An estimate of $1.5 million will be required for the initial reclamation bond based on the initial construction plan and prior estimates for site closure for the Stockpile Project. An additional $3.5 million is estimated to be required to close the planned facilities and bonding will be adjusted as new facilities are added, particularly the Phase 2 leach pad. Closure funding is expected to be supplemented by resale of the modular SX/EW plant and other infrastructure and equipment, with a salvage value consideration of $5 million.

20.3 Hydrogeology

Stantec completed a review of hydrogeologic information and completed a numerical groundwater flow model to assess the groundwater flow conditions and potential for dewatering associated with mine expansion options. The numerical groundwater flow model was developed from conceptual model data from multiple sources, including the Pinal AMA regional model (Wickam and Corkhill, 1989; Liu and others, 2014), and the Sacaton Mine Initial Hydrogeologic Characterization Study (Tetra Tech, 2017). This model was utilized to evaluate predictive flow scenarios from the anticipated mine operations.

20.3.1 Background

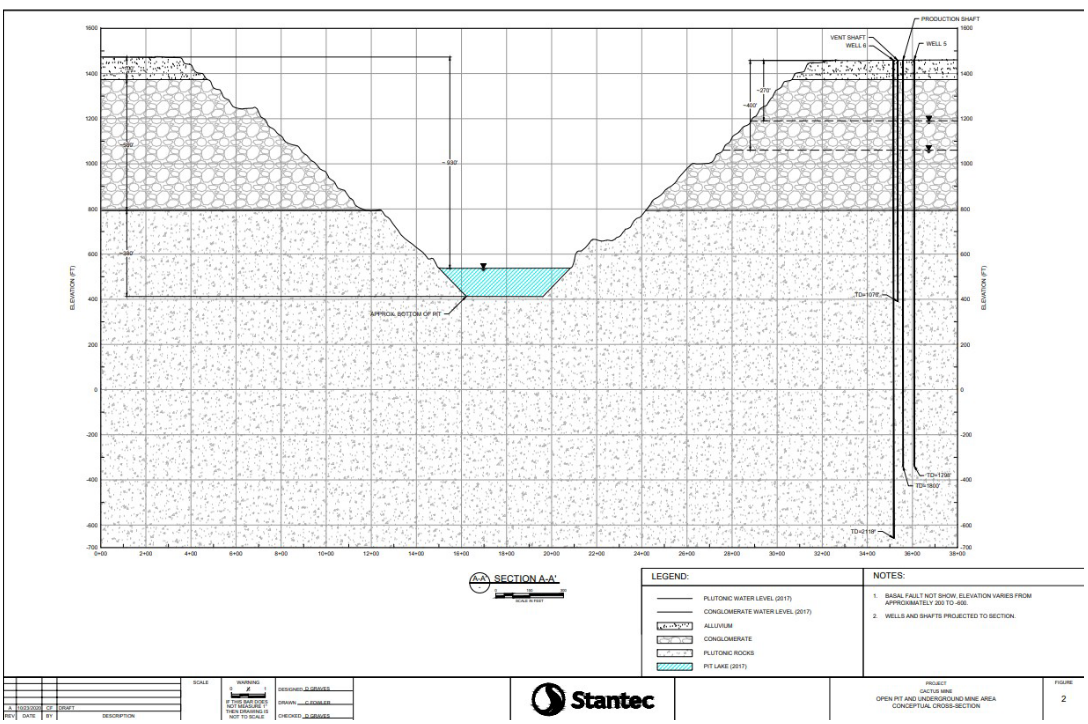
The Cactus Project (i.e., Mine) is in Pinal County, Arizona, approximately 6 miles northwest of Casa Grande, 40 miles south of Phoenix, and 10 miles west of Interstate 10 (I-10) freeway. The Mine property boundary covers approximately 2,500 acres. The average elevation at the Mine is 1,450 ft (442 m) above sea level. This site is in the Basin and Range Physiographic Province of south-central Arizona and near the Eloy and Maricopa-Stanfield sub-basins of the Pinal AMA.

The mine was previously owned and operated by ASARCO and known as the Sacaton Mine, with mining operations beginning in the 1970s. During operation, the mine consisted of the pit, crushing facilities and coarse ore stockpile, a 9,000 tpd flotation mill, a (TSF) that covered approximately 300 acres, a return water impoundment, an overburden dump, and a WRD that covered approximately 500 acres. The mine was permanently closed 31 March 1984. The property was acquired by Arizona Sonoran in 2018.

20.3.2 Conceptual Model

The geologic setting of the Mine and surrounding area includes basin fill deposits consisting of unconsolidated alluvial fill and conglomerate bedrock of approximately 300-600 ft thickness. The unconsolidated alluvium is the principal aquifer in this region, but these deposits are relatively thin at the Mine property and are above the groundwater table. The conglomerate formation has approximately 300 ft of saturated thickness below the Mine property. These sedimentary units overly plutonic crystalline bedrock. The existing open pit from previous mining operations extends to approximately 1,040 ft (317 m) depth below surrounding grade, while enriched mineralized material has been identified down to approximately 1,800 ft below existing grade. The plutonic crystalline bedrock and mineralized deposit lies above the Basal Fault, a regional geologic feature that confines an aquifer in the underlying Pre-Cambrian basement bedrock (fractured bedrock aquifer system). Wells and shafts have been completed in the conglomerate, plutonic crystalline rocks, and Pre-Cambrian basement rock below the Basal Fault. Potentiometric head of the basement rock (below the Basal Fault) is under pressure and is higher than the water table elevation of the overlying conglomerate and plutonic crystalline rock, indicating confined aquifer conditions below the Basal Fault. Mining operations are not intended to extend below the Basal Fault. Figure 20-2 provides a conceptual cross section of the pit.

Figure 20-2: Conceptual Cross Section



The bedrock formations at the Mine are poorly transmissive. This is evidenced by the low stage of the existing pit lake with an elevation (530 ft [161.5 m] amsl) approximately 700 ft (213 m) below the water table in the surrounding bedrock aquifer. The conglomerate unit has been reported as poorly productive, yielding only small quantities of water to wells in the area and has been observed only as seeps in the pit wall (M&A, 1986). The hydraulic conductivity of the underlying plutonic crystalline bedrock is estimated at approximately three orders of magnitude lower than the conglomerate bedrock unit, and therefore is unlikely to contribute any significant in-flow to the pit (or future underground operations). Tetra Tech (2017) analyzed isotopic samples which revealed the water in the pit to be enriched in deuterium and oxygen-18 relative to the nearby groundwater. This evidence indicates that the open pit is not a flow-through system and that expanding the pit or nearby underground operations are not likely to be subjected to a large inflow of groundwater.

The regional groundwater flow in the aquifer is from southeast to northwest. Figure 20-3 shows a “pre-development” (1941) groundwater elevation map. Although stresses and pumping in the basin since pre-development conditions have resulted in localized changes to the flow conditions, the regional gradient remains from southeast to northwest. However, localized groundwater flow at the Mine has been observed to be NE to SW.

Figure 20-4 shows the 2019 groundwater elevation map for the Mine. Depth to water is approximately 300 ft (91 m) below surface. There are no perennial surface water features near the Mine, only washes that flow intermittently following precipitation and runoff.

Figure 20-3: Pre-Development Groundwater Map

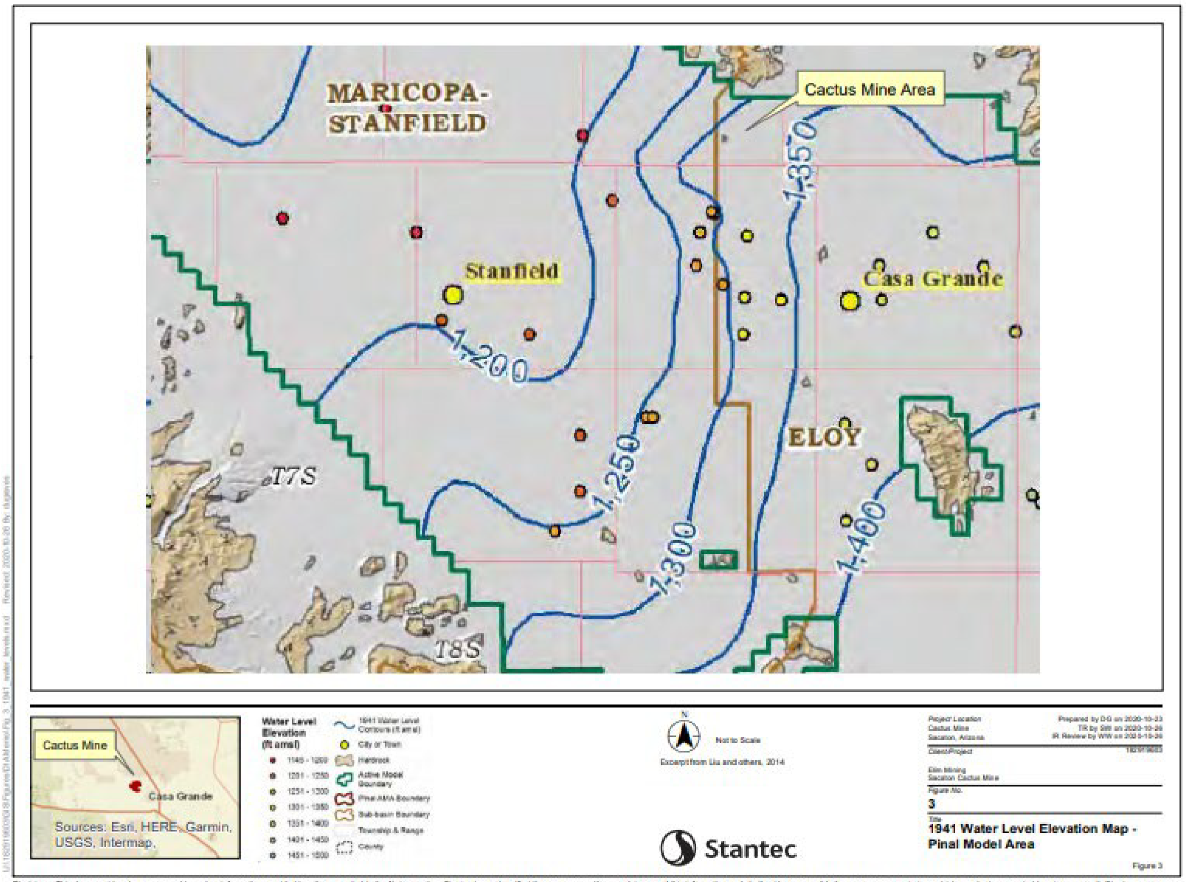
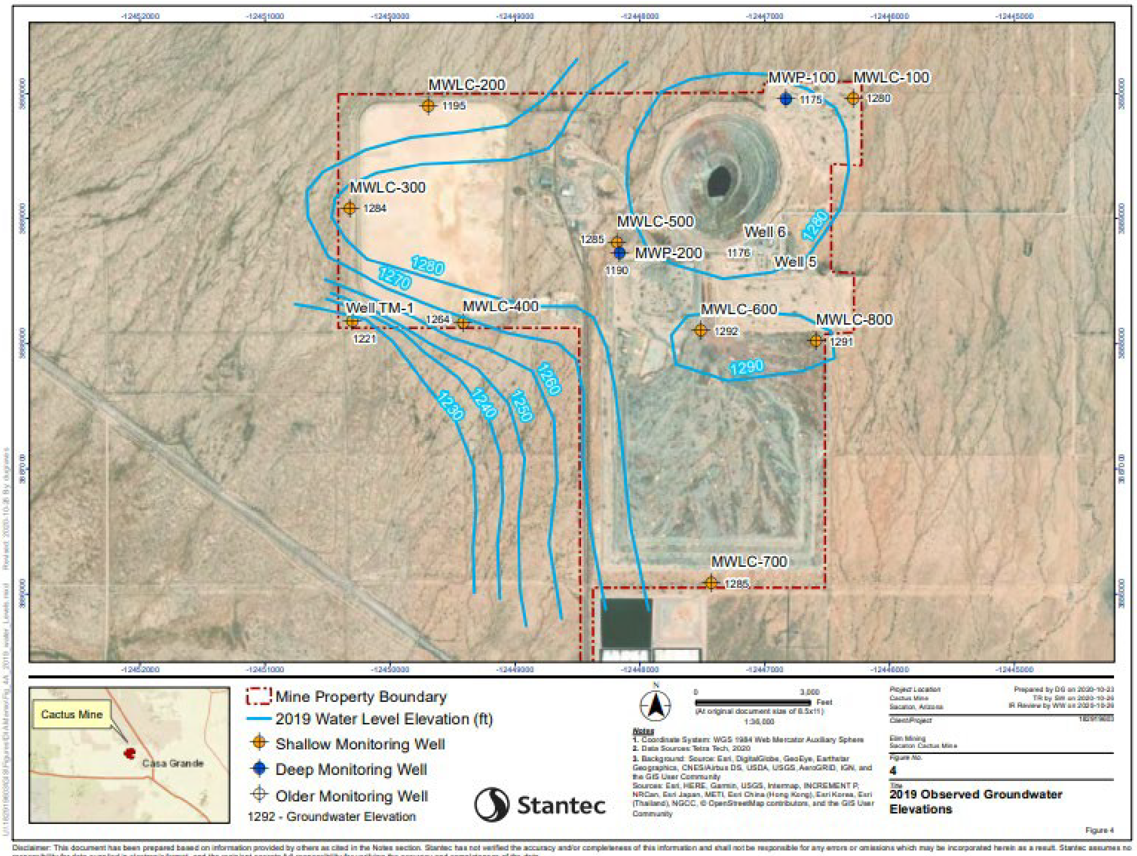


Figure 20-4: 2019 Groundwater Elevation



Hydraulic conductivity (K) of the alluvium, conglomerate, and plutonic crystalline bedrock was estimated from the Pinal AMA model (Liu and others, 2014) and from specific capacity data from nearby wells reported from Montgomery and Associates (M&A, 1986). The alluvium has a higher K than the underlying conglomerate, ranging from approximately 2-50 ft/d; however, these deposits are thin and unsaturated at the Mine. The K values of the conglomerate range from 0.5-2 ft/day, while the plutonic crystalline bedrock is estimated at 0.003-0.005 ft/d. The low K values of these formations near the Mine is evidenced by the low stage of the pit lake (approximately 530 ft amsl) as compared to surrounding groundwater elevations (approximately 1,250 ft amsl), suggesting rapid pit inflow does not occur. The specific yield and specific storage of the aquifer units were taken from the Pinal AMA model and are assigned as 0.11 for alluvium / conglomerate (specific yield) and ranges from 0.02 and 0.000002 for crystalline bedrock (specific storage).

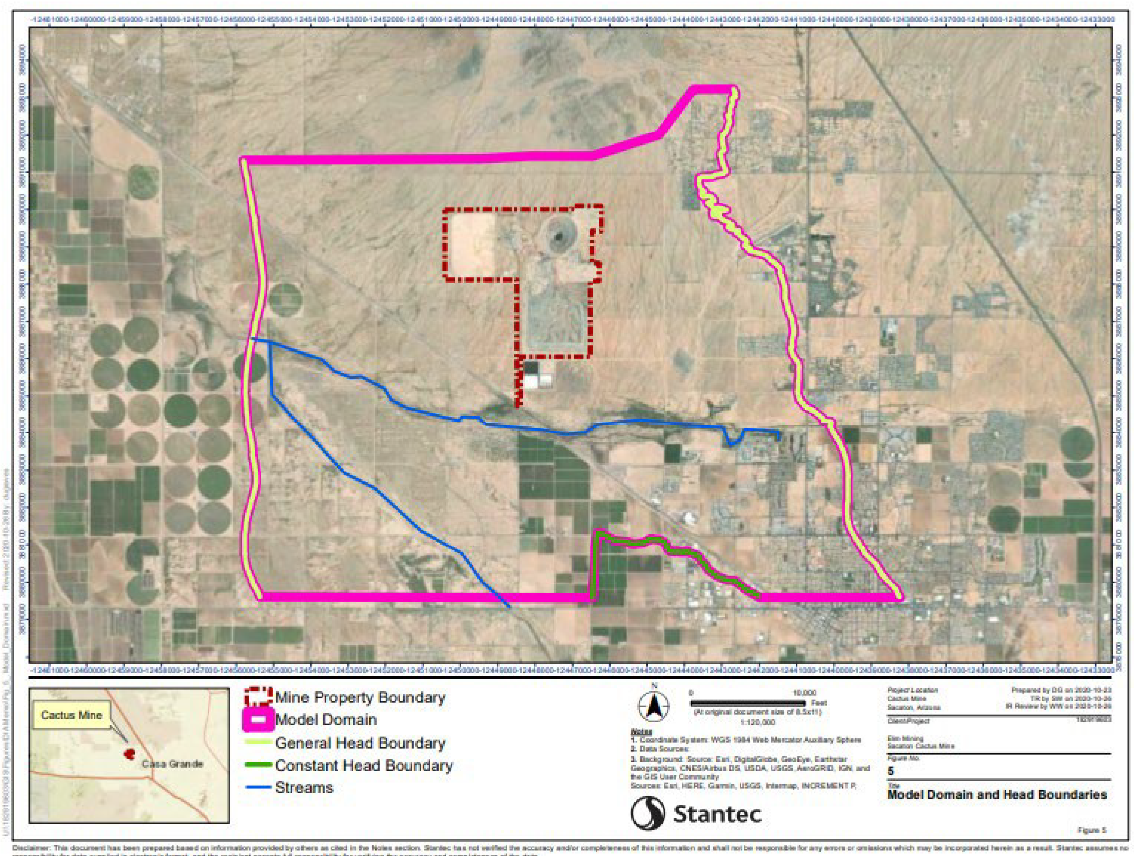
20.3.3 Numerical Groundwater Model

Groundwater flow was simulated using MODFLOW 2005, a modular three-dimensional finite difference groundwater flow model developed by the USGS. The modeling was conducted using the pre- and post-processor groundwater modeling system (GMS), a powerful, graphical interface that allows for model construction, analysis, calibration, and visualization.

The model domain is approximately 47 mi2 with a grid cell size of 100 ft (30 m) × 100 ft (30 m) and was established at the regional scale so that boundary conditions would not impact simulated flow near the Mine. Head boundaries were placed in the southeast and northwest portions of the model domain and used to establish the flow field and hydraulic gradient to closely match the regional data, established from the Pinal AMA.

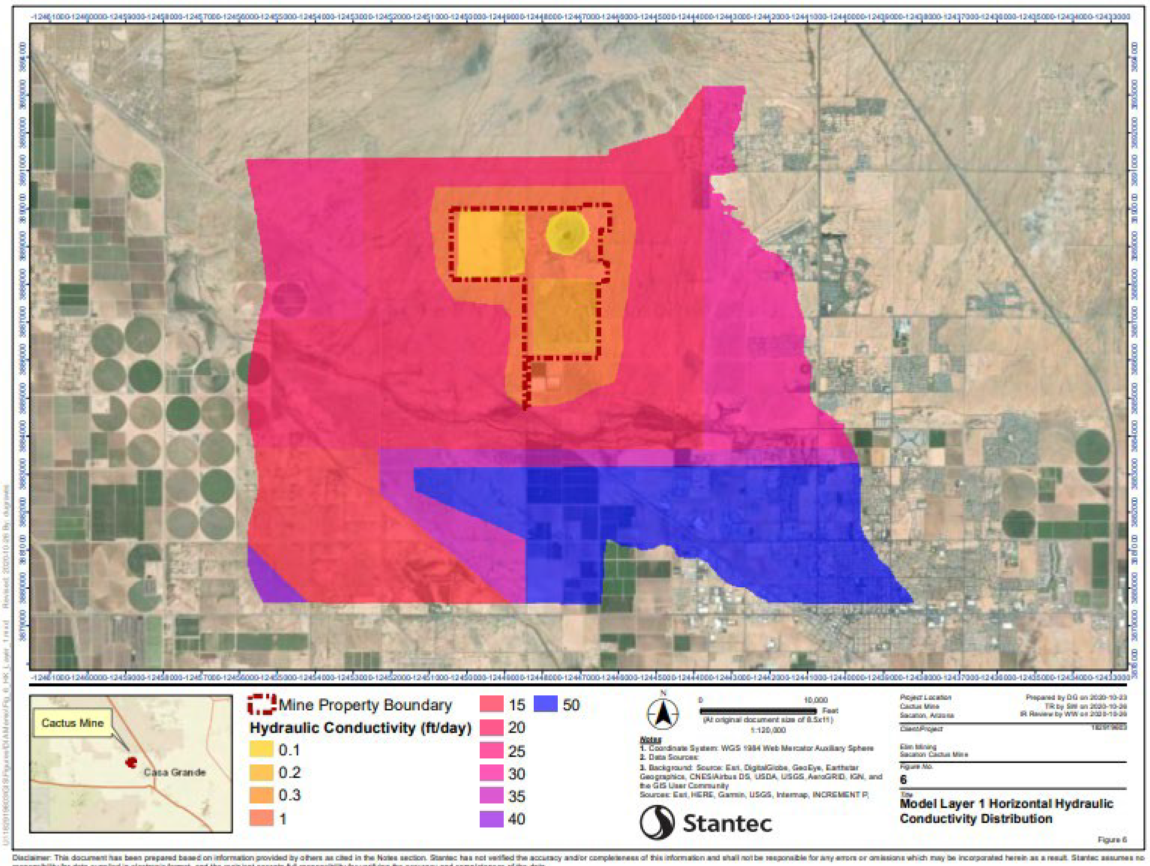
Model (Liu and others, 2014). The remainder of the model domain perimeter was established as a no-flow boundary, generally representing the basin boundaries.

Figure 20-5: Model Domain and Head Boundaries



For the numerical model development, the surficial alluvium and conglomerate bedrock are combined as Layer 1, while the underlying plutonic crystalline bedrock (ore body target) is Layer 2. The bottom of the model (Layer 2) was assigned to the approximate elevation of the Basal Fault near the Mine, as planned operations are not targeted below this fault. The distribution of K for Layer 1 is shown on Figure 20-6. A constant K value of the plutonic crystalline bedrock (model Layer 2) was assigned at 0.003 ft/day based on estimates of specific capacity near the Mine (M&A, 1986).

Figure 20-6: Hydraulic Conductivity Distribution



A steady-state model was constructed for pre-development conditions (pre-1950) and calibrated to water level observations to this time-period, available from ADWR well records. Figure 20-7 shows the modeled potentiometric surface from the calibrated steady state model and Figure 20-8 shows the steady state calibration results. The mean absolute error (MAE) for head values is 27.8 ft (8.5 m) for all calibration points, or approximately 9% of the total head change across the model domain which is considered an acceptable calibration.

Figure 20-7: Simulated Groundwater Elevations

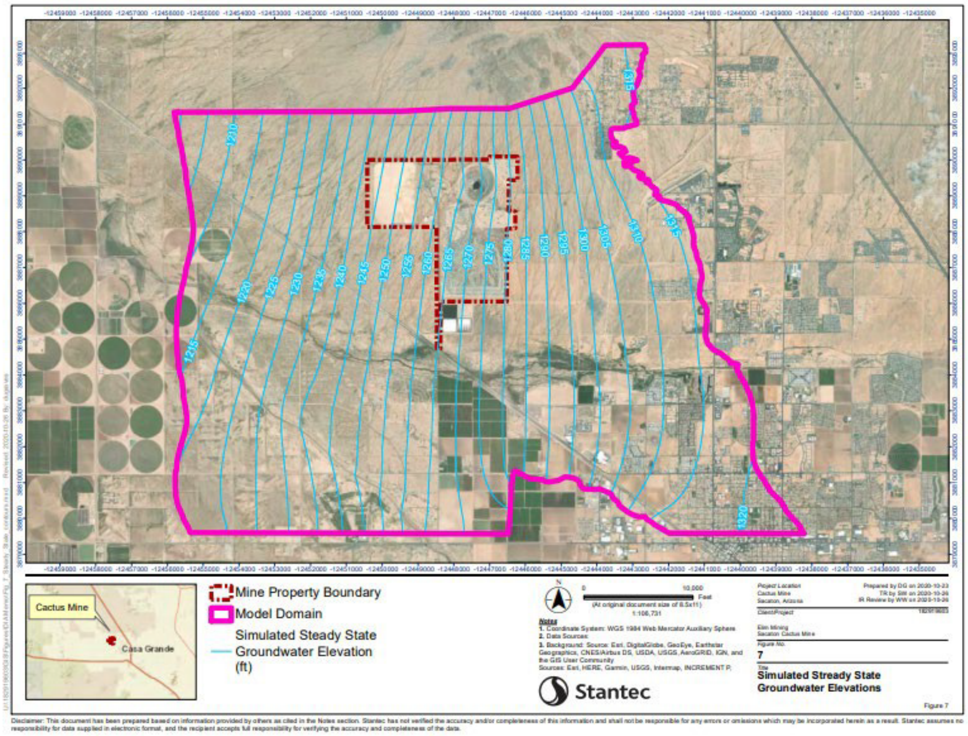
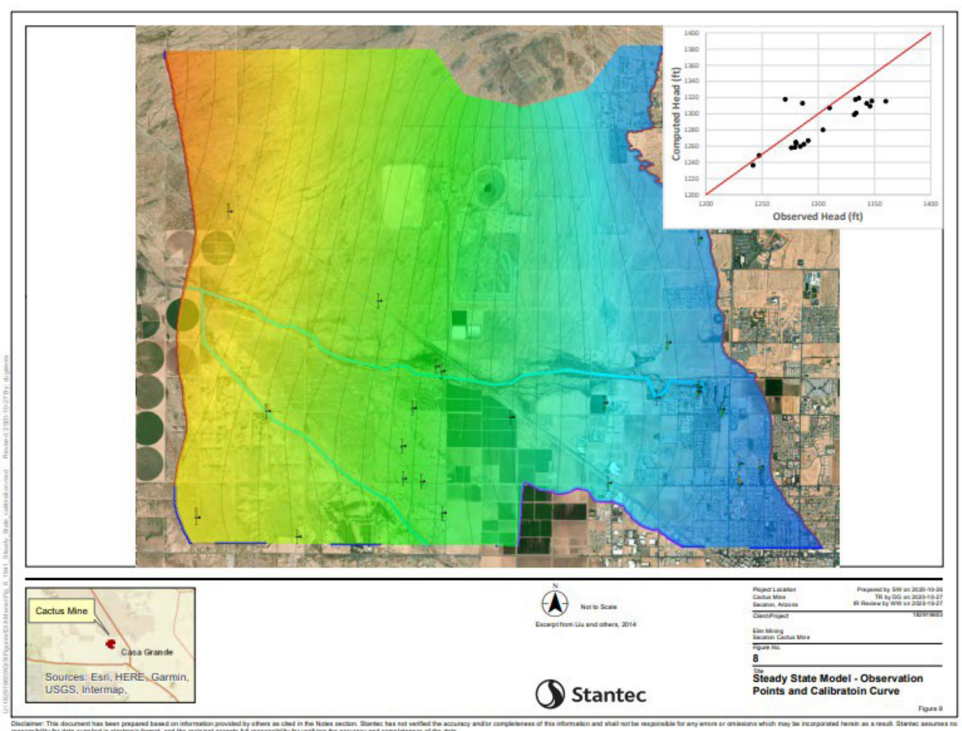
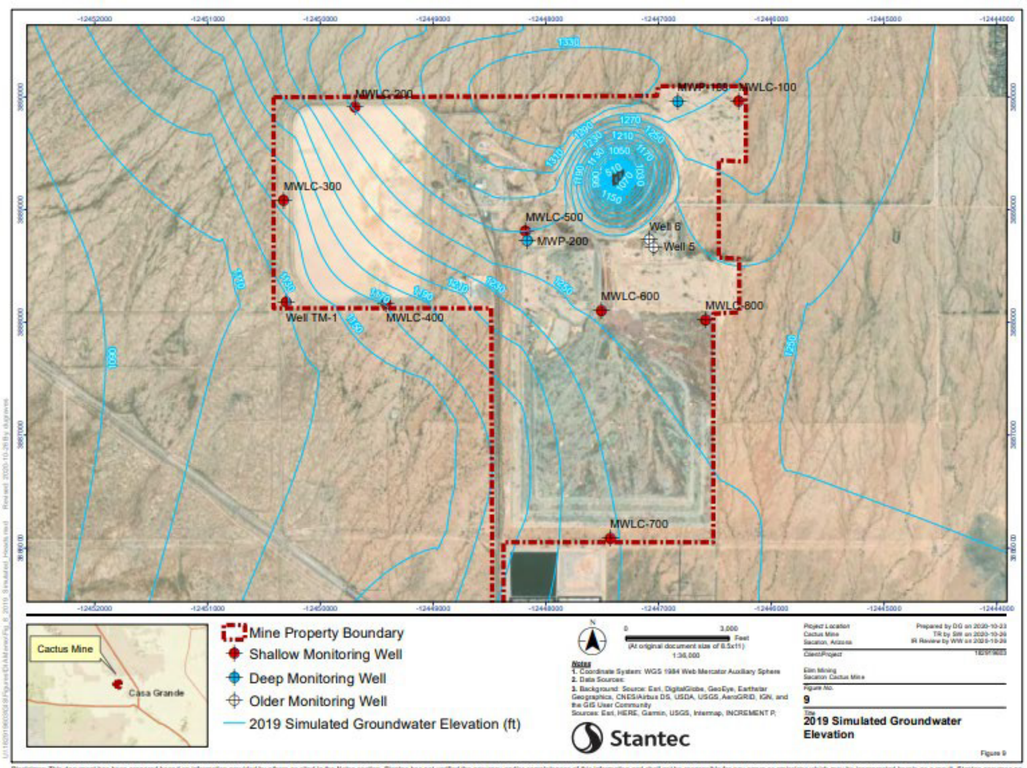


Figure 20-8: Steady State Model



Following steady-state calibration, pumpage was assigned to the model from the registered wells within the model domain using the reported pumping values per ADWR records and a transient model was established to run from 1960 to 2019 (current conditions). Additional model calibration was performed to match the groundwater flow gradient near the Mine to recent observed conditions. To simulate the pit lake, the MODFLOW Drain package was used to assign the water elevation of the lake stage. This method was effective for simulating the hydrologic sink of the pit while maintaining reasonable model calibration in the surrounding aquifer (i.e., matching simulated heads to observed heads with a MAE of approximately 32.0 ft (9.8 m), or approximately 15% of total head change across the mine site). Figure 20-9 shows the modeled 2019 potentiometric surface from the transient model. The resulting heads (2019 potentiometric surface) from the current transient model were used to establish starting heads for a predictive model scenario for the Mine to simulate a 30-year operation and a 20-year post-closure period.

Figure 20-9: 2019 Simulated Groundwater Elevation



Predictive Scenarios

For future mining operations and water management, two options were evaluated using the numerical groundwater model: (1) dewatering via wells; and (2) pit extraction via sump pumping.

Dewatering via groundwater production wells was assessed in several model simulations. Due to the low K of the bedrock formations, significant production from wells could not be achieved, which also limits the cone of depression from expanding laterally from the wellbore. Therefore, a relatively large number of wells would be anticipated to dewater the bedrock aquifer and the resulting open pit. Model simulations resulted in approximately 18 wells of 200 gallons per minute (gpm) each required to dewater the conglomerate formation (Layer 1) surrounding the pit (the more productive formation that contributes inflow to the pit).

As an alternative, extraction via pumping from the existing and future pit areas was simulated. Because the open pit is not in direct communication with the groundwater flow system and infiltration rates are low (as evidenced by the pit lake stage approximately 700 ft [213 m] below the groundwater table), mine water management should be effective via sump pumping from the open pit or underground operations. This was simulated by assigning the pit elevations via drains and evaluating the model output to estimate the outflow (i.e., required pumpage) of the drain features. These results indicate approximately 300-450 gpm of pumping from the future pit areas may be required, depending on the selected mining method and progression schedule.

Model Results

MODPATH particle tracks were used to estimate the capture zone during the simulated operations (and pump extraction) from the open pits. Figure 20-10 shows the potentiometric surface in 2050, following a 30-year simulated operation (pumping from open pits) and resulting groundwater flow particle tracks at the termination of the 30-year period. Although the pit is a hydrologic sink, the surrounding groundwater gradient is from NE to SW, similar to recent observations. Figure 20-11 shows the resulting potentiometric surface in 2070, to represent approximately 30 years of operation and 20 years of mine closure. The simulated groundwater flow conditions remain similar to the 2019 and 2050 conditions.

Figure 20-10: 2050 Simulated Groundwater Elevation

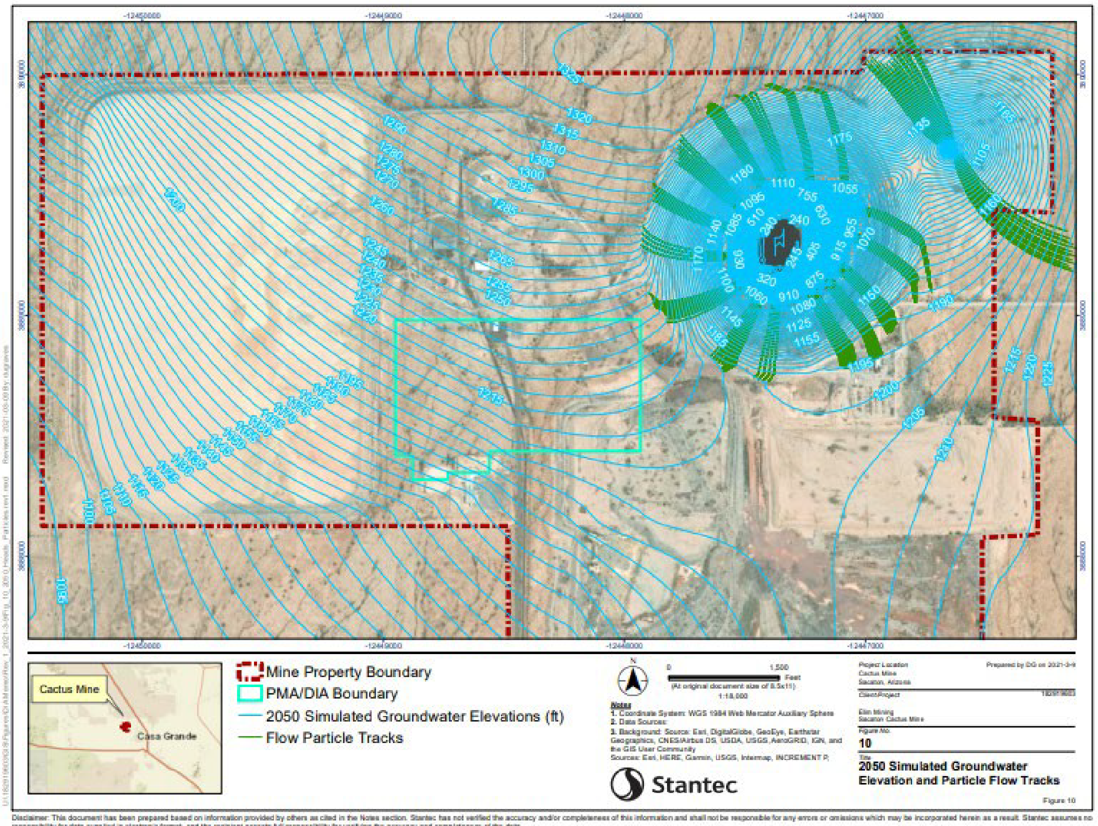
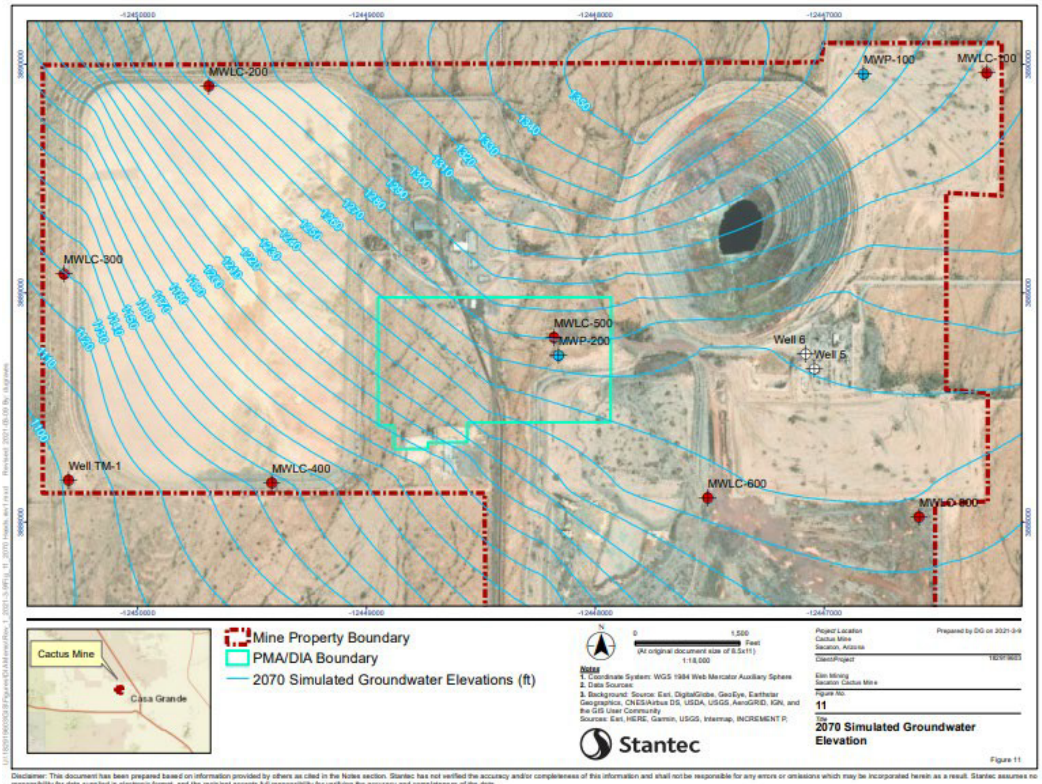


Figure 20-11: 2070 Simulated Groundwater Elevation



20.3.4 Water Quality and Water Management

Groundwater quality below the site has been reported as slightly saline (relatively high specific conductance or total dissolved solids [TDS]), while pit water has been reported as poor quality (Tetra Tech, 2017). Historic groundwater sample data indicates relatively high concentrations of several parameters including arsenic (ranging from approximately 0.02-2.1 mg/L), fluoride (ranging from approximately 3-19 mg/L), TDS (ranging from approximately 550-10,000 mg/L), nitrate (ranging from non-detect to approximately 68 mg/L), and uranium (ranging from non-detect to approximately 0.1 mg/L). Many of the sample result concentrations exceed respective Arizona Aquifer Water Quality (AWQ) standards; however, the ranges of concentrations for these parameters is not uncommon in aquifers of central and southern Arizona.

The pit lake is a terminal hydrologic sink, and seepage inflow is subjected to evaporation which enriches the mineral content of the water. Historic pit lake water quality (M&A, 1986, Tetra Tech, 2017) indicates elevated concentrations of arsenic (0.06 mg/L), fluoride (6-84 mg/L), nitrate (6.5-48 mg/L), and TDS (8,400 mg/L), and a relatively low pH (4.1). Although other metals and ion concentrations are higher than the surrounding groundwater concentrations (enriched within the pit lake water), they are unlikely to cause concern for water management.

If mine operation water management includes dewatering the pit lake, water quality considerations should be factored into the management design strategy. If pumped water will be applied to the heap leach and captured (operations adhere to the BADCT), water quality considerations may not be imperative. However, if any discharge is anticipated, permit conditions may dictate water quality thresholds and treatment technology may be necessary for compliance.

20.4 Social or Community Impact

In keeping with Arizona Sonoran's community engagement and partnership standards, the Cactus Project will be developed with a plan to establish and maintain the support of our host communities.

Arizona Sonoran has commenced early-stage community outreach and is currently evaluating partnerships within the community. As the Cactus Project's permits will involve a public process and is based on the permit submission and review schedule, Arizona Sonoran plans to elevate outreach during the permitting process and throughout the life of the mine.

Some steps have been completed and others have been delayed due to COVID-19. The following actions have been completed or are planned.

1. Creation of a Conduit for Concerns
   Establishment of a website, email, and phone conduit for members of the community to contact with concerns. All interactions are logged, and Arizona Sonoran takes steps to address legitimate concerns as expediently as possible. A record of all steps taken to address concerns is maintained by the external relations team.

2. Community Partnerships
   Formation of partnerships with community service organizations to identify the needs of the community. This is achieved through the creation of a community partnership foundation with grant and in-kind standards—a committee of community members and Arizona Sonoran representatives is planned for goal setting and decision-making. We have also identified several high-visibility projects that will help the community.

3. Interact with Local Government
   Attendance of virtual (and in-person post COVD-19 when it is safe and permitted to do so) local council and county board of supervisors' meetings and present project updates. Capture pressing issues, both related and unrelated to mining, with local governments. Reach out and offer site tours and briefings to all interested local and county officials. Create a schedule and develop a plan for frequency of visits with local officials. Keep community leaders and elected officials up to date on project developments.

4. Interaction with Opposition Groups
   With acceptance that some groups might not change their position on mining, Arizona Sonoran will take steps to create a constructive and friendly dialogue, and address attainable concerns.

5. Participate in Local Events
   Create and maintain a calendar of local events. Sponsor, support, and attend events as frequently as possible. Keep a record of events and sponsorships.

6. Advertise Local Partnerships and Sponsorships
   Make sure that the community is aware of your investments.
   The following steps will be completed post-COVID-19, when it is safe, permitted, and prudent to do so.

7. Open Houses
   Lead periodic project open houses, which differ from any potential ADEQ or other agency statutory requirements. Open houses should be advertised to drive community attendance.

8. Community Support Coalition
   Identify enthusiastic community members that will be willing to voice support via person-to-person interactions, comment letters, editorials, and social media posts. Convene meetings of these individuals as needed and make sure they are informed of project development and milestones.

9. Downtown Office
   Plan on a small community office in Casa Grande with enough space to hold and host meetings and maintain a visible presence within the community. Advertise office hours in local publications. The office may be offered to community service organizations as a resource for meetings.

Arizona Sonoran conducted public opinion research related to the re-development of the site in 2019. The data showed significant support for the Project in the region. Arizona Sonoran has committed to maintaining and growing their support over the life of the operation. The polling data also provided Arizona Sonoran with useful information regarding messages resonate with the community. Arizona Sonoran has developed a comprehensive environmental, social and governance framework which aims to address any community concerns and operate Cactus in a socially responsible manner. The framework includes the following attributes.

Responsible Operations

Arizona Sonoran is committed to investing in technological efficiencies, including low-carbon and water-efficient technologies. Arizona Sonoran aspires to design all facilities to meet or exceed BADCT standards as well as ensure strict adherence to any operating permit limits, including aquifer protection, stormwater discharge and air quality allowances. Arizona Sonoran also plans to operate with zero wastewater discharge. Arizona Sonoran continues to explore the use of renewable energy for its operations and in this context, it is exploring renewable energy concepts to support its operations. Arizona Sonoran's ability to reduce its carbon footprint will be further aided by the APS, which plans to produce up to 65% of its total energy from renewable resources by 2030, and up to 100% by 2050.

Arizona Sonoran has taken a proactive approach towards limiting environmental impacts and in this context, to improve biodiversity, Arizona Sonoran has commenced conducting biodiversity surveys on its properties. While Arizona Sonoran has not found any protected species on its properties, Arizona Sonoran has created a proactive plan to protect and enhance natural habitat for Cactus Wren, Saguaros, and Ironwoods, three iconic species in the State of Arizona. Additionally, in early 2021, Arizona Sonoran planted 220 trees along Bianco Road, the main access to the Cactus Project property, to increase biodiversity, reduce dust and beautify the landscape.

Positive Work Culture

Arizona Sonoran aspires to provide meaningful work opportunities and prioritize worker wellbeing and safety. Arizona Sonoran’s success is directly linked to the health and safety practices at its operations. All Arizona Sonoran employees, contractors, and visitors are required to adhere to best practices in health and safety. Arizona Sonoran conducts daily safety briefings with all employees and contractors working on site to build a culture of safety and vigilance. This has included specific protocols to protect employees against the spread of COVID-19. Arizona Sonoran is continuing to build its safety program as it transitions to development of the Cactus Project. In 2020, Arizona Sonoran had 40,387 hours worked with zero fatalities and zero lost-time incidences.

Beyond workplace safety, Arizona Sonoran strives to provide rewarding work and development opportunities. Arizona Sonoran provides competitive wages and benefits and promotes work-life balance. It is also committed to creating a diverse, equitable and inclusive workplace where human rights are respected and enforced through its diversity and inclusion policy.

Part of the Community

Arizona Sonoran is committed to supporting local economic development and an open dialog with all stakeholders. Arizona Sonoran regularly meets with local community leaders and, regional and state level lawmakers and officials and heads of educational institutions, to share its plans and to better understand community needs. The company also maintains a public hotline to answer any questions and solicits community partnerships and local procurements.

Arizona Sonoran will complete a stakeholder identification and mapping exercise during the permitting process. This will help Arizona Sonoran identify and prioritize outreach activities. The information collected should feed into a useful index of information that aids in the eventual development of the stakeholder map. The stakeholder mapping process will identify key stakeholders among a wide range of individuals and groups. Attention will be paid to the process of information gathering, making sure that it is across a wide spectrum of inputs.

The stakeholder map will guide and prioritize stakeholder interactions based on a range of factors.

Arizona Sonoran recognizes that with operations in dry and arid climates, dust can be an issue. Arizona Sonoran will take additional measures to ensure that it is in compliance with its Air Quality permit by using soil stabilizers, watering and pavement to addressing potential dust generating areas.

Noise from mining activity is also another factor that Arizona Sonoran will look at addressing by installing engineering controls during the design of the facilities.

21.0 CAPITAL AND OPERATING COSTS

The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as at 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report.

21.1 CAPEX

The estimated initial preproduction capital cost for the Cactus Project is $127 million with details presented in Table 21-1.

Table 21-1: Initial Capital Cost Estimate

Capital Costs			-2	-1	0	1
Project Infrastructure	US$	-				
Leachpad Infrastructure	US$	(24,500,000)			(20,000,000)	(4,500,000)
SX-EW Facilities	US$	(74,000,000)			(50,000,000)	(24,000,000)
Flotation Processing Facilities	US$	-				
Tailings Facilities	US$	-				
Capitalised Drilling – Cactus Orebodies	US$	(7,833,238)	(5,013,878)	(2,819,359)		
Capitalised Drilling – Stockpile	US$	-				
Technical Studies	US$	(4,100,543)	(2,696,543)	(1,404,000)		
Project/Other Costs	US$	(2,582,841)	(1,003,000)	(1,579,841)		
OP- Capitalised Stripping	US$	(47,085,000)				(20,835,000)
UG-Capitalised Development	US$	(29,124,000)				-
Mobile Mine Equipment (OP_UG)	US$	-				
Mine Equipment (OP_UG_	US$	-				
Sustaining Capital – Leachpad Facilities	US$	(74,600,000)				
Sustaining Capital – SX-EW Facilities	US$	(26,000,000)				
Sustaining Capital – Open Pit	US$	(130,979,500)				-
Sustaining Capital – UG	US$	(108,752,000)				-
Exploration	US$	-				
Land Acquisitions	US$	(27,475,000)	(7,000,000)	(7,525,000)	(7,950,000)	
TAGC Founders Fee	US$	(1,100,000)			(300,000)	(500,000)
Cash Reclamation	US$	(5,000,000)				
Salvage Value	US$	5,000,000				
Total CAPEX	**US$**	**(558,132,122)**	**(15,713,421)**	**(13,328,201)**	**(78,250,000)**	**(49,835,000)**

The capital cost estimate was put together by Stantec, Samuel Engineering, and Arizona Sonoran based on industry benchmarking, historical information recovered for the site, 2020 project resource drilling and analysis, preliminary metallurgical bottle roll and column testing of fresh mineralized material and Stockpile Project samples, preliminary flowsheet, conceptual heap leach and SX/EW processing facilities.

The costs reflect the construction capex required to bring the project into production and includes $23 million for other costs such as land payments. Another $99 million is allocated for initial SX/EW and leach pad facilities. The construction cost does not include the cost of open pit stripping for the first year ($21 million) or prefeasibility and feasibility stage work (totaling $16 million as of start of July 2021).

A contingency of 15% has been included in the capital cost for ancillary mine equipment, leach pad infrastructure and the SX/EW facility. Contingency is an allowance to cover unforeseeable costs that may arise during the project execution, which reside within the scope-of-work but cannot be explicitly defined or described at the time of the estimate due to lack of information. It is assumed that contingency will be spent; however, it does not cover scope changes or Project exclusions.

The estimate is expressed in second-quarter 2021 United States dollars. No provision has been included to offset future escalation.

21.1.1 Process CAPEX

For the process capital cost, a distributed percentage factoring technique has been used which is often employed when developing an estimate for a process facility at a preliminary stage where there is a lack of design data and specific requirements from which to base costs.

Table 21-2 details the initial capital required to build process facilities to support initial copper production of 22,000 tpa.

Table 21-2: Process Initial Capital Expenditure

Direct and Indirect Cost Components	Leach Pads, Ponds, and Pipelines	SX/EW Facility	Total Capital Cost
Description	**Cost (USD)**	**Cost (USD)**	**Cost (USD)**
Directs			
Mechanical Equipment	0	24,545,000	**24,545,000**
Civil	16,638,000	1,849,000	**18,487,000**
Foundations	68,000	2,369,000	**2,437,000**
Structures	0	1,386,000	**1,386,000**
Buildings	0	1,849,000	**1,849,000**
Piping	1,013,000	8,318,000	**9,331,000**
Electrical	706,000	3,882,000	**4,588,000**
Instruments	0	1,035,000	**1,035,000**
Miscellaneous	0	665,000	**665,000**
Subtotal Directs	**18,425,000**	**45,898,000**	**64,323,000**
Indirects			
Contractor Indirect	Included Above	4,720,000	**4,720,000**
Construction Equipment	Included Above	2,360,000	**2,360,000**
Surveying and Testing Services	666,000	225,000	**891,000**
EP Services	1,105,000	4,049,000	**5,154,000**
Construction Management	921,000	3,179,000	**4,100,000**
Vendor Reps	0	555,000	**555,000**
Spare Parts	0	277,000	**277,000**
Initial Fills	0	500,000	**500,000**
Commissioning	0	443,000	**443,000**
Freight	368,486	2,803,000	**3,171,486**
Mining Equipment	0	0	**0**
Owner's Cost	Excluded	Excluded	**Excluded**
Taxes	Excluded	Excluded	**Excluded**
Subtotal Indirects	**3,060,486**	**19,111,000**	**22,171,486**
Contingency	3,008,000	9,036,000	**12,044,000**
Total Cost (USD) 22 ktpy (initial)	**24,493,486**	**74,045,000**	**98,538,486**

21.1.2 Open Pit and Underground Development CAPEX

A total initial capitalized stripping cost of $47 million is included in the Economic Model for the first two years in respect of the open pit. This reflects 16 benches mined for a total of 32 million tons of waste at a blended cost of $1.50/ton, based on the shorter haulage distance to waste dump facilities.

Underground development costs of $29 million are also included as development CAPEX for years 5 and 6 reflecting twin ramp development and associated infrastructure at an equivalent cost of $90/ton mined.

21.1.3 Land Acquisition Costs

Arizona Sonoran has entered into binding obligations to make payments for land acquisitions in relation to the Project for a total outstanding amount of approximately $23 million. These payments are included in the Project total capital cost for the purposes of the economic model.

21.1.4 Sustaining Capital

Underground

Sustaining capital of $109 million has been included based on the LOM schedule and includes mine ramp spiral and level development and associated infrastructure at an equivalent cost of $70/ton mined.

Open Pit

A total open pit sustaining capital of $131 million has been included based on the mine schedule. Open pit sustaining capital has been developed based on the basis of mineralized material access with unit sustaining costs rising as the pit deepens. Applicable unit costs for relevant years are as noted in Table 21-3.

Table 21-3: Open Pit Sustaining CAPEX

2025 and 2026	$1.75/ton
2027	$2.00/ton
2028-2031	$2.15/ton

Leach Pad/SX EW

Construction costs for future phases of leach pads at approximately $74 million, and a plant expansion to 35k ton Cu of $25 million are carried as sustaining costs in the economic model.

No other sustaining capital is anticipated for the facilities given the nature of the planned expansions along with contractor mining.

21.1.5 Estimate Lead and Methodology

For process construction, and in factored estimates such as this, the supply cost of the mechanical equipment for the facilities is used as the basis for calculating the overall cost of the facility. Various percentages of the equipment costs are then applied to obtain values for each of the prime commodity accounts, which include earthwork, concrete, structural steel, mechanical, piping, electrical, and instrumentation.

Budgetary pricing for all mechanical equipment was obtained as either a packaged supply from Metalex or from recent project quotation for similar equipment and factored to the project scale as necessary.

The basis of mechanical equipment costs used in this estimate include budgetary equipment pricing from vendors, in-house historical data, and costs from Mine & Mill.

The distributive percentage factoring is applied to both the labor for installation as well as for the cost of materials within each prime commodity account.

Project execution will follow a typical EPCM approach. The execution timeframe considered is approximately 16 months from notice to proceed through commissioning completion.

Project ramp-up will be commensurate with heap leaching pad development.

Installation cost of the equipment is based on a percentage of the equipment cost. The percentages range from 15%-35%, depending on equipment type.

An allowance of 5% of the identified mechanical equipment cost is included for miscellaneous equipment not yet identified at this stage of the Project.

No quantity takeoffs for materials and have been performed. All direct costs other than the mechanical equipment, including the civil, concrete, steel, buildings, piping, electrical, and instrumentation components of the SX/EW facility have been derived as factored percentages of the mechanical equipment cost.

All direct costs other than the mechanical equipment has been factored and distributed as percentages of the mechanical equipment cost.

Owner's costs (e.g., PM, permitting, land acquisition (save as otherwise mentioned above), insurances, etc.) or external risk factors (e.g., escalation, political, weather, force majeure, etc.) have not been addressed in this estimate.

21.1.6 Mechanical Equipment Basis

All mechanical equipment is assumed to be procured by either the engineer or Arizona Sonoran and provided free issue to the construction contractor for installation; thereby avoiding any third-party markup.

Costs assume that equipment and materials will be purchased on a competitive basis, and installation contracts will be awarded in well-defined packages.

The cost of mechanical process equipment is the backbone of the SX/EW facility estimate and the other costs within the plant facility are factored percentages of the equipment.

A preliminary mechanical equipment list was derived from a basic set of flowsheets which was used to locate pricing for similar size / duty items from recent historical project cost data.

Metalex, a company based in Santiago, Chile, designs and supplies small, modular, relocatable, and standard SX and EW plants for the recovery of copper has provided the bulk of the equipment list and budgetary pricing for this capital cost.

Metalex has based the SX/EW equipment for Cactus on the designs from two other operating facilities, Benkala Copper Mine and Andacollo. Metalex has endeavored to combine the best features of each in order to provide Arizona Sonoran with a package with that maximizes amount of preassembly that can be done, thereby minimizing the time needed onsite for field installation.

To round out the mechanical equipment supply, Samuel Engineering has attempted to identify and price from historical data any other major equipment that has not been included with the Metalex equipment package.

21.1.7 Estimate Accuracy

Minimal design has been performed on the facilities other than preliminary flowsheets and rough plot plan layouts. The design will continue to evolve throughout future studies.

Construction materials, quantities, equipment selection and sizing as well as other design development issues are not resolved at this stage. Costs will increase and decrease as designs develop and the scope is narrowed.

The order of magnitude capital cost has been developed to a level sufficient to assess / evaluate the project concept and overall viability. The estimate can be classified as an AACE Class 5 estimate.

As a final check, the Lang Factor (ratio of the overall capital cost divided by the process equipment cost), has been reviewed against the historical norms and the particular attributes of the facility under analysis. In the case of a fluids type processing facilities, the generally accepted Lang Factor is around 4.74. The overall Lang Factor of the SX/EW in this estimate is about 4.0. The general complexity level of the project relatively low, and much of the equipment is intended to be modularized.

21.1.8 Mining Equipment (Contract Mining Base Case)

The capital cost estimate assumes that the mining contractor will provide all the equipment, temporary facilities and dedicated infrastructure required to perform the mining services within the rates included in the operating cost estimate. No additional mining related capital has been included.

21.1.9 Assumptions and Exclusions

The following assumptions have been made in developing the Project's capital cost.

- Assumes contractor mining and no additional equipment is required for the mining contractor.
- Mobile light duty equipment is assumed to be leased not purchased.
- Pursuant to recent land acquisitions, new fencing around the facilities is required. Some minor repairs or new gates may be necessary.
- It is assumed that there will be no buried interferences. No allowance has been made in the estimate for any utility relocations or demolition. Additionally, no allowances have been made for encountering hazardous waste or other buried items.
- There are sufficient water rights available sourced from both off-site and on-site wells that can be used to supply fresh water to the plant.

Exclusions

Items not included in the capital estimate are as follows.

- Mobile equipment (except cathode forklift).
- Utility power transmission lines and substation, including the main transformer.
- Access roads.
- Ancillary buildings and/or refurbishment of other existing buildings.
- Allowance for special incentives (schedule, safety, etc.).
- Taxes.
- Working capital, sustaining capital, interest, and financing cost.
- Force majeure occurrences, such as risk due to labor disputes, permitting delays, etc.

21.2 Preliminary Project Execution and Schedule

Project execution will follow a typical EPCM approach. The execution timeframe considered is approximately 16 months from notice to proceed through commissioning completion.

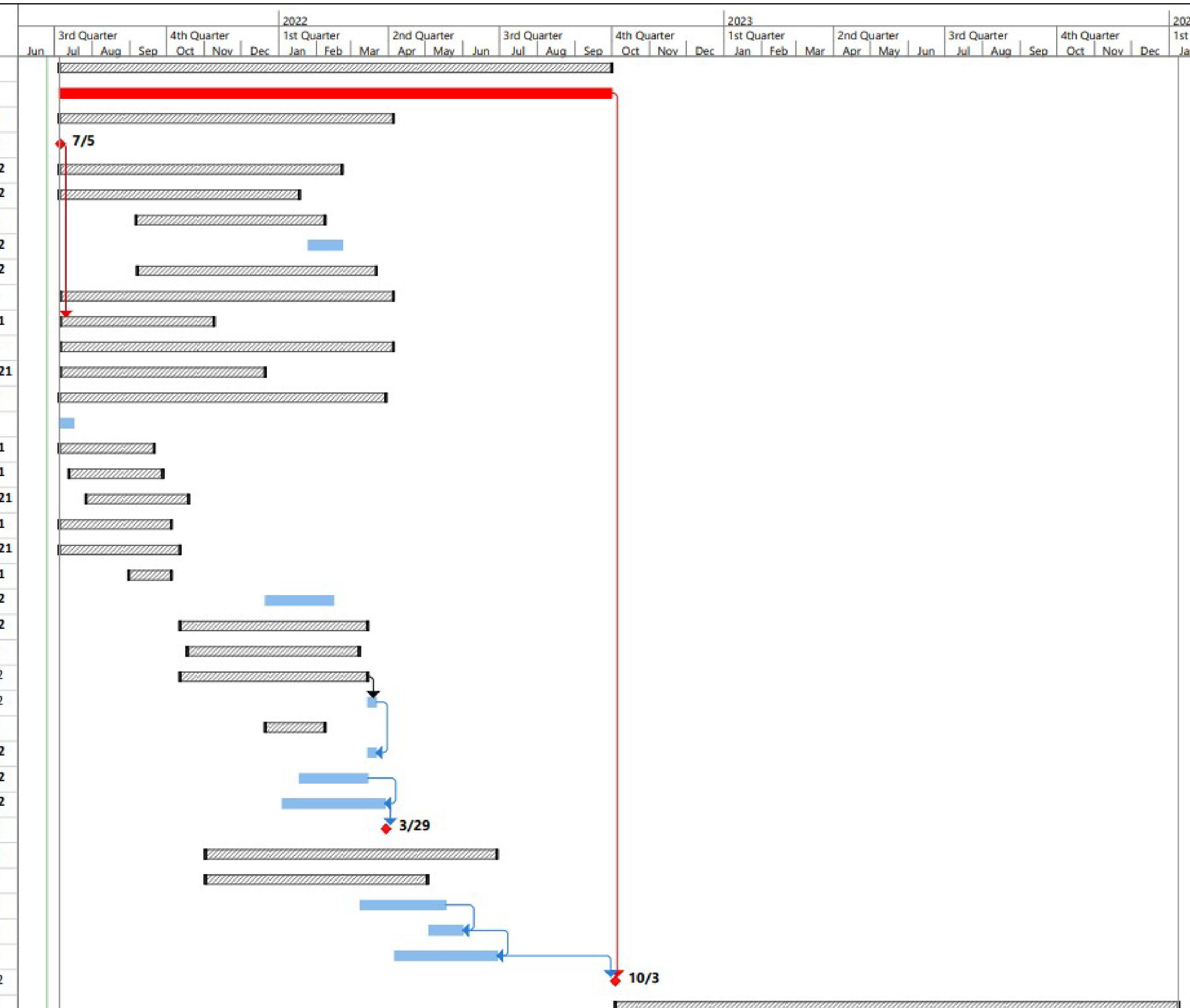
Project ramp-up will be commensurate with heap leaching pad development. A preliminary development schedule is included in Figure 21-1.

Permitting and long lead order timelines are the highest risks to the proposed schedule development plan. Equipment delivery times, particularly the rectifier-transformer units, is expected to be over 6-8 months based on Metalex's budget estimate. Equipment delivery will drive the timeline for completion of the Project.

The schedule is as extracted from the 2021 Cactus PEA.

Figure 21-1: Preliminary Project Execution Schedule

ID	Task Name	Duration	Start	Finish
1	Project Execution Plan	325 days?	Mon 7/5/21	Fri 9/30/22
2	Project Permitting	65 wks	Mon 7/5/21	Fri 9/30/22
3	Integrated Pre-Feasibility Study	196 days?	Mon 7/5/21	Mon 4/4/22
4	Pre-Fesibilty Study Start	1 day?	Mon 7/5/21	Mon 7/5/21
5	Resource Development	166 days	Mon 7/5/21	Mon 2/21/22
6	Drilling Program	141 days	Mon 7/5/21	Mon 1/17/22
9	Resource Model Finalization	111 days	Mon 9/6/21	Mon 2/7/22
12	Reporting	4 wks	Tue 1/25/22	Mon 2/21/22
13	Mine Engineering & Planning	140 days	Tue 9/7/21	Mon 3/21/22
19	Metallurgical Development	195 days	Tue 7/6/21	Mon 4/4/22
20	Stockpile Column FS Testing	90 days	Tue 7/6/21	Mon 11/8/21
24	Cactus West/East Column Testing	195 days	Tue 7/6/21	Mon 4/4/22
41	Geotechnical Design Development	120 days	Tue 7/6/21	Mon 12/20/21
50	PFS Engineering Development	192 days?	Mon 7/5/21	Tue 3/29/22
51	Project Design Basis	2 wks	Mon 7/5/21	Fri 7/16/21
52	Process Engineering	56 days?	Mon 7/5/21	Mon 9/20/21
64	Mechanical Engineering	55 days	Tue 7/13/21	Mon 9/27/21
69	Plant Design	60 days	Tue 7/27/21	Mon 10/18/21
74	Structural/Architectural Engineering	66 days	Mon 7/5/21	Mon 10/4/21
80	Electrical Engneering	71 days	Mon 7/5/21	Mon 10/11/21
88	Instrumentation & Controls Engineering	25 days	Tue 8/31/21	Mon 10/4/21
93	Project Execution Plan	8 wks	Tue 12/21/21	Mon 2/14/22
94	Capital Estimate	110 days	Tue 10/12/21	Mon 3/14/22
95	Basis of Estimate	101 days	Mon 10/18/21	Mon 3/7/22
98	Estimate Development	110 days	Tue 10/12/21	Mon 3/14/22
101	Contingency Analysis	1 wk	Tue 3/15/22	Mon 3/21/22
102	Execution Schedule	35 days	Tue 12/21/21	Mon 2/7/22
105	Risk Analysis	1 wk	Tue 3/15/22	Mon 3/21/22
106	Financial Analysis	8 wks	Tue 1/18/22	Mon 3/14/22
107	NI 43-101 Report Development	12 wks	Tue 1/4/22	Mon 3/28/22
108	Report Complete	1 day?	Tue 3/29/22	Tue 3/29/22
109	Integrated Definitive Feasibility Study	171 days?	Tue 11/2/21	Tue 6/28/22
110	FEED Engineering	130 days?	Tue 11/2/21	Mon 5/2/22
116	Updated Cost Estimates	10 wks	Wed 3/9/22	Tue 5/17/22
117	Updated Financial Analysis	4 wks	Wed 5/4/22	Tue 5/31/22
118	NI 43-101 Report	12 wks	Wed 4/6/22	Tue 6/28/22
119	Project Decision	1 day?	Mon 10/3/22	Mon 10/3/22
120	EPCM Proejct Development	330 days	Tue 10/4/22	Mon 1/8/24



Project: Cactus Mine Project De
Date: Fri 6/25/21

Task
Split
Milestone
Summary
Project Summary
Inactive Task
Inactive Milestone
Inactive Summary
Manual Task
Duration-only
Manual Summary Rollup
Manual Summary
Start-only
Finish-only
External Tasks
External Milestone
Deadline
Critical
Critical Split
Progress
Manual Progress

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21.3 OPEX

The operating costs for the Project were developed based on a combination of benchmarks, direct build-up from metallurgical parameters, typical unit consumption and costs for similar operations and factoring.

For the SX/EW plant and based on an initial plant size of 22,000 t annual copper production, the direct operating costs are expected to average $0.59/lb of copper cathode produced through the first six years of production, as presented in Table 21-4.

No contingency has been included in the operating costs presented. Taxes are considered in the financial analysis model.

Table 21-4: Processing Annual Operating Cost Estimate Summary

Average Yr 1-6 Cactus Mine Operating Costs - 23.2 ktpy Cu & Combined Tons									
	Units	Unit Consmptn	Consumption Rate		Unit Price		Annual Cost	$/ton Processed	$/lb Copper
Power	**kWh/lb**	**1.60**	**8485**	**kWh**	**$ 0.058**	**$/kWh**	**$ 4,278,900**	**$ 0.24**	**$ 0.09**
EW	kWh/lb	1.00	5303	kWh	$ 0.058		$ 2,674,312	$ 0.15	$ 0.06
SX/TF	kWh/lb	0.45	2387	kWh	$ 0.058		$ 1,203,441	$ 0.07	$ 0.03
Utilities/Misc.	kWh/lb	0.15	796	kWh	$ 0.058		$ 401,147	$ 0.02	$ 0.01
SX/Reagents							**$ 1,953,259**	**$ 0.11**	**$ 0.04**
Extractant	kg/kg Cu	0.005	289	kg/d	$ 9.95	$/kg	$ 1,048,399	$ 0.06	$ 0.02
Acid			2	tons/d	$ 120.00	$/ton	$ 87,600	$ 0.00	$ 0.00
EW Reagents (Cobalt, Guar, FC1100)			$ 0.015	$/lb Cu			$ 660,000	$ 0.04	$ 0.01
Diluent					15%	% of Ext $	$ 157,260	$ 0.01	$ 0.00
MTCE/Misc.					**$ 0.05**	**$/lb Cu**	**$ 2,200,000**	**$ 0.12**	**$ 0.05**
Direct Labor		**49**	staff				**$ 3,536,000**	**$ 0.20**	**$ 0.08**
	Gen Frmn	1			$ 120,000	$/yr	$ 120,000	$ 0.01	$ 0.00
	Metallurgist	1			$ 100,000	$/yr	$ 100,000	$ 0.01	$ 0.00
	Ops Frmn	4			$ 95,000	$/yr	$ 380,000	$ 0.02	$ 0.01
	Mntce Frmn	4			$ 95,000	$/yr	$ 380,000	$ 0.02	$ 0.01
	Shift Operator	12			$ 70,000	$/yr	$ 840,000	$ 0.05	$ 0.02
	EW Crew	4			$ 70,000	$/yr	$ 280,000	$ 0.02	$ 0.01
	Laboratory	6			$ 50,000	$/yr	$ 300,000	$ 0.02	$ 0.01
	Mech/Pipe	4			$ 83,000	$/yr	$ 332,000	$ 0.02	$ 0.01
	Elect	2			$ 83,000	$/yr	$ 166,000	$ 0.01	$ 0.00
	Tech/Instr.	3			$ 86,000	$/yr	$ 258,000	$ 0.01	$ 0.01
	Labor	4			$ 50,000	$/yr	$ 200,000	$ 0.01	$ 0.00
	Security	4			$ 45,000	$/yr	$ 180,000	$ 0.01	$ 0.00
SXEW TOTAL							**$ 11,968,159**	**$ 0.67**	**$ 0.26**
Acid (Net)	**lbs/ton ore**	**13.4**	**298**	**tons/d**	**$ 120.00**	**$/t**	**$ 13,067,875**	**$ 0.73**	**$ 0.28**
Oxide Ore	lbs/ton ore	14.5							
Enriched Ore	lbs/ton ore	1.0							
MTCE/Misc.					**$ 0.025**	**$/t**	**$ 444,650**	**$ 0.03**	**$ 0.01**
Power	**100**	**kW**	**2400**	**kWh**	**$ 0.058**	**$/kWh**	**$ 1,210,227**	**$ 0.07**	**$ 0.03**
Water (all Areas)			**1,534**	**ac-ft/y**	**$ 10.00**	**$/a-ft**	**$ 15,340**	**$ 0.00**	**$ 0.00**
Labor		**11**	staff				**$ 640,000**	**$ 0.04**	**$ 0.01**
	Leach Frmn	1			$ 100,000	$/yr	$ 100,000	$ 0.01	$ 0.00
	Eq. Operator	2			$ 70,000	$/yr	$ 140,000	$ 0.01	$ 0.00
	Leach Labor	8			$ 50,000	$/yr	$ 400,000	$ 0.02	$ 0.01
LEACHING TOTAL							**$ 15,378,092**	**$ 0.86**	**$ 0.33**
Direct OPEX		**60**	**staff**				**$ 27,346,251**	**$ 1.54**	**$ 0.59**

With a plant expansion reflecting 35,000 t of annual copper production, the direct operating costs are expected to average $0.26/lb of copper cathode produced, as presented in Table 21-5.

Table 21-5: Processing Operating Cost Details

Average Yr 7-17 Cactus Mine Operating Costs - 32.5 ktpy Cu & Combined Tons									
	Units	Unit Consmptn	Consumption Rate		Unit Price		Annual Cost	$/ton Processed	$/lb Copper
Power	**kWh/lb**	**1.60**	**11875**	**kWh**	**$ 0.058**	**$/kWh**	**$ 5,987,955**	**$ 0.98**	**$ 0.09**
EW	kWh/lb	1.00	7422	kWh	$ 0.058		$ 3,742,472	$ 0.61	$ 0.06
SX/TF	kWh/lb	0.45	3340	kWh	$ 0.058		$ 1,684,112	$ 0.28	$ 0.03
Utilities/Misc.	kWh/lb	0.15	1113	kWh	$ 0.058		$ 561,371	$ 0.09	$ 0.01
SX/Reagents							**$ 2,434,817**	**$ 0.40**	**$ 0.04**
Extractant	kg/kg Cu	0.005	404	kg/d	$ 9.95	$/kg	$ 1,467,145	$ 0.24	$ 0.02
Acid			2	tons/d	$ 120.00	$/ton	$ 87,600	$ 0.01	$ 0.00
EW Reagents (Cobalt, Guar, FC1100)			$ 0.015	$/lb Cu			$ 660,000	$ 0.11	$ 0.01
Diluent					15%	% of Ext $	$ 220,072	$ 0.04	$ 0.00
MTCE/Misc.					**$ 0.05**	**$/lb Cu**	**$ 2,200,000**	**$ 0.36**	**$ 0.03**
Direct Labor		**52**	staff				**$ 3,739,000**	**$ 0.61**	**$ 0.06**
	Gen Frmn	1			$ 120,000	$/yr	$ 120,000	$ 0.02	$ 0.00
	Metallurgist	1			$ 100,000	$/yr	$ 100,000	$ 0.02	$ 0.00
	Ops Frmn	4			$ 95,000	$/yr	$ 380,000	$ 0.06	$ 0.01
	Mntce Frmn	4			$ 95,000	$/yr	$ 380,000	$ 0.06	$ 0.01
	Shift Operator	12			$ 70,000	$/yr	$ 840,000	$ 0.14	$ 0.01
	EW Crew	5			$ 70,000	$/yr	$ 350,000	$ 0.06	$ 0.01
	Laboratory	6			$ 50,000	$/yr	$ 300,000	$ 0.05	$ 0.00
	Mech/Pipe	4			$ 83,000	$/yr	$ 332,000	$ 0.05	$ 0.01
	Elect	3			$ 83,000	$/yr	$ 249,000	$ 0.04	$ 0.00
	Tech/Instr.	3			$ 86,000	$/yr	$ 258,000	$ 0.04	$ 0.00
	Labor	5			$ 50,000	$/yr	$ 250,000	$ 0.04	$ 0.00
	Security	4			$ 45,000	$/yr	$ 180,000	$ 0.03	$ 0.00
SXEW TOTAL							**$ 14,361,771**	**$ 2.35**	**$ 0.22**
Acid (Net)	**lbs/ton ore**	**3.4**	**14**	**tons/d**	**$ 120.00**	**$/t**	**$ 602,684**	**$ 0.10**	**$ 0.01**
Oxide Ore	lbs/ton ore	6.0							
Enriched Ore	lbs/ton ore	1.0							
MTCE/Misc.					**$ 0.025**	**$/t**	**$ 152,950**	**$ 0.03**	**$ 0.00**
Power	**100**	**kW**	**2400**	**kWh**	**$ 0.058**	**$/kWh**	**$ 1,210,227**	**$ 0.20**	**$ 0.02**
Water (all Areas)			**1,061**	**ac-ft/y**	**$ 10.00**	**$/a-ft**	**$ 10,610**	**$ 0.00**	**$ 0.00**
Labor		**11**	staff				**$ 640,000**	**$ 0.10**	**$ 0.01**
	Leach Frmn	1			$ 100,000	$/yr	$ 100,000	$ 0.02	$ 0.00
	Eq. Operator	2			$ 70,000	$/yr	$ 140,000	$ 0.02	$ 0.00
	Leach Labor	8			$ 50,000	$/yr	$ 400,000	$ 0.07	$ 0.01
								$ -	
LEACHING TOTAL							**$ 2,616,471**	**$ 0.43**	**$ 0.04**
								0	0
Direct OPEX		**63**	staff				**$ 16,978,243**	**$ 2.78**	**$ 0.26**

A total of 49 direct operating staff and 11 attributed G&A staff is initially anticipated for the operations running 24 hours per day, seven days per week and 365 days per year. Labor costs include a 30% benefits consideration.

Power has been considered from Arizona Public Service Company at a fully built-up rate of $0.058/kWh.

Water will be sourced from four wells, two off-site, and two on-site to fulfill anticipated yearly consumption of 1,061 acre-ft. ASCU has secured water rights totaling 3,736 afy via a historic Grandfathered Water Rights Type 2 Non-Irrigation grandfather rights (Certificate 58-100706.0005) for 136 afy and a Mineral Extraction and Process Permit No. 59-233782.000 from the ADWR for 3,600 afy. This right is for 50 years.

Contract mining costs for the Stockpile Project, open pit, and underground were derived from either benchmarking and/or zero-based principles using cost inputs from the local area including operating and maintenance labor rates and diesel price. Consumables such as tire and ground engaging tools are included in maintenance costs and are calculated as cost per hour. Productivities of the mining equipment are based on OEM performance curves and the fleet has been matched to average production rates and corresponding haulage. A 20% contractor premium has been applied to all costs.

Conventional support equipment including water trucks, graders and dozers will support the mining activity by maintaining roads and controlling dust, dumping, and loading areas, including both mineralized material and waste. Water required for the mining operation dust control is included in the contractor rate at the site water cost.

For the life of the project, surface material movements average $2.09/t and include mineralized material and waste movements of the Stockpile Project, open pit, and underground. The underground unit mining rate of $28.93/t is separate and reflects a benchmark cost of mining TLS.

21.3.1 General and Administrative

An allowance equal to approximately 7% of direct operating costs has been included for General and Administrative (G&A) costs for the Project. These costs are people related and include the following.

- Following are the G&A staffing directly related to the project.
- Accounting
- Human Resources
- Purchasing

This staff will be allocated to the leach project as well as the total mine site. Arizona Sonoran's corporate staff will serve as backup to these staff.

In addition, off-site costs will be offices, computer, and office supplies for staff. G&A also includes associated insurance and state and local taxes.

22.0 ECONOMIC ANALYSIS

The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as at 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report.

22.1 Cautionary Statement

Certain information and statements contained in this section and in the report are forward-looking in nature. Forward-looking statements include, but are not limited to, statements with respect to the economic and study parameters of the Project; Mineral Resource estimates; the cost and timing of any development of the Project; the proposed mine plan and mining methods; dilution and extraction recoveries; processing method and rates and production rates; projected metallurgical recovery rates; infrastructure requirements; capital, operating and sustaining cost estimates; the projected LOM and other expected attributes of the project; the (NPV) and IRR after-tax and payback period of capital; capital; future metal prices; the timing of the environmental assessment process; changes to the project configuration that may be requested as a result of stakeholder or government input to the environmental assessment process; government regulations and permitting timelines; estimates of reclamation obligations; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. Material assumptions regarding forward-looking statements are discussed in this report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this report, the forward-looking statements in this report are subject to the following assumptions.

- There being no signification disruptions affecting the development and operation of the Project.
- The availability of certain consumables and services and the prices for power and other key supplies being approximately consistent with assumptions in the report.
- Labor and materials costs being approximately consistent with assumptions in the report.
- Permitting and arrangements with stakeholders being consistent with current expectations as outlined in the report.
- All environmental approvals, required permits, licenses and authorizations will be obtained from the relevant governments and other relevant stakeholders.

- Certain tax rates, including the allocation of certain tax attributes, being applicable to the Project.
- The availability of financing for Arizona Sonoran's planned development activities.
- The timelines for exploration and development activities on the Project.
- Assumptions made in Mineral Resource estimate and the financial analysis based on that estimate, including, but not limited to, geological interpretation, grades, commodity price assumptions, extraction and mining recovery rates, hydrological and hydrogeological assumptions, capital and operating cost estimates, and general marketing, political, business, and economic conditions.

The production schedules and financial analysis annualized cash flow table are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the Project assumptions as discussed in this Report and may result in changes to the calendar timelines presented.

The preliminary economic assessment provided herein is preliminary in nature and is partly based on 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 assessment based on these Mineral Resources will be realized.

22.2 Methodology Used

A discounted cash flow analysis of the Project was prepared using technical and cost inputs developed by Stantec, Samuel Engineering, and Arizona Sonoran. These inputs have been reviewed in detail by Stantec and are accepted as reasonable.

The discounted cash flow analysis was performed on a stand-alone project basis with annual cash flows discounted on an end-of-year basis. The economic evaluation used a real discount rate of 8% and was performed as of July 2021 (denoted as Year -2 of the Project) using Q2 2021, US dollars. While all costs prior to the start of construction are considered as "sunk costs", these are still included in the economic analysis for the purpose of a project valuation.

This economic analysis is a direct result of those costs as well as the capital cost estimate and is therefore considered to have the same level of accuracy minus 20% to plus 35%.

22.3 Financial Model Parameters

Technical-economic parameters used in the model are summarized in the following sections.

Table 22-1 presents the model inputs used in the economic analysis based on second quarter, 2021 US dollars.

Table 22-1: Model Input

Area	Description	Units	Values
	Construction period	Years	1.3
	Mine life (after preproduction)	years	18
	Average Annual Production Rate Copper	t × 1,000	28,216
Metal Pricing	Copper Price	US$/lb	3.35
Cost Criteria	Estimate Basis	US$	Second Quarter 2021
	Inflation / Currency Fluctuation		None
	Leverage	% Equity	100
Income Tax	United States Corporate Income	% Profit	21
	Arizona Corporate IncomeArizona Mining Severance	% Profit % Profit	6.9 2.5
Royalties / Payments	None	n/a	3.18%
Transportation, Smelting, and Refining Charges	Shipping, Handling, and Fees	US$/lb Copper	0.04

Details of the assumptions and the outcome of the analysis are provided in Table 22-2.

Table 22-2: Financial Assumptions and Results

Assumption / Outcome	Value / Results
Copper Price	$3.35/lb
Total Mineralized Material Mined	179 Million Tons
Annual Average Processing Rate Over LOM	10 million tons per annum
Average Recovery Rates Over LOM	Stockpile Project: CuAS: 90%, CuCN: 40% OP/UG: CuAS: 90%, CuCN: 72%
Average Production Over LOM	28 ktpa
Operating Costs (per Ton Processed)	$9.06/ton
Average Cash Cost (C1) and All-In Sustaining Cost (C1 Cost+ Sustaining CAPEX)	C1: US$1.55/lb AISC: US$1.88/lb
Sustaining CAPEX Over LOM (OP and UG, SXEW and Leach pad expansions)	$340 Million
LOM Free Cash Flow (FCF) (Post Tax Undiscounted)	$960 million
Post Tax NPV8	$312 million

Post Tax IRR	33%

22.3.1 Mineral Resource, Mineral Reserve, and Mine Life

The Mineral Resource estimate is provided in Section 14.0 of the report. A subset of these Mineral Resources is used in the PEA mine plan (refer to Section 22.4). Stantec Engineering provided a mine production schedule on an annualized basis.

The process schedule was prepared on an annualized basis by Samuel Engineering. It includes the mine production with copper grade from the mine production plan and adds plant processing data. The product for sales is reported as the copper metal production. The table uses recoveries from the metallurgical test work from Section 13.5, and payables from expected payment terms outlined in Section 19.0.

22.3.2 Smelting and Refining Terms

The smelting and refining terms assumed in the financial analysis were included in Table 22-1. The product of the plant will be cathode LME Grade A copper and will require no further refining. The small brokerage fee is expected with the buyer taking ownership at the Cactus Project site and shipping to the market.

22.3.3 Metal Price

Metal pricing at $3.35/lb is based on current market factors including benchmarking undertaken for comparable studies. As this value is lower than the current market spot price, sensitivities were run mostly above the considered valuation price.

22.3.4 Capital Costs

The capital cost estimate basis and the sustaining costs were provided in Section 21.1.

22.3.5 Operating Costs

The operating cost estimate basis was provided in Section 21.3.

Note that for process, the operating cost metrics will not match between those calculated in the economic model and the operating cost table. The difference exists because the two metrics are calculated on a different time basis. As the production plan indicates, copper is leached from the mineralized material over the course of two years, therefore operations will continue one year after the completion of material placement. Although the mine life is 17 years, operations will last 18 years. In the model this increases the total administrative costs of the operation, therefore increasing the cost metric per ton of material.

22.3.6 Taxes and Royalties

Income Tax

Taxation for the Cactus Project will be as a result of copper or metal sales income. Generally, the rates are as follows.

- US Income Tax Rate is 21%
- State of Arizona Income Tax Rate is 6.9%
- Arizona Mining Severance Tax Rate is 2.5%

Rather than simply add the rates to get 30.4% total off net income, we have assumed that the net income will be reduced by approximately 20% based on exploration and depletion, which have currently not been quantified, write offs and the actual severance tax rate will be one half of what they are at 1.25%. For this reason, it has been assumed that a total tax rate of 24% would be used to account for these deductions in income.

Depreciation

All initial and sustaining capital costs have been depreciated on the basis of 10 year sum of years depreciation considering remaining project life where relevant.

Royalties

A 3.18% royalty is assumed to be applicable to the Project based on current contractual arrangements.

22.3.7 Closure Costs

Closure costs were estimated by Arizona Sonoran at $5 million. The estimate is based upon a value allotted to the closure of $1.5 million for the Stockpile Project facilities with the addition of $3.5 million estimated for the closure of the remaining site facilities.

22.3.8 Salvage Value

A salvage value of $5 million was utilized, as it was taken into consideration when estimating the closure cost of the processing facility.

22.3.9 Financing

The financial model presents an unlevered case where no financing is assumed.

22.3.10 Inflation

Inflation is not included in the financial model or the capital and operating cost estimates.

22.4 Economic Analysis

22.4.1 Preliminary Economic Analysis Results

The Cactus project's after-tax economic results for the PEA evaluation are summarized in Table 22-3 and show an-tax (NPV) of $312 million at an 8% discount rate, an IRR after-tax of 33%. Table 22-3 presents the cashflow on an annualized basis.

Table 22-4 summarizes key unit assumptions in the plan. Table 22-5 presents the detailed cash flow for the Project.

Table 22-3: Summary, Financial Analysis (After-Tax; Base Case is Highlighted)

Financial Results	Units	Value
Cumulative Cashflow (LOM)	US$ million	960.0
Net Present Value (4%)	US$ million	540.0
Net Present Value (8%)	US$ million	312.0
Net Present Value (10%)	US$ million	238.0
Internal Rate of Return (IRR after-tax)	%	33.0
Payback	Years	3.5
Initial Capital Construction Costs	US$ million	124.0

Table 22-4: Key Assumptions for Table 22-2

Item	Units	Value
Ownership	%	100.00
Brokerage Fee	US$/lb Cu	0.04
Overall Tax Rate	%	24.00

Table 22-5: Cash Flow

Parameters	Unit	Life of Mine Total	Year -2	Year -1	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
KEY INPUTS																									
Metal Prices																									
Cu	US$/lb	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35
MINING SCHEDULE	Unit	Total	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Stockpile																									
Waste Mined	k tons	22,823	-	-	-	13	3,600	7,930	8,650	2,630	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ore Mined	k tons	81,242	-	-	-	14,560	19,730	19,530	19,690	7,732	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Material Mined	k tons	104,065	-	-	-	14,573	23,330	27,460	28,340	10,362	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cu Grade	**%**	**0.14%**	0.00%	0.00%	0.00%	0.16%	0.15%	0.13%	0.13%	0.12%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Cu	**k tons**	**114**	-	-	-	23	30	26	25	9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Open Pit																									
Waste Mined	**k tons**	**101,890**	-	-	-	13,890	17,500	18,600	14,660	11,200	9,730	11,110	1,830	760	480	500	1,060	570	-	-	-	-	-	-	-
OPEX	k tons	2,610	-	-	-	-	-	-	-	-	-	-	-	-	480	500	1,060	570	-	-	-	-	-	-	-
Development	k tons	31,390	-	-	-	13,890	17,500	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining	k tons	67,890	-	-	-	-	-	18,600	14,660	11,200	9,730	11,110	1,830	760	-	-	-	-	-	-	-	-	-	-	-
Prime Stockpile	k tons	1,825	-	-	-	-	-	-	-	0	74	40	126	117	209	517	237	505	-	-	-	-	-	-	-
Ore Mined	**k tons**	**69,862**	-	-	-	442	1,189	3,113	4,540	9,982	9,662	9,455	7,353	5,464	3,987	4,094	6,305	4,276	-	-	-	-	-	-	-
Oxide	k tons	46,730	-	-	-	400	730	2,090	2,150	8,550	6,300	6,720	6,450	4,270	1,590	1,080	4,590	1,810	-	-	-	-	-	-	-
Enriched	k tons	23,132	-	-	-	42	459	1,023	2,390	1,432	3,362	2,735	903	1,194	2,397	3,014	1,715	2,466	-	-	-	-	-	-	-
Total Material Mined	**k tons**	**171,752**	-	-	-	14,332	18,689	21,713	19,200	21,182	19,392	20,565	9,183	6,224	4,467	4,594	7,365	4,846	-	-	-	-	-	-	-
Cu Grade	%	**0.26%**				0.120%	0.210%	0.193%	0.194%	0.188%	0.237%	0.241%	0.256%	0.313%	0.446%	0.410%	0.263%	0.355%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Contained Cu	**k tons**	**185**	-	-	-	1	3	6	9	19	23	23	19	17	18	17	17	15	-	-	-	-	-	-	-
Oxide	k tons	87	-	-	-	0	1	2	3	14	12	13	13	10	3	2	9	4	-	-	-	-	-	-	-
Enriched	k tons	98	-	-	-	0	2	4	6	5	11	10	5	7	14	15	7	11	-	-	-	-	-	-	-
Underground																									
Waste Mined	**k tons**	**2,157**	-	-	-	-	-	-	-	129	194	194	194	194	194	194	194	194	194	194	86	-	-	-	-
OPEX	k tons	280	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	194	86	-	-	-	-
Development	k tons	324	-	-	-	-	-	-	-	129	194	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining	k tons	1,554	-	-	-	-	-	-	-	-	-	194	194	194	194	194	194	194	194	-	-	-	-	-	-
Ore Mined	**k tons**	**27,524**	-	-	-	-	-	-	-	-	630	1,266	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,419	529	-	-
Oxide	k tons	6,317	-	-	-	-	-	-	-	-	418	863	1,746	55	14	51	704	262	256	255	484	1,209	-	-	-
Enriched	k tons	21,208	-	-	-	-	-	-	-	-	212	403	774	2,465	2,506	2,469	1,816	2,258	2,264	2,265	2,036	1,210	529	-	-
Ore Mined - Primary	k tons	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Material Mined	**k tons**	**29,681**	-	-	-	-	-	-	-	129	824	1,461	2,714	2,714	2,714	2,714	2,714	2,714	2,714	2,714	2,606	2,419	529	-	-
Total Ore Milled	**k tons**	**27,524**	-	-	-	-	-	-	-	-	630	1,266	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,419	529	-	-
Cu Grade	%	1.27%	-	-	-	-	-	-	-	-	1.15%	1.16%	1.15%	1.13%	1.15%	1.17%	1.45%	1.39%	1.36%	1.42%	1.31%	1.30%	0.86%	-	-
Contained Cu	**k tons**	**349**	-	-	-	-	-	-	-	-	7	15	29	29	29	30	37	35	34	36	33	32	5	-	-
Oxide	k tons	74	-	-	-	-	-	-	-	-	5	10	20	1	0	1	9	3	3	3	6	14	-	-	-
Enriched	k tons	275	-	-	-	-	-	-	-	-	2	5	9	28	29	29	28	32	31	33	27	17	5	-	-
PROCESSING SCHEDULE	Unit	Total	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Ore Processed - Heap Leach Stockpile Oxide	k tons	81,242	-	-	-	14,560	19,730	19,530	19,690	7,732	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ore Processed - Heap Leach OP/UG Oxide	k tons	53,047	-	-	-	400	730	2,090	2,150	8,550	6,718	7,583	8,196	4,325	1,604	1,131	5,294	2,072	256	255	484	1,209	-	-	-
Ore Processed - Heap Leach OP/UG Enriched	k tons	44,340	-	-	-	42	459	1,023	2,390	1,432	3,574	3,138	1,677	3,659	4,903	5,483	3,531	4,724	2,264	2,265	2,036	1,210	529	-	-
Ore Processed - Total	**k tons**	**178,628**	-	-	-	15,002	20,919	22,643	24,230	17,714	10,292	10,721	9,873	7,984	6,507	6,614	8,825	6,796	2,520	2,520	2,520	2,419	529	-	-
Recovered Metals																									
Cu Recovery - Stockpile CuAS	%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	
Recovered Cu	**k tons**	**84**	-	-	-	13	21	20	19	10	2	-	-	-	-	-	-	-	-	-	-	-	-	-	
Cu Recovery - Stockpile CuCN	%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	
Recovered Cu	k tons	**8**	-	-	-	1	2	2	2	1	0	-	-	-	-	-	-	-	-	-	-	-	-	-	
Cu Recovery -OP/UG CuAS	%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	
Recovered Cu	k tons	**190**	-	-	-	0	1	2	3	9	14	19	27	17	12	10	18	15	9	8	9	13	4	0	
Cu Recovery -OP/UG CuCN	%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	72.0%	
Recovered Cu	k tons	**226**	-	-	-	0	1	2	3	4	8	9	10	19	24	25	22	24	20	19	17	12	5	1	
Cu Recovery - Total	%	78.6%	-	-	-	60.6%	75.1%	81.1%	79.5%	86.4%	80.1%	77.1%	76.5%	79.7%	76.4%	76.6%	75.4%	78.3%	84.9%	74.9%	77.7%	78.1%	191.4%	-	
Recovered Cu	k tons	**509**	-	-	-	14	25	26	27	24	24	29	37	36	36	36	40	39	29	27	26	25	9	1	
NET SMELTER RETURN	Unit	Total	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	
Payable Metals																									
	%		100.0%	100.0%	-	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	
Payable Cu Cathode	k tons	509	-	-	-	14	25	26	27	24	24	29	37	36	36	36	40	39	29	27	26	25	9	1	
	US$' k	3,408,585	-	-	-	96,381	165,221	174,611	180,594	162,724	161,885	193,633	245,384	243,427	239,945	238,153	268,288	263,863	195,131	179,579	171,190	164,816	58,060	5,700	

Total Cu tons	Cu k tons	**508**	-	-	-	14	25	26	27	24	24	29	37	36	36	36	40	39	29	27	26	25	9	1	
Total Payable Metals	US$ k	**3,408,585**	-	-	-	96,381	165,221	174,611	180,594	162,724	161,885	193,633	245,384	243,427	239,945	238,153	268,288	263,863	195,131	179,579	171,190	164,816	58,060	5,700	
Transportation & Refining Costs																									
Copper Treatment Charge	US$ dmt		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	US$ k	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Copper Refining Charge	US$/payable lb	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	
	US$ k	**(407)**	-	-	-	(12)	(20)	(21)	(22)	(19)	(19)	(23)	(29)	(29)	(29)	(28)	(32)	(32)	(23)	(21)	(20)	(20)	(7)	(1)	
Net Smelter Return	US$ k	**3,408,178**	-	-	-	96,370	165,201	174,590	180,573	162,705	161,865	193,610	245,355	243,398	239,916	238,124	268,256	263,831	195,108	179,557	171,170	164,796	58,053	5,700	
Net Smelter Return	US$/ton	**19.08**	-	-	-	6.42	7.90	7.71	7.45	9.19	15.73	18.06	24.85	30.49	36.87	36.00	30.40	38.82	77.42	71.25	67.93	68.14	109.69	-	
Total revenues	US$	**3,408,178**	-	-	-	96,370	165,201	174,590	180,573	162,705	161,865	193,610	245,355	243,398	239,916	238,124	268,256	263,831	195,108	179,557	171,170	164,796	58,053	5,700	
OPERATING COSTS	Unit	Total	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	
Mining - Ore on Stockpile	US$/t mined	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	
	US$ k	(63,369)	-	-	-	(11,357)	(15,389)	(15,233)	(15,358)	(6,031)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Mining - Waste on Stockpile	US$/t mined	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	
	US$ k	(11,412)	-	-	-	(7)	(1,800)	(3,965)	(4,325)	(1,315)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Mining - Open Pit Ore	US$/t mined	2.45	2.45	2.45	2.45	1.75	1.75	1.75	2.00	2.15	2.45	2.45	2.45	2.45	2.45	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	
	US$ k	(170,875)	-	-	-	(774)	(2,081)	(5,448)	(9,080)	(21,461)	(23,672)	(23,165)	(18,015)	(13,387)	(9,768)	(12,282)	(18,915)	(12,828)	-	-	-	-	-	-	
Mining - Open Pit Waste	US$/t mined	0.12	2.45	2.45	2.45	1.75	1.75	1.75	2.00	2.15	2.45	2.45	2.45	2.45	2.45	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	
	US$ k	(12,730)	-	-	-	-	-	-	-	(0)	(182)	(98)	(309)	(285)	(1,689)	(3,050)	(3,892)	(3,224)	-	-	-	-	-	-	
Mining - UG Ore	US$/t mined	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	28.93	
	US$ k	(796,269)	-	-	-	-	-	-	-	-	(18,226)	(36,635)	(72,905)	(72,903)	(72,903)	(72,904)	(72,904)	(72,903)	(72,903)	(72,904)	(72,902)	(69,968)	(15,311)	-	
Mining - UG Waste	US$/t mined	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	
	US$ k	(8,397)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(5,826)	(2,571)	-	-	-	
Royalty	US$/t milled	0.61	-	-	-	0.20	0.25	0.25	0.24	0.29	0.50	0.57	0.79	0.97	1.17	1.14	0.97	1.23	2.46	2.27	2.16	2.17	3.49	-	
	US$ k	(108,380)	-	-	-	(3,065)	(5,253)	(5,552)	(5,742)	(5,174)	(5,147)	(6,157)	(7,802)	(7,740)	(7,629)	(7,572)	(8,531)	(8,390)	(6,204)	(5,710)	(5,443)	(5,241)	(1,846)	(181)	
Heap Leach (Stockpile, OP, UG)	US$/t milled	0.94	0.86	0.86	-	0.86	0.86	0.86	0.86	0.86	0.86	0.65	0.43	0.43	0.43	0.43	0.43	0.43	0.43	0.43	0.43	0.43	0.43	0.43	
	US$ k	(126,813)	-	-	-	(12,902)	(17,990)	(19,473)	(20,838)	(15,234)	(8,851)	(6,969)	(4,245)	(3,433)	(2,798)	(2,844)	(3,795)	(2,922)	(1,084)	(1,084)	(1,084)	(1,040)	(228)	-	-
Heap Leach - Oxides	US$/t milled	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	US$ k	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Heap Leach - Enriched (OP, UG)	US$/t milled	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	US$ k	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
SX-EW Process	US$/t milled	1.26	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67	1.51	2.35	2.35	2.35	2.35	2.35	2.35	2.35	2.35	2.35	2.35	2.35	2.35	
	US$ k	(224,626)	-	-	-	(10,051)	(14,016)	(15,171)	(16,234)	(11,868)	(6,896)	(16,189)	(23,202)	(18,762)	(15,291)	(15,543)	(20,739)	(15,971)	(5,922)	(5,922)	(5,922)	(5,684)	(1,244)	-	
Site General & Administration	US$/t milled	0.53	-	-	-	0.16	0.16	0.16	0.16	0.17	0.96	1.00	1.03	1.03	1.03	1.05	1.05	1.05	0.89	0.89	0.89	0.89	0.89	-	
	US$ k	(94,612)	-	-	-	(2,348)	(3,274)	(3,544)	(3,962)	(2,971)	(9,925)	(10,717)	(10,212)	(8,258)	(6,730)	(6,943)	(9,264)	(7,134)	(2,237)	(2,237)	(2,237)	(2,147)	(470)	-	
Other	US$/t milled		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	US$ k	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Total OPEX	**US$/t milled**	**9.06**	-	-	-	2.70	2.86	3.02	3.12	3.62	7.08	9.32	13.84	15.63	17.95	18.32	15.64	18.15	35.06	37.18	35.78	34.76	36.09	-	
	US$ k	**(1,617,482)**	-	-	-	(40,503)	(59,804)	(68,386)	(75,539)	(64,055)	(72,899)	(99,930)	(136,690)	(124,769)	(116,808)	(121,138)	(138,039)	(123,372)	(88,351)	(93,683)	(90,159)	(84,079)	(19,098)	(181)	
C1 Cost	**US$/lb**	**1.59**	-	-	-	1.41	1.21	1.31	1.40	1.32	1.51	1.73	1.87	1.72	1.63	1.70	1.72	1.57	1.52	1.75	1.76	1.71	1.10	0.11	
Total Cost	**US$/lb**	**2.14**	-	-	-	3.14	1.75	2.43	1.88	2.12	2.82	2.83	2.11	2.26	1.82	1.90	1.89	1.74	1.75	1.75	1.76	1.71	1.10 -	1.95	
INCOME	Unit	Total	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	
Gross Pre-Tax Operating Income	**US$ k**	**1,790,697**	-	-	-	55,867	105,397	106,204	105,033	98,650	88,967	93,680	108,665	118,630	123,108	116,986	130,217	140,460	106,757	85,874	81,010	80,717	38,955	5,518	
	US$/t milled	**10.02**	-	-	-	3.72	5.04	4.69	4.33	5.57	8.64	8.74	11.01	14.86	18.92	17.69	14.76	20.67	42.36	34.08	32.15	33.37	73.61	-	
CAPITAL COSTS	Unit	Total	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	
Project Infrastructure	US$ k	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Leachpad Infrastructure	US$ k	(24,500)	-	-	(20,000)	(4,500)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
SX-EW Facilities	US$ k	(74,000)	-	-	(50,000)	(24,000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Flotation Processing Facilities	US$ k	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings Facilities	US$ k	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capitalised Drilling - Cactus Orebodies	US$ k	(7,833)	(5,014)	(2,819)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capitalised Drilling - Stockpile	US$ k	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Technical Studies	US$ k	(4,101)	(2,697)	(1,404)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Project/Other Costs	US$ k	(2,583)	(1,003)	(1,580)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
OP- Capitalised Stripping	US$ k	(47,085)	-	-	-	(20,835)	(26,250)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
UG-Capitalised Development	US$ k	(29,124)	-	-	-	-	-	-	-	(11,646)	(17,478)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mobile Mine Equipment (OP_UG)	US$ k	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine Equipment (OP_UG_	US$ k	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital - Leachpad Facilities	US$ k	(74,600)	-	-	-	-	-	(25,500)	-	-	(24,940)	-	-	(24,160)	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital - SX-EW Facilities	US$ k	(26,000)	-	-	-	-	-	-	-	-	-	(26,000)	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital - Open Pit	US$ k	(130,980)	-	-	-	-	-	(32,550)	(25,655)	(22,400)	(20,920)	(23,887)	(3,935)	(1,634)	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital - UG	US$ k	(108,752)	-	-	-	-	-	-	-	-	-	(13,594)	(13,594)	(13,594)	(13,594)	(13,594)	(13,594)	(13,594)	(13,594)	-	-	-	-	-	-
Exploration	US$ k	-	0	0	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Land Acquisitions	US$ k	(27,475)	(7,000)	(7,525)	(7,950)	-	-	-	-	(5,000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TAGC Founders Fee	US$ k	(1,100)	-	-	(300)	(500)	(300)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cash Reclamation	US$ k	(1,500)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(1,500)	(3,500)
Salvage Value	US$ k	5,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5,000	-
Total CAPEX	**US$ k**	**(558,132)**	(15,713)	(13,328)	(78,250)	(49,835)	(26,550)	(58,050)	(25,655)	(39,046)	(63,338)	(63,481)	(17,529)	(39,388)	(13,594)	(13,594)	(13,594)	(13,594)	(13,594)	-	-	-	-	3,500	(3,500)
TAXES	Unit	Total	-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20

Total Taxes	**US$ k**	**(272,323)**	-	-	-	-	(14,577)	(16,083)	(15,727)	(13,648)	(9,887)	(9,851)	(14,562)	(16,951)	(19,340)	(18,844)	(22,775)	(25,873)	(18,170)	(15,064)	(15,632)	(17,020)	(8,178)	(141)	-
Cash Flow Analysis																									
Cash Flow	Unit	Total	**-2**	**-1**	0	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**
Pre-Tax (Undiscounted)																									
Net Pre-Tax Cash Flow	US$ k	1,232,565	(15,713)	(13,328)	(78,250)	6,032	78,847	48,154	79,378	59,604	25,629	30,200	91,137	79,242	109,514	103,392	116,623	126,866	93,163	85,874	81,010	80,717	38,955	9,018	(3,500)
Cumulative Pre-Tax Cash Flow	US$ k	1,232,565	(15,713)	(29,042)	(107,292)	(101,260)	(22,412)	25,742	105,121	164,725	190,354	220,554	311,691	390,932	500,446	603,838	720,461	847,327	940,490	1,026,363	1,107,374	1,188,091	1,227,046	1,236,065	1,232,565
Post-Tax (Undiscounted)																									
Net Post-Tax Cash Flow	US$ k	960,242	(15,713)	(13,328)	(78,250)	6,032	64,271	32,072	63,652	45,957	15,743	20,349	76,574	62,290	90,173	84,548	93,848	100,993	74,992	70,810	65,379	63,698	30,777	8,877	(3,500)
Cumulative Post-Tax Cash Flow	US$ k	960,242	(15,713)	(29,042)	(107,292)	(101,260)	(36,989)	(4,917)	58,734	104,691	120,433	140,782	217,357	279,647	369,820	454,368	548,216	649,209	724,201	795,011	860,390	924,088	954,865	963,742	960,242

ECONOMIC RESULTS	Unit			
Pre-Tax NPV @ 8%	**US$ k**	**417,054**	**IRR**	**39%**
Post-Tax NPV @ 8%	**US$ k**	**312,099**	**IRR**	**33%**

22.4.2 Cash Costs

Following are the financial results.

- Post start-up average LOM C1 cash cost: $1.55/lb copper.
- Post start-up average LOM all-in sustaining cash cost: $1.88/lb copper.
- Post start-up average LOM total costs: $2.06/lb copper.

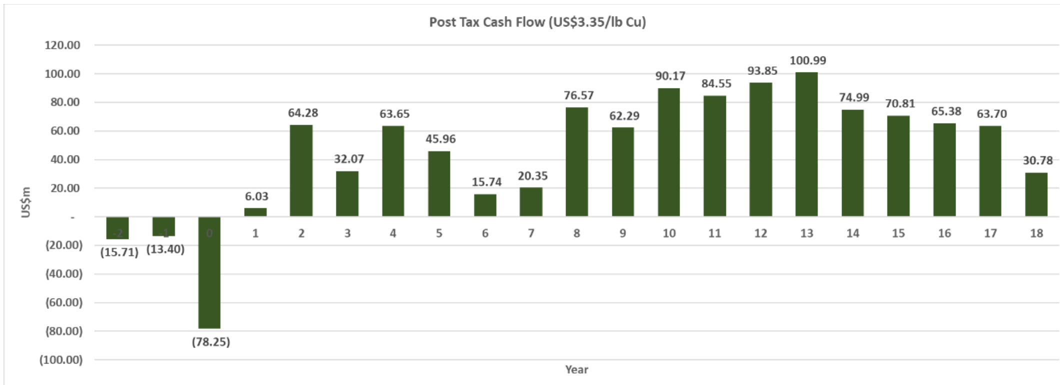
Cash cost includes all direct and indirect costs associated with the physical activities that would generate concentrate products for sale to customers, including mining to gain access to mineralized materials, mining of mineralized materials and waste, milling, third-party related treatment, refining and transportation costs, on-site administrative costs, and royalties. Cash cost does not include depreciation, depletion, amortization, exploration expenditures, reclamation and remediation costs, financing costs, income taxes, or corporate general and administrative costs not directly or indirectly related to the Project. Cash cost is divided by the number of payable copper pounds generated by the plant for the period to arrive at the cash costs per pound of copper.

All-in sustaining cost includes cash cost and sustaining CAPEX. This is divided by the number of payable copper pounds generated by the plant for the period to arrive at the all-in sustaining costs per pound of copper.

Total cost includes all costs associated with the project each year (including all initial and expansion CAPEX). This is divided by the number of the payable copper pounds generated by the plant for the period to arrive at the total costs per pound of copper.

Figure 22-1 captures life-of-mine cash flows on a post-tax basis using a flat copper price of US$3.35/lb, with positive cashflow commencing in Year 1 post development capital investment.

Figure 22-1: Life-of-Mine Post Tax Cash Flow



22.5 Sensitivity Analysis

Table 22-6Figure 22-2, Table 22-6, and Table 22-7 present project post-tax NPV and IRR sensitivities to copper price while keeping current project financials such as capital cost, mineralized material copper grade, metallurgical recovery, and operating costs constant.

Figure 22-2: Copper Price versus Post Tax NPV and IRR Sensitivities

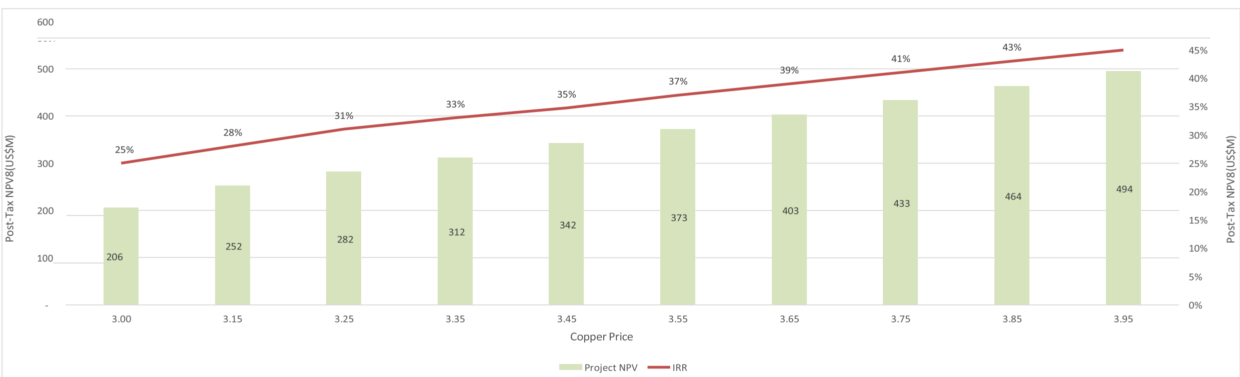


Table 22-6: Project CAPEX Versus Post Tax NPV and IRR Sensitivities

Total Project Capex	Project 8 NPV (Post Tax, $M) (US$3.35/lb Cu)	Post Tax IRR
15%	273.14	27%
10%	286.13	29%
0%	312.10	33%
-10%	338.07	37%
-15%	351.05	40%

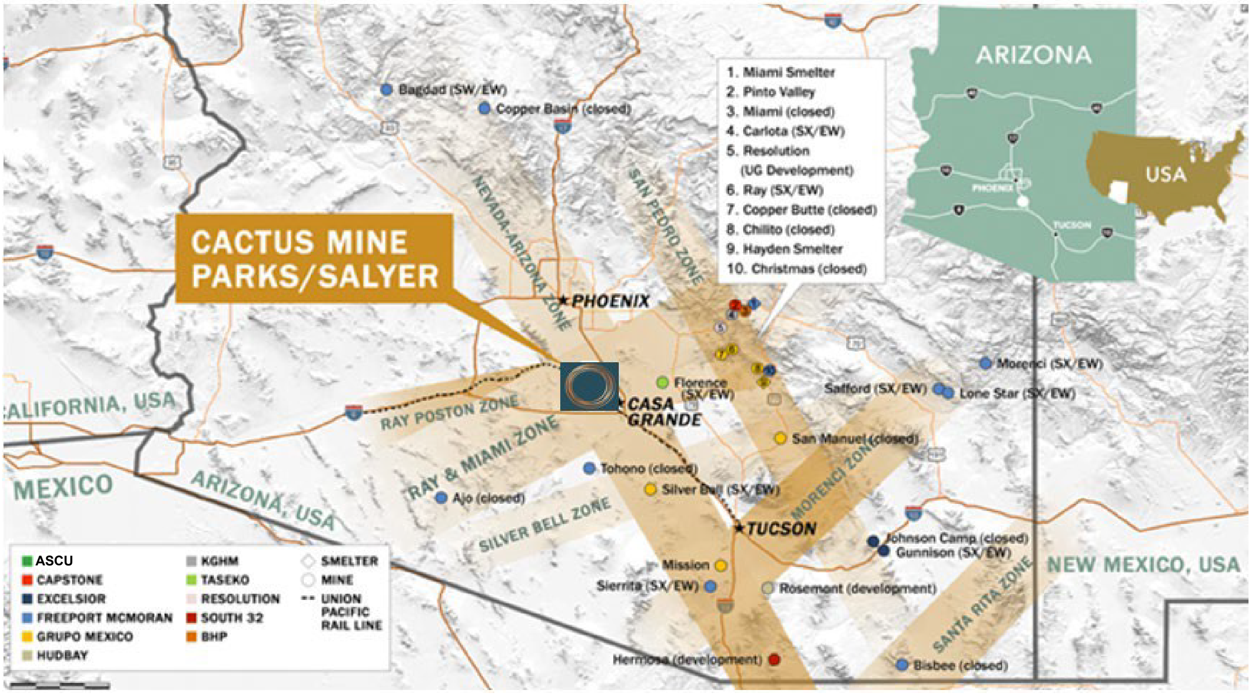
Table 22-7: OPEX versus Post Tax NPV and IRR Sensitivities

LOM Opex	Project 8 NPV (Post Tax, $M) (US$3.35/lb Cu)	Post Tax IRR
15%	239.40	28%
10%	263.64	30%
0%	312.10	33%
-10%	360.56	36%
-15%	384.79	37%

23.0 ADJACENT PROPERTIES

The Project, as shown in Figure 23-1, is surrounded by other, current and past-producing, copper deposit mines and similar processing facilities.

Figure 23-1: Regional Copper Mines and Processing Facilities



The nearest adjacent mineral property is the Santa Cruz copper porphyry deposit approximately 6 miles southeast of the Cactus site and 7 miles west of Casa Grande, Arizona. Deposit information, obtained from an abstract of the Geology of the Santa Cruz Porphyry Copper Deposit Henry G. Keis, ASARCO, Incorporated, Tucson, Arizona, reports associated alteration and mineralization in the Santa Cruz copper porphyry, including that of fault displaced portions (such as the Cactus Project), is about 7 miles long and about a mile wide. The QP has been unable to verify the information concerning the adjacent property and that such information is not necessarily indicative of the mineralization of the Cactus Project.

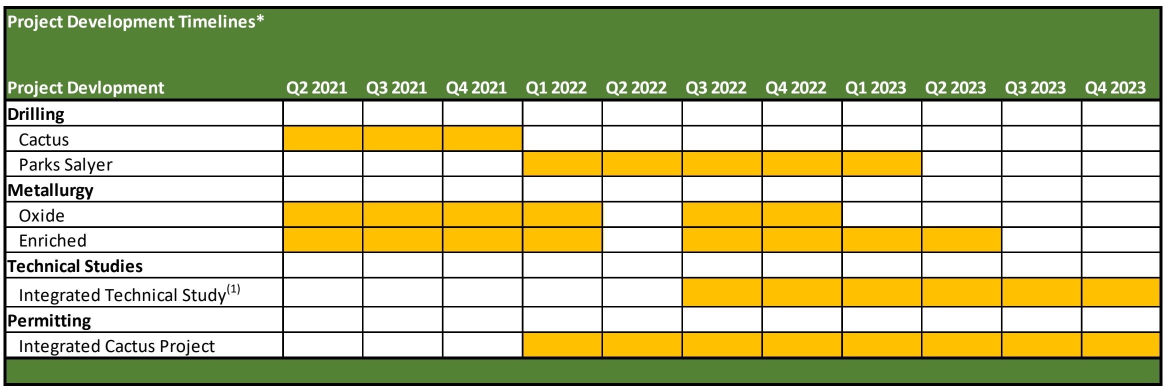
Within Pinal County there are currently two operating copper mines. These mines are the Florence Copper Mine, owned and operated by Taseko Mines Ltd. (approximately 25 miles ENE) and the Ray Mine, owned, and operated by ASARCO LLC, a subsidiary to Grupo Mexico (approximately 50 miles ENE) of the Cactus Mine.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

The following is a high-level expected development timeline for the Cactus Project which includes the Cactus Deposits, Cactus Stockpile, and potentially the Parks/Salyer Deposit reflecting the integrated technical studies. The execution plan remains conceptual and is subject to various factors outside of Arizona Sonoran's control (Figure 24-1). The project execution schedule provided in 21.2 only reflects the Cactus Project expected timelines.

Figure 24-1: Project Development Timelines



Notes:
*Updated only to reflect recommendations as included in Section 26
(1) Integrated Technical Study work is expected to incorporate Cactus, Stockpile, and Parks/Salyer

25.0 INTERPRETATION AND CONCLUSIONS

For certain chapters in this report, text and figures have been taken directly from the 2021 Cactus PEA. The mineral resource estimate for the Parks/Salyer Project as described in this report was not included in the 2021 Cactus PEA and it does not have a negative impact on or otherwise adversely affect the mineral resource estimate that formed the basis of the 2021 Cactus PEA. The date of the Cactus Resource is as of 01 March 2021 and the inputs and assumptions used for economic assessment are valid as of 31 August 2021. The results and conclusions of the 2021 Cactus PEA are still considered current and therefore have been carried over for this report. The current study is considered scoping in nature and suitable for inclusion in a PEA as defined and allowed in NI 43-101 guidelines. The Cactus Project (without incorporation of Parks/Salyer) as contemplated in the study work to date presents the following attributes.

- Private land package, 100% ownership with a 3.18% royalty attached based on current contractual arrangements.
- Permitting limited to State of Arizona processes and county level permits, no US Federal agencies or processes involved.
- Indicated resources of 151.8 million tons at a grade of 0.531% CuT (1.6 billion pounds) and inferred resources of 228.9 million tons at a grade of 0.384% CuT (1.76 billion pounds). Additional Inferred resources of 75.5 million tons at a grade of 0.168% total copper (0.145% soluble copper) contained in the Stockpile Project (223 million pounds).
- Copper recovery projected from preliminary metallurgical testing of 90% for all CuAS material for the Stockpile Project and open pit / underground, and 40% for all CuCN oxide material and 72% on open pit/underground CuCN enriched material with an average net acid consumption of 10.5 lb of H_2SO_4 per ton of material leached for the Stockpile Project and 7.9 lb of H_2SO_4 per ton of material leached for open pit ore.
- Conventional copper heap leach, modular design SX/EW processing facilities to produce an average 28,216 tpa of LME Grade A quality cathode product over LOM.
- Initial capital construction cost of project approximately $124 million, basic project infrastructure and utilities (power, water) already exist within the project site (including PFS program and costs).
- Operating Cash Costs estimated at an average $1.55/lb of copper produced, All-in sustaining costs of an average of $1.88 of copper produced and total costs of $2.06/lb of copper produced.
- Project economic analysis at $3.35/lb Copper, NPV @ 8% Discount Rate $312 million, after-tax IRR 33%, with a 3.5 year payback period.

The Parks/Salyer deposit provides exploration potential and may be incorporated into future studies to enhance the project economics. An infill diamond drilling program should be undertaken to reduce the drill spacing of the deposit to 250ft in conjunction with drilling supporting geotechnical, metallurgical, and hydrological studies.

25.1 Mineral Tenure, Surface Rights, Water Rights, Royalties, Agreements

The Cactus copper deposits are part of a large porphyry copper system that has been dismembered and displaced by Tertiary extensional faulting. It is similar in most regards to the model proposed by Lowell and Guilbert (1970) and these concepts will guide exploration. The deposit has a complex weathering history including oxidation and leaching which resulted in the formation of a chalcocite blanket. The chalcocite blanket in the mineralized deposit is irregular in thickness, grade, and continuity. These irregularities are caused by tilting, post-enrichment oxidation, and possibly by fault offsets. The thickness of leached capping varies from less than 100 ft (30 m) to over 650 ft (198 m), with the thicker intercepts on the north side. The later stage of oxidation and leaching modified the blanket by oxidizing portions of it in place and mobilizing some of the chalcocite to a greater depth. Substantial quantities of oxidized copper minerals are found in the oxidized zone.

Arizona Sonoran's understanding of mineral zoning in general and characteristics of the supergene oxidized, and enriched zones, will help in the interpretation of exploration drill results and aid in understanding the distribution of mineralization in both the Cactus deposits and the Stockpile Project. The current Stockpile Project was created through dumping of defined waste material from the historic Sacaton open pit mine operations by ASARCO during the period 1972 to 1984. All oxide copper mineralization, and sulfide copper mineralization below the working grade control cutoff of 0.3% Cu, as well as non-mineralized Gila Conglomerate from the west and east sides of the open pit, was directed to the WRD.

Arizona Sonoran acquired historic Type 1 Non-Irrigation grandfather rights (Certificate 58-100307) for 45.36 afy. In addition to the grandfathered rights Arizona Sonoran has obtained its permit from the ADWR for an additional 3,554.64 afy under a Permit to Withdraw Groundwater for Mineral Extraction and Metallurgical Processing within an Active Management Area (A.R.S. § 45-514). This entitlement is expected to be sufficient for LOM as outlined in this PEA. If additional water is required Arizona Sonoran has options to purchase this water from either GRWS or Global.

25.2 Exploration, Drilling and Analytical Data Collection Supporting Mineral Resource Estimation

The Cactus deposits have been drilled historically under ASARCO and recently by Arizona Sonoran. Core drilling has been undertaken in mineralized zones defining two zones of economic mineralization in Cactus West and Cactus East. Cactus West was mined through 1972 to 1984 prior to closure of the mine. An underground shaft and development were underway in the 1980s prior to the closure. Arizona Sonoran performed significant verification work on the historical drillholes to support the use of this data in the PEA. In addition, Arizona Sonoran drilled 22 core holes on the project (20 within the resource area) to confirm mineralization characteristics, attain metallurgical test samples, and expand the resource. Samples undertaken on 10 ft (3.0 m) lengths except where geological contacts or alteration determined otherwise. Samples were logged and photographed on site.

To drill test the mineral potential of the Stockpile Project, Arizona Sonoran designed a program of sonic drilling, using a Boart Longyear LS 600 sonic drill to drill 6-inch diameter vertical test holes through the lifts into the underlying paleo surface (anywhere from 40 ft (12.2 m) to 105 ft (32.0 m) below lift surface). Two hundred six sonic holes have been drilled on the Stockpile Project to infill to approximately 400 ft (121.9 m) centers. The core was bagged by the drillers at 2.5 ft (0.76 m) intervals using tubular plastic bags; each bag was marked with drill hole and interval footage. The drill holes were logged geologically on site, identifying primary lithology (barren conglomerate, alluvium, or mineralized waste) for selection of samples to be sent for assay; alluvial samples were not assayed.

Use of QA/QC measures such as blind analytical standards and blanks as well as blind preparatory blanks aided in the verification of analytical accuracy for data use in both the Cactus Project deposits and Stockpile Project resources.

25.3 Metallurgical Testwork

The preliminary scoping testing has provided sufficient indications as to the potential metallurgical performance using industry proven methodology. The results are in line with benchmarked projects in Arizona employing similar methodologies for ROM acid heap leaching. Limited historical information is available from ASARCO; however, the information tends to agree with the projections for acid consumption. The preliminary tests are qualitative in nature, but lack the quantitative data related to the performance over time in the leach system.

Continuing to refine the sequential assays for the drill hole composites and new holes to be drilled in future will provide more confidence in the understanding of the In Situ condition of the materials in the Stockpile Project.

Additional metallurgical testing, in the form of column testing as outlined in this report is required to advance the level of confidence in leaching performance criteria. Column testing materials should be obtained from all lifts of the Stockpile Project as well as critical depths of the open pit and underground across a broader area to ensure an adequate understating of variability.

As with resource definition, the ability to obtain demonstrated representative samples from the stockpile facility is compromised. An inherent risk exists as to the representativeness of the samples tested to date or in future exists.

The particle size and fineness in the materials excavated are both a risk and opportunity. Finer particle sizes are likely to delivery improved copper recovery sooner. Ponding observed on the Stockpile Project from a recent weather event also indicates the potential for short circuiting other flow problems in leach in the pad from the finer material and compaction potential.

Underground deposits require critical metallurgical and hydrodynamic testing to verify heap leaching performance expectations for the materials defined. While the Parks/Salyer and Cactus underground deposits are near each other, initial indications from geologic logging and physical observations indicate potentially significant mineralogical and geologic differences that may result in differing metallurgical performances.

25.4 Mine Plan

The mine plan referred to in this report consists of a truck and shovel / loader operation operated by contractors. This mining method is a standard method used for this kind of material movement consistent with other operations in the United States.

25.5 Recovery Plan

Recovery estimates proposed for the Cactus Project are believed to be reasonable and appropriate for the current level of study. An average recovery of 74% of CuT given the high degree of oxide and CuAS contained is recommended at this time.

Oxide materials demonstrate a relatively rapid copper extraction potential, with copper extractions within two months achieved in column tests completed to date. A 3-month leach cycle has been considered for these materials. A one-year distribution of the recovery values used has been employed to account for heap inefficiencies, stacking planning and solution management activities. This will be refined with kinetic testing of the Stockpile Project and Cactus Project open pit materials.

Sulfide leaching completed to date indicates longer leaching cycles will be required. The materials will also be placed in a separate leach pad area that can be managed for bio-leaching kinetics and the longer cycle times required. A two-year distribution of the recovery values used has been employed to account for heap inefficiencies, stacking planning and solution management activities. This will be refined with kinetic testing of the Cactus Project open pit materials.

25.6 Infrastructure

Infrastructure required for the project is well understood, with much of the major components in place including power, water, and access. Risks associated with the current condition of the substation on site will be addressed in the next stage of study. This level of definition is advanced for this stage of study and provides higher confidence in this area.

25.7 Environmental, Permitting, and Social Considerations

Permitting is entirely within the State of Arizona and county level agencies and processes. Although less complex given Federal processes are not also required, uncertainty still exists related to timing of approvals and final requirements.

There are no known fatal flaws with respect to the site conditions. As a legacy clean-up site, the Cactus project will reduce the risks posed by the existing waste dump as it currently sits. Other legacy areas such as the tailings facility will be closed by the ASARCO Trust as part of the SIP agreement prior to purchase closing.

Permitting timing is considered aspirational but achievable.

Social license has been evaluated by Arizona Sonoran concurrently with technical study and strategies are continually assessed at this stage of investigation. The area is well exposed similar types of mining operations and have benefitted historically. There is no known organized opposition to the project and public announcements by Arizona Sonoran appear to be favorably received so far.

25.8 Markets and Contracts

The assumptions and provisions for contracted services and consumables is based on local understanding and existing or proposed agreements. The rates and pricing for power are confirmed to a higher level of certainty and based on defined terms.

H2SO4 is the most significant issue due to the sensitivity to the operation of the Hayden smelter for lower cost acid. Pricing has been assumed based on discussion with two similar operations and the unit costs is in line with current contracts. Alternative supply options and competitive alternatives should be investigated for the next phase of study.

25.9 Capital Cost Estimates

The methodology to develop the capital cost estimates are appropriate for this level of study and utilize budget pricing from vendors and recently estimated projects. As a further effort to confirm the accuracy ranges, the estimate information also compared to a more detailed recent project estimate for reasonableness on major components, materials, and costs.

Geotechnical understanding for the heap leach pad area proposed is still required to be completed to confirm ground conditions and soil under liner suitability.

The contingency of 15% represents a higher level of confidence in the mechanical equipment cost and sizing basis, contract mining strategy, significant existing infrastructure at the project site, and beneficial location of the project for contractor skills and workforce.

25.10 Operating Cost Estimates

The methodology used to develop the operating cost estimates are appropriate for this level of study.

Mining costs were built up using a combination of benchmarking and first principles with a contractor premium applied. Mining haul distances are reasonably certain given the existing nature of the Stockpile Project. Some uncertainty exists with the internal waste re-handling component of the Stockpile Project and proportion of waste left in place.

Processing costs were developed from a combination of direct build-up of costs based on metallurgical parameters for acid and water consumption. Electric power is factored from similar plants and published information.

Manpower estimates for the process facilities were developed from first principles and payrates benchmarked to similar projects. A highly skilled workforce exists in the area familiar with mining and copper hydrometallurgical process plant backgrounds. The location of the Cactus project should be favorable to attracting a high-quality staff.

25.11 Economic Analysis

The economics have been generated using a copper price of US$3.35/lb. These results show a robust project with a 33% after-tax IRR and a NPV (8%) of over $312 million at this copper price, but the project does appear to be quite sensitive to the copper price. At a current prevailing price of $4.00/lb these results increase significantly to 45% after-tax IRR and $510 million, respectively. Given the robustness of the Project at conservative copper price assumptions, we believe that the Project demonstrates a significant value proposition.

25.12 Risks and Opportunities

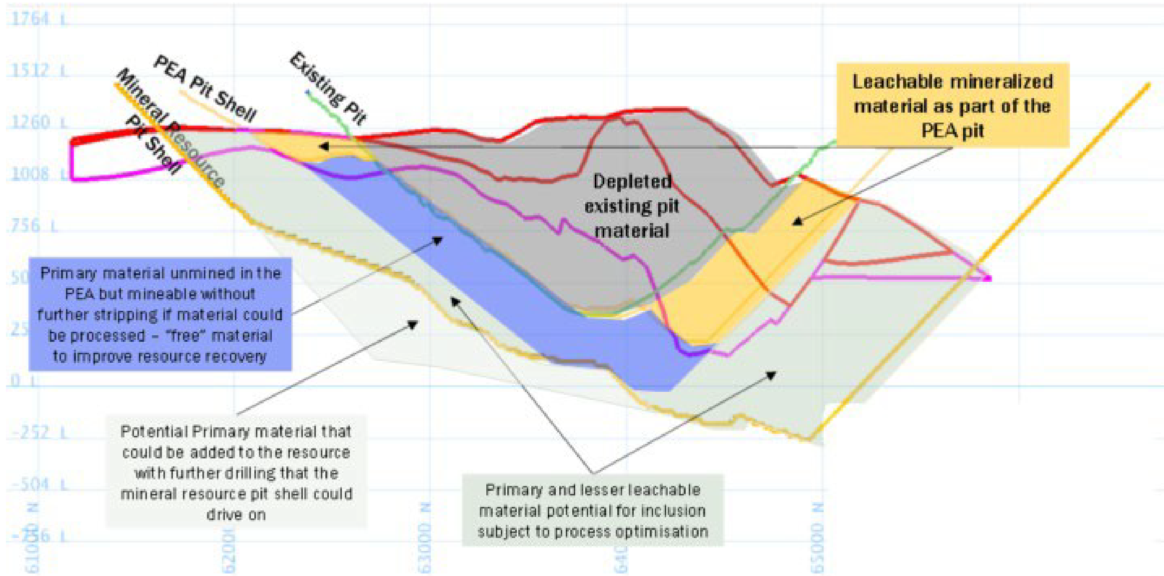
25.12.1 Opportunities

The following are key opportunities for the Project.

Resource Expansion

- In-Pit Potential: Based on the current level of exploration and planning, the Cactus West and East deposits comprise 2 billion pounds of leachable copper material. Only 1.28 billion pounds of leachable resource has been included within the PEA LOM, as the current pit mine plan has reached its natural limits for strip ratio due to increasing waste and decreasing grades on the periphery. Being able to process the primary material through sulfide leaching or flotation, which sits in the final pit floor, could add significant upside without additional waste stripping cost. This could result in depth expansion of the existing pit footprint, but also drive pit economics to support further pit expansions.
- Ex-Pit Potential: Based on the current level of exploration and planning, there is potential to add to the resource base through testing and conversion of material currently characterized as waste north of the Cactus West deposit.
- Figure 25-1 represents a cross-sectional view of the Cactus West pit. The green outline is the existing pit reflecting depletion. The PEA pit shell contains the leachable resource contemplated for that shell. The Mineral Resource pit shell captures all leachable and primary material as reflected in the Mineral Resource.
- Parks/Salyer: There may be further potential to expand the Parks/Salyer mineral resource declared herein through further drilling. The mine trend between Parks/Salyer and Cactus West may hold potential for a down-dropped fault block of porphyry mineralisation similar in nature to Cactus East.

Figure 25-1: Cross Section Looking North Reflecting Depleted Material and Current Resource



Process Optimization

- Further metallurgical testing should be done to refine acid consumption and copper recoveries by source.
- Further metallurgical testing of sulfide recoveries could also demonstrate alternate process facilities thereby resulting substantial expansion of production rates.
- Upside from production of copper sulfate:
- Improved metallurgical performance (kinetics, acid consumption and copper recovery) and an alternative processing to an intermediate copper sulfate product are also potential opportunities to be pursued.
- A preliminary investigation was conducted regarding alternative processing routes. This included a site visit and discussion with the management team at a nearby processing facility in Arizona. The potential for producing copper sulfate and shipping 180 miles round trip was considered. Preliminary results indicate the following.
- Producing copper sulfate and sending it to existing and unused Electro-Winning capacity at nearby facilities could provide savings in CAPEX of about $20 million to the Project by not building an EW circuit.
- A development schedule improvement of up to 3 months could be realized based on eliminating long lead items.
- Net Cactus site based OPEX savings, including shipping to nearby facilities, would be about $0.04/lb.
- Potential processing charges to recover external EW costs plus profit is assumed to be about $0.085/lb. No cost discussions have taken place.

- The likely net overall operating cost increase to Arizona Sonoran could be $0.05/lb ($1.32 million per year) against the $20 million in capital savings.
- Emerging technologies for improved leaching of sulfide copper ores are being developed, in particular a proprietary catalytic bio-heap leaching technology that may provide an alternative approach to improving the leach performance of primary sulfide content in the leach materials considered in this report and the primary sulfides presently not considered economically suitable for commercial heap leaching operation.
- The future potential for a copper concentrator for primary sulfide materials should also continue to be investigated.

Stockpile Project Sequencing

Significant opportunities exist to further enhance the Stockpile Project in the areas of mining sequencing and heap leach feed grade distribution, Table 25-1 shows a preliminary mineral resource estimate for grade by lift in the Stockpile Project. Due to the uncertainty in possible low/no grade pockets within the lifts until more infill drilling is completed, an economic case has not been established at this time.

Table 25-1: Preliminary Estimate for Grade by Lift

Inferred Resources	Cu Sol Cutoff	Tons (million tons)	Cu Grade (%)				Pounds Cu (million pounds)			
			CuAS	CuCN	CuSol	TCu	CuAS	CuCN	CuSol	TCu
Lift 4	0.095	0.5	0.246	0.063	0.309	0.346	2.6	0.7	3.3	3.7
Lift 3	0.095	34.1	0.132	0.026	0.158	0.184	90.4	17.7	108	125.6
Lift 2	0.095	28.8	0.108	0.027	0.135	0.158	62.2	15.4	77.6	90.8
Lift 1	0.095	14	0.098	0.026	0.123	0.150	27.3	7.2	34.5	42
Total*	**0.095**	**77.4**	**0.118**	**0.026**	**0.144**	**0.169**	**182.5**	**40.9**	**223.5**	**262.2**

* Figures may not add up due to rounding

Project Schedule

Assuming permitting can be achieved as indicated, the project schedule could be brought forward 6-8 months by reducing the equipment delivery timeframe noted and starting the leach pad construction immediately upon receipt of permits. Any early execution or equipment purchase would be at the risk of project delays.

High Copper Commodity Environment

The Project development timeline driven by private land permitting is shorter, relative to other copper projects, which could see the Project developed in a higher copper price environment. For the purposes of LOM modelling, resources included in the open pit mine plan reflect an optimization run at a conservative $2.27/lb copper price to present a robust initial mine plan, maximize grade inputs and consequently project value. There is significant room to expand the existing mineral inventory should US$+3.00/lb copper prices continue to prevail. There is potential room to expand the Integrated Cactus PEA inventory through improving strip ratios for certain areas adding approximately 10%-15% additional contained copper (resulting in +20 year mine life and increased production in the near term) and optimizing recovery methods for primary ore. Further trade-off studies in this context will also be pursued during the upcoming work programs.

25.12.2 Risks

Risks associated with the uncertainty of permitting processes and timing, resource definition confidence in WRD and the dispute over land ownership are the most significant risks identified. The following are the identified risks for this Project.

Stockpile Project Resource

- Unusual resource risks are associated with defining mineral content of waste rock facilities. Limited resource definition is available to be included in the estimates grade and tonnage made. Historic dump plans and information is not available for review and interpretation. Additional definition is required to ascertain a higher level of confidence in the resources included in this report. An average tons and grade approach have been used.
- As with resource definition, the ability to obtain truly representative samples from the Stockpile Project, or waste rock facility is somewhat compromised. An inherent risk exists as to representativeness of the samples tested to date or in future. Sequential assaying methodology provides a broader interpretation spatially with in the Stockpile Project related to recovery expectations.
- The potential for crushing larger materials may be required to achieve the recovery results projected and assessed against costs.
- Mitigation measures for the potential leach hydrodynamics may need to consider conveyor stacking as a means to avoid surficial compaction and associated leach solution flow distribution and effectiveness.

Existing Litigation

- Ramm had expressed interest in developing a pumped hydro renewable energy project at the site and had previously publicly announced that it would apply for a FERC license so that it could use FERC's eminent domain authority to acquire the property. The application was not contested and, consistent with its practice to issue preliminary permits to uncontested applications, by order of 19 July 2018, FERC granted the preliminary permit.

- The preliminary permit gives Ramm no rights in the site or rights to develop their project. The preliminary permit only initiates the longer permitting process. On 15 January 2020, Ramm began the formal licensing process by filing its NOI and Pre-Application Document (PAD), together with a Letter Requesting Use of TLP. The ASARCO Multi-State Environmental Trust, to which Arizona Sonoran is under contract to acquire the property from, Arizona Sonoran, and the Arizona Department of Environmental Quality all filed comments opposing Ramm's initiation of the licensing process. On 04 March 2020, FERC rejected Ramm's NOI and PAD as "patently deficient". FERC determined the pre-application document relied upon a single study conducted for the purpose of remediating a copper mine site, lacked agency or tribal consultation, and was therefore incomplete. FERC also cited the public comments received from ASCU that Ramm does not have rights to access the site to conduct the required studies.
- However, by 10 June 2020, ASCU was notified of a FERC application filed by REA for a preliminary permit for Project No. 15010-000 to study the feasibility of developing an approximately 200 MW closed-loop, pumped- storage hydro project near Casa Grande in Pinal County, Arizona. Note that REA is a direct affiliation of Ramm. As portrayed in the Application, approximately 50-100 acres of the Project's site (Casa Grande Hydro Site) would overlap with land ASCU purchased in July 2020 from the ASARCO Multi-State Environmental Custodial Trust (the Trust). On 08 August 2020, ASCU filed their response with FERC, again outlining plans to develop a copper mine on the Mine Site (Cactus Project), further reiterating that REA has no permission to access the property. The Casa Grande Hydro Site would encroach on the mine shaft of the Cactus Project materially impeding underground extraction activities. On 09 July 2021, Ramm requested a two-year extension of its preliminary permit. On 12 August 2021, FERC denied the request because Ramm filed the request after the deadline. FERC noted, however, that the rejection does not preclude Ramm from filing for an entirely new preliminary permit for the project. FERC typically only issues new preliminary permits to former permittees in extraordinary circumstances.

Permitting

Permitting for mining projects in the western US and Arizona 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. Although the Cactus Project is on private lands, these risks remain.

Geotechnical

Geotechnical risks associated with the Cactus Project, including the proposed heap leach pad locations, open pit and underground wall stability have not been fully assessed and will require extensive test work to confirm current work and assumptions.

Metallurgical Testing

- The testing as outlined in this report is required to advance the level of confidence in leaching performance criteria such as recoveries, acid consumption, leach flow rates and hydrodynamic flow both for Cactus Project mineralization as well as the Stockpile Project.
- As with resource definition, the ability to obtain truly representative samples from the waste rock facility is somewhat compromised. An inherent risk exists as to representativeness of the samples tested to date or in future. Sequential assaying methodology provides a broader interpretation spatially with in the Stockpile Project related to recovery expectations.
- Leach solution hydrodynamic performance risks in the heap leach pads due to excess fine materials, clays in intermixed alluvial materials, varying ore grades, complex mineralogies, and other factors are a risk to leaching metallurgical performance and heap stability. Testing and evaluation of these considerations should be conducted to confirm practical leaching parameters and reduce the potential risks.

Tax Rates

- The Project economics vary with the tax rate used in the evaluation. The all-in rate assumption of 24% is reasonable for this level of study, given that the depletion values have not been quantified. Should the full tax rate of 30.5% be applied to the project, the after-tax IRR reduces from 28% to 26%.

25.13 Conclusions

The resource estimates established for both the Stockpile Project and Cactus Project combined with associated metallurgical testing appear adequate for this PEA, with additional work warranted to continue to investigate the Project. The resource estimate for Parks/Salyer deposit further warrants additional drilling such that it can be included into an integrated PFS/technical study.

The primary goals of future work programs should be as follows.

- In-fill drill programs of the current resource volume to convert inferred material to indicated and measured resource categories.
- Continue to expand the current resource through additional, step-out drilling.
- Continue to explore the mineralized targets away from the deposit to evaluate the potential for additional deposits to add to the medium-term expansion potential.
- Conduct additional metallurgical testing as outlined in this PEA.
- Complete an integrated Cactus PFS of the project based on the positive outcome from this PEA and the Parks/Salyer Mineral Resource.

Based on the outcomes of the scoping level study, considering the above, and the absence of fatal or serious flaws, the project is worthy of continued development to a PFS level of confidence and consequently DFS level to advance the understanding of the technical risks associated with resource confidence, metallurgical performance, and project development costs.

26.0 RECOMMENDATIONS

As set out in the 2021 Cactus PEA, the QPs to this report recommend the completion of a PFS to advance the development of the Project. As set out in the 2021 Cactus PEA recommendations for further work study programs have been divided into two phases to better define the goals and objectives and assist in planning and budgeting the work. Phase 1 is the completed PFS and Phase 2 is advancing the project to a DFS. Phase 2 is dependent on positive results from Phase 1.

Table 26-1 captures all Phase 1 costs required to complete a PFS, whereas Table 26-2 reflects the additional Phase 2 costs do the DFS, including final detailed engineering and initial exploration drilling on Parks/Salyer and NE Extension. The budget has been estimated for project expenditures commencing in Q4 2021 for the next two phases of the work program. The results of the lab testing, particularly Metallurgical, will form the basis to proceed the study to a DFS. The results of additional drilling will be required prior to a scoping level evaluation of the economics for Parks/Salyer and are not included in the costs below.

Table 26-1: Phase 1, Prefeasibility Study Costs

Budget Category	Estimate Cost (US$ 000)	
	Q3, 2021	Q4, 2021
Drilling	2,782	1,232
Project Support	396	276
Technical Studies	750	750
Lab Testing (Assaying and Metallurgical)	493	198
Permitting	59	80
Land Payments	7,000	
Exploration – Adjacent Properties		
Total	**11,479**	**2,535**

Table 26-2: Phase 2, Definitive Feasibility Study Costs

Budget Category	Estimate Cost (US$ 000)
Drilling	3,128
Project Support	750
Technical Studies	652
FEED Engineering	800
Lab Testing (Assaying and Metallurgical)	398
Permitting	124
Land Payments	7,900
Exploration – Adjacent Properties	2,916
Total	**16,669**

The following specific tasks should be undertaken as part of Phase 1, PFS work program.

- Sustainability
  - Continue permitting activities and land acquisition as planned.
  - While adequate for this PEA, further hydrogeologic study is required to better quantify aquifer levels and impacts from mining.
- Geotechnical
  - Develop geotechnical information required for engineering design.
  - For example, the proposed pillar between open pit high wall and underground stopes is fairly represented in the PEA but needs geotechnical verification once additional data becomes available.
- Drilling
  - The present Cactus West and East deposit outlines appear to be drill limited to the north and east. Continued step out drilling in these areas could very well extend the limits of known mineralization.
  - Continue metallurgical sample drilling across the Project area.
  - Condemnation/step-out drilling to be completed to confirm the placement of dumps, leach pads and plant facilities.
  - If the decision is made to go underground at the Cactus East, plans should be made to have a close spaced definition drilling program to provide a more detailed understanding of mineralized material zone boundaries for stope design purposes.

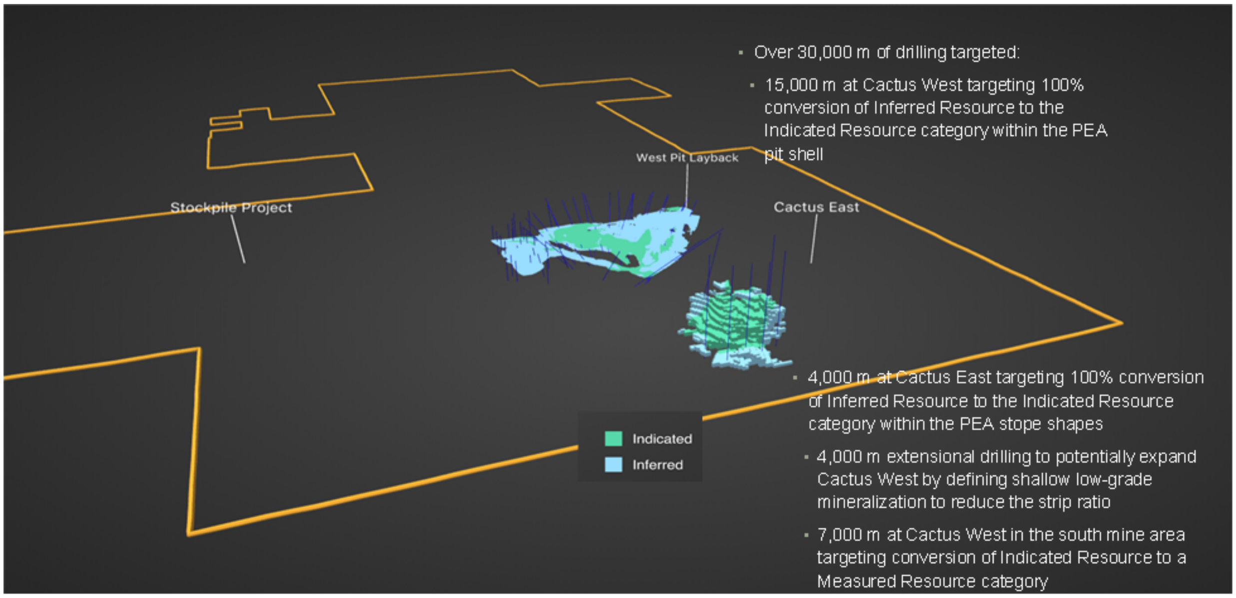
- Lab Testing
  - Significant additional column testing, particularly large columns, recovery by size fraction to determine merits of crushing / agglomeration and importance of isolating oxides and sulfides from open pit, leaching characteristics of mixed oxides and sulfides will be required.
  - Reduce the number of calculated soluble grades in the model through assaying of historical pulps (currently 30% of composites use calculated CuAS and CuCN grades based on CuT grades and mineralization domains).
- Mine Design

Regarding the Cactus East underground:

- While current plans do not expect Cactus East to be operated as an in situ leach operation, this proposed leaching method should be considered further with the existing core and resource information. In Situ leach may be an alternative to underground mining in a low copper price environment, thereby still realizing high value ore.
  - The proposed Transverse Longhole Stopping mining method is suited for the deposit and the primary/secondary sequence with access from sublevels at 75 ft (23 m) spacing is logical. An economic trade-off study that envisions Avoca style TLS should be commissioned. With the relatively wide dimensions of the mineralized deposit, additional opposite side access to set up Avoca mining (continuous mining and backfilling) may prove to add enough additional productivity gains to offset the additional development costs.
  - If the timing of the open pit layback schedule is not conducive to commence portal excavation in a timely manner, then access from the surface, which lengthens the development declines, should be considered.
- Costs and Schedule
  - The mining costs seem reasonable and sufficient for a PEA-level evaluation but will need a higher level of detail and productivity analysis in the next stage. This will include a total buildup of equipment, personnel, and materials.
  - A more detailed production and development schedule is required to verify the mines' ability to achieve the mining schedules presented for the Stockpile Project and Cactus Project.

A graphical representation of the drill plan is as provided in Figure 26-1.

Figure 26-1: Cactus Drill Plan



27.0 REFERENCES

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28.0 CERTIFICATES

CERTIFICATE OF AUTHOR

This Certificate of Author has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects, Part 8.1.

a) Name, Address, Occupation:
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b) Title and Effective Date of Technical Report:
This certificate applies to the Technical Report entitled "Mineral Resource Estimate and Technical Report – Arizona Sonoran Copper Company Inc. (Parks / Salyer)" for the Cactus Project with an effective date of 10 November 2022 (the "Technical Report").

***c)* Qualifications:**
I am a graduate of the University of Alberta, Edmonton, Alberta, Canada, in 2001 with a Bachelor of Science degree in Mining Engineering. I am registered as a Professional Engineer in the Province of Ontario (Reg. No. 1001106839) and in the State of Arizona (Reg. No. 68634). I have worked as a mining engineer for a total of 20 years since my graduation. I fulfill the requirements of a Qualified Person as defined in NI 43-101.

d) Site Inspection:
I have visited the property that is subject of this Technical Report on 03 March 2020 and 21 April 2022.

e) Responsibilities:
I am responsible for authoring Sections 2, 3, 4, 5, 6, 15, 18, 19, 20, 21, 22, 23 and 24 of the Technical Report along with those sections of the Summary pertaining thereto.

I am responsible for co-authoring Sections 1, 16, 25 and 26 of the Technical Report

f) Independence:
I am independent of Arizona Sonoran Copper Company applying the tests as set out in Section1.4 on National Instrument 43-101 and form 43-101F1.

g) Prior Involvement:
I have no prior involvement with the property that is the subject of this Report and I hold no interest in, nor do I expect to receive any interests, direct or indirect from Arizona Sonoran Copper Company or any associated or affiliated company.

h) Compliance with NI 43-101:
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.

i) Disclosure:
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.

Dated November 10, 2022.

Jason Sexauer P. Eng, .PE.
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This certificate applies to the Technical Report entitled "Mineral Resource Estimate and Technical Report – Arizona Sonoran Copper Company Inc. (Parks / Salyer)" for the Cactus Project with an effective date of 10 November 2022 (the "Technical Report").

***c)* Qualifications:**
I am a graduate of Michigan Technological University, Houghton, Michigan, United States of America, in 1997 with a Bachelor of Science degree in Mining Engineering and in 2006 with a Doctor of Philosophy in Mining Engineering. I am a graduate of the University of Michigan, Ann Arbor, Michigan, United States of America, in 1999 with a Master of Science in Engineering degree in Civil & Environmental Engineering. I am registered as a Professional Engineer in the State of Michigan (License No. 6201309292, exp. 03/01/2023). I have worked as a mining engineer for a total of 14 years since my graduation. I fulfill the requirements of a Qualified Person as defined in NI 43-101.

d) Site Inspection:
I have visited the property that is subject of this Technical Report on 03 March 2020 to review drill core and geotechnical drilling, then on 07 June 2022 to inspect upper pit walls.

***e)* Responsibilities:**
I am responsible for co-authoring Sections 16, 25 and 26 of the Technical Report along with those sections of the summary pertaining thereto.

f) Independence:
I am independent of Arizona Sonoran Copper Company applying the tests set out in Section 1.4 on NI 43-101 and form 43-101F1.

g) Prior Involvement:
I have no prior involvement with the property that is the subject of this Report and I hold no interest in, nor do I expect to receive any interests, direct or indirect from Arizona Sonoran Copper Company or any associated or any associated or affiliated company.

h) Compliance with NI 43-101:
I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument form.

i) Disclosure:
As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated November 10, 2022.

W.M.G.

Wilhelm Max-Otto Greuer Ph.D., P.E. Geotechnical Consultant
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c) Qualifications:

I am a graduate of the Lakehead University, Thunder Bay, Ontario, Canada, in 1979 with a Bachelor of Science degree in Geology. I am registered as a Certified Professional Geologist with the American Institute of Professional Geologists (No CPG-11758) and a Registered Member of the Society of Mining, Metallurgy and Exploration (SME # 04164071). I have worked as a mining geologist for a total of 42 years since my graduation. I fulfill the requirements of a Qualified Person as defined in NI 43-101.

d) Site Inspection:

I visited the Property that is the subject of this Technical Report on 13 August 2019, 03 October 2019, 03 March 2020, 25 August 2020, 20 January 2021, 21 April 2022, and 19 October 2022 to review ongoing drilling, core and chip logging, sampling, sample security and transport of assays to the assay lab. I have visited Skyline labs on 27 August 2019, 03 October 2019, 03 March 2020, and 02 April 2021 to review sample security, sample prep, assay methodologies, and QAQC work.

e) Responsibilities:

I am responsible for authoring Sections 7, 8, 9, 10, 11, 12, and 14 of the Technical Report along with those sections of the Summary pertaining thereto.

I am responsible for co-authoring Sections 1, 25, and 26 of the Technical Report along with those sections of the Summary pertaining thereto.

***f)* Independence:**

I am independent of Arizona Sonoran Copper Company applying the tests set out in Section 1.4 on NI 43-101 and form 43-101F1.

***g)* Prior Involvement:**

I have no prior involvement with the property that is the subject of this Report and I hold no interests in, nor do I expect to receive any interests, direct or indirect Arizona Sonoran Copper Company or any associated or affiliated company.

***h)* Compliance with NI 43-101:**

I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that Instrument and Form.

***i)* Disclosure:**

As of the 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.

Dated November 10, 2022.

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CERTIFICATE OF AUTHOR

This Certificate of Author has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Minerals Projects, Part 8.1.

a) Name, Address, Occupation:

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b) Title and Effective Date of Technical Report:

This certificate applies to the Technical Report entitled "Mineral Resource Estimate and Technical Report – Arizona Sonoran Copper Company Inc. (Parks / Salyer)" for the Cactus Project with an effective date of 10 November 2022 (the "Technical Report").

c) Qualifications:

I am a graduate of the I am a graduate of the Colorado School of Mines with the following degrees:

Metallurgical Engineer, 1963.
Master of Science, Metallurgical Engineering, 1967.
Doctor of Philosophy, Metallurgical Engineer, 1969.
I was a Member of the Mining and Metallurgical Society of America, Member #01216.
I am a Founding Registered Member of the Society for Mining, Metallurgy, and Exploration, Inc.Number-1802650RM, 2006.
I am a Registered Professional Engineer, Metallurgical Engineering, and State Board of Technical Registration, State of Arizona, USA, and Certificate Number 10560, 1976.
I have worked as a Metallurgical Engineer for 51 years.
I fulfill the requirements of a Qualified Person as defined in NI 43-101.

d) Site Inspection:

I have visited the Cactus project site on 25 February 2021 and 25 March 2021 to observe geological sampling, auger sampling, selection, transportation, and storage of samples in Reno Nevada for metallurgical testing. I have visited McClelland Laboratory in Reno Nevada to observe the metallurgical testing of Cactus samples, being tested under the guidance of Mr. James L. Sorensen.

e) Responsibilities:

I am responsible for authoring Sections 13 and 17 of the Technical Report along with those sections in the Summary pertaining thereto.

I am responsible for co-authoring Sections 25 and 26 of the Technical Report along with those sections of the Summary pertaining thereto.

f) Independence:

I am independent of Arizona Sonoran Copper Company applying the tests set out in Section 1.4 on NI 43.101 and form 43-101F1.

g) Prior Involvement:

I have no prior involvement with the property that is the subject of this Report and I hold no interests in, nor do I expect to receive any interests, direct or indirect from Arizona Sonoran Coper Company or any associated or affiliated company.

h) Compliance with NI 43-101:

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.

i) Disclosure:

As of the 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.

Dated 10 November 2022

Dr. Martin C. Kuhn PE, SME-RM

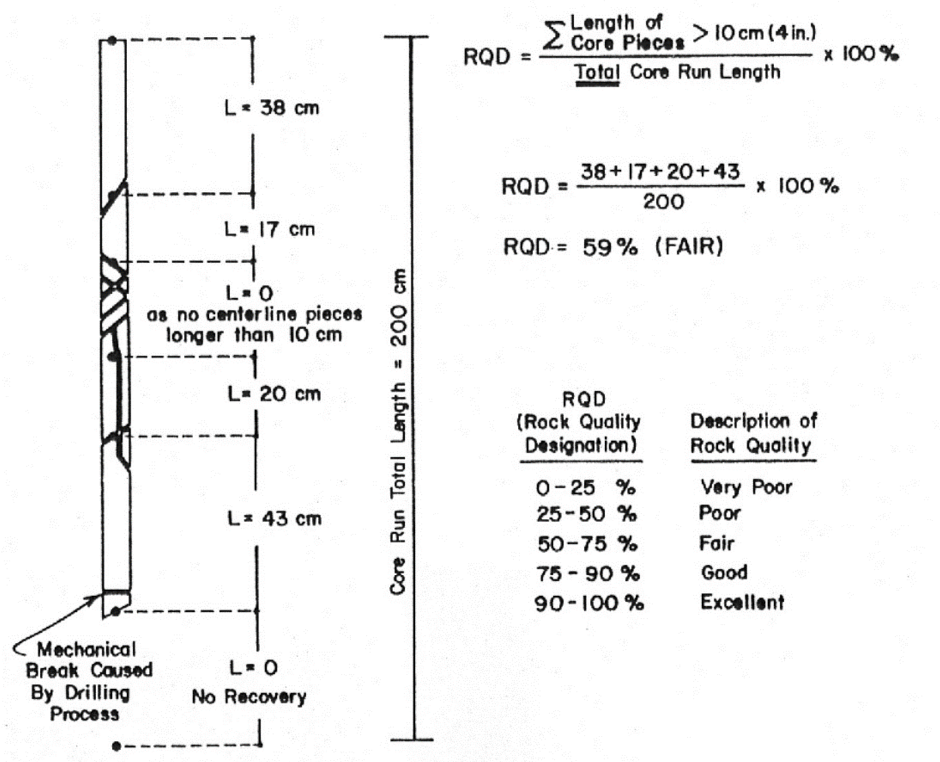
Registered Professional Engineer (Metallurgical)
CERTIFICATE NO.
10560
MARTIN C.
KUHN
Date Signed 11/10/2022
ARIZONA, U.S.A.

Expires 6/30/2024

Appendix A
Drill Data and Rock Properties

RQD is a basis of rock mass characterization and evaluating the stability of excavations in rock and is measured from core (Figure A1.1).

Figure A1.1: Rock Quality Designation (RQD, Deere & Deere, 1989)



Figures A1.2 and A1.3 show RQD measured from ECW-010. The RQD is mostly Fair to Excellent from the collar down to about 600 meter above sea level (masl) elevation, with a poor zone at approximately 950 masl which is where the drill hole would intercept the Western Fault. Poor and Very Poor RQD is encountered below 500 masl, where the drill hole is penetrating the West Ore Body.

Figures A1.4 and A1.5 show RQD measured from ECE-016. The RQD is mostly Fair to Excellent from 500 masl elevation to sea level, where the South Fault and contact between the host rock and East Ore Body are located. The RQD below sea level is Poor and Very Poor, where core was retrieved from the East Ore Body.

For pit wall stability analysis and differential stress impacts on the pit wall pillar thickness, engineering properties of the host lithologies were derived from RocScience RocData library of characteristic rock properties. Figure A1.6 shows a screen shot from RocData, displaying the host rock properties of the conglomerate and granite/monzonite porphyry formations.

Figure A1.2: ECW-010 RQD – Plan View

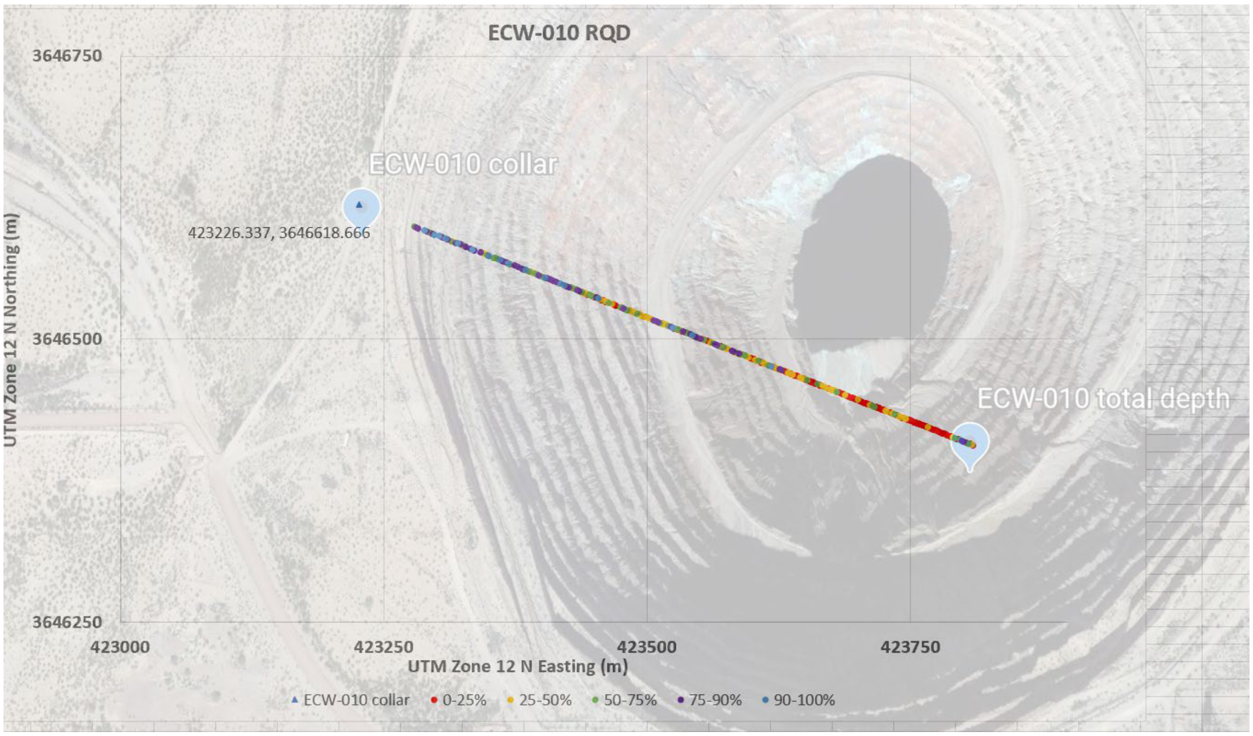


Figure A1.3: ECW-010 RQD – Profile View Facing North

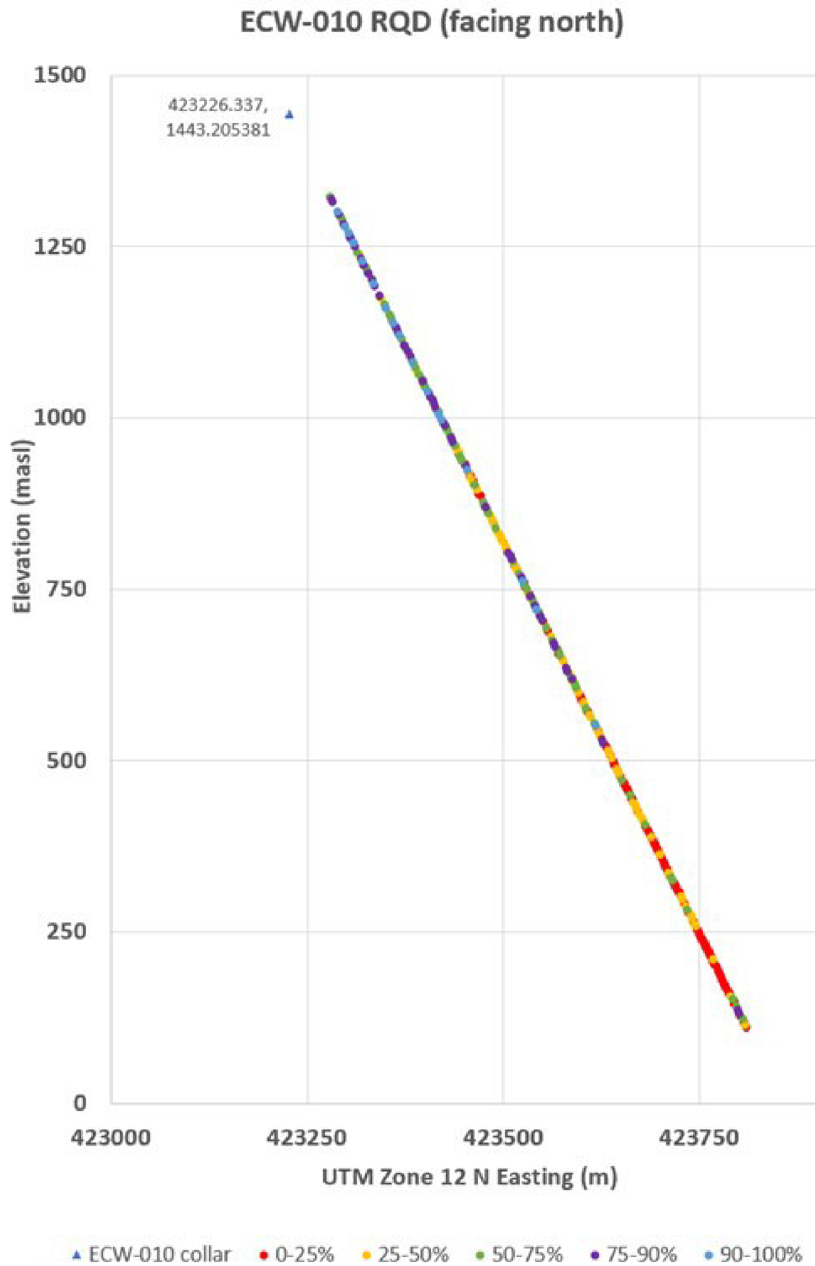


Figure A1.4: ECE-016 RQD – Plan View

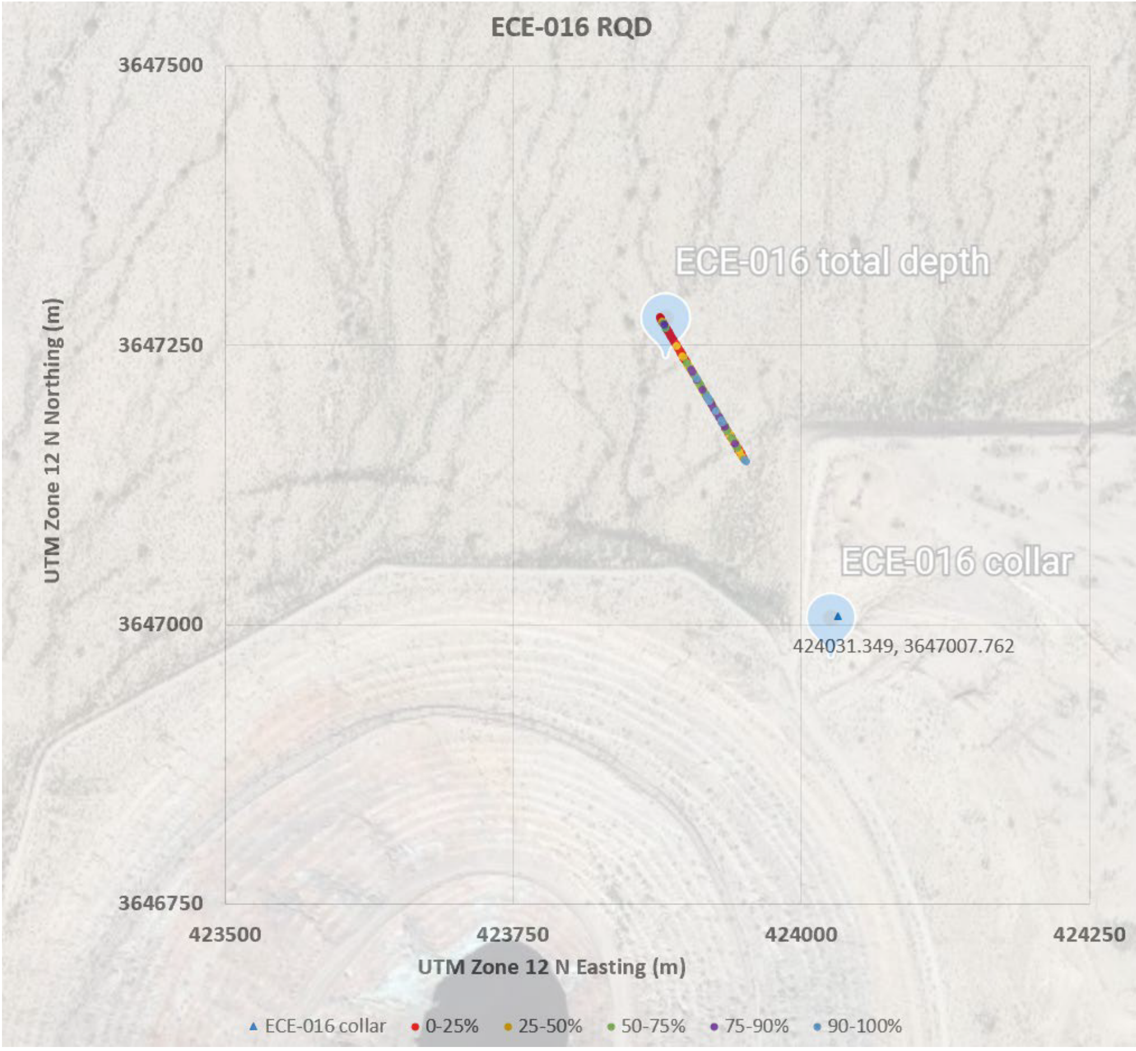


Figure A1.5: ECE-016 RQD – Profile View Facing North

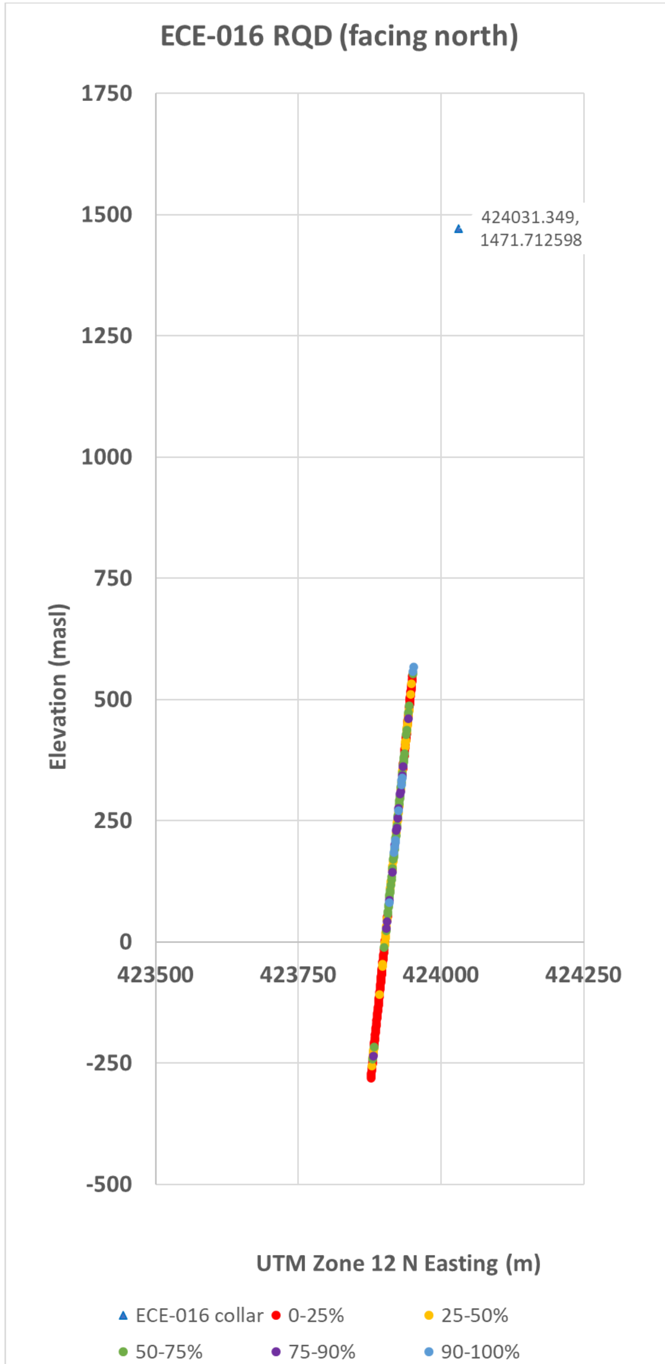


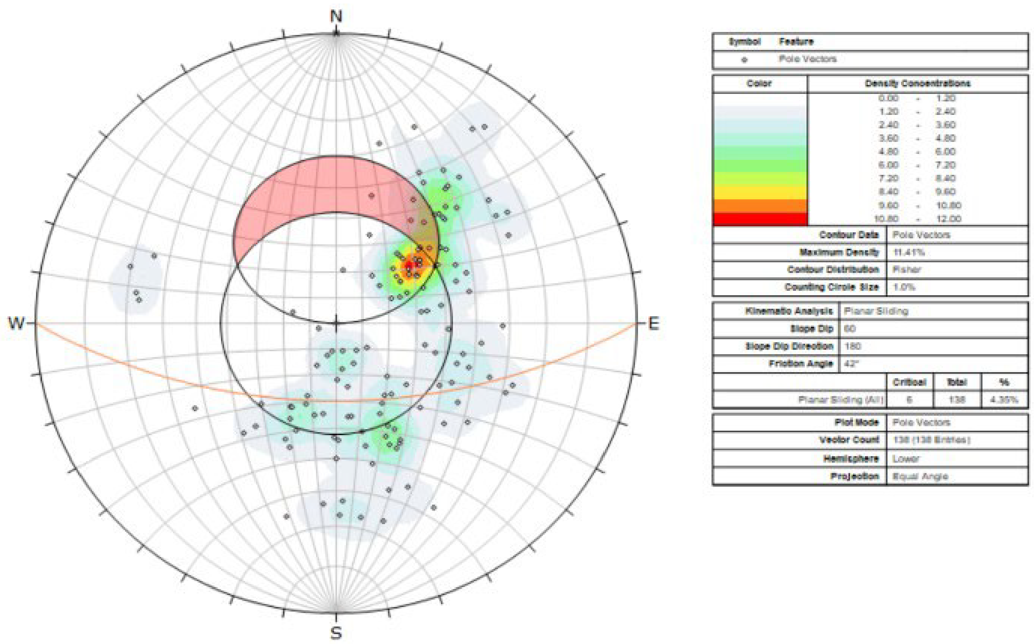
Table A1.6: Host Rock Engineering Properties

	Conglomerate	iranite/Monzonite Porphyr
Hoek Brown Classification		
intact uniaxial compressive strength	58 MPa	175 MPa
GSI	67	67
mi	21	32
disturbance factor	0	0
intact modulus	20300 MPa	74375 MPa
modulus ratio	350	425
Hoek Brown Criterion		
mb	6.462	9.847
s	0.026	0.026
a	0.502	0.502
Failure Envelope Range		
application	general	general
sig3max	14.5 MPa	43.75 MPa
Mohr Coulomb Fit		
cohesion	4.598 MPa	15.419 MPa
friction angle	42.003 deg	45.62 deg
Rock Mass Parameters		
tensile strength	-0.229 MPa	-0.454 MPa
uniaxial compressive strength	9.215 MPa	27.805 MPa
global strength	20.655 MPa	75.605 MPa
modulus of deformation	13680.799 MPa	50123.618 MPa

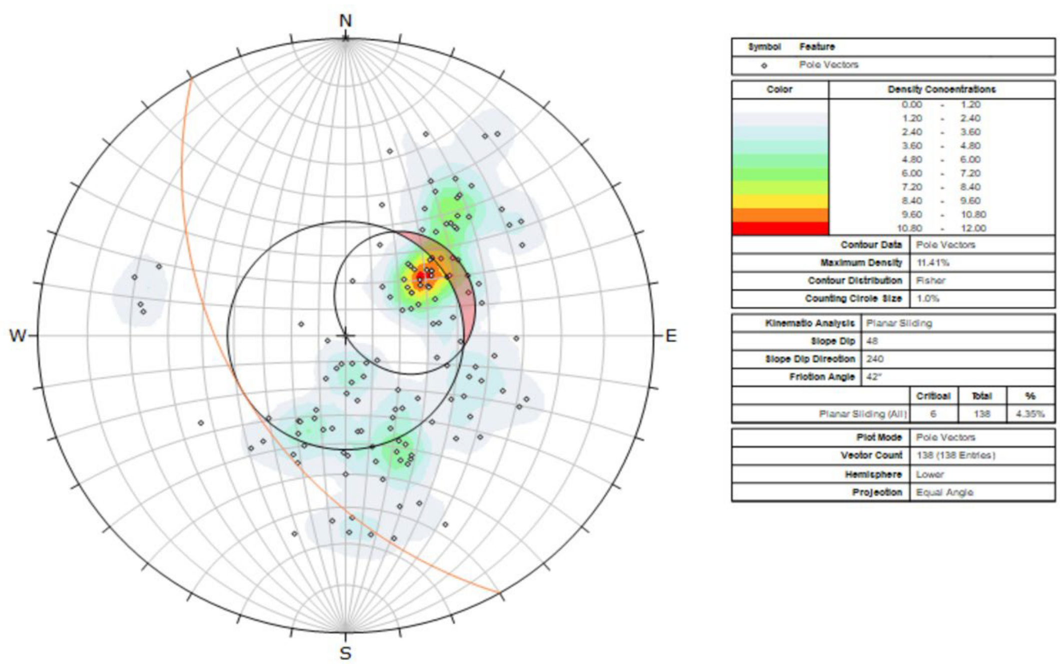
Appendix B
Kinematic Analysis

Planar Sliding

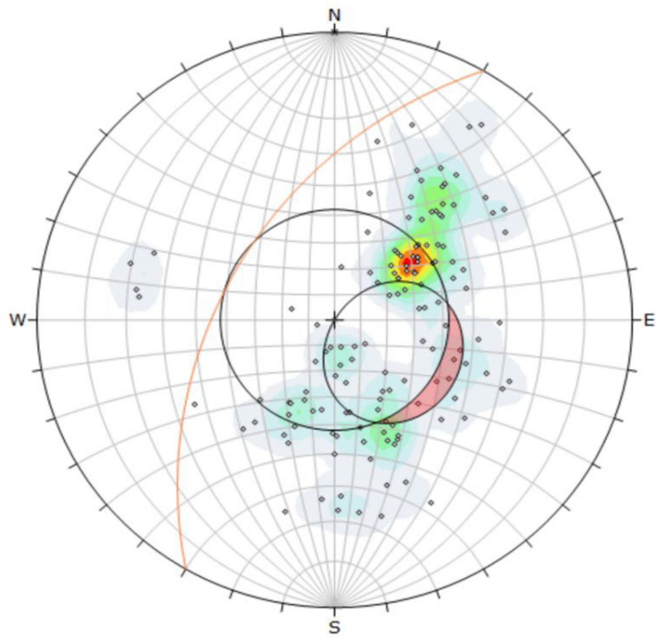
Section 1: 330°-30°, Dip Direction = 180°



Section 2: 30°-90°, Dip Direction = 240°



Section 3: 90°-150°, Dip Direction = 300°



Symbol	Feature
◇	Pole Vectors

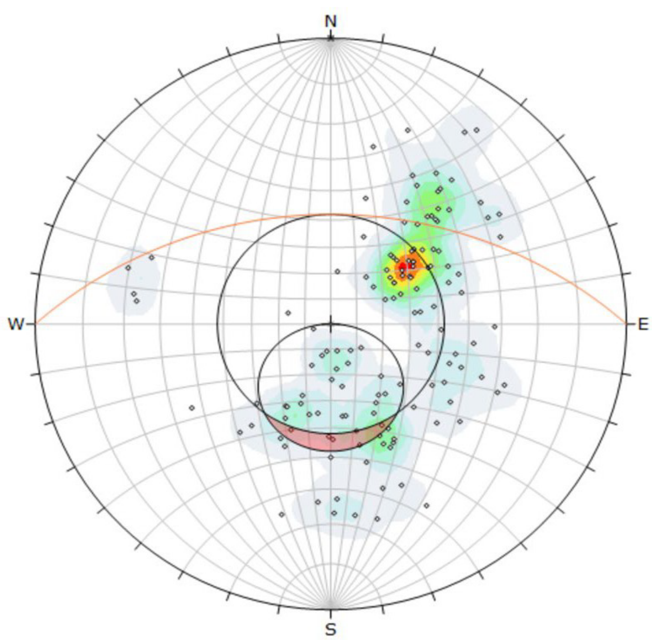
Color	Density Concentrations
	0.00 - 1.20
	1.20 - 2.40
	2.40 - 3.60
	3.60 - 4.80
	4.80 - 6.00
	6.00 - 7.20
	7.20 - 8.40
	8.40 - 9.60
	9.60 - 10.80
	10.80 - 12.00
Contour Data	Pole Vectors
Maximum Density	11.41%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Planar Sliding
Slope Dip	49
Slope Dip Direction	300
Friction Angle	42°

	Critical	Total	%
Planar Sliding (All)	6	138	4.35%

Plot Mode	Pole Vectors
Vector Count	138 (138 Entries)
Hemisphere	Lower
Projection	Equal Angle

Section 4: 150°-210°, Dip Direction = 0°



Symbol	Feature
◇	Pole Vectors

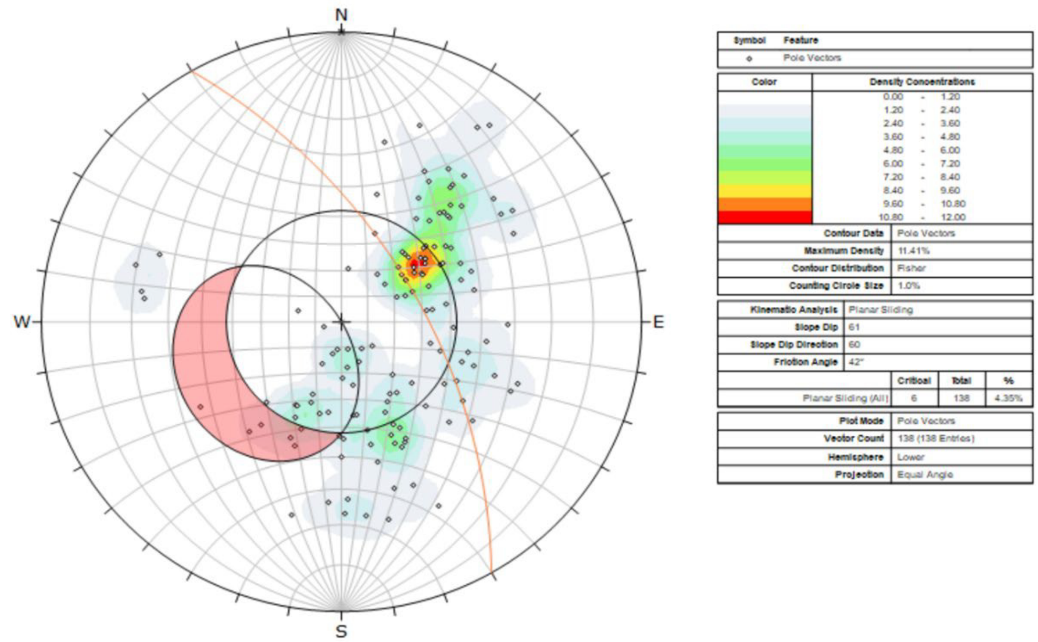
Color	Density Concentrations
	0.00 - 1.20
	1.20 - 2.40
	2.40 - 3.60
	3.60 - 4.80
	4.80 - 6.00
	6.00 - 7.20
	7.20 - 8.40
	8.40 - 9.60
	9.60 - 10.80
	10.80 - 12.00
Contour Data	Pole Vectors
Maximum Density	11.41%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Planar Sliding
Slope Dip	48
Slope Dip Direction	0
Friction Angle	42°

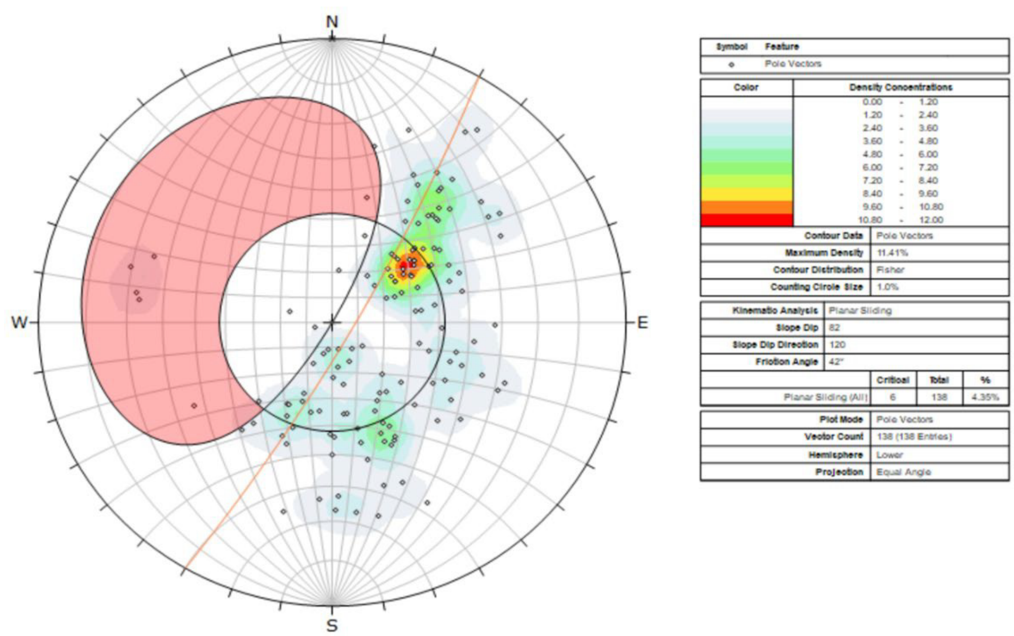
	Critical	Total	%
Planar Sliding (All)	6	138	4.35%

Plot Mode	Pole Vectors
Vector Count	138 (138 Entries)
Hemisphere	Lower
Projection	Equal Angle

Section 5: 210°-270°, Dip Direction = 60°

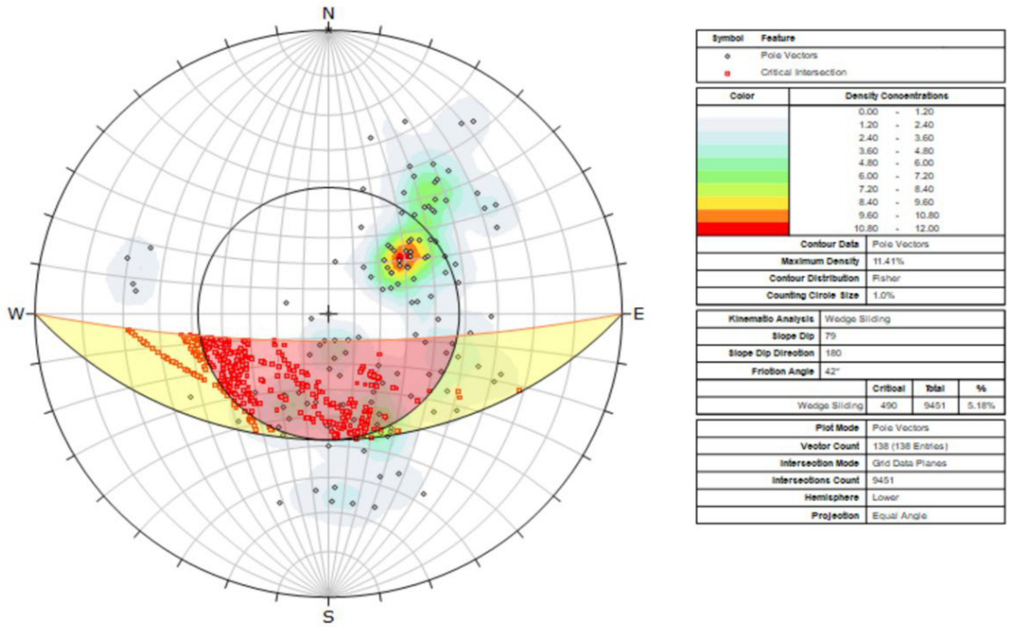


Section 6: 270°-330°, Dip Direction = 120°

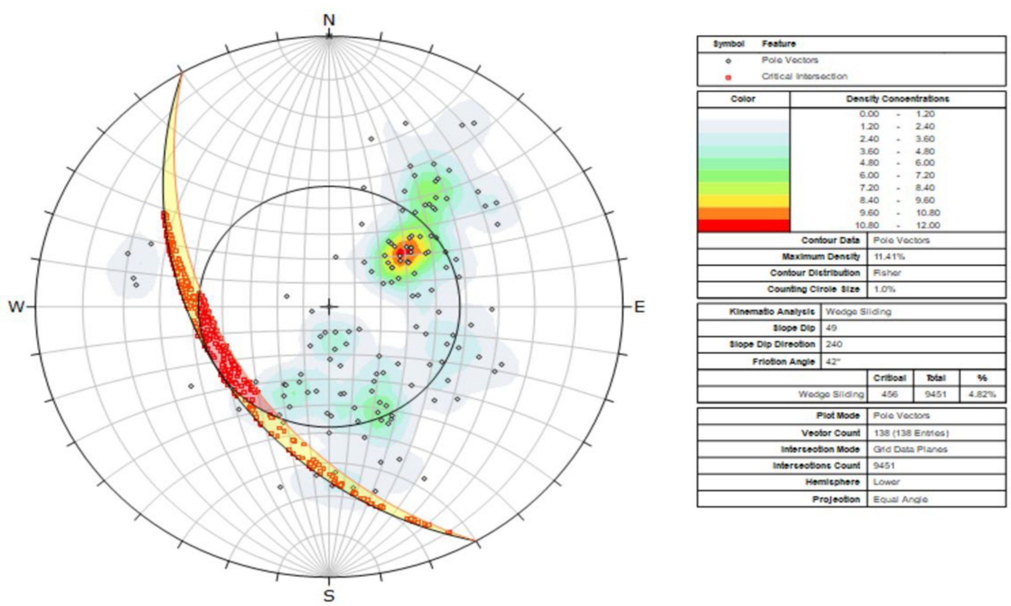


Sliding Wedge

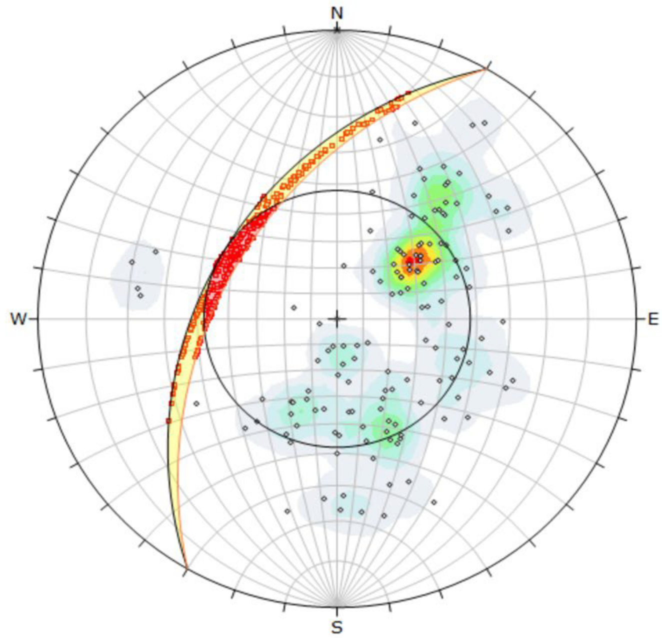
Section 1: 330°-30°, Dip Direction = 180°



Section 2: 30°-90°, Dip Direction = 240°



Section 3: 90°-150°, Dip Direction = 300°



Symbol	Feature
◇	Pole Vectors
■	Critical Intersection

Color	Density Concentrations
	0.00 - 1.20
	1.20 - 2.40
	2.40 - 3.60
	3.60 - 4.80
	4.80 - 6.00
	6.00 - 7.20
	7.20 - 8.40
	8.40 - 9.60
	9.60 - 10.80
	10.80 - 12.00

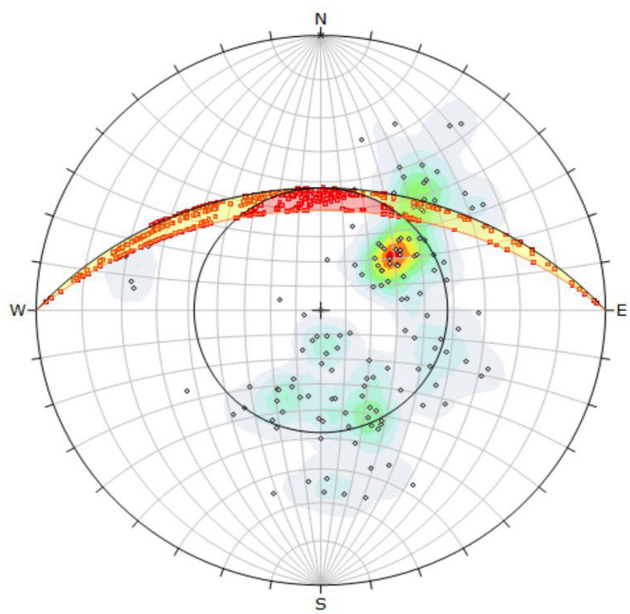
Contour Data	Pole Vectors
Maximum Density	11.41%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Wedge Sliding
Slope Dip	48
Slope Dip Direction	300
Friction Angle	42°

	Critical	Total	%
Wedge Sliding	426	9451	4.51%

Plot Mode	Pole Vectors
Vector Count	138 (138 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	9451
Hemisphere	Lower
Projection	Equal Angle

Section 4: 150°-210°, Dip Direction = 0°



Symbol	Feature
◇	Pole Vectors
■	Critical Intersection

Color	Density Concentrations
	0.00 - 1.20
	1.20 - 2.40
	2.40 - 3.60
	3.60 - 4.80
	4.80 - 6.00
	6.00 - 7.20
	7.20 - 8.40
	8.40 - 9.60
	9.60 - 10.80
	10.80 - 12.00

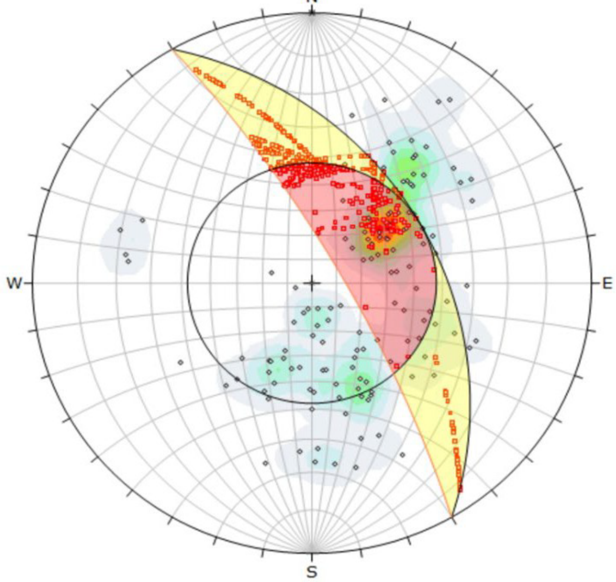
Contour Data	Pole Vectors
Maximum Density	11.41%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Wedge Sliding
Slope Dip	50
Slope Dip Direction	0
Friction Angle	42°

	Critical	Total	%
Wedge Sliding	448	9451	4.74%

Plot Mode	Pole Vectors
Vector Count	138 (138 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	9451
Hemisphere	Lower
Projection	Equal Angle

Section 5: 210°-270°, Dip Direction = 60°



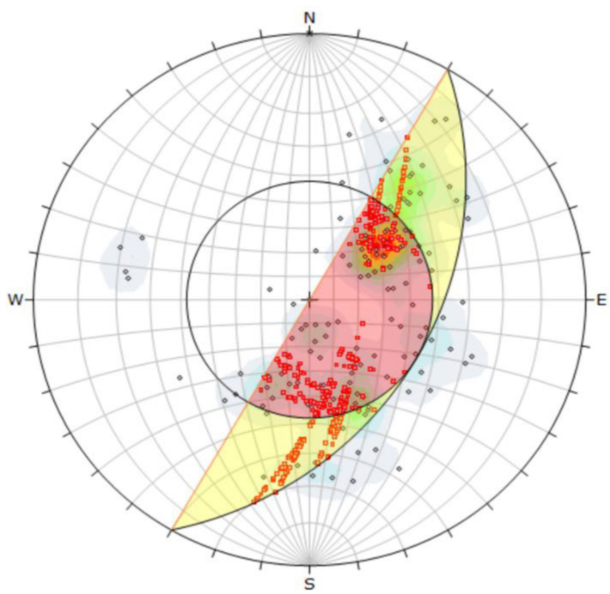
Symbol	Feature
◇	Pole Vectors
■	Critical Intersection

Color	Density Concentrations
	0.00 - 1.20
	1.20 - 2.40
	2.40 - 3.60
	3.60 - 4.80
	4.80 - 6.00
	6.00 - 7.20
	7.20 - 8.40
	8.40 - 9.60
	9.60 - 10.80
	10.80 - 12.00
Contour Data	Pole Vectors
Maximum Density	11.41%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Wedge Sliding		
Slope Dip	78		
Slope Dip Direction	60		
Friction Angle	42°		
	Critical	Total	%
Wedge Sliding	468	9451	4.95%

Plot Mode	Pole Vectors
Vector Count	138 (138 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	9451
Hemisphere	Lower
Projection	Equal Angle

Section 6: 270°-330°, Dip Direction = 120°



Symbol	Feature
◇	Pole Vectors
■	Critical Intersection

Color	Density Concentrations
	0.00 - 1.20
	1.20 - 2.40
	2.40 - 3.60
	3.60 - 4.80
	4.80 - 6.00
	6.00 - 7.20
	7.20 - 8.40
	8.40 - 9.60
	9.60 - 10.80
	10.80 - 12.00
Contour Data	Pole Vectors
Maximum Density	11.41%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Wedge Sliding		
Slope Dip	90		
Slope Dip Direction	120		
Friction Angle	42°		
	Critical	Total	%
Wedge Sliding	390	9451	4.13%

Plot Mode	Pole Vectors
Vector Count	138 (138 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	9451
Hemisphere	Lower
Projection	Equal Angle

Appendix C
Pit Wall Pillar Numerical Modeling

Figure A3.3 shows the magnitudes of the stress contours computed by the RS3 modeling software. Input rock properties are given in Table A1.6, where the pillar between the pit wall and the nearest stopes is part of the granite/monzonite formation. Three section profiles (a-a', b-b', & c-c') were taken to display stress contours in the rock between the pit wall and the nearest stopes, at the general locations of the nearest stopes (Figures A3.1 and A3.2).

Figure A3.1: Contour Profile Planes (Plan View)

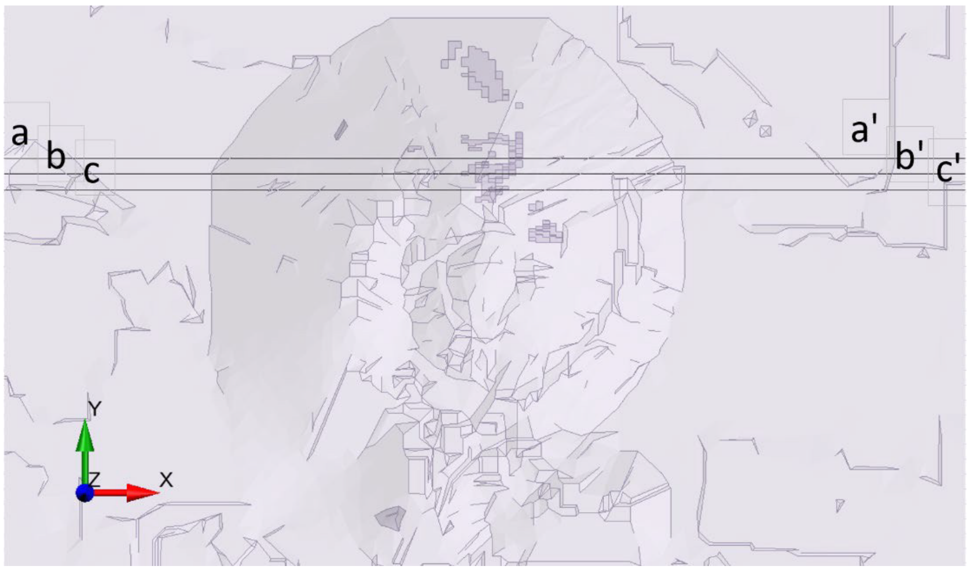
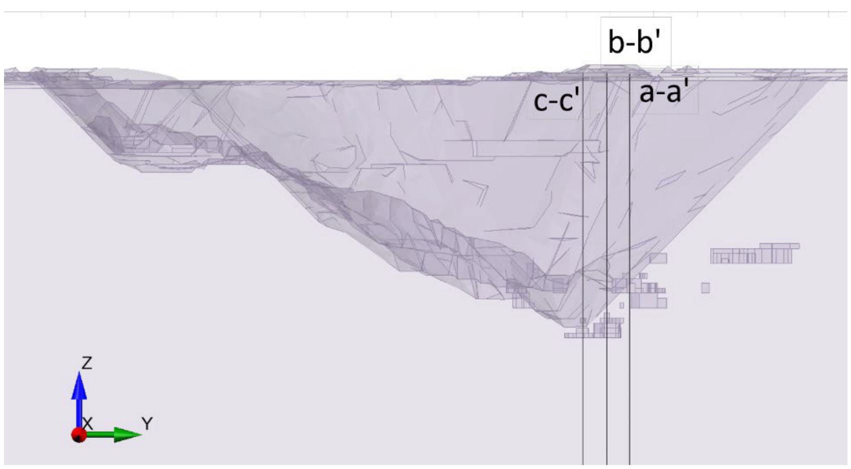


Figure A3.2: Contour Profile Planes (Section View)



The differential stress is the difference between the major principal stress σ1, acting on a stress element, and the minor principal stress σ3. For fracture propagation to occur in rock, the ratio of differential stress to intact unconfined compressive strength (UCSi) generally exceeds 0.3 (Bieniawski, 1967).

$$0.3 \geq (\sigma1 - \sigma3)/UCSi \qquad \text{Equation 16.1}$$

The model assumes no backfill, which provides confinement of fractured rock at the excavation boundaries. Figures A3.4 through A3.9 show σ1 and σ3 contours on section profiles a-a', b-b', & c-c'.

Figure A3.3: σ1 and σ3 Model Interpretation Stress Magnitudes

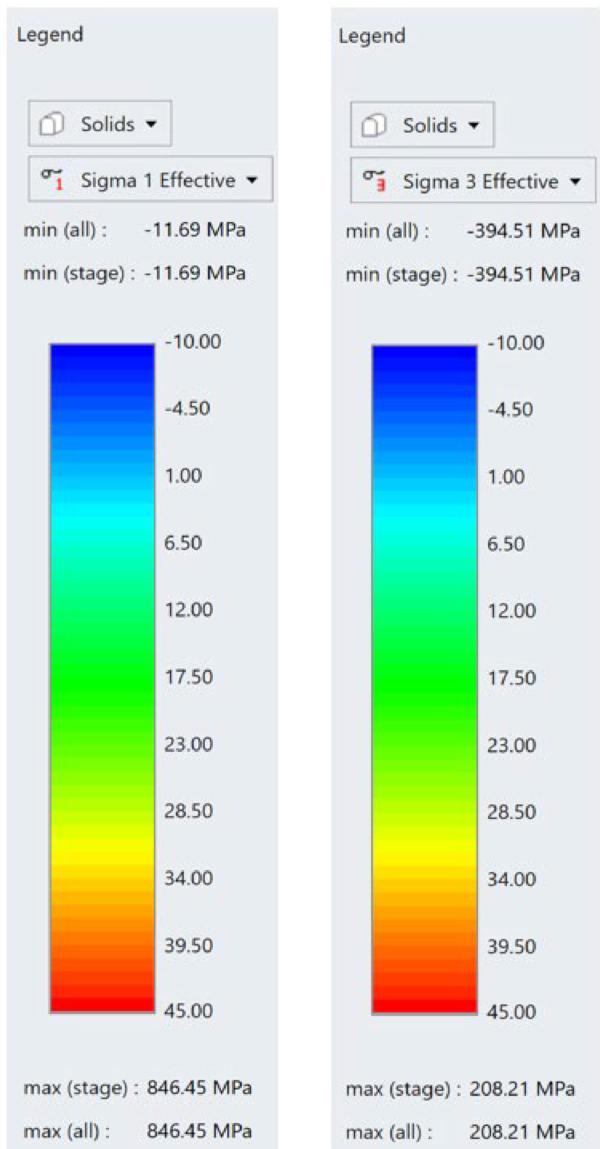


Figure A3.4: Section a-a' σ

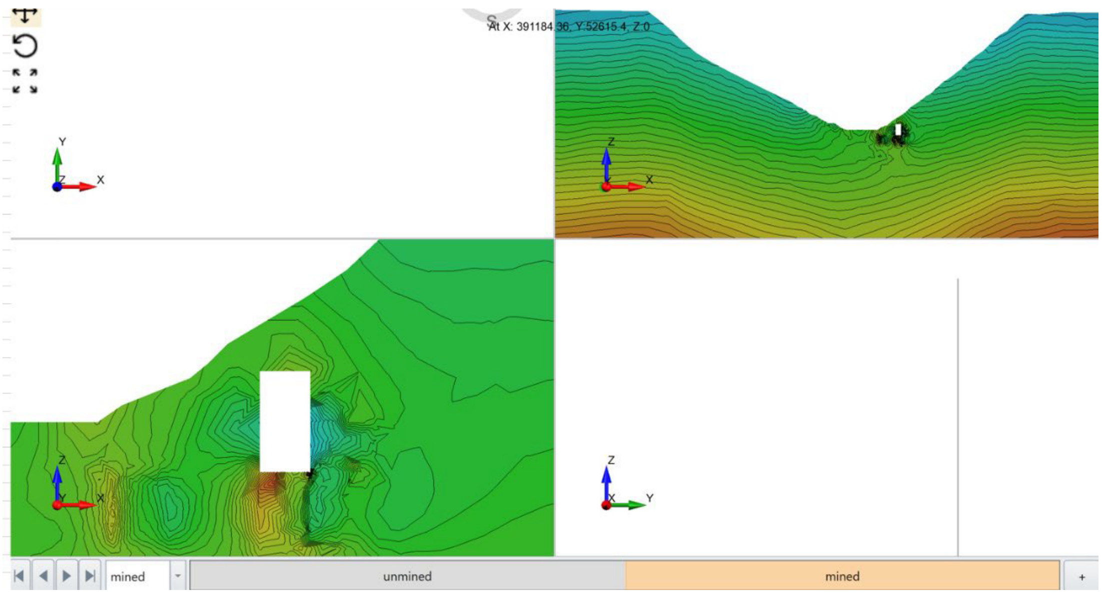


Figure A3.5: Section a-a' σ3

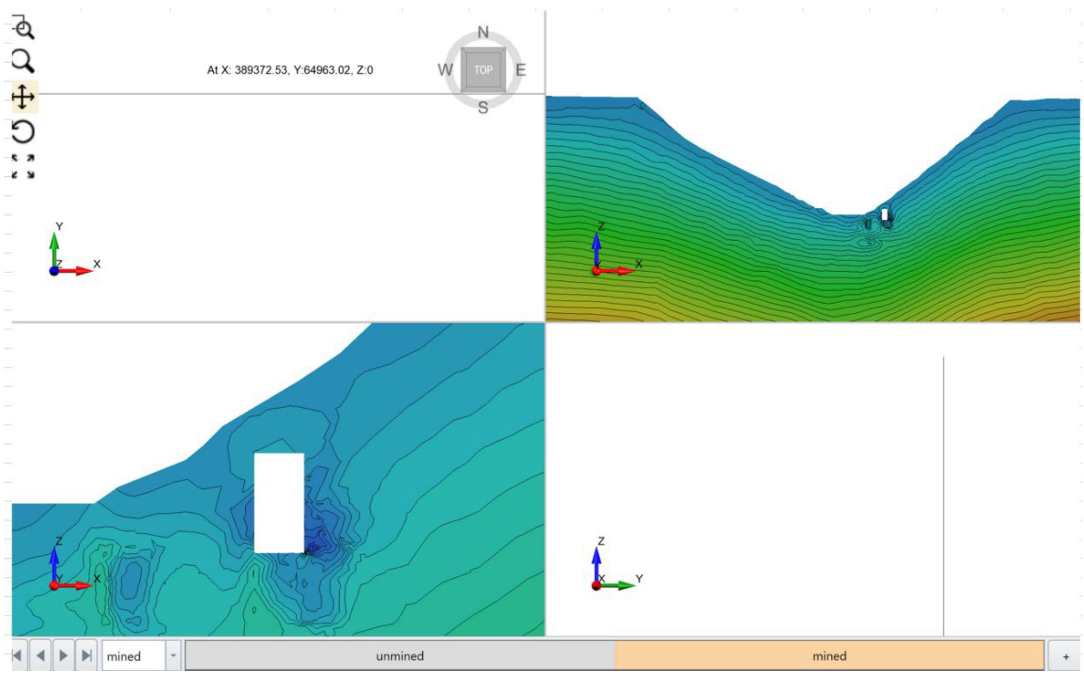


Figure A3.6: Section b-b' σ_1

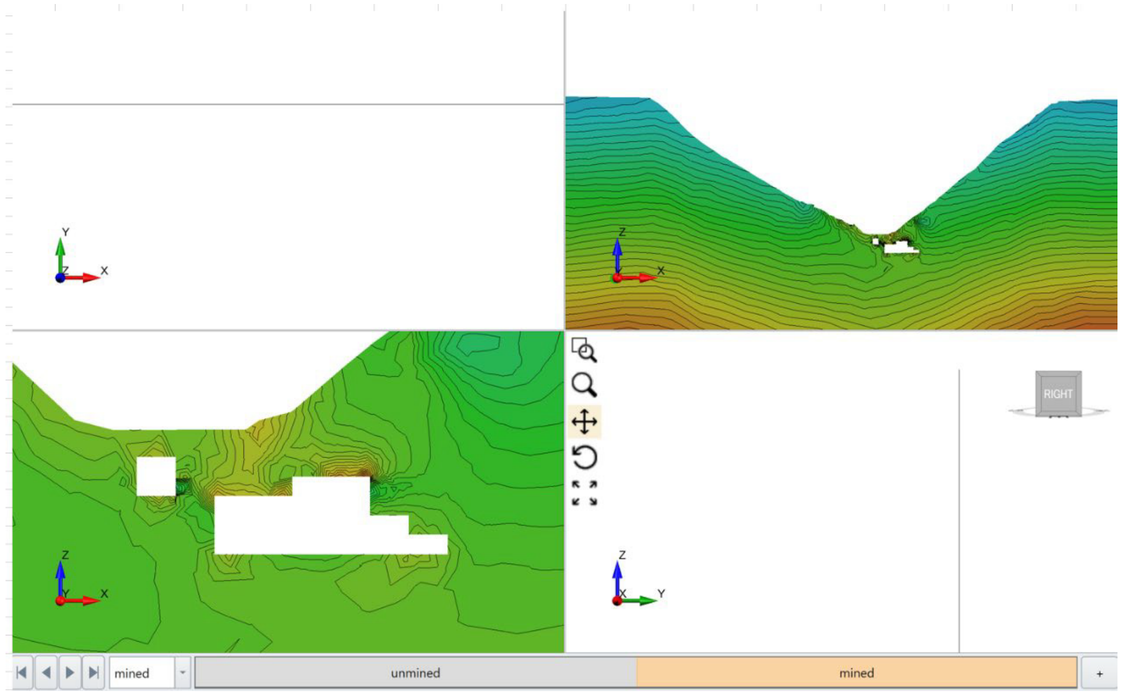


Figure A3.7: Section b-b' σ_3

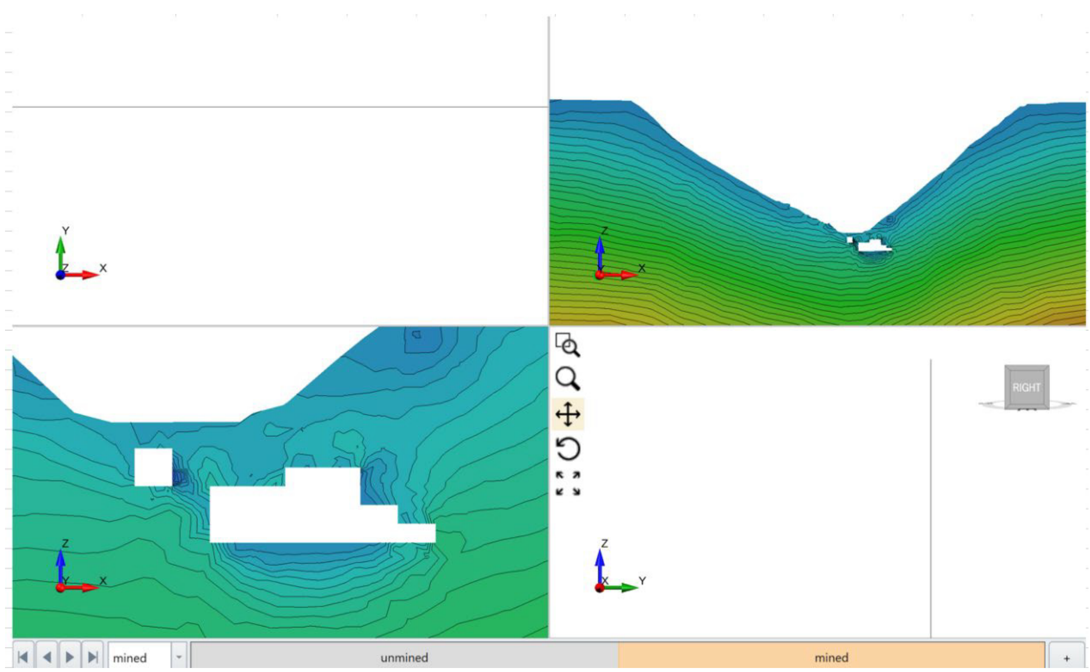


Figure A3.8: Section c-c' σ_1

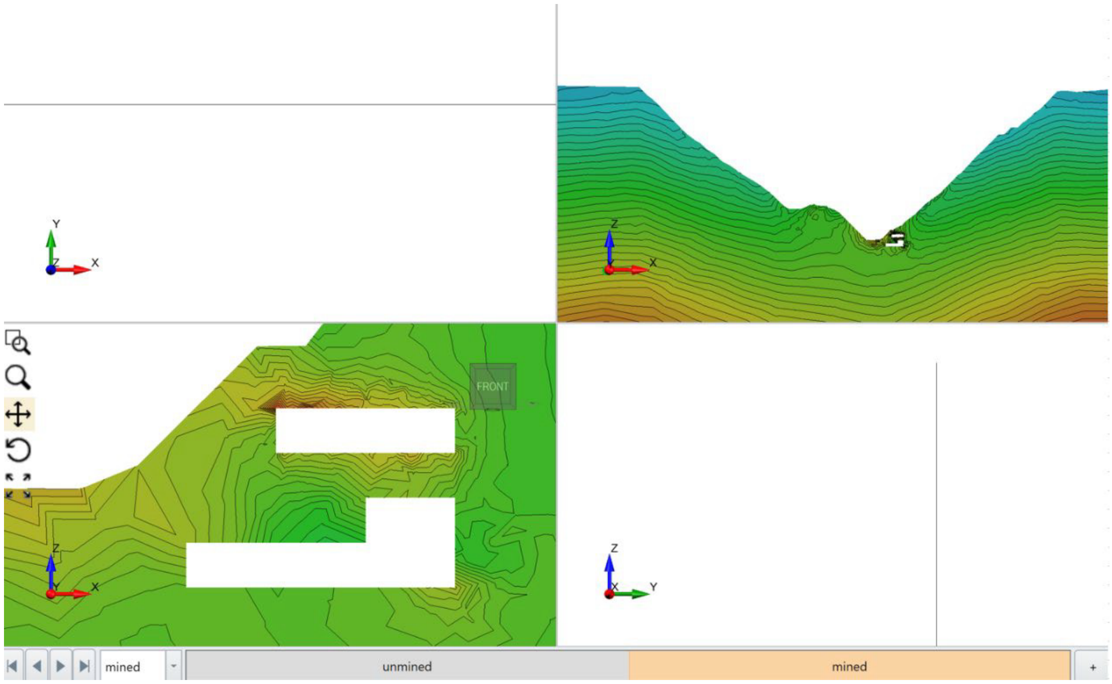
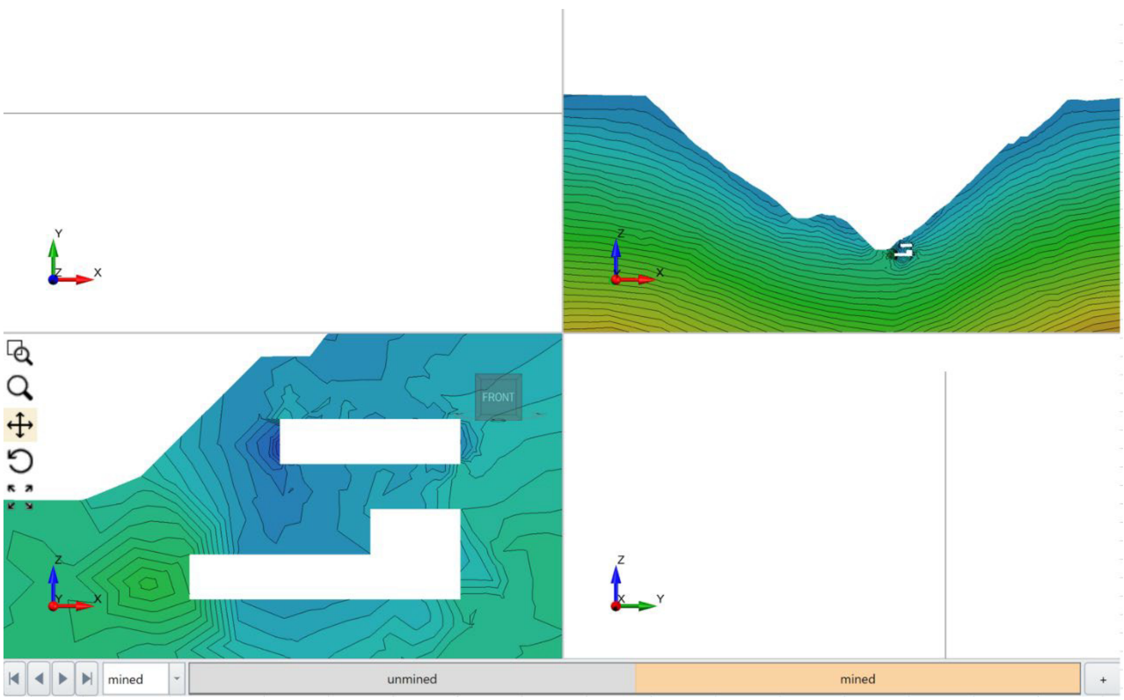


Figure A3.9: Section c-c' σ_3



Section a-a' shows a minimum pillar thickness of 35 m. The major principal stress σ1 in the pillar rock is approximately 20 MPa, and the minor principal stress σ3 is approximately -2 MPa (tension). The pillar rock is granite/monzonite, where UCSi is 175 MPa. The ratio of differential stress (σ1 – σ3) to UCSi is 0.125, well below the 0.3 threshold for fracture propagation.

Section b-b' shows a minimum pillar thickness of 50 m. The major principal stress σ1 in the pillar rock is approximately 25 MPa, and the minor principal stress σ3 is approximately MPa (tension). The pillar rock is granite/monzonite, where UCSi is 175 MPa. The ratio of differential stress (σ1 – σ3) to UCSi is 0.154, well below the 0.3 threshold for fracture propagation.

Section c-c' shows a minimum pillar thickness of 40 m. The major principal stress σ1 in the pillar rock is approximately 37 MPa, and the minor principal stress σ3 is approximately MPa (tension). The pillar rock is granite/monzonite, where UCSi is 175 MPa. The ratio of differential stress (σ1 – σ3) to UCSi is 0.23, well below the 0.3 threshold for fracture propagation. This section shows higher σ1 magnitudes in stopes nearest the pit wall due to a wider open span than in the stopes in sections a-a' and b-b' nearest the pit wall.

Appendix D
Mathews-Potvin Stability Graph Analysis

Stability is evaluated in terms of excavation dimensions versus ground conditions. The modified rock quality index (Q') is defined in the following equation.

$$Q' = (RQD / J_n) \times (J_r / J_a) \qquad \text{Equation A4.1}$$

Where Jn is the joint density and RQD/Jn is a measure of the block size. Joint surface roughness is Jr, and joint alteration is Ja. The ratio Jr/Ja is a measure of joint surface strength and stiffness. The modified index Q' is isolated from the impact of stress and hydraulic pressure from the normal Q-value and only accounts for the rock mass properties.

The classification of the rock mass and estimates of appropriate stope sizes are accomplished with the Modified Stability Graph Method through the Modified Stability Number N', which is defined as follows.

$$N' = Q' \times A \times B \times C \qquad \text{Equation A4.2}$$

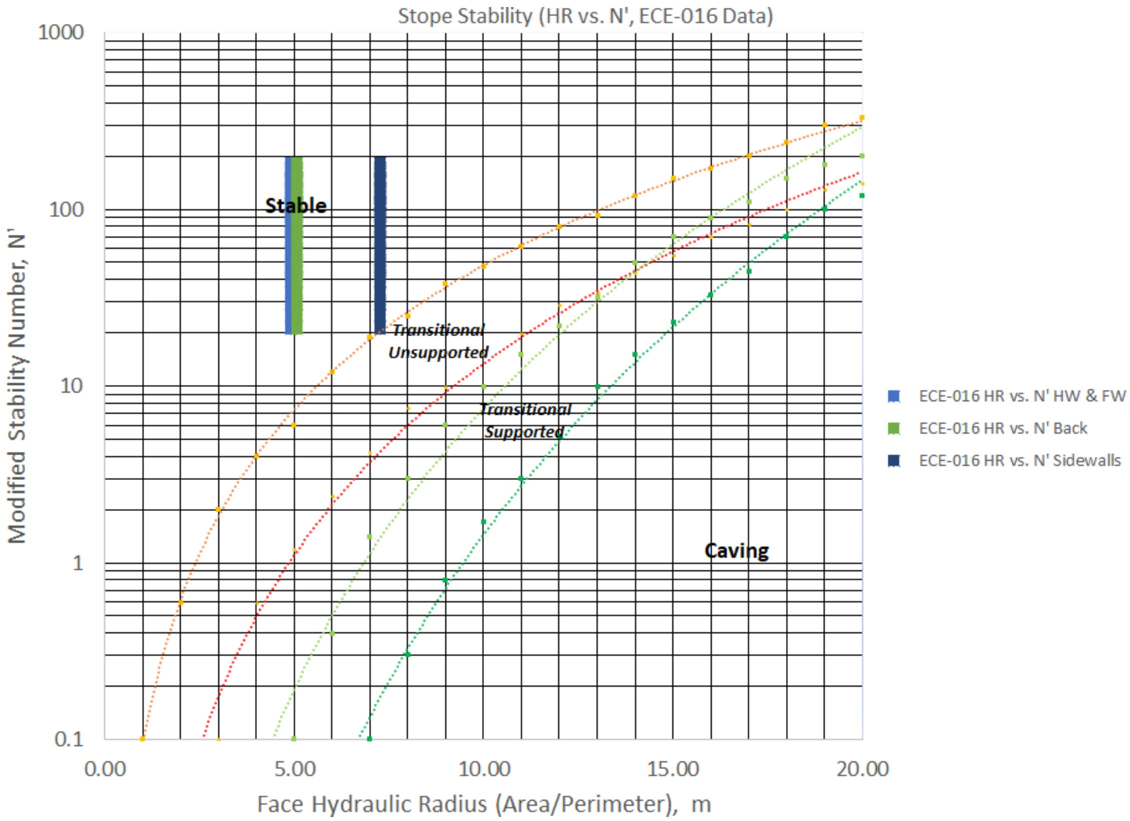
Where A is a measure of the stress impact on the rock mass, B is a measure of the orientation of discontinuities relative to the excavation surface, and C is a measure of the influence of gravity on stability.

The maximum stress (σmax) at pit bottom elevation is estimated to be 65 MPa, at which the σmax/UCSi ratio in the granite/monzonite formation has an A value of 0.38. For predominantly horizontal orientations of structure (Fig. 16.9), the B value is 0.3. For a 0°-30° critical joint dip and a 90° stope face dip, the C value is 8.

The hydraulic radius (HR) is a measure of exposure and the outcome of the Modified Stability Graph method. It is defined as dividing the area of the stope face by the perimeter of that face. A more open HR incorporating greater span or exposure will be more unstable than an equivalent area with narrower exposure.

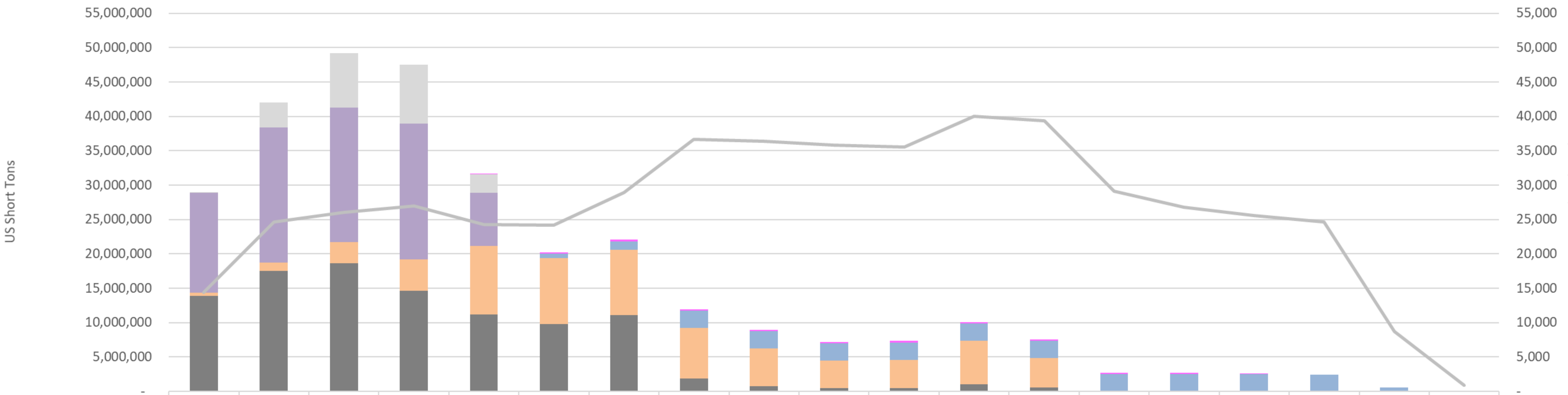
Stope design dimensions are given as 50 ft (15.2 m) width, 75 ft (22.8 m) height, and an initial length of 100 ft (30.4 m), to be adjusted as required. For stopes of these dimensions, HR for the hanging wall and footwall (HRHW&FW) is 4.9, HR for the back (HRback) is 5.1, and HR for the sidewalls (HRSW) is 7.3. Figure A4.1 shows the range of N' derived from ECE-016 data and where this range falls on the Mathews-Potvin Stability Graph for HRHW&FW, HRback, and HRSW. The N' range represents ground conditions in host rock, the mineralized deposit, and the mineralized/waste contact zone, with the lower values in mineralized deposit and the contacts. The range is in the stable unsupported zone, with the lowest N' value (presumably the ore/waste contact) at the unsupported transitional zone for the sidewalls, implying that 100 ft is the maximum stable unsupported stope length before an impending potential for progressive deterioration of unsupported sidewalls at the contacts, with some dilution to be expected. Other stopes may be longer, based on N'.

Figure A4.1: Mathews-Potvin Stability Graph for Stope Sizing



Appendix E
Life of Mine Schedule

Cactus Production Schedule



	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
UG Waste	-	-	-	-	129,400	194,200	194,200	194,200	194,200	194,200
UG Ore	-	-	-	-	-	630,005	1,266,31	2,520,04	2,519,98	2,519,96
External Stk - Waste	13,421	3,600,00	7,930,00	8,650,00	2,630,00	-	-	-	-	-
External Stk - Ore	14,560,0	19,730,0	19,530,0	19,690,0	7,731,94	-	-	-	-	-
Pit - Ore	442,000	1,189,00	3,113,00	4,540,00	9,982,00	9,662,00	9,455,00	7,353,00	5,464,00	3,987,00
Pit Waste	13,890,0	17,500,0	18,600,0	14,660,0	11,200,0	9,730,00	11,110,0	1,830,00	760,000	480,000
Recovered Cu Tons	14,385	24,660	26,061	26,954	24,287	24,162	28,900	36,625	36,332	35,813

	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19
UG Waste	194,200	194,200	194,200	194,200	194,200	85,700			
UG Ore	2,519,99	2,519,99	2,519,97	2,519,99	2,520,02	2,519,95	2,418,51	529,226	
External Stk - Waste	-	-	-	-	-	-	-	-	-
External Stk - Ore	-	-	-	-	-	-	-	-	-
Pit - Ore	4,094,00	6,305,00	4,276,00	-	-	-	-	-	-
Pit Waste	500,000	1,060,00	570,000	-	-	-	-	-	-
Recovered Cu Tons	35,545	40,043	39,383	29,124	26,803	25,551	24,599	8,666	851

CACTUS MINE LEACH TONS SCHEDULE

			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Stockpile																				
Waste		**t**	**13,421**	**3,600,000**	**7,930,000**	**8,650,000**	**2,630,000**	-	-	-	-	-	-	-	-	-	-	-	-	-
Ore		**t**	**14,560,000**	**19,730,000**	**19,530,000**	**19,690,000**	**7,731,940**	-	-	-	-	-	-	-	-	-	-	-	-	-
		Tsol	**0.159%**	**0.154%**	**0.134%**	**0.127%**	**0.120%**													
		Cu tons	**23,193**	**30,333**	**26,120**	**25,102**	**9,308**	-	-	-	-	-	-	-	-	-	-	-	-	-
Pit																				
Waste		t	13,890,000	17,500,000	18,600,000	14,660,000	11,200,000	9,730,000	11,110,000	1,830,000	760,000	480,000	500,000	1,060,000	570,000					
Prime Stockpile			-	-	-	-	100	74,300	40,200	126,000	116,500	209,300	516,700	237,400	504,700					
			-	-	-	-	0.425%	0.361%	0.306%	0.285%	0.313%	0.316%	0.302%	0.341%	0.310%					
			-	-	-	-	0	268	123	359	365	661	1,560	810	1,565					
Ore		t	442,000	1,189,000	3,113,000	4,540,000	9,982,000	9,662,000	9,455,000	7,353,000	5,464,000	3,987,000	4,094,000	6,305,000	4,276,000					
		Tsol	0.12%	0.21%	0.19%	0.19%	0.19%	0.24%	0.24%	0.26%	0.31%	0.45%	0.41%	0.26%	0.36%					
		Cu tons	530	2,500	6,000	8,800	18,800	22,900	22,800	18,800	17,100	17,800	16,800	16,600	15,200					
Oxide		t	400,000	730,000	2,090,000	2,150,000	8,550,000	6,300,000	6,720,000	6,450,000	4,270,000	1,590,000	1,080,000	4,590,000	1,810,000					
		Tsol	0.110%	0.096%	0.105%	0.126%	0.159%	0.186%	0.193%	0.208%	0.227%	0.214%	0.213%	0.200%	0.232%					
		Cu tons	440	700	2,200	2,700	13,600	11,700	13,000	13,400	9,700	3,400	2,300	9,200	4,200					
Enriched		t	42,000	459,000	1,023,000	2,390,000	1,432,000	3,362,000	2,735,000	903,000	1,194,000	2,397,000	3,014,000	1,715,000	2,466,000					
		Tsol	0.214%	0.392%	0.371%	0.255%	0.363%	0.333%	0.358%	0.598%	0.620%	0.601%	0.481%	0.431%	0.446%					
		Cu tons	90	1,800	3,800	6,100	5,200	11,200	9,800	5,400	7,400	14,400	14,500	7,400	11,000					
UG																				
Waste		t					129,400	194,200	194,200	194,200	194,200	194,200	194,200	194,200	194,200	194,200	194,200	85,700		
Ore		t					-	630,005	1,266,316	2,520,043	2,519,981	2,519,962	2,519,997	2,519,997	2,519,979	2,519,996	2,520,027	2,519,956	2,418,515	529,226
		Tsol						1.151%	1.160%	1.155%	1.131%	1.155%	1.175%	1.448%	1.393%	1.361%	1.421%	1.306%	1.303%	0.855%
		Cu tons					-	7,250	14,688	29,100	28,500	29,100	29,600	36,500	35,100	34,300	35,800	32,900	31,503	4,527
Oxide		t					-	418,005	862,842	1,745,643	54,781	13,762	51,497	704,197	261,679	256,396	255,227	483,956	1,208,515	
		Tsol						1.172%	1.178%	1.134%	1.095%	1.453%	1.165%	1.278%	1.185%	1.170%	1.254%	1.157%	1.167%	
		Cu tons					-	4,900	10,168	19,800	600	200	600	9,000	3,100	3,000	3,200	5,600	14,103	
Enriched		t					-	212,000	403,474	774,400	2,465,200	2,506,200	2,468,500	1,815,800	2,258,300	2,263,600	2,264,800	2,036,000	1,210,000	529,226
		Tsol						1.108%	1.120%	1.201%	1.132%	1.153%	1.175%	1.514%	1.417%	1.383%	1.439%	1.341%	1.438%	0.855%
		Cu tons					-	2,350	4,520	9,300	27,900	28,900	29,000	27,500	32,000	31,300	32,600	27,300	17,400	4,527